Goliath Gold Complex

NI 43-101 Technical Report and Prefeasibility Study

Kenora District, Ontario, Canada

Effective Date: February 22, 2023

Prepared for:
Treasury Metals Inc.
15 Toronto Street, Suite 401
Toronto, Ontario, Canada M5C 2E3

Prepared by:
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List of Qualified Persons:
Tommaso Roberto Raponi, P.Eng., Ausenco Engineering Canada Inc.
Dr. Gilles Arseneau, P.Geo., SRK Consulting Inc.
Sean Kautzman, P.Eng., SRK Consulting Inc.
Colleen MacDougall, P.Eng., SRK Consulting Inc.
David Ritchie, P.Eng., SLR Consulting Ltd.
Luis Vasquez, P.Eng., SLR Consulting Ltd.
Debbie Dyck, P.Eng., Minnow Environmental Inc.
Kathy Kalenchuk, P.Eng., RockEng Inc.
Kristen Gault, P.Geo., WSP E&I Canada Limited

CERTIFICATE OF QUALIFIED PERSON

Tommaso Roberto Raponi, P.Eng.

I, Tommaso Roberto Raponi, P. Eng., certify that I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc., (Ausenco), with an office address of Suite 1550 - 11 King St West, Toronto, ON M5H 4C7. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from the University of Toronto with a Bachelor of Applied Science degree in Geological Engineering with specialization in Mineral Processing in 1984. I am a Professional Engineer registered with the Professional Engineers Ontario (#90225970), Engineers and Geoscientists British Columbia (#23536) and NWT and Nunavut Association of Professional Engineers and Geoscientists (#L4508) and with Professional Engineers and Geoscientists Newfoundland and Labrador (#10968).

2. I have practiced my profession continuously for over 38 years with experience in the development, design, operation and commissioning of mineral processing plants, focusing on gold projects, both domestic and internationally. My project design and development experience include the generation of capital and operating costs for mineral processing plants and associated infrastructure and financial modeling of project economics.

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

4. I visited the Goliath Complex from July 7-8, 2021, for two days.

5. I am responsible for Sections 1.1, 1.2, 1.3, 1.10, 1.14, 1.15.1, 1.17 to 1.22.1, 1.22.2, 2, 3, 13, 17, 18.1 to 18.5, 19, 21.1, 21.2.1 to 21.2.4, 21.2.6 to 21.2.13, 21.3.1 (except 21.3.1.2), 21.3.2, 21.4.1, 21.4.2, 21.5, 21.6.1, 21.6.3, 21.7, 22, 23, 24, 25.1, 25.5, 25.8, 25.9, 25.11, 25.12.2, 25.13.2, 25.14, 25.15.1, 25.16, 26.1, 26.2, 26.3 and 27 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I acted as a qualified person for the technical report "N.I. 43-101 and Preliminary Economic Assessment of the Goliath Complex, Ontario, Canada" with an effective date of January 28, 2021.

8. 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: March 27, 2023

"Signed and sealed"

Tommaso Roberto Raponi, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Dr. Gilles Arseneau, P.Geo.

I, Dr. Gilles Arseneau, P.Geo., certify that I am employed as an associate consultant with SRK Consulting Inc (Canada), with an office address at 320 Granville St #2600, Vancouver, BC. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from the University of New Brunswick with a B.Sc. (Geology) in 1979, the University of Western Ontario with an M.Sc. (Geology) in 1984, and the Colorado School of Mines with a Ph.D. (Geology) in 1995. I am a Professional Geoscientist registered as a member, in good standing, with the Association of Professional Engineers & Geoscientists of British Columbia (License #23474).

2. I have practiced my profession for 35 years. I have worked on similar gold projects in North America. I have over 25 years experience in mineral resources estimation and 10 years working as an exploration geologist supervising exploration programs including field mapping and drilling programs in the Precambrian Shield of Canada

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

4. I visited the Goliath Complex between July 7-8, 2021, for two days.

5. I am responsible for Sections 1.4 to 1.9, 1.11, 4 to 12, 14, 25.2 to 25.4, 25.6, and 25.15.2 of the Technical Report.

6. I am independent of Treasury Metal Inc. and the Goliath Gold Complex as independence is defined in Section 1.5 of NI 43-101.

7. I have had no previous involvement with the Goliath Gold Complex.

8. 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: March 27, 2023

"Signed and sealed"

Dr. Gilles Arseneau, P.Geo.

CERTIFICATE OF QUALIFIED PERSON

Sean Kautzman, P.Eng.

I, Sean Kautzman, P.Eng., certify that I am employed as a Principal Consultant with SRK Consulting (Canada) ("SRK"), with an office address of 2A – 69 Young Street, Sudbury, Ontario, Canada, P3E 3G5. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from the University of Missouri – Rolla in Rolla, Missouri, USA in 2000 with a Bachelor of Science in Mining Engineering. I am a Professional Engineer of Professional Engineers Ontario (License #100159892).

2. I have practiced my profession for 22 years. I have been directly involved in underground mining engineering projects worldwide, including technical reviews, audits, and technical studies for precious metal, base metal, bulk commodities, and industrial mineral projects and operations.

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

4. I visited the Goliath, Goldlund and Miller project sites between July 7-8, 2021, for two days.

5. I am responsible for Sections 1.12 to 1.13, 15, 16.1, 16.5, 16.6, 21.2.5, 21.6.2, 25.7, 25.12.1, 25.13.1 and 25.15.3 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have had no previous involvement with Goliath Gold Complex.

8. 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: March 27, 2023

"Signed and sealed"

Sean Kautzman, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Colleen MacDougall, P.Eng.

I, Colleen MacDougall, P.Eng., certify that I am employed as a Principal Consultant with SRK Consulting (Canada) ("SRK"), with an office address of Suite 1500, 155 University Avenue, Toronto, Ontario, Canada. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from McGill University in Montreal, Quebec, Canada in 2006 with a B.Eng. in Mining Engineering. I am a Professional Engineer of Professional Engineers Ontario (License #100530936).
2. I have practiced my profession for 16 years. I have been directly involved in open pit mining engineering projects worldwide, including technical reviews, audits, and technical studies for precious metal, base metal, bulk commodities, and industrial mineral projects and operations.
3. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
4. I visited the Goliath, Goldlund and Miller project sites between July 7-8, 2021, for two days.
5. I am responsible for Sections 1.12 to 1.13, 15, 16.1, 16.4, 16.6, 21.2.5, 21.6.2, 25.7, 25.12.1, 25.13.1, and 25.15.3 of the Technical Report.
6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.
7. I have had no previous involvement with Goliath Gold Complex.
8. 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: March 27, 2023

"Signed and sealed"

Colleen MacDougall, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

David Ritchie, P.Eng.

I, David Ritchie, P.Eng., certify that I am employed as a Managing Principal with SLR Consulting (Canada) Ltd. with an office located at 55 University Avenue, Suite 501, Toronto, ON, M5J 2H7. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from Ryerson Polytechnic University, Toronto, Canada with a B.Eng. in Civil Engineering in 1995, and from University of Western Ontario, London, Canada with a M.Eng. in Geotechnical Engineering in 2000. I am a Professional Engineer registered with Professional Engineers Ontario (Licence #90488198).

2. I have practiced my profession for 27 years. I have been directly involved continuously with tailings site selection and tailings management alternatives studies, tailings deposition planning and water management, geotechnical investigations and dam design, and closure and rehabilitation.

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

4. I visited the Goliath project site on July 7 and 8, 2021 for a visit duration of two days.

5. I am responsible for Sections 1.15.2, 1.22.4, 18.6, 18.7 (except 18.7.5), 21.3.1.2, and 26.4 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have had no previous involvement with the Goliath Gold Complex.

8. 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: March 27, 2023

"Signed and sealed"

David Ritchie, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Luis Vasquez, P.Eng.

I, Luis Vasquez, M.Sc., P.Eng., certify that I am employed as a Principal Hydrotechnical Engineer with SLR Consulting (Canada) Limited, with an office address of Suite 501, 55 University Ave., Toronto, ON M5J 2H7. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from Universidad de Los Andes, Bogotá, Colombia, in 1998 with a B.Sc. degree in Civil Engineering and a Master of Science – Water Resources Engineering in 1999. I am a registered member of the Professional Engineers Ontario (Registration No. 100210789).

2. I have practiced my profession for 24 years since my graduation. I have been directly involved in:

- Development of water management plans and engineering design of water management infrastructure for mining projects,
- Engineering design an operational support related to water management for various mine sites,
- Authoring sections of several NI 43-101 Technical Reports.

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

4. I visited the Goliath Complex Site between July 7 and 8, 2021 for a visit duration of two days.

5. I am responsible for Sections 1.15.3, 1.22.3, 18.8.1, 18.8.2, and 26.3 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have had no previous involvement with the Goliath Gold Complex Project.

8. 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: March 27, 2023

"Signed and sealed"

Luis Vasquez, M.Sc., P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Debbie Dyck, P.Eng.

I, Debbie Dyck, P.Eng., certify that I am employed as an Environmental Engineer, Principal Environmental Approvals Specialist, with Minnow Environmental Inc., a Trinity Consultants Company ("Minnow"), with an office address of 2 Lamb Street, Georgetown, Ontario, Canada, L7G 3M9. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from the University of Waterloo, Ontario, in 1990 with a B.A.Sc. in chemical engineering with an option in management science. I am a registered member in good standing of the association of Professional Engineers of Ontario (#90353145) since 1993.

2. I have practiced my profession for 32 years as an environmental consultant, focussing on federal and provincial regulatory and permitting aspects specific to the mining industry. I was directly involved in the subject property as a technical advisor for baseline water quality and preliminary site water balance aspects during the latter period of the federal environmental assessment process, in 2018, and have been directly involved since early 2022 with respect to environmental and provincial permitting aspects required for future site development.

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

4. I visited the Goliath and Goldlund project sites between October 25 to 26, 2022 for a visit of two days; I have not visited the Miller project site.

5. I am responsible for Sections 1.16, 20 (except 20.2.1.6, and 20.3.1.6), 20.4.1 and 25.10 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have had no previous involvement with the Goliath Gold Complex.

8. 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: March 27, 2023

"Signed and sealed"

Debbie Dyck, P.Eng.

CERTIFICATE OF QUALIFIED PERSON

Kathy Kalenchuk, P.Eng., P.E.

I, Kathy Kalenchuk, P.Eng., P.E., certify that I am President and Principal Consultant, with RockEng Inc. located at 920 Princess St., Suite 310, Kingston, Ontario, Canada. This certificate applies to the technical report titled "Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada" that has an effective report date of February 22, 2023 (the "Technical Report").

1. I graduated from the University of Alberta, Edmonton, Canada in 2004 with a bachelor's degree in mining engineering and from Queen's University, Kingston, Ontario, Canada in 2007 with a master's degree in geomechanical engineering and in 2010 with a Ph.D. in geomechanical engineering. I am registered as a Professional Engineer in the provinces of Ontario (#100164661), British Columbia (#189685), Saskatchewan (# 66741), and Manitoba (#44426), and the state of Montana (#PEL-PE-LIC-59819).

2. I have worked as a geotechnical engineer for 13 years continuously since my Ph.D. graduation. Relevant work experience includes:

    a. Due Diligence Reviews: Review of mine plans and designs for investors as well as third-party review for technical quality and validity of geomechanical studies.
    b. Underground Mine Design: All geomechanical components of underground mine design such as, mining method, mine sequencing, stope sizing, excavation stability, static and dynamic ground support, hazard identification and risk management, infrastructure siting, and construction method evaluations.
    c. Open Pit Design: All geomechanical components of open pit mine design such as bench design, catchment capacities, highwall stability, instrumentation, hazard identification and risk management.
    d. Operational Geomechanics and Ground Control: Routine site support for operating mines, including audits and inspections, incident investigations, regular design reviews, support evaluations, instrumentation data interpretation, development and review of Ground Control Management Plans, and provision of ground control training to site personnel.
    e. Site Characterization: Core logging, geotechnical mapping, damage mapping, data analysis, domain delineation, material parameterization, in situ stress.
    f. Participation and author of several NI 43-101 Technical Reports.

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

4. I have not visited the Goliath project.

5. I am responsible for Sections 16.2, 16.3, and 16.4 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have been involved with the Goliath Gold Complex as follows:
   a. RockEng, 2022. Report # 22009-105 Treasury Metals Goliath PFS.
   b. RockEng, 2022. Report # 22009-102 Miller PFS Geotechnical Slope Design.
   c. RockEng, 2021. Report # 21058-104 Treasury Metals Goldlund Pit Pre-Feasibility Study.
   d. RockEng, 202. Report #20048-103 Geomechanics Review of Goliath PEA.
   e. MDEng, 2018. Report #18001-101 Minimum Inter-lens Pillar Dimensions for Underground and Supported Scenarios.
   f. MDEng, 2014. Geotechnical Analysis for the Open Pit and Underground Mine Plans of the Treasury Metals' Goliath Gold Project Dryden, ON.
8. 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: March 27, 2023

"Signed and sealed"

Kathy Kalenchuk, P.Eng., P.E.

CERTIFICATE OF QUALIFIED PERSON

Kristen Gault, P.Geo.

I, Kristen Gault, P.Geo., certify that I am employed as an Associate Geochemist with WSP E&I Canada Limited (“WSP”), with an office address of 2020 Winston Park Drive, Suite 600, Oakville Ontario, L6H 6X7. This certificate applies to the technical report titled “Goliath Gold Complex NI 43-101 Technical Report and Prefeasibility Study, Kenora District, Ontario, Canada” that has an effective report date of February 22, 2023 (the “Technical Report”).

1. I graduated from the University of Ottawa, Ontario, Canada in 2010 with an M.Sc. in Earth Sciences. I am registered member of the with the Professional Geoscientists Ontario (License #2577)

2. I have practiced my profession for 13 years. I have been directly involved in metal leaching and acid rock drainage (ML/ARD) assessment and prediction for mining projects, including geochemical characterization studies and water quality / geochemical modelling for mine development, operations, and closure.

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

4. I have not visited the Goliath, Goldlund or Miller property.

5. I am responsible for Sections 1.22.5, 18.7.5, 20.2.1.6, 20.3.1.6, 20.4.1, and 26.5 of the Technical Report.

6. I am independent of Treasury Metals Inc. as independence is defined in Section 1.5 of NI 43-101.

7. I have been involved with the ML/ARD assessment of the Goliath, Goldlund, and Miller projects from 2020 to present.

8. 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: March 27, 2023

“Signed and sealed”

Kristen Gault, P.Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Treasury Metals Inc. (Treasury Metals) by Ausenco Engineering Canada Inc. (Ausenco), SRK Mining Consulting Canada Inc. (SRK), *SLR Consulting (Canada) Ltd*. (SLR), Minnow Environmental Inc. (Minnow), RockEng Inc. (RockEng), and WSP E&I Canada Limited (WSP), 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 Treasury Metals subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party's sole risk.

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1 SUMMARY

1.1 Introduction

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Treasury Metals Inc. (Treasury Metals or the Company) to summarize the results of a Prefeasibility Study (PFS) of the Goliath Gold Complex. The Goliath Gold Complex is comprised of three projects, namely: the Goliath project, Goldlund project and Miller project. The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 Standards of disclosure for Mineral Projects (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by Treasury Metals to manage and coordinate the work related to the technical report. Ausenco also developed the PFS-level design and cost estimate for the process plant and general site infrastructure.
- SRK Mining Consulting Canada Inc. (SRK) was commissioned to complete the mineral resource and reserve estimates for the Goliath, Goldlund and Miller projects, and to design the open pit and underground mine plan, mine production schedule, and mine capital and operating costs.
- SLR Consulting Limited (SLR) was commissioned to develop the prefeasibility-level design for the tailings storage facility and site water management infrastructure for the Goliath project.
- Stantec Inc. (Stantec) was commissioned to develop prefeasibility-level design for the site water management infrastructure for the Goldlund project.
- Minnow Environmental Inc. (Minnow) was commissioned to review and summarize the various environmental baseline studies, outline the federal and provincial permitting requirements, provide an overview of anticipated closure measures and related costs, summarize community and Indigenous consultation efforts, and address environmental considerations and potential environmental liabilities for the Goliath, Goldlund and Miller projects.
- RockEng Inc was commissioned to provide geomechanical analyses and design constraints for the Goliath open pit and underground, Goldlund open pit, and Miller open pit projects.
- WSP E&I Canada Limited (WSP) was commissioned to assess the metal leaching and acid rock drainage (ML/ARD) potential of tailings and mine rock associated with the Goliath, Goldlund and Miller projects.

1.2 Terms of Reference

The report supports disclosures by Treasury Metals in a news release dated February 22, 2023 entitled, "Treasury Metals Completes Pre-Feasibility Study for Goliath Gold Complex".

All measurement units used in this report are metric unless otherwise noted. Costs are expressed in Canadian Dollars (currency: CAD; symbol: C$). The report uses Canadian English. United States dollars, where referenced, are represented by "USD" to denote currency and "US$" as a symbol.

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

The Goliath Gold Complex contains three deposits: Goliath, Goldlund and Miller. Treasury Metals owns 100% of Goliath Gold Complex.

1.3 Project Setting

The Goliath project is in the Kenora Mining Division in northwestern Ontario, approximately 4 km northwest of the Village of Wabigoon, 20 km east of Dryden and 2 km north of Trans-Canada Highway 17. The Goldlund and Miller projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath project, off Highway 72.

Access to the Goliath project is north from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Anderson and Tree Nursery Roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals. Access to the Goldlund site is east from Highway 72 via Goldlund Mine Road. The Miller project site is accessed via forestry road east off Highway 72. Access roads for the Goldlund and Miller sites are maintained by the Sustainable Forest Licence Holder (Domtar) for the area.

All major industrial services and supplies are available in Dryden and Sioux Lookout and the area is serviced by both the Dryden Airport and Sioux Lookout Airport. The Goliath project is located 20 km from Dryden, which has a population of 5,586 according to the Statistics Canada 2016 census. The Goldlund and Miller projects are located 43 km and 35 km, respectively, south of Sioux Lookout, which has a population of 5,272. The Goliath Gold Complex is located about 300 km northwest of the City of Thunder Bay, which is a major economic centre along the Trans-Canada Highway and port at the northwest head of the St. Lawrence Seaway on Lake Superior.

At the time of writing, Treasury Metals holds the sufficient surface rights necessary for any potential future mining operations including tailings storage areas, waste disposal areas, and a processing plant.

The location of the Goliath Gold Complex is shown in Figure 1-1.

Figure 1-1: Location of the Goliath Gold Complex

Source: Treasury Metals (2021).

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

The Goliath property covers approximately 7,601 ha and consists of 284 mining claims totalling approximately 6,254 ha; four mining leases totalling 359.25 ha; and 28 land parcels (includes patented claims) totalling 1,347.189 ha. Of the 1,347.18 ha of the patents and leases, 90.2 ha are surface rights only from seven land parcels. Of the 284 mining claims, 267 are single-cell mining claims, eight are boundary cell mining claims, and nine are multi-cell mining claims. The mineral rights are 100% held by Treasury Metals and all mineral rights are in good standing.

Treasury Metals, a former subsidiary of Laramide Resources Ltd. (Laramide), was spun out of Laramide as a dividend to Laramide's shareholders. Treasury Metals was listed and began trading on the Toronto Stock Exchange (TSX) exchange on August 19, 2008 under the trade symbol "TML".

The Goliath property consists of two historic properties that were consolidated into one: the larger Thunder Lake property, purchased from Teck and Corona, and the Laramide property.

The Goliath property is held 100% by Treasury Metals, subject to certain underlying royalties and payment obligations on 13 of the 21 land parcels, totalling approximately $103,500 per year.

Treasury Metals also has an option agreement pursuant to which Treasury Metals has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson property – 40.8711 hectares) located immediately west and contiguous to the Goliath project.

The Goldlund and Miller properties together consist of 1,349 mining claims totalling approximately 26,634 ha, 26 patented claims totalling 390.97 ha, one mining lease of 48.56 ha, and one licence of occupation of 74.84 ha. The patented claims and mining lease allow for both mineral rights and surface rights, while the Licence of Occupation allows for mineral rights only.

The Goldlund property is subject to the Goldlund Mines Limited Royalty Agreement, covering six patented claims as well as the three patented claims covered by the Mining Lease. Goldlund Mines will receive a 1% NSR on any ore mined above 50 m below the existing shaft collar as of the date of the agreement. The Goldlund property is also subject to the Rio Algom Limited Option Agreement whereby the Property owner will pay a 2.5% NSR and will have the right but not the obligation to purchase the NSR in its entirety for a one-time payment of $2.5 million with a 10-day notification of intent to exercise the purchase right and a 1.5% NSR with First Mining (which was purchased by an affiliate of Sprott Resources Streaming and Royalty Corp. (Sprott)) for the Goldlund property, which the Company can repurchase 0.5% for a one-time payment of $5 million.

The Goliath and Goldlund properties are subject to a royalty with an affiliate of Sprott Resources Streaming and Royalty Corp (Sprott) whereby Sprott will receive a 2.2% NSR on all minerals produced on the Goliath Gold Complex for the life of the project. The Company has the right to repurchase 50% of this royalty until December 31, 2028 for various purchase prices, at the Company's sole discretion, and the royalty also reduces by 50% upon the production of 1.5 million ounces of gold.

Proceeding the effective date of the resource, in February 2022 Treasury Metals had amalgamated many of the Goldlund claims from single-cell claims to multi-cell claims. While this reduces the total number of claims of the property, the effective area has remained unchanged as a result of this process. The claims shown and listed in this report reflect the property as it was at the effective date of the resource. After the amalgamation, many of the claim numbers and internal claim boundaries will have changed.

1.5 Geology and Mineralization

The Goliath Gold Complex is in the Archean Eagle-Wabigoon-Manitou greenstone belt in the Wabigoon Subprovince of the Superior Province. In the immediate area of the Goliath deposit, a 100 to 150 m thick unit of intensely deformed and variably altered, fine- to medium-grained, muscovite-sericite schist and biotite-muscovite schist with minor metasedimentary rocks hosts the most significant concentrations of gold in the Main and C Zones of the deposit.

Native gold and silver are associated with finely disseminated sulphides, coarse-grained pyrite and very narrow light grey translucent "ribbon" quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite. The alteration consists of primarily sericitization and silicification in association with the gold mineralization.

At Goliath, the gold-bearing zones strike from 090° to 072° with dips that are consistently between 72° and 78° south or southeast. The mineralized zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. In the Goliath deposit, higher grade gold mineralization occurs in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west. The main area of gold, silver and sulphide mineralization and alteration occurs up to a maximum drill-tested vertical depth of ~805 m, over a drill-tested strike-length of more than 2,500 m. The mineralized zones remain open at depth.

The Goldlund project is situated in northwestern Ontario approximately 60 km by road east of the town of Dryden, with a land package that covers a strike-length of over 50 km of greenstone belt in the Archean Wabigoon Subprovince. Historical gold production from the Goldlund and Windward mines is reported to be 18,000 oz of gold, with mining activities carried out between 1982 and 1985 using both open pit and underground mining methods.

Gold mineralization is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. The stockwork zones are hosted in albite-trondhjemite to diorite (granodiorite) strata-parallel sills, which dip from vertical to -80° southward and range in thickness from 14 m to 60 m. The stockwork zones form bands within the granodiorite sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine- to medium-grained pyrite and magnetite.

The mineralized sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the project.

The gold mineralization has been interpreted as a series of nine northeast-trending sub-parallel zone wireframes, considering a nominal 0.1 g/t Au threshold. Wireframes of Zones 1, 7, and 5 consist principally of gold mineralization associated with the stockwork veins in the large granodiorite sills, while wireframes of Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralization associated with stockwork veins that are hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only a minor contribution from the granodiorite sills.

1.6 History

The first gold mining on record in the region was in Van Horne Township in the early 1900s with very limited gold production from auriferous veining in biotite schist within the regional Wabigoon fault system. Sporadic exploration was carried out along the belt throughout the 1900s with only limited documentation of exploration activity conducted on the property.

1.6.1 Goliath Property

The earliest known government report covering the larger Dryden-Sioux Lookout Belt is the Ontario Department of Mines Report and Geology Map by Satterly (1941). In 1956-57, Compton-Wabigoon conducted geological mapping, magnetometer surveys, and the completion of two diamond drillholes totalling 458 m to explore the mineral potential of the major iron formation unit located in Lots 1-4, Concession V and VI, along the northern boundary of the property. Also in 1956, G.L. Pidgeon completed surface work and one shallow drillhole (drilled south) testing a sphalerite showing in the south half of Lot 6, Concession IV (Fraser Option legacy claim 0134).

Three major mining companies conducted exploration work on the Thunder Lake gold deposit (Goliath deposit) from 1989 to 1999 (last field work 1998). These are Teck Exploration Ltd. (Teck), Corona Gold Corporation (Corona), and Laramide Resources Ltd. (Laramide). At that time, the property held by all three companies covered more than 1,300 ha. Teck held the majority of the property and all of the surface exposure.

Exploration and resource development work at Goliath was undertaken by Teck from 1989 to 1999 on what was then called the "Thunder Lake Property". During this period, the property was divided into two properties called "Thunder Lake East" and "Thunder Lake West". The property was optioned to Corona, previously called Continental Caretech Corporation (CCC), in which CCC could earn an interest in the project under terms of an initial agreement dated January 3, 1994. Corona funded the exploration work from 1994 to 1999, but Teck remained the project operator both designing and running all field exploration activities.

In 1998, Teck completed an underground exploration and bulk sampling program at a cost of $1,929,071. This entire underground program, from surface site preparation through final closure plan, was completed between May 15 and September 15, 1998. The underground work consisted of a 27 m long inclined trench provided a 9 m high outcrop face suitable for the construction of a portal collar. A decline was prepared at a grade of 15% with the portal located just north of Norman Road and the north boundary of the Laramide property. Four bulk samples from the Main Zone (No. 1 and No. 2 shoots) totalling 2,375 tonnes were excavated consisting of blasted muck from drift rounds and slashed and material from a 400 tonne take-down-back test mining area grading above 3 g/t Au. After the underground work was completed, the portal was sealed and the area contoured, reseeded, and fully remediated in late 1999.

1.6.2 Goldlund Property

Exploration activities on the Goldlund project date from the 1940s, where in 1941, A. Ward and R. Lundmark (two prospectors working for the Mosher group) discovered gold mineralization in the southwestern part of Echo Township (Page, 1984). From 1946 to 1952 there were significant exploration activities carried out on the Newlund Mines Limited and Windward Gold Mines prospects. The Newlund prospect was extensively explored by 4,570 m of underground drifts and crosscuts on four levels (200 ft, 350 ft, 500 ft, and 800 ft), and 6,220 m of core drilling from a 255 m deep vertical shaft. The 200 ft level on the Newlund prospect was extended more than 3.2 km to the west to connect with the 68 m vertical shaft on the Windward prospect, crossing the entire Windward claim block (Page, 1984). From 1952 to 1973, there was only limited exploration activities carried out on the Echo Township gold prospects.

In 1974, Goldlund Mines Limited and Rayrock Mines Limited entered into an agreement and rehabilitated the surface facilities. This work included installing a new headframe and hoist and dewatering the underground workings to the second level (350 ft). A program of bulk sampling, underground chip sampling, and core drilling of 41 holes totalling 4,932 ft (approximately 1,500 m) was carried out. No further activities were carried out, as the prospect was deemed uneconomic given the gold price at that time (Page, 1984).

In total, approximately 143,825 m of drilling has been completed in 808 surface drillholes, and approximately 18,624 m of drilling has been completed in 480 underground holes. Additionally, Tamaka Gold Corporation (Tamaka) carried out a trenching program in 2012 that included the excavation, stripping, mapping, channel sampling and a detailed structural analysis.

From mid-1982 to early 1985, Campbell Resources Inc. (Campbell Chibougamau), through its wholly owned subsidiary Goldlund Mines Limited, operated an underground mine and an open pit mine and processed material through the mill at the site. Pieterse (2005) compiled the production records that show underground mine production of 100,000 tons (approximately 90,700 tonnes) at an estimated grade of 0.15 oz/ton Au (approximately 5.14 g/t Au) and open pit production of 43,000 tons (approximately 39,000 t) at an estimated grade of 0.17 oz/ton Au (approximately 5.83 g/t Au.

1.6.3 Miller Property

There has been no historical exploration or drilling activities on the Miller deposit prior to 2018. In 2018 and 2019, First Mining completed two drill programs on Miller, as described in Section 10 of this report.

1.7 Exploration

Since 2008, Treasury Metals has focused its exploration work on the western half of the property to evaluate the gold potential of the Goliath deposit. During this 12-year period, exploration activities consisted of re-establishing the former Teck exploration grid, geological mapping and sampling, prospecting, the completion of structural studies, trenching and channel sampling, the completion of a ground IP geophysical survey and two airborne geophysical surveys, downhole IP and tomography surveys, metallurgical testing, and mineral resource estimations of the main deposit (including carrying out preliminary economic analyses in 2012, 2017 and 2021).

1.8 Drilling and Sampling

The mineralization was sampled over the years with multiple campaigns of core drilling by Teck-Corona and Treasury Metals since the 1990s. The drill database is now a mix of historical data and more recent data collected by Treasury Metals from 2008 through to 2021. Both data types were used in the resource estimate. The mineral resource estimate for Goliath is supported by 904 surface drillholes with an aggregated length of 290,6856 m.

For the Goldlund deposit, the dataset consists of 1,934 core holes representing 250,861 m of core (1,454 surface holes and 480 underground drillholes). In addition, the Goldlund data includes 246 underground channel samples representing 3,637 m and 188 trenches and one pit for 1,444 m of sampling. Of these, 1,375 core holes contributed to the Goldlund mineral resource estimate. The underground channel and trench samples were not considered for grade estimate but were included in the modelling of mineralized zones.

There are 61 drillholes in the Miller database totalling 10,370 m of drilling. Of these, 49 drillholes (7,964 m) contributed to the Miller resource estimate.

Drilling continued in 2022, where 53 drillholes with a length of 17,706 metres were completed on exploration targets as well as 8 drillholes with a length of 2,597 metres in support of geotechnical and metallurgical studies. The results of this additional drilling were not included in the PFS as it did not result in a significant variance from the resource published January 17, 2022.

1.9 Data Verification

Details of qualified persons site visit are provided in Section 2.4. The site visits allowed the QPs to review the property access and site facilities at Goliath and Goldlund. The surface geology at Goliath, Goldlund and Miller was examined. As part of the site visits, the local geology and exploration history of the project was reviewed with Treasury metals staff. Drillhole collar locations were examined in the field and compared with locations provided in the digital database for the project. A total of 36 drill collars were verified with hand-held GPS, and all collars were found to be within the margins of error allowed by NI 43-101.

In addition to drill collars, the core logging and sampling procedures, as well as quality control and quality assurance measures, were reviewed for all three deposits.

For the three deposits, a total of 18 holes were reviewed and geological logging was validated along with sampling methodology. Drill core was compared with core logging sheets and procedures which included commentary on typical lithologies, alteration and mineralization styles, and contact relationships at the various lithological boundaries.

1.10 Metallurgical Testwork

Metallurgical testwork programs have been conducted on samples from Treasury Metals since 2011. The following historical sources of technical and project information were referenced in developing the process plant design for the prefeasibility study, along with the current prefeasibility metallurgical testwork program:

1. 2011 G&T Metallurgical Services Ltd. Pre-Feasibility Metallurgical Testing Goliath Gold Project. KM 2906.
2. 2012 ALS Metallurgy (formerly G&T Metallurgy), Feasibility Metallurgical Testing, Treasury Metals Incorporated. KM3406.
3. 2013 SGS Scoping Study and Comminution testing on samples From the Goldlund Project. 13665-001.
4. 2017 ALS Metallurgy, Metallurgical Testwork on Goliath Gold Samples, Treasury Metals Incorporated. KM5262.
5. 2017 Base Metallurgical Laboratories, Metallurgical Testing of Goliath Project. BL0172.
6. 2020 Technical Report Re-Issue, Goldlund Gold project, Sioux Lookout, Ontario.
7. 2020 Metallurgical Testing of the Goliath Gold Project. BL0697.

The metallurgical program for the prefeasibility study was conducted in 2021 at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as project BL840. Tests were performed on samples from the Goliath, Goldlund, and Miller deposits. The testwork program included head analysis, comminution testwork, and cyanidation testing of gravity tailings on composites from all three deposits, and mineralogy, extended gravity recovery gold testing, whole ore cyanidation, cyanide detoxification, and thickening testwork on samples selected from the Goliath and Goldlund deposits. Gold deportment tests were also conducted on two master composites from the Goldlund deposit.

The comminution tests demonstrated that the Goliath samples have medium hardness and low abrasivity with a design SAG mill comminution (SMC) test Axb of 37.0, Bond ball work index of 10.9 kWh/t, and abrasion index of 0.077 g. In comparison, the Goldlund samples are very hard and have high abrasivity, demonstrating a design SMC Axb result of 26.2, a Bond ball work index of 16.0 kWh/t, and an abrasion index of 0.564 g. The one Miller composite demonstrated a Bond ball work index of 14.5 kWh/t.

Extended gravity recoverable gold (E-GRG) tests were completed on two Goliath composites and three Goldlund composites. The E-GRG test results for the two Goliath samples indicate a fair amount of coarse gold and a high amenability to gravity gold recovery with high recoveries ranging from 52% to 55%, whereas the two Goldlund samples (from the main pit) show less amenability to gravity with recoveries of 28.5% and 43.5%. A third Goldlund composite taken from the west pit shows higher amenability to gravity concentration with an estimated gravity recovery of 54%.

Leach testwork was conducted to optimize the comminution circuit grind size and determine the most favourable leach conditions. The grind size did not have a significant effect on the leach residue grade of the two Goliath composites or two Goldlund main pit composites, but it did demonstrate a strong correlation for the Goldlund west pit composite.

Variability testwork was then performed on the gravity tailings of all composites. Goliath tests were performed with a retention time of 24 hours at a grind size of 100 µm, with varying lime and cyanide additions. The variability tests yielded recoveries of 93.9% and 94.8% for the Main Zone and Central Zone, respectively. Goldlund variability tests were conducted over a 48-hour retention time and a grind size of 90 µm with various lime and cyanide additions. Recoveries of 87.8% and 93.0% were achieved for the main pit and west pit composites, respectively.

Recovery ranges for each deposit based on the predicted head grades are summarised in Table 1-1.

Table 1-1: Summary of Recovery Ranges per Deposit

Deposit	Maximum Head Grade (g/t)	Minimum Head Grade (g/t)	Maximum Gold Recovery (%)	Minimum Gold Recovery (%)
Goliath	3.55	1.39	96.4	94.5
Goldlund, Zone 1	1.83	1.23	92.0	89.6
Remaining Goldlund Zones	1.83	1.23	95.6	93.7
Miller	1.18	0.86	94.2	93.6

1.11 Mineral Resource Estimate

The mineral resource estimates presented in this report represent an update from the resources presented in the March 10, 2021 technical report for the Goliath Gold Complex. The mineral resources for the Goliath and Goldlund deposits were prepared by Dr. Gilles Arseneau and Ms. Sheila Ulansky of SRK (Canada) Inc. The mineral resources for the Miller deposit were prepared by Dr. Arseneau, who is the qualified person for all three mineral resource statements presented in this technical report.

Mineral resources for the Goliath Gold Complex are reported as being potentially extractable by open pit and underground operations. The mineral resources statements were prepared by Dr. Gilles Arseneau, P. Geo., associate consultant with SRK. The mineral resources are based on 2,899 drillholes measuring 551,916 meters for the Goliath, Goldlund and Miller deposits, including 176 drillholes and 41,072 meters from the 2021 drilling campaign.

The mineral resources are prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). The estimated mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The effective data of the mineral resource statement is January 17, 2022.

The Goliath open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.25 g/t gold based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of $93.873*Au(g/t)^{0.021}$ and 60%, respectively.

The underground mineral resources are reported inside shapes generated from Deswik Mining Stope Optimizer (MSO) at a cut-off grade of 2.2 g/t gold based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of $93.873*Au(g/t)^{0.021}$ and 60%, respectively. Table 1-2 summarizes the 2022 mineral resources for the Goliath and Goliath East deposits.

Table 1-2: Goliath and Goliath East Mineral Resource Statement, SRK, January 17, 2022

Type	Classification	Cut-off	Tonnes	Au (g/t)	Au (Oz)	Ag (g/t)	Ag (Oz)
Open Pit	Measured	0.25	6,223,000	1.20	239,500	4.70	940,600
	Indicated	0.25	23,081,000	0.75	559,400	2.53	1,878,500
	Measured + Indicated	0.25	29,304,000	0.85	798,900	2.99	2,819,100
	Inferred	0.25	3,330,000	0.66	70,200	0.80	85,200
Underground	Measured	2.20	170,000	6.24	34,100	22.34	122,100
	Indicated	2.20	2,550,000	3.55	291,000	7.08	580,800
	Measured + Indicated	2.20	2,720,000	3.72	325,100	8.04	702,900
	Inferred	2.20	48,000	2.95	4,600	4.06	6,300
Total	Measured		6,393,000	1.33	273,600	5.17	1,062,700
	Indicated		25,631,000	1.03	850,400	2.98	2,459,300
	Measured + Indicated		32,024,000	1.09	1,124,000	3.42	3,522,000
	Inferred		3,378,000	0.69	74,800	0.84	91,500

Notes: **1.** Mineral resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc. Mineral resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. **2.** Mineral resource effective date January 17, 2022. **3.** Goliath open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.25 g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60%, respectively. **4.**Goliath underground mineral resources are reported inside shapes generated from Deswik Mining Stope Optimizer (DSO) at a cut-off grade of 2.2g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60%, respectively. **5.** Gold and silver assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.5 m for Goliath. **6.** Excludes unclassified mineralization located within mined-out areas. **7.** Silver grade and ounces are derived from the Goliath tonnage only. **8.** Goliath open pit cut-off is 0.25 g/t. **9.** All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

The Goldlund open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold based on a gold price of US$1,700/oz and a gold processing recovery of $90.344xAu(g/t)^{0.0527}$.

Underground mineral resources are reported inside MSO shapes at a cut-off grade of 2.2 g/t gold based on a gold price of US$1,700/oz and a gold processing recovery of $90.344xAu(g/t)^{0.0527}$. Table 1-3 summarizes the mineral resources for the Goldlund deposit.

Table 1-3: Goldlund Deposit Mineral Resource Statement, SRK, January 17, 2022

Type	Classification	Cut-off	Tonnes	Au (g/t)	Au (Oz)
Open Pit	Measured	0.30	0	0.00	0
	Indicated	0.30	33,353,000	0.85	911,000
	Measured + Indicated	0.30	33,353,000	0.85	911,000
	Inferred	0.30	28,833,000	0.73	680,200
Underground	Measured	2.20	0	0.00	0
	Indicated	2.20	222,000	4.06	29,000
	Measured + Indicated	2.20	222,000	4.06	29,000
	Inferred	2.20	222,000	3.26	23,300
Total	Measured		0	0.00	0
	Indicated		33,575,000	0.87	940,000
	Measured + Indicated		33,575,000	0.87	940,000
	Inferred		29,055,000	0.75	703,500

Notes: 1. Mineral resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc. Mineral resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral resource effective date January 17, 2022. 3. Goldlund open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 4. Goldlund underground mineral resources are reported inside DSO shapes at a cut-off grade of 2.2 g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 5. Gold assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 2.0 m for Goldlund. 6. Excludes unclassified mineralization located within mined-out areas. 7. Goldlund open pit cut-off grade is 0.30 g/t, 8. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

The Miller open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold that on a gold price of US$1,700/oz and a gold processing recovery of $93.873*Au(g/t)^{0.021}$. Table 1-4 summarizes the mineral resources for the Goldlund deposit; Table 1-5 shows the mineral resource estimate for the Goliath Gold Complex.

Table 1-4: Miller Deposit Mineral Resource Statement, SRK, January 17, 2022

Type	Classification	Cut-off	Tonnes	Au (g/t)	Au (Oz)
Open Pit	Measured	0.30	0	0	0
	Indicated	0.30	2,112,000	1.10	74,600
	Measured +Indicated	0.30	2,112,000	1.10	74,600
	Inferred	0.30	138,000	1.01	4,500

Notes: 1. Mineral resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc. Mineral resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral resource effective date January 17, 2022. 3. Miller open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 93.873*Au(g/t)^0.021. 4. Gold assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.0 m for Miller. 5. Miller open pit cut-off grade is 0.30 g/t 6. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

Table 1-5: Goliath Gold Complex Mineral Resource Estimate

Type	Classification	Cut-off	Tonnes	Au (g/t)	Au (Oz)	Ag (g/t)	Ag (Oz)
Open Pit	Measured	0.25 / 0.3	6,223,000	1.20	239,500	4.70	940,600
	Indicated	0.25 / 0.3	58,546,000	0.82	1,545,000	2.53	1,878,500
	Measured + Indicated	0.25 / 0.3	64,769,000	0.86	1,784,500	2.99	2,819,100
	Inferred	0.25 / 0.3	32,301,000	0.73	754,900	0.80	85,200
Underground	Measured	2.20	170,000	6.24	34,100	22.34	122,100
	Indicated	2.20	2,772,000	3.59	320,000	7.08	580,800
	Measured + Indicated	2.20	2,942,000	3.74	354,100	8.04	702,900
	Inferred	2.20	270,000	3.21	27,900	4.06	6,300
Total	Measured		6,393,000	1.33	273,600	5.17	1,062,700
	Indicated		61,318,000	0.95	1,865,000	2.98	2,459,300
	Measured + Indicated		67,711,000	0.98	2,138,600	3.42	3,522,000
	Inferred		32,571,000	0.75	782,800	0.84	91,500

Notes: 1. Mineral Resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc., Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral Resource effective date January 17, 2022. 3. Goliath Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.25g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60% respectively. 4. Goldlund Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.3g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 5. Miller Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 93.873*Au(g/t)^0.021. 6. Goliath Underground Mineral Resources are reported inside shapes generated from Deswick Mining Stope Optimiser (DSO) at a cut-off grade of 2.2g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60% respectively. 7. Goldlund Underground Mineral Resources are reported inside DSO shapes at a cut-off grade of 2.2g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 8. Gold and Silver assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.5 m for Goliath, 2.0 m for Goldlund and 1.0 m for Miller. 9. Excludes unclassified mineralization located within mined out areas. 10. Silver grade and ounces are derived from the Goliath tonnage only. 11. Goliath Open Pit and Goldlund/Miller cut-off grades are 0.25g/t and 0.30g/t, respectively. 12. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

1.12 Mineral Reserve Estimate

The qualified persons accepting the professional responsibility for the open pit and underground mineral reserve estimates are Ms. Colleen MacDougall, P. Eng., and Mr. Sean Kautzman, P. Eng., respectively. Ms. MacDougall undertook open pit mine planning work to support the preparation of the mineral reserve statement for the Goliath, Goldlund and Miller open pits. Mr. Kautzman undertook underground mine planning work supporting the preparation of the mineral reserve statement for the Goliath underground project.

Mineral reserves for the Goliath Complex consist of open pit mineral reserves at Goliath, Goldlund and Miller, and underground mineral reserves at Goliath, with effective date of December 31,2022 are founded on and included within the mineral resource estimates with an effective date of January 17, 2022.

Project base case economic analysis shows that the life-of-mine plan founded on the mineral reserve estimates provides a positive present value of the net cash flow, confirming that the mineral reserves are economically viable and that economic extraction can be justified.

The QPs are not aware of any additional mining, metallurgical, infrastructure, permitting, or other factors not presented in this report that could materially affect the mineral reserve estimate. The mineral reserve estimate is presented in Table 1-6.

Table 1-6: Mineral Reserves Estimate

Classification	Quantity (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)
Open Pit – Goliath					
Proven	3,969	1.05	134	3.22	410
Probable	5,580	0.67	119	2.20	395
Proven & Probable	9,549	0.83	254	2.62	805
Open Pit – Goldlund					
Proven	-	-	-	-	-
Probable	16,256	1.19	621	-	-
Proven & Probable	16,256	1.19	621	-	-
Open Pit – Miller					
Proven	-	-	-	-	-
Probable	738	1.03	24	-	-
Proven & Probable	738	1.03	24	-	-
Underground – Goliath					
Proven	596	3.96	76	16.73	321
Probable	3,180	2.85	292	5.85	598
Proven & Probable	3,776	3.03	368	7.56	918
Total					
Proven	4,565	1.43	210	4.98	731
Probable	25,754	1.28	1,057	1.20	993
Proven & Probable	30,319	1.30	1,267	1.77	1,724

Notes: **1.** Mineral reserves with an effective date of December 31, 2022 are founded on and included within the mineral resource estimates, with an effective date of January 17, 2022. **2.** Mineral reserves were developed in accordance with CIM Definition Standards (2014). **3.** Open pit mineral reserves incorporate 10%, 7% and 9% dilution for Goliath, Goldlund and Miller, respectively. Open pit mineral reserves include 1% loss for Goliath and Miller, no losses are included for Goldlund. Goliath underground mineral reserves include 5% dilution and 0% loss for development. For stopes at Goliath underground, the mineral reserves include 15% dilution (both downhole and uphole stopes) and 90% (downhole) and 80% (uphole) recovery. **4.** Open pit mineral reserves are reported based on open pit mining within designed pits above cut-off values of C$15.22/t, C$16.00/t and C$23.63/t for Goliath, Goldlund and Miller, respectively. Goliath underground mineral reserves are reported based on underground mining within designed underground stopes above a mill feed cut-off value of C$107.66/t (inclusive of 15% mining dilution). The cut-off values are based on a gold price of US$1,550/oz Au, a silver price of US$22, transportation costs of C$5/oz Au, payabilities of 99% Au and 97% Ag, LOM average gold recoveries of 94.2% for Goliath, 94.3% for Goldlund and 94.0% for Miller, and a silver recovery of 60% for Goliath. **5.** Underground mineral reserves following Year 13 have been removed from the LOM plan and thus are excluded in the mineral reserve table above. Some low grade Goldlund material above cut-off is not fed to the plant and therefore not included in the mineral reserves. **6.** The Qualified Person for the open pit mineral reserve estimate is Colleen MacDougall, Peng; and the Qualified Person for the underground mineral reserve estimate is Sean Kautzman, Peng, both are SRK Consulting (Canada) Inc. employees. **7.** Rounding may result in apparent summation differences between tonnes, grade and contained metal.

1.13 Mining Methods

The Goliath deposit will be mined using open pit and underground mining methods, while the Goldlund and Miller deposits will be mined by open pit methods. The operations will feed a single processing facility located at Goliath at a rate of

6,460 t/d or 2,358 kt/a. Feed from Miller will be hauled to Goldlund with mining trucks. Both Goldlund and Miller feed will be hauled by contractor in highway trucks to Goliath.

The open pit operations will consist of conventional drilling, blasting, loading, and hauling. Loading will be undertaken on 10 m benches with one 11 m^3 excavator and along with two 6 m^3 excavators, which will paired with 63 t haul trucks in the pit. Mining will commence at Goliath with one year of pre-production and two years of production. Production will move to Goldlund in Year 2 until Year 7. The final three stages of Goliath will be mined in Year 7 to 9. Miller will be mined in Years 8 and 9. High-grade feed will be fed preferentially throughout the mine life, with lower grades used to fill the plant to capacity. Mining from the pits will end in Year 9, after which the plant will be fed from the low-grade stockpiles from Goliath and Goldlund until then end of mine life in Year 13. Total material movement from the open pit operations average 14 Mt/a for the first 8 years.

Underground mining will be conducted using a long hole open stoping method following a longitudinal retreat approach, with stopes extracted in a bottom-up sequence. The mining fleet will be supplied and operated via a contractor and will consist of modern mobile equipment typically used in narrow-vein long hole open stoping scenarios. Development will begin after the Goliath open pit has started, with production nearest the crown pillar targeted early in the life of mine such that those stopes are extracted and backfilled prior to deposition of tailings in the open pit. First ore is achieved in Year 1 with sustained commercial production in Year 3. The peak annual ore tonnage is scheduled for Year 7 with a steady decline in production in successive years as the number of active working faces decreases. After mining ceases in Year 13 it will enter its closure stage, with approximately 3.8 Mt of ore processed from the underground mine.

1.14 Recovery Methods

Metallurgical testing results (from the prefeasibility program and historical results) were analyzed, and several processing options were reviewed. Based on the analysis, a conventional leach and carbon-in-leach process route was chosen to be most suitable for the deposit and project economics (see Figure 1-2).

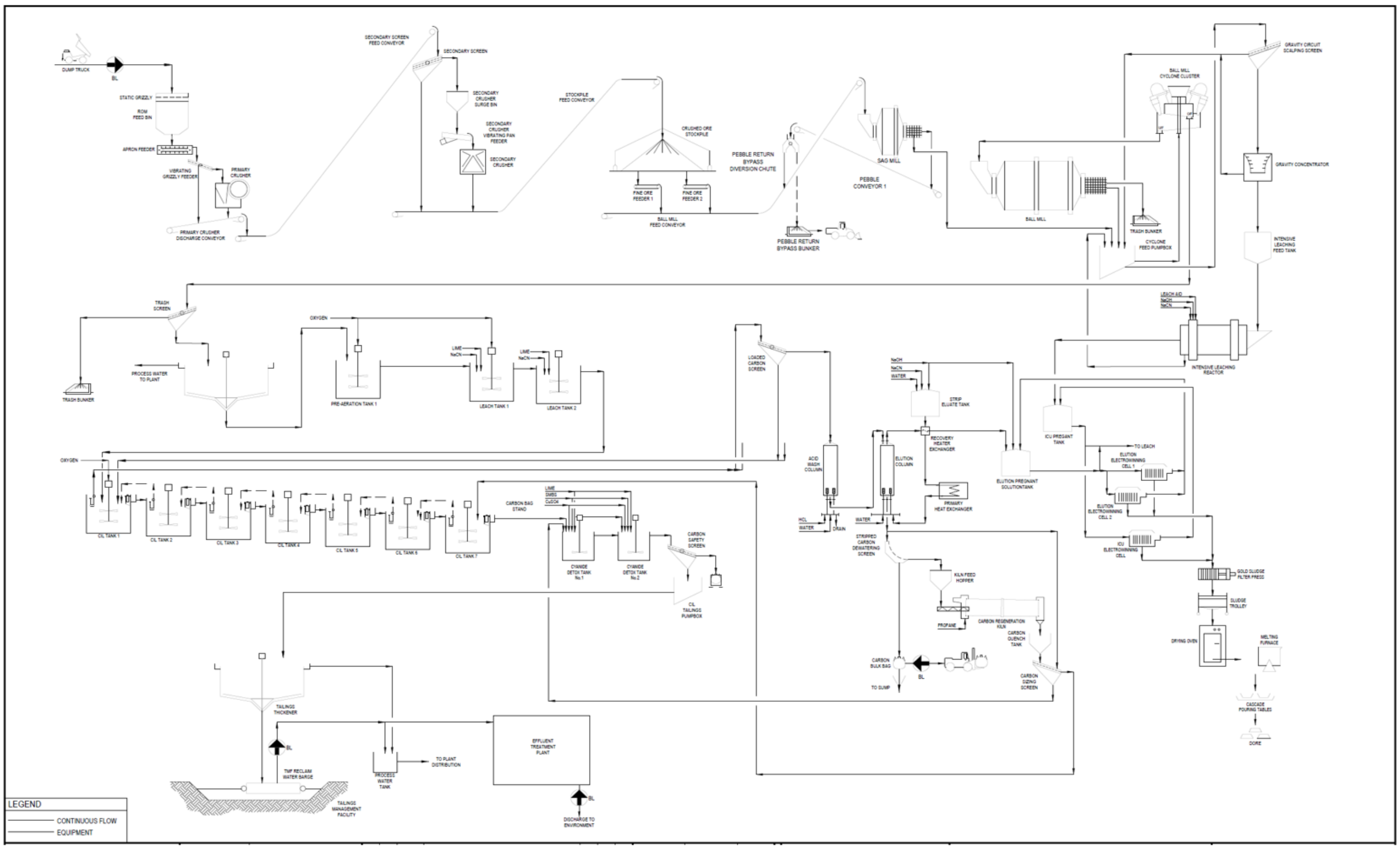
The process plant was designed using conventional processing unit operations to treat up to 6,460 t/d (2.36 Mt/a) based on an availability of 8,059 hours per year or 92%. The crushing plant section design is set at 67% availability. and the gold room availability is set at 52 weeks per year. The gold room will operate one shift per day, five days per week, and will produce gold/silver doré bars.

Ore will be hauled from the mine to the primary crushing facility equipped with an apron feeder, grizzly feeder, and jaw crusher. The crushed ore will be conveyed to the secondary scalping screen, where undersize material will bypass the secondary cone crusher while oversize will be crushed. The two streams will be combined and conveyed to the covered stockpile. The crushed ore will be ground by a SAG mill followed by a closed-circuit ball mill with hydro-cyclone classification. The cyclone feed pump will feed the cluster of hydro-cyclones and a second pump will feed the gravity circuit. The gravity circuit will consist of one scalping screen and a centrifugal batch concentrator. The scalping screen undersize will feed the centrifugal concentrator, and the concentrate will be collected and subsequently leached by the intensive cyanidation reactor circuit. The scalping screen oversize, gravity concentrator tailings, and the intensive cyanidation reactor tailings will recirculate to the cyclone feed pump box. The cyclone overflow will flow to the high-rate pre-leach thickener prior to the conventional leach and CIL circuit with a final grind size of 80% passing 85 µm. The cyclone underflow will report back to the ball mill.

Gold and silver adsorbed in the CIL circuit will be recovered onto activated carbon and eluted using an AARL carbon elution circuit followed by electrowinning in the gold room. The gold-silver electrowinning sludge will be dried in an oven and mixed with fluxes and smelted in a furnace to pour gold/silver doré bars.

Carbon will be reactivated in a carbon regeneration kiln before being returned to the CIL circuit. CIL tails slurry will be treated in cyanide destruction using the SO_2/air process before reporting to a final tailings thickener. Thickener underflow will be pumped to the tailings storage facility while tailings thickener overflow reports to process water.

Figure 1-2: Overall Process Flow Diagram



Source: Ausenco, 2023.

1.15 Project Infrastructure

1.15.1 General

Infrastructure to support the Goliath Gold Complex will consist of site civil work, buildings and facilities, water management systems, a tailings storage facility, an electrical substation, and power distribution. Mine facilities and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication, and sanitary systems as required. The processing plant and tailings storage facility will be located at the Goliath property, along with most ancillary project infrastructure.

The Goliath, Goldund and Miller properties may provide sufficient area to establish mine infrastructure (such as tailings and waste storage areas) and a processing plant site.

1.15.2 Tailings Storage Facility

Tailings are deposited at a thickened, non-segregating consistency of 63% solids by mass with a 3% surface slope contained by rockfill embankment dams with HDPE geomembrane-faced liners keyed into natural clayey soils. The non-segregating nature of the tailings means that only a very small pond can be maintained within the tailings storage facility (TSF) and surplus water reports by gravity to an external reclaim pond that is excavated and also lined with geomembrane.

This concept allows the reclaim pond to be maintained at a low elevation relative to the ground surface mitigating dam safety and seepage risks. Seepage from the base of the TSF is inhibited by naturally occurring clayey lacustrine soils that have been determined to be extensive within the TSF footprint, the relatively low permeability homogeneous tailings, and absence of a pond within the TSF.

The TSF containment dams are constructed initially of locally borrowed sand with an HDPE geomembrane on the upstream face and they are raised progressively in the downstream manner using non potentially acid generating / non-metal leaching rockfill. Both upstream and downstream toe stabilization berms are provided for the dams to meet stability criterion.

1.15.3 Site Water Management

Contact water will be collected in ditches and ponds. Water transfer between ponds and/or facilities such as the mill and the effluent treatment plant (ETP) will be achieved with pumps and pipelines, although gravity drainage will be prioritized when the topography allows. A total of eight water management ponds are proposed as part of the site-wide water management strategy. Non-contact water diversion berms and/or ditches will be implemented, if required. Water collected in the reclaim pond adjacent to the TSF will be the primary source of mill make-up water.

The engineering design intent for water management is to have a flexible water management system that provides alternatives to collect and treat contact water. Treasury made the commitment for mine water effluent to meet Provincial Water Quality Objectives (PWQO) for the protection of aquatic life at final discharge to the environment under the range of flow magnitude and climatic conditions considered for water balance modelling and design.

The water treatment concept involves the implementation of a modular ETP to allow expansion of the flow capacity by implementing additional modules as needed. At the time of commissioning the mill (i.e., beginning of operations) the ETP should have the capacity to treat to PWQO all flows collected within the project site when production starts and the disturbed footprint is smaller. Expansion of the flow capacity will be implemented in the future for a larger disturbed footprint, as required by operations and in accordance with permitting requirements.

The project site exhibits a net positive water balance. Surplus collected water must be discharged to the environment under dry, average, and wet annual climatic conditions to avoid net accumulation of water on an annual basis. Under dry conditions, water should be stored seasonally to assure availability of water as required to meet the water demand of the mine operation. According to the water balance modelling results, enough water would be collected from the project footprint to meet the make-up water demand on an annual basis for process plant operation.

1.16 Environmental, Permitting and Social Considerations

The Goliath, Goldlund, and Miller projects are three distinct properties and be subject to individual permitting processes as each site is developed. Accordingly, the following subsections present a summary of the environmental aspects and anticipated permitting efforts required for the proposed life of mine for each of the three projects. An overview of Treasury Metals' social and community programs is also briefly summarized.

1.16.1 Regulatory Framework Overview and Status

Treasury Metals initiated the federal environmental assessment (EA) process in November 2012 in accordance with the *Canadian Environmental Assessment Act, 2012* (CEAA 2012). The CEAA 2012 process involved an initial submission of the Goliath Project Environmental Impact Statement (EIS) in April 2015 and a final revised EIS in April 2018 for the construction, operation, decommissioning, and abandonment of an open pit and underground gold mine and associated infrastructure. After two rounds of regulatory review and stakeholder comment periods, the Canadian Environmental Assessment Agency released their Draft Environmental Assessment Report in June 2019. In August 2019, the Minister of the Environment issued their Decision Statement concluding, with required mitigative measures and conditions, that the Goliath project is not likely to cause significant adverse environmental effects, and the Goliath project was thereby permitted to proceed under CEAA 2012.

The Goliath project, as presented in this technical report, has undergone optimizations since the 2018 EA/EIS, including optimized designs of the tailings facility, mine rock stockpiles and water management systems. The milling of ore from the Goldlund and Miller project sites at the Goliath project site is also being proposed. In accordance with the Decision Statement, however, Treasury Metals is obligated to notify the federal and relevant authorities and consult with specified Indigenous groups regarding any changes to the Goliath project that may result in potential adverse environmental effects, including proposed mitigation measures, and follow-up requirements to be implemented pertaining to the proposed changes. Additional environmental data continue to be collected to support the proposed changes, Indigenous consultation is ongoing, and preparation of the required federal agency notification, as stipulated in the Decision Statement, is currently in progress.

The currently proposed design changes at the Goliath project site are not anticipated to significantly affect the overall description of the Goliath project as assessed under the federal CEAA 2012 process, nor are they expected to trigger a review of the environmental impact assessment under the current Canadian *Impact Assessment Act*, which came into force in August 2019.

Based on current proposed designs for the Goldlund and Miller projects, neither is expected to require completion of a federal impact assessment.

1.16.2 Goliath Project Site

1.16.2.1 Baseline / Environmental Studies

Environmental data collection for the Goliath project was initiated in 2008, with additional environmental studies undertaken to support the federal environmental assessment process. Since issuance of the federal Decision Statement in August 2019, Treasury Metals continues to collect supplemental environmental baseline monitoring data while optimization and engineering designs for the Goliath project are progressed.

To date, baseline environmental studies have been conducted addressing aspects of surface water quality, aquatic resources (including sediment quality, benthic invertebrate community, fish community and fish habitat), hydrology, hydrogeology and groundwater quality, terrestrial resources, geochemistry, air quality and noise. Avian species at risk detected within the local study area during terrestrial baseline studies conducted between 2011 and 2016 included common nighthawk, barn swallow, Canada warbler and olive-sided flycatcher, although no evidence of active nesting within the project site area was observed, except for active barn swallow nests at the Treasury Metals office location during additional 2022 surveys. No whip-poor-wills were detected during these surveys. Three bat species were detected at the project site area; big brown bats, little brown myotis, and northern myotis, the latter two being considered species at risk.

Geochemical test results indicate that almost all project mine rock (i.e., waste rock and ore) is potentially acid generating (PAG; at a neutralization potential ratio of less than 1 [NPR<1]) with a very low neutralization potential content (on the order of 5 to 10 kg $CaCO_3$/t). Humidity cell testing also indicated that these materials may become net-acid generating after several months to one year of exposure. Similarly, the Goliath tailings are PAG and metal leaching, with a short lag time to acid onset (i.e., one year). Therefore, the tailings storage facility and tailings deposition will need to be designed to physically isolate PAG and/or metal leaching tailings to limit their contact with oxygen and surface runoff. Site contact waters will need to be properly collected, managed and treated prior to discharge to the environment.

The tailings storage facility and tailings deposition will therefore need to be designed to minimize oxidation of the tailings solids (during mine operations and over the long term following closure), and site contact waters will need to be properly collected, managed and treated prior to discharge to the environment.

The Decision Statement for the Goliath project stipulates a number of conditions that require Treasury Metals to undertake various environmental monitoring programs at the Goliath project site during the construction, operation, closure, and post-closure phases of the project. The existing environmental baseline monitoring programs conducted to date will provide the basis for the monitoring frameworks and may be modified to meet regulatory and reporting requirements as the project continues to move through the permitting phase.

1.16.2.2 Permitting Considerations

The Goliath project is prepared to move into the next permitting phase, which will be supported by the substantial baseline information gathered during the federal environmental assessment process. Provincial permitting will involve acquiring environmental permits and approvals primarily from the Ministry of the Environment, Conservation and Parks (MECP), the Ministry of Natural Resources and Forestry (MNRF), and the Ministry of Mines (MINES).

Typical provincial environmental approvals are expected for construction and operation of the Goliath project, including Environmental Compliance Approval – Industrial Sewage Works for water management, treatment and discharge; Environmental Compliance Approval for air and noise emissions; Permits to Take Water; Work Permits and Land Use Permits for construction of roads, water crossings, work on/near shorelines and watercourse realignments; and Forest Resource Licences.

Two tributaries of Blackwater Creek will be partially overprinted by project infrastructure, resulting in the unavoidable harm to fish and fish habitat, infilling of waters frequented by fish, and/or reduction of flow. The loss of fish habitat will require a Fisheries Act Authorization (FAA), including development and implementation of an offsetting plan for compensation of the lost fish habitat, pursuant to the *Fisheries Act*. A regulatory amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER), will also be required for mine waste facilities that overprint fish habitat. As part of the FAA, Treasury Metals will need to provide financial assurance to the department of Fisheries and Oceans Canada (DFO) until it can be demonstrated that the fish habitat offsets are constructed and functioning as intended.

1.16.2.3 Closure and Reclamation Planning

Closure of the Goliath project will be governed by the *Ontario Mining Act* and its associated regulations and codes. A Closure Plan will be required, detailing measures for temporary suspension, state of inactivity and final closure, including financial assurance required to implement the closure measures.

Conventional methods of closure are expected to be employed at the Goliath project site. The closure measures for the TSF will be designed for long-term chemical and physical stabilization of the deposited tailings, including placement of a vegetated soil cover to prevent erosion and dust generation. The potentially acid-generating waste rock stockpiles at the Goliath project will also be covered with a low permeability cover and seeded, with drainage direct to the open pit. The open pit will be allowed to flood, and natural drainage will be re-established to the extent practicable. The remaining disturbed site area will be revegetated. Monitoring at appropriate sampling locations, including those established during baseline studies and operations, will be conducted after closure to confirm performance.

1.16.3 Goldlund Project Site

1.16.3.1 Baseline / Environmental Studies

Environment baseline data collection for the Goldlund project site was initiated in 2020, which built upon basic scoping level aquatic information gathered in 2017. To date, baseline environmental studies have been conducted addressing aspects of surface water quality, aquatic resources (including sediment quality, benthic invertebrate community, fish community and fish habitat), hydrology, hydrogeology and groundwater quality, terrestrial resources, and geochemistry.

The local area of interest delineated for avian and wildlife surveys conducted in 2020 to 2021 was relatively large, covering more than a 5 km buffer around the Goldlund project site. Avian species at risk detected in the local area of interest included bald eagle, barn swallow, Canada warbler, common nighthawk, eastern whip-poor-will, eastern wood pewee, evening grosbeak, and short-eared owl, with only two barn swallows observed within the proposed project footprint at the former mine site. Bat species recorded throughout the local area of interest included eastern red bat, hoary bat, silver-haired bat, little brown myotis and evidence of tri-colored bat; with little brown myotis (recorded at 75% of the 2020 bat survey locations) and tri-colored bat being identified as provincially listed. Black ash, a recently assigned Ontario species at risk, was identified at a number of ecosite stands within the local area of interest and are primarily located southwest of the proposed project site in the area of Tablerock Lake. The MECP has paused the protection of black ash until 2024 while protection and recovery plans are developed.

Geochemical testing of drill core indicated that most of the tested samples (i.e., approximately 98%) were classified as non-potentially acid generating (NPAG, NPR>2). Kinetic testwork is in progress to confirm the potential for metal leaching of project materials.

The existing environmental baseline monitoring programs conducted to date will provide the basis for future monitoring and will be modified to support engineering design work and to meet regulatory and reporting requirements as the project moves through the permitting phase.

1.16.3.2 Permitting Considerations

A number of provincial and federal permitting approvals and authorizations will be required to develop the Goldlund project. Most of these permits and approvals will be the same as those required for the Goliath project. Additional agencies that may be involved in permitting the Goldlund project include the Ministry of Transportation for highway upgrades and/or site entrance requirements, and possibly the Ministry of Tourism, Culture and Sport.

Various infrastructure (e.g., overburden, mine rock, and ore stockpiles) for the Goldlund site is being designed and located to avoid overprinting of watercourses. As such, the requirement for an FAA or Schedule 2 listing under requirements of the MDMER is currently not envisioned.

1.16.3.3 Closure and Reclamation Planning

Previous mining and milling activities at the Goldlund project site ceased in 1985, prior to the enactment of the closure regulations (O.Reg. 240/00) under the *Ontario Mining Act* in 2000. As such, some physical hazards currently remain that will require rehabilitation. Accordingly, a Closure Plan and financial assurance will be required for rehabilitation of the proposed Goldlund project, as presented herein, as well as any remaining physical and/or chemical hazards from earlier mining and milling activities.

Conventional methods of closure are expected to be employed at the Goldlund project site. The waste rock stockpiles will be progressively rehabilitated, with drainage directed to the open pits. The open pits will be allowed to flood, and natural drainage will be re-established to the extent practicable. The remaining disturbed site area will be revegetated. Monitoring at appropriate sampling locations, including those established during baseline studies and operations, will be conducted after closure to confirm performance.

1.16.4 Miller Project

1.16.4.1 Baseline / Environmental Studies

To date, baseline and supporting studies at the Miller project site have been limited and have involved preliminary monitoring programs addressing aspects of surface water quality, aquatic resources (including sediment quality, benthic invertebrate community, fish community and fish habitat), and geochemistry. Preliminary geochemical testing of 50 drill core samples indicated that most of the tested samples (i.e., approximately 98%) were classified as non-potentially acid generating (NPAG, NPR>2). Kinetic testwork is proposed to assess the potential for metal leaching of project materials.

The existing environmental baseline monitoring programs conducted to date will provide the basis for future monitoring and will be modified to support engineering design work and to meet regulatory and reporting requirements as the project moves through the permitting phase. Additional baseline studies, including hydrology, hydrogeology, and terrestrial resources, will also be required.

1.16.4.2 Permitting Considerations

A number of provincial and federal permitting approvals and authorizations will be required for the development of the Miller project. Most of these permits and approvals will be the same as those required for the Goliath project. Additional agencies that may be involved in permitting the Miller project include the Ministry of Transportation for any highway upgrades and/or site entrance requirements, and possibly the Ministry of Tourism, Culture and Sport.

Various infrastructure (e.g., overburden, mine rock, and ore stockpiles) for the Miller site will be designed and located to avoid any overprinting of watercourses. As such, the requirement for an FAA or Schedule 2 listing under requirements of the MDMER is currently not envisioned.

1.16.4.3 Closure and Reclamation Planning

Conventional methods of closure are expected to be employed at the Miller project site. The waste rock stockpiles will be progressively rehabilitated, with drainage directed to the open pit. The open pit will be allowed to flood, and natural drainage will be re-established to the extent practicable. The remaining disturbed site area will be revegetated. Monitoring at appropriate sampling locations, including those established during baseline studies and operations, will be conducted after closure to confirm performance.

1.16.4.4 Social Considerations

Treasury Metals has actively engaged local and regional communities, First Nations, and other stakeholders to gain an understanding of their issues and interests, identify potential partnerships, and build social acceptance for the three projects. Stakeholders involved in project consultations to date include those with a direct interest in the projects, and those who provided data for baseline studies. The involvement of stakeholders will continue throughout the various project stages.

Non-Indigenous public interest groups were identified as part of past, present and future consultation and engagement efforts. This includes the Village of Wabigoon, City of Dryden, Town of Sioux Lookout, and other regional partners and stakeholders.

The three project sites are located within the Treaty 3 (1873) area of Ontario, which affords hunting, trapping and fishing rights and protections, and it has been shared with Treasury Metals that there are areas within the Goliath Gold Complex property boundaries that are used by Indigenous communities for traditional land and resource use. Treasury Metals is committed to working collaboratively with Indigenous and regional communities to ensure informed and engaged dialogue throughout the life of the projects. To date, Treasury Metals has participated in consultation and engagement activities with the following Indigenous communities:

- Asubpeeschoseewagong Netum Anishinabek (Grassy Narrows First Nation)
- Eagle Lake First Nation
- Lac Des Mille Lacs First Nation
- Lac Seul First Nation
- Métis Nation of Ontario
- Naotkamegwanning First Nation
- Wabauskang First Nation
- Wabigoon Lake Ojibway Nation.

Treasury Metals will endeavour to maximize participation with its Indigenous partners wherever possible. Treasury Metals is focused on building and strengthening relationships, integrating traditional knowledge into its decision-making frameworks, and actively communicating and sharing information in a transparent manner.

1.17 Capital Cost Estimates

The capital cost estimate was developed in Q3 2022 from budgetary quotations, Ausenco's in-house database of projects and studies, and experience from similar operations to a level of accuracy of ±25% (Class 4). The level of accuracy is in accordance with the Association for the Advancement of Cost Engineering International (AACE International) for prefeasibility studies. The estimate includes mining, processing, utilities, tailings storage facility, project site infrastructure, and project delivery.

The capital cost summary is presented in Table 1-7. The total initial capital cost for the Goliath Gold Complex is $335.0 million and life-of-mine sustaining costs are $197.6 million. Closure costs are additional and are estimated at $28.9 million.

The capital cost estimates are based on the following assumptions and parameters:

- For material sourced in US dollars, an exchange rate of 1.34 Canadian dollar to 1.00 US dollar was assumed.
- No allowance has been made for exchange rate fluctuations.
- No escalation has been added to the estimate.
- A growth allowance was included.

Data for the estimates have been obtained from numerous sources, including the following:

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

Table 1-7: Capital Cost Estimate Summary

WBS2	WBS Description	Initial Capital (C$M)	Sustaining Capital (C$M)	Total Capital (C$M)
11000 12000 21000	Goldlund Mine Miller Mine Goliath Open Pit Mine	63.4	41.7	105.1
22000	Goliath Underground Mine	3.7	91.3	95.0
Mining Total		67.1	133.0	200.1
13000	Goldlund-Miller On-Site Infrastructure	0.0	11.5	11.5
Goldlund-Miller Property Total		$0.0	11.5	11.5
23000	Process Plant	98.6	0.0	98.6
24000	On-Site Infrastructure	75.3	36.6	112.0
Goliath Property Total		173.9	36.6	210.6
31000	Main Access Road Diversion	0.0	1.1	1.1
32000	HV Line Tie-In	0.1	0.0	0.1
33000	Goldlund-Goliath Transport Connection	0.0	5.8	5.8
34000	Watercourse Realignment	2.0	0.0	2.0
35000	Water Management Pipeline	1.8	0.0	1.8
Off-Site Infrastructure Total		3.9	6.9	10.8
41000	Temporary Construction Facilities and Services	20.6	9.7	30.3
42000	Commissioning Reps and Assistance	0.3	0	0.3
43000	Spares (Commissioning, Initial and Insurance)	2.1	0.0	2.1
44000	First Fills & Initial Charges	1.0	0.0	1.0
Project Indirects Total		24.0	9.7	33.7
50000	Project Delivery	14.3	0.0	14.3
60000	Owner's Costs	16.6	0.0	16.6
Project Delivery and Owner's Costs Total		30.9	0.0	30.9
71000	Contingency	35.1	0.0	35.1
Grand Total		$335.0	$197.6	$532.7

1.18 Operating Cost Estimates

The operating cost estimate is presented in Q1 2023 Canadian dollars. The estimate was developed to have an accuracy of ±25%. The estimate includes mining, processing, contracted hauling, and general and administration (G&A) costs. As the Goliath and Goldlund projects have separate power, consumable, and reagent requirements, these elements were derived separately from the shared process plant operating costs. Table 1-8 provides a summary of the project operating costs.

The overall life-of-mine operating cost is C$1,446.6 million over 13 years, or an average of C$47.7/t milled in a typical year. Of this total, processing and G&A account for C$343.9 million and mining accounts for C$995.3 million.

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2022 pricing without allowances for inflation.

- For material sourced in US dollars, an exchange rate of 1.34 Canadian dollar to 1.00 US dollar was assumed.
- The estimated cost for diesel is C$1.0/L.
- Annual power costs were calculated using a unit price of C$0.07/kWh.
- Labour is assumed to come from the local area of highly skilled workers in Dryden, Wabigoon and Dinorwic.

Table 1-8: Operating Cost Estimate Summary

Cost Centre	C$/t Milled (Over LOM)	LOM Operating Cost (C$M)
Mining		
Open Pit Mining	15.4	467.2
Underground Mining	13.5	409.2
Goldlund Ore Haulage to Mill	3.9	119.0
Mining Subtotal	32.8	995.3
Process Plant		
Reagents	3.2	97.7
Consumables	2.7	81.4
Plant Maintenance	0.6	17.9
Power	2.4	72.7
Laboratory	0.1	2.1
Labour – Process Plant	2.4	72.1
Process Plant Subtotal	11.3	343.9
G&A		
G&A Expenses	1.5	45.1
Mobile Equipment	0.2	6.7
Effluent Treatment Plant	1.8	55.6
G&A Subtotal	3.5	107.4
Total Project Operating Costs	47.7	1,446.6

Note: *Mining operating unit cost is listed on a per tonne milled basis. The mining operating unit cost per tonne mined is listed in Table 1-8.

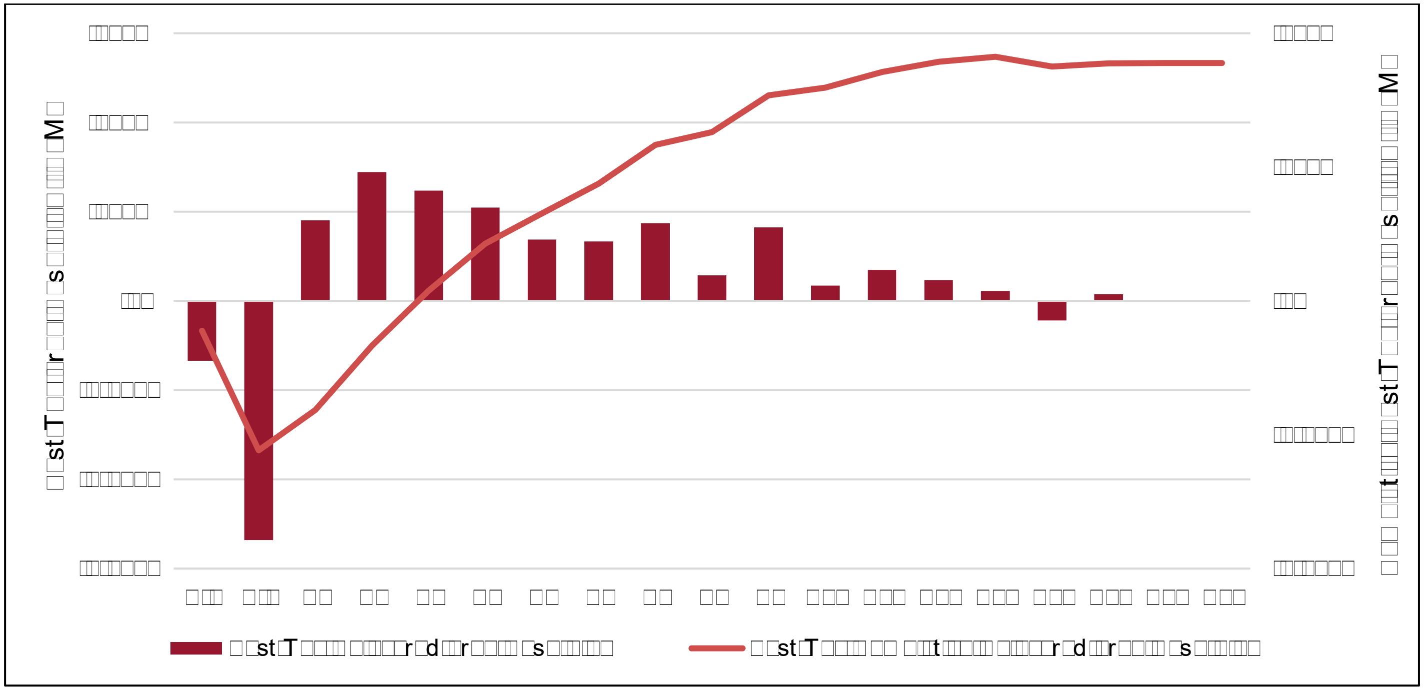
1.19 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. Cash flows have been discounted to the start of construction, assuming that the project execution decision will be taken, and major project financing will be carried out at this time.

The pre-tax NPV discounted at 5% is $469 million; the IRR is 29.3%, and payback period is 2.8 years. On a post-tax basis, the NPV discounted at 5% is $336 million; the IRR is 25.4%; and the payback period is 2.8 years.

A summary of the post-tax project economics is shown graphically in Figure 1-3 and listed in Table 1-9.

Figure 1-3: Post-Tax Project Economics



Source: Ausenco, 2023.

Table 1-9: Economic Analysis Summary

Description	Unit	Life-of-Mine Total / Average
General Assumptions		
Gold Price	US$/oz	1,750
Silver Price	US$/oz	21
Discount Rate	%	5.0%
Exchange Rate	USD:CAD	0.75
Production		
Mill Head Grade Au	g/t	1.30
Mill Head Grade Ag	g/t	1.77
Mill Recovery Rate Au	%	92.8%
Mill Recovery Rate Ag	%	60.0%
Total Mill Ounces Recovered Au	koz	1,175
Total Mill Ounces Recovered Ag	koz	1,034
Total Average Annual Production Au	koz	90
Total Average Annual Production Ag	koz	80
Operating Costs		
Open Pit Mining Cost	C$/t mined	4.22
Underground Mining Cost	C$/t mined	61.23
Mining Cost (Open Pit + Underground)	C$/t milled	32.83
Goldlund Ore Haulage to Mill	C$/t milled	7.00
Processing Cost	C$/t milled	11.34
G&A Cost	C$/t milled	3.54
Refining and Transport Au	C$/oz Au	5.00
Refining and Transport Ag	C$/oz Ag	0.26
Total Operating Costs	C$/t milled	47.71
Cash Costs and All-in Sustaining Costs (By-Product Basis)		
Operating Cash Costs*	US$/oz Au	935
All-in Sustaining Cost **	US$/oz Au	1,072
Capital Expenditures		
Initial Capital Cost	C$M	335
Sustaining Capital Cost	C$M	198
Closure Capital Cost	C$M	29
Salvage Value	C$M	10
Economics		
Pre-tax NPV @ 5%	C$M	469
Pre-tax IRR	%	29.3%
Pre-tax Payback	years	2.8
Post-tax NPV @ 5%	C$M	336
Post-tax IRR	%	25.4%
Post-tax Payback	years	2.8

Note: * Cash costs consist of mining costs, processing costs, G&A and refining charges and royalties. Cash cost is calculated on a by-product basis. ** AISC includes cash costs plus sustaining capital, closure costs and salvage value. AISC is calculated on a by-product basis. Source: Ausenco, 2023.

A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using the following variables: gold price, foreign exchange, total operating cost, initial capital cost. Table 1-10 summarizes the post-tax sensitivity analysis results.

Table 1-10: Post-Tax Sensitivity Summary

Gold Price	Post-Tax NPV(5%)	Initial Capital Cost		Total Operating Cost		Foreign Exchange	
US$/oz	Base Case	(-20%)	(+20%)	(-20%)	(+20%)	(-20%)	(+20%)
$1,550	$178	$242	$114	$321	$30	$486	($52)
$1,600	$218	$282	$153	$361	$73	$535	($4)
$1,750	$336	$400	$271	$479	$192	$682	$103
$1,900	$453	$518	$389	$596	$310	$829	$202
$2,000	$532	$596	$467	$675	$389	$928	$268
Gold Price	**Post-Tax IRR**	**Initial Capital Cost**		**Total Operating Cost**		**Foreign Exchange**	
US$/oz	**Base Case**	**(-20%)**	**(+20%)**	**(-20%)**	**(+20%)**	**(-20%)**	**(+20%)**
$1,550	16.6%	23.8%	11.4%	24.0%	7.2%	33.2%	1.0%
$1,600	18.9%	26.5%	13.5%	26.1%	10.3%	35.6%	4.8%
$1,750	25.4%	34.2%	19.3%	32.1%	17.9%	42.5%	11.9%
$1,900	31.6%	41.4%	24.6%	37.7%	24.7%	49.3%	18.0%
$2,000	35.4%	45.9%	28.0%	41.4%	29.0%	53.7%	21.7%

Source: Ausenco, 2023.

1.20 Risks and Opportunities

Section 25.15 identifies project risks and opportunities.

Project risks include:

- Inflation of costs beyond the growth allowances, contingencies and discount rates used in the project financial assessment,
- Gold prices below those anticipated in the project financials and sensitivities,
- Site geotechnical and hydrologic conditions that require a change to foundation and TSF design, and
- Requirements for water management beyond the assumptions used in the study.

Project opportunities Include:

- Improvements in gold recovery and reduced costs through value engineering and metallurgical studies of ore processing.
- Reduced open pit mining costs through optimized pit slope designs after geotechnical studies of the Goldlund pit.
- Reduced underground sustaining capital costs through further optimization of underground infrastructure designs and level spacings.

1.21 Interpretation and Conclusions

As detailed in Section 25 of this report, the Goliath Gold Complex contains an economic resource that has been converted to a mineral reserve through application of pre-feasibility levels of engineering design and project costing. The undertaking carries an acceptable level of risk and generates free cash flow and return on investment at current estimates of cost and revenue.

1.22 Recommendations

1.22.1 Metallurgical Testwork

Additional comminution tests (e.g., SMC, Bond ball work index, and abrasion index) are recommended on samples from both the Goliath and Goldlund deposits over a range of lithologies or zones to minimize risk in the crushing and grinding circuit design.

Further leaching testwork should be conducted at a grind size of 105 um for Goliath and 85 um for Goldlund to confirm the findings of this report which were interpolated over a range of grind sizes. Tests for the Goldlund deposit should be run at both telluride and conventional cyanide leaching conditions and include measurement of the deportment of gold within telluride bearing minerals. Oxygen uptake testing should also be completed these samples to confirm the oxygen consumption requirements in leaching across the deposit.

The feasibility study metallurgical testwork program should also include additional point samples representing a variety of mineralogies in the two primary deposits to understand the recovery behaviour. It is recommended that these samples undergo gravity-leach testwork to investigate reagent addition and recovery behaviour, and that the cyanidation tailings should complete vendor thickener tests to ensure accurate equipment sizing. Specific attention should be given to understanding the metallurgical responses within both Zones 1 and 4.

1.22.2 Recovery Methods

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

- Material handling testing to support process design. The results and recommendations should be incorporated into the crushing and stockpile circuit detailed design.
- Conduct additional comminution testing on additional variability samples from Goliath and Goldlund to better understand hardness variability and minimize throughput risk in the crushing and grinding circuit designs.
- Conduct validation testing at the selected grind size to confirm the interpretation used for process design and recovery estimates.
- Additional downstream testwork on telluride-bearing zones and lithologies to understand recovery behaviour at various operating conditions.

The cost of these items is covered under metallurgical testwork, as a result zero estimation is considered for this section.

During the next phase of study, additional process design work should be performed to produce capital and operating cost estimates with an accuracy of ±15% (AACE International Class 4).

1.22.3 Water Management

The following work related to water balance and water management is recommended for completion during the feasibility-level study:

- Identify specific effluent water treatment requirements and solutions based on the ongoing geochemical characterization and the results of the water quality model to be updated with the new geochemical data.
- Conduct groundwater numerical modelling to inform pit groundwater inflow rates through the operating and closure phases based on modelling predictions. The model will determine the transient inflow rate during both the initial stages of dewatering and as the pit depth is increased.
- Conduct a better definition of underground mine dewatering rates.
- Develop a stochastic flow (water balance) model to simulate the water management strategy with greater detail incorporating decision making functions and considering probabilistic climatic scenarios.
- Carry out a geomorphology assessment for Blackwater Creek to define the allowable effluent discharge flow and/or period to release treated water to the creek.
- Identify potential alternative discharge locations via a piped outlet to Wabigoon Lake and carry out assimilative capacity assessment downstream of the potential discharge location.
- Prepare water quality estimates for the mine closure phase including the open pit and other key mine features.

1.22.4 Tailings Management

Work that will be required to advance the design includes the following:

- Hydrogeological investigations and dam seepage modelling to ensure compatibility between the dam seepage control measures and the assimilative capacity of the receivers.
- Tailings rheology and geotechnical characterization.
- Geotechnical and geophysical investigations and analyses to confirm the natural clay continuity, constructability of the geomembrane anchor trench, and potential for seismic liquefaction of the outwash sand dam foundations.

1.22.5 Geochemistry

The following recommendations are provided to advance the Project ML/ARD assessment

- Continue to advance ongoing geochemical studies for the Goliath, Goldlund and Miller projects to refine currently available data regarding the ML/ARD potential of mine materials and associated mine waste / water management needs for operations and closure. This should include laboratory testwork, field testwork, and geochemical modelling.
- Undertake further assessment of the potential risk of producing a PAG and/or metal leaching tailings with proposed ore feed profiles.

- Conduct additional tailings geochemical assessment to evaluate the proposed deposition strategy to mitigate ML/ARD risks during mine operations and the potential performance of closure cover concepts to maintain long term geochemical stability of the tailings.

- Evaluate geochemically suitable (i.e., non-potentially acid generating and non-metal leaching) sources of rockfill and borrow materials for use in TSF construction.

- Advance water quality estimates including updates to the Goliath water quality estimates when additional geochemical data are available and prepare water quality estimates for Goldlund and Miller projects.

Additional geochemical assessment, beyond what is outlined here, may be required pending the outcome of these recommendations.

2 INTRODUCTION

2.1 Introduction

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) for Treasury Metals Inc. (Treasury Metals) to summarize the results of a prefeasibility study (PFS) of the Goliath Gold Complex. The individuals presented in Section 2.3, by virtue of their education, experience, and professional association, are considered Qualified Persons (QPs) as defined by NI 43-101 (CIM, 2014). The QPs meet the requirement of independence defined in NI 43-101.

The responsibilities of the engineering consultants are as follows:

- Ausenco Engineering Canada Inc. (Ausenco) was commissioned by Treasury Metals to manage and coordinate the work related to the PFS study based on NI 43-101. Ausenco also develop the PFS-level design and cost estimate for the process plant and general site infrastructure.
- SRK Mining Consulting Canada Inc. (SRK) was commissioned to complete the mineral resource estimate for the Goliath, Goldlund and Miller projects, and to design the open pit and underground mine plan, mine production schedule, and mine capital and operating costs.
- SLR Consulting (Canada) Ltd (SLR) was commissioned to complete and tailings storage and site wide water management for the Goliath, Goldlund, and Miller projects to design and estimate different water modeling.
- Minnow Environmental Inc. (Minnow) was commissioned to complete environmental studies and permitting management for the Goliath, Goldlund, and Miller projects.
- RockEng Inc. (RockEng) was commissioned to complete mining geotechnical studies for the Goliath, Goldlund, and Miller projects to provide information regarding geotechnical aspects of open pit and underground mining.
- WSP E&I Canada Limited (WSP) was commissioned to complete the geochemistry studies for the Goliath, Goldlund, and Miller projects to conduct and assess the geochemical tests on tailings.

The MRE was prepared in accordance with NI 43-101 Standards of Disclosure for Mineral Projects and are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

2.2 Qualified Persons

The qualified persons (QPs) for this technical report, as defined in NI 43-101 and in accordance with Form 43-101F1, are listed in Table 2-1.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Discovery?	Report Section
Tommaso Roberto Raponi	P. Eng (ON)	Senior Mineral Processing Specialist	Ausenco Engineering Canada, Inc.	Yes	1.1, 1.2, 1.3, 1.10, 1.14, 1.15.1, 1.17 to 1.22.1, 1.22.2, 2, 3, 13, 17, 18.1 to 18.5, 19, 21.1, 21.2.1 to 21.2.4, 21.2.6 to 21.2.13, 21.3.1 (except 21.3.1.2), 21.3.2, 21.4.1, 21.4.2, 21.5, 21.6.1, 21.6.3, 21.7, 22, 23, 24, 25.1, 25.5, 25.8, 25.9, 25.11, 25.12.2, 25.13.2, 25.14, 25.15.1, 25.16, 26.1, 26.2, 26.3 and 27
Dr. Gilles Arseneau	P. Geo.	Associate Consultant	SRK Consulting Inc. (Canada)	Yes	1.4 to 1.9, 1.11, 4 to 12, 14, 25.2 to 25.4, 25.6, 25.15.2
Sean Kautzman	P. Eng	Principal Consultant	SRK Consulting Inc. (Canada)	Yes	1.11 to 1.13, 15, 16.1, 16.3, 16.6, 21.3.3, 21.4.1, 21.6.2, 25.7, 25.12.1, 25.13.1, 25.15.2, 25.15.3
Colleen MacDougall	P. Eng	Principal Consultant	SRK Consulting Inc. (Canada)	Yes	1.12 to 1.13, 15, 16.1, 16.4, 16.6, 21.2.5, 21.6.2, 25.7, 25.12.1, 25.13.1, 25.15.3
David Ritchie	P. Eng	Managing Principal	SLR Consulting (Canada) Ltd.	Yes	1.15.2, 1.22.4, 18.6,18.7 (except 18.7.5), 21.3.1.2, 26.4
Luis Vasquez	P. Eng	Principal Hydrotechnical Engineer	SLR Consulting (Canada) Ltd.	Yes	1.15.3, 1.22.3, 18.8.1, 18.8.2, 26.3
Debbie Dyck	P. Eng	Environmental Engineer, Principal Environmental Approvals Specialist	Minnow Environmental Inc.	Yes	1.16, 20 (except 20.2.1.6, and 20.3.1.6), 20.4.1, and 25.10
Kathy Kalenchuk	P. Eng	President and Principal Consultant	RockEng Inc.	Yes	16.2, 16.3, 16.4
Kristen Gault	P.Geo.	Associate Geochemist	WSP E&I Canada Limited	Yes	1.22.5, 18.7.5, 20.2.1.6, 20.3.1.6, 20.4.1, 26.5

Source: Ausenco, 2023.

2.3 Terms of Reference

The report supports disclosures by Treasury Metals in a news release dated February 22, 2023 entitled, "Treasury Metals Completes Pre-Feasibility Study for Goliath Gold Complex".

All measurement units used in this Report are SI units unless otherwise noted. Currency is expressed in Canadian dollars (currency abbreviation: CAD; symbol: C$).

The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and in accordance with the requirements of Form 43-101 F1. Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (2019).

The Goliath Gold Complex contains three projects—Goliath, Goldlund and Miller—which each have their own deposit. Treasury Metals owns 100% of the Goliath Gold Complex.

2.4 Site Visits and Scope of Personal Inspection

Qualified site inspections are listed in Table 2-2 and briefly described below.

Table 2-2: Qualified Person Site Inspections

Qualified Person	Dates of Site Visit	Days on Site	Project
Tommaso Roberto Raponi	July 07 and 08 2021	2	Goliath, Goldlund and Miller
Dr. Gilles Arseneau, P. Geo.	July 07 and 08 2021	2	Goliath, Goldlund and Miller
Sean Kautzman	July 07 and 08 2021	2	Goliath, Goldlund and Miller
Colleen MacDougall	July 07 and 08 2021	2	Goliath, Goldlund and Miller
David Ritchie	July 07 and 08 2021	2	Goliath, Goldlund and Miller
Luis Vasquez	July 07 and 08 2021	2	Goliath, Goldlund and Miller
Debbie Dyck, P.Eng.	October 25 and 26 2022	2	Goliath and Goldlund

- Tommaso Roberto Raponi visited the Goliath complex on July 7 and 8, 2021.
- Dr. Gilles Arseneau visited the Goliath Complex from July 7 to 8, 2021 to review the property geology, exploration program, drillhole collar locations, drilling program, core handling and sample protocols, and diamond drill core were examined for all three deposits. Dr. Arseneau was accompanied by Mr. Adam Larsen, Exploration Manager for Treasury Metal.
- Sean Kautzman visited the Goliath, Goldlund and Miller project sites between July 7-8, 2021 for a visit duration of two days.
- Colleen MacDougall visited the Goliath, Goldlund and Miller project sites between July 7-8, 2021 for a visit duration of two days.

- David Ritchie visited the Goliath Complex in July 2021 for a visit duration of two days.
- Luis Vasquez visited the Goliath complex site between July 7 and 8, 2021 for a visit duration of two days.
- Debbie Dyck, P.Eng., visited the Goliath and Goldlund project sites between October 25 and 26, 2022 for a visit duration of two days; she did not visit the Miller project site.

2.5 Effective Dates

The effective date of this report is February 22, 2023.

2.6 Information Sources and References

This report is based on internal company reports, maps, published government reports, and public information, as listed in Section 27 of this report. It is also based on the information cited in Section 3.

Treasury Metals has supplied the list of mineral rights and mineral claim maps presented in this report. The QP examined the online GIS website of the Ontario Ministry of Energy, Northern Development and Mines (MENDM), as well as the online site of the Mining Lands Administration System[1] (MLAS), to selectively review, but not verify, these mineral rights.

2.7 Previous Technical Reports

The Goliath Gold Complex and its individual project components have been the subject of several previous technical reports, as summarized in Table 2-3.

[1] https://www.gisapplication.lrc.gov.on.ca/Html5Viewer261/index.html?viewer=mlas.mlas&locale=en-US

Table 2-3: Previous Technical Reports on the Goliath, Goldlund and Miller Projects

Reference	Company	Name
Goliath Gold Complex		
Ausenco, 2021	Treasury Metals	NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex
Goliath Project		
P&E, 2020	Treasury Metals	Amended Updated Mineral Resource Estimate for the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (August 2020)
CSA, 2017	Treasury Metals	Preliminary Economic Assessment Update on the Goliath Gold Project, Kenora Mining Division, Ontario (April 2017)
P&E, 2015	Treasury Metals	Technical Report and Updated Resource Estimate for the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (October 2015)
Roy et al., 2012	Treasury Metals	Preliminary Economic Analyses of the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (July 2012)
Roy et al., 2011	Treasury Metals	Technical Report and Mineral Resource Update on the Goliath Gold Project, Kenora Mining Division, Northwestern Ontario (November 2011)
Roy, 2010	Treasury Metals	Updated Mineral Resource Estimate for the Goliath Deposit (2010)
Roy et al., 2008	Treasury Metals	Report on the Goliath Project, Kenora Mining Division, Northwestern Ontario, Canada (December 2008)
Wetherup, 2008	Treasury Metals	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (April 2008)
Wetherup et al., 2008	Treasury Metals	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (February 2008)
Wetherup et al., 2007	Laramide Resources	Independent Technical Report, Thunder Lake Property, Goliath Project, Kenora Mining Division, Northwestern Ontario (November 2007)
Goldlund and Miller Projects		
WSP, 2020	First Mining	Technical Report Re-issue; Goldlund Gold Project, Sioux Lookout, Ontario (July 2020)
WSP, 2019	First Mining	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (April 2019)
WSP, 2017	Tamaka	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (February 2017)
WSP, 2015	Tamaka	Technical Report and Resource Estimation Update on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (March 2015)
Tetra Tech, 2014	Tamaka	Technical Report and Resource Estimate Update on the Goldlund Deposit, Goldlund Project. Sioux Lookout, Ontario (January 2014)
Tetra Tech, 2013	Tamaka	Technical Report and Resource Estimate on the Goldlund Deposit, Goldlund Project, Sioux Lookout, Ontario (January 2013)
Tetra Tech, 2012	Tamaka	Technical Report and Resource Estimate for the Goldlund Gold Deposit, Sioux Lookout, Ontario (March 2012)
Wardrop, 2011	Tamaka	Technical Report on the Goldlund Property Sioux Lookout, Ontario
Wardrop, 2010	Tamaka	Technical Report and Resource Estimate on the KRP Deposit, Sioux Lookout Ontario (January 2011)
PK Geological Services, 2009	Tamaka	Technical Report and Mineral Inventory Estimate for the Goldlund Group Property, Echo Township, Northwestern Ontario (April 2009)
RPA, 2006	Tamaka	Technical Report on the Goldlund Gold Property, Ontario, Canada (June 2006)

2.8 Units and Name Abbreviations

Table 2-4: Acronyms and Abbreviations

Acronym	Definition
AP	acidification potential
ARD	acid rock drainage
BWi	ball mill work index
CCME	Canadian Council of Ministers of the Environment
CEAA	Canadian Environmental Assessment Act
CFE	concentration of frequent effects
CIP	carbon-in-pulp
CoG	Cut-off grade
CWi	crusher work index
DFO	Oceans and Fisheries Canada
DL	detection limit
DTW	down the hole
E-GRG	Extended gravity recoverable gold
ECCC	Environmental and Climate Change Canada (Federal)
EIA	Environmental Impact Assessment
EQA	Environmental quality act
ETP	Effluent Treatment Plant
GRG	Gravity recoverable gold
GW	ground water
HARD	half absolute relative difference
LOM	Life-of-Mine
M&I	Measured and Indicated
MDMER	Metal and Diamond Mining Effluent Regulations
ML	metal leaching
MRE	Mineral Resource Estimate
NNP	net neutralization point
NP/AP	neutralizing potential / acid potential
NAG	not potentially acid generating
NPV	net present value
NSP	net smelter price
NSR	net smelter return
OEM	Original Equipment Manufacturer
OP	open pit
OVB	Overburden
PEA	Preliminary Economic Assessment
PGA	Potential generator of acid
PFS	Prefeasibility Study
PNN	net neutralizing power
PN	neutralization potential
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
ROM	Run of mine
RWi	bond rod mill work index
SG	specific gravity
TSF	Tailings Storage Facility
UG	underground
W:O	Waste to ore ratio

Table 2-5: Units of Measurement

Abbreviation	Definition
CAD	Canadian dollar (symbol: C$)
USD	United States dollar (symbol: US$)
°	degrees
oz/ton Au	ounces of gold per ton
g/t Au	grams of gold per tonne
cm	centimeter
dBA	A-weighted decibels
ft	feet
g	gram
g/t	grams per tonne
ha	hectare
K	Thousand (C$)
km	kilometer
koz	thousand ounces
kt	thousand tonnes
kt/d	thousand tonnes per day
m	meter
mm	millimeter
Mt/a	Million tonnes per annum
Moz	million ounces
Mt	million tonnes
Mt/y	million tonnes per year
t/d	tonnes per day

3 RELIANCE ON OTHER EXPERTS

3.1 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The QPs have not verified the legal status or legal title to any claims and the legality of any underlying agreements that may exist concerning the Goliath, Goldlund and Miller projects, as described in Section 4 of this report.

The QP's relied upon spreadsheets entitled "Treasury Schedules January, 2022" and "Goldlund Schedules January 2022" as supplied by email from Treasury Metals on March 16, 2023. The QPs relied on these in section 4 and 19.

"Title review, Goliath Property, Province of Ontario" by Cassels, Brock & Blackwell, LLP, January 25, 2022.

3.2 Taxation

Treasury Metals' management compiled the tax calculations for the Goliath Gold Complex with assistance from third party taxation experts.

This information has been relied upon in Sections 1, 22.3, 22, 22, and 25.

3.3 Royalties

The QPs have relied upon the following information when considering royalties in the economic analysis:

- "Ausenco Royalty Memo" received by email sent by Treasury Metals on February 10, 2023.

The information has been relied upon in Sections 1, 22.3.1, 22.6, 22.8, and 25.13.

4 PROPERTY DESCRIPTION AND LOCATION

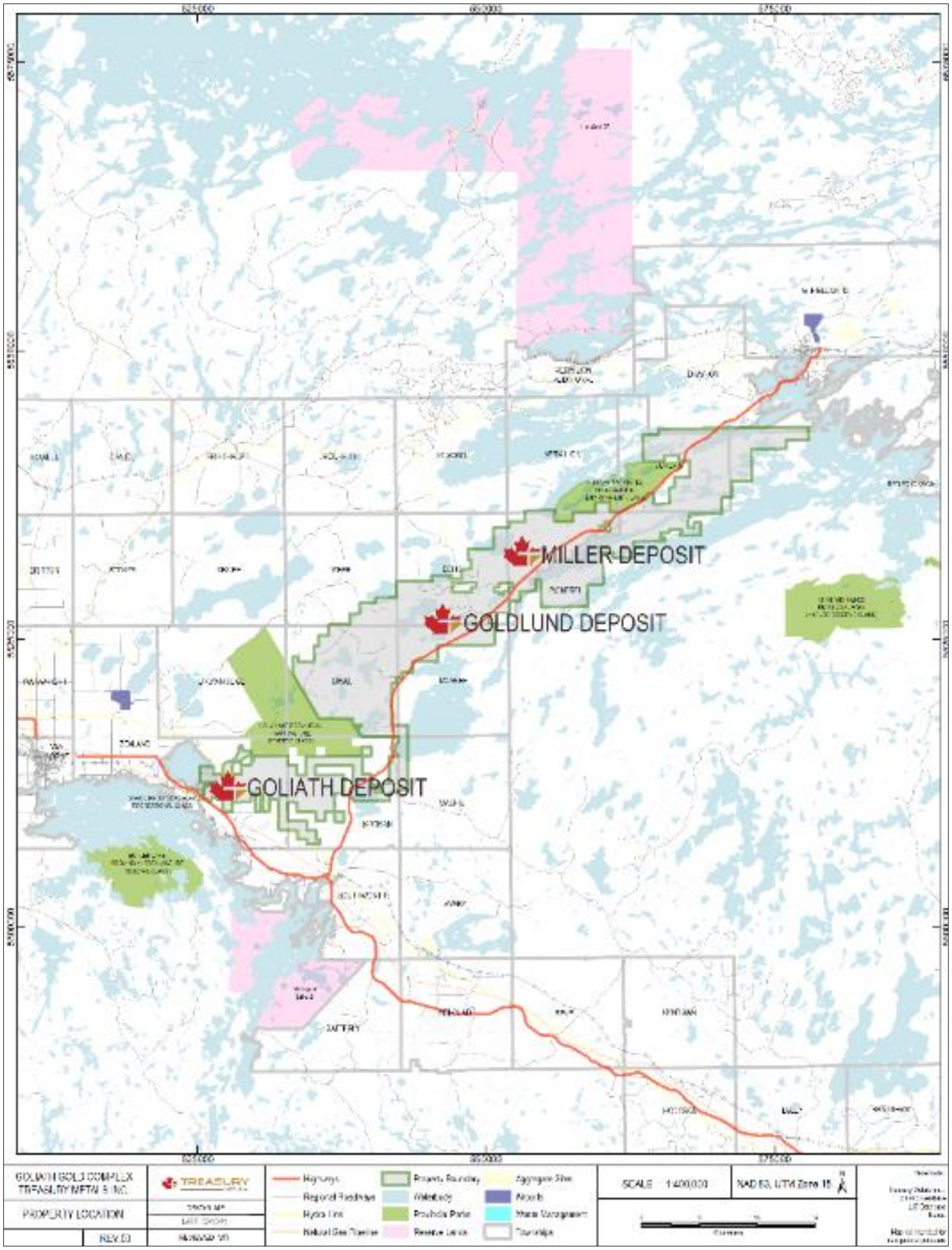
4.1 Introduction

The Goliath Gold Complex is comprised of the Goliath and Goldlund-Miller properties, which together cover approximately 34,719 ha. As shown on Figure 4-1, the Goliath Gold Complex is located approximately 350 km northwest of Thunder Bay in the Northwest Ministry of Natural Resources (MNR) Region. The complex can be found on 1:250,000 scale Mapsheets National Topographic System (NTS) 052F (Dryden) and 052K (Lac Seul). Figures 4-2 and 4-3 on the following pages show the location and tenure of the properties.

Figure 4-1: Location of the Goliath Gold Complex

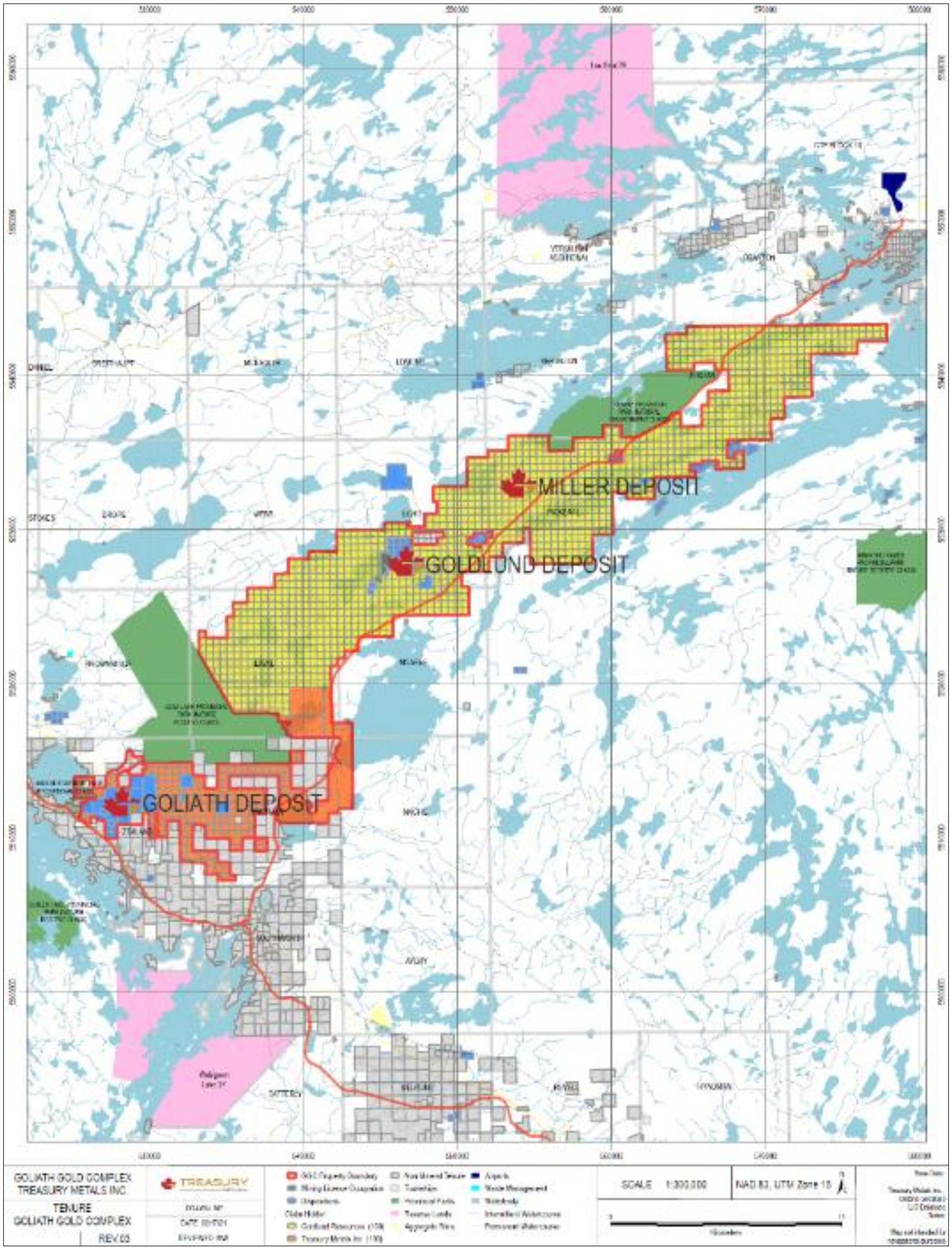
Source: Treasury Metals (2021).

Figure 4-2: Location of the Goliath, Goldlund and Miller Deposits



Source: Treasury Metals (2021).

Figure 4-3: Tenure of the Goliath and Goldlund-Miller Mineral Claims



Source: Treasury Metals (2021).

4.2 Property and Title in Ontario

4.2.1 Mining Cell Claims

In Ontario, Crown lands were available to licensed prospectors for the purposes of mineral exploration prior to 2018. Traditional claim staking in Ontario (post and blazed lines) came to an end on January 8, 2018, and on April 10, 2018 the MNDM converted all existing ground or map-staked mining claims (legacy claims) into one or more cell claims or boundary claims as part of their new provincial grid system. A cell claim was created when one or more legacy mining claims in a cell were held by the same owner. A boundary claim was created when there were multiple legacy claims in cell held by different claim holders. The provincial grid is based on latitude/longitude and is comprised of more than 5.2 million cells ranging in size from 17.7 ha in the north up to 24 ha in the south.

A mining claim remains valid provided the claim holder properly completes and files the assessment work as required by the Mining Act, and the Minister approves the assessment work. A claim holder is not required to complete any assessment work within the first year of recording a mineral claim. In order to keep an unpatented mining claim current, the claim holder must perform (a minimum) $400 worth of approved assessment work per mining claim unit, per year; immediately following the initial staking date, the claim holder has two years to file one year's worth of assessment work. Mining claims are forfeited if the assessment work is not completed.

A claim holder may prospect or carry out mineral exploration on the land under the claim. However, the land covered by these claims must be converted to leases before any development work or mining can be performed.

4.2.2 Mining Lease

Mining leases grant the owner title and ownership to the land and the ability to extract and sell extracted resources. The exact rights conferred under a mining lease vary depending upon the type of lease issued (either mining rights only, surface rights only or both mining and surface rights) and will usually be described in detail, including reservations, under the lease patent document executed by the Crown. Mining leases are granted for 21 years and may be renewed for a further 21 years if the application is made within 90 days of the expiry date. Mining Leases are maintained by an annual rental fee of $3.00/ha (Mining Act, Ontario Regulation 45/11).

Prior to bringing a mine into production, the lessee must comply with all applicable federal and provincial legislation.

4.2.3 License of Occupation

Prior to 1964, Mining Licences of Occupation (MLO) were issued, in perpetuity, by the MNDM to permit the mining of minerals under the beds of bodies of water. MLOs were associated with portions of mining claims overlying adjacent land. As an MLO is held separate and apart from the related mining claim, it must be transferred separately from the transfer of the related mining claim. The transfer of an MLO requires the prior written consent of the Ministry.

MLOs are maintained by an annual rental fee of $5.00/ha (Mining Act, Ontario Regulation 45/11).

4.2.4 Mining Patent

Mining patents are freehold mining claims that permit the patentee to all of the Crown's title to the subject lands and to all mines and minerals relating to such lands, unless something to the contrary is stated in the patent. A mining patent can include surface and mining rights or mining rights only. Since mining patents convey freehold interest in the land subject to the patent, no consents are required for the patentee to transfer or mortgage those lands.

Mining patents were granted to perpetuity provided the taxes on these lands are paid annually.

4.2.5 Goliath Property

The Goliath property covers approximately 7,601 ha and is defined by mineral rights and surface rights that are 100% held by Treasury Metals. Of this total, the mineral rights cover approximately 7,511 ha. The Goliath property has two deposits, the Goliath and Goliath East deposit, and is located as follows:

- on 1:50,000 scale NTS Mapsheets 052F/09 (Dyment), 10 (Wabigoon), 15 (Dryden), and 16 (Big Sandy Lake)
- at approximately 49°45.4′ North and 92°33.0′ West
- at approximately 532,441 mE; 5,511,624 mN, Zone 15U (NAD83 datum) Universal Transverse Mercator (UTM) coordinates
- in the Kenora Mining Division
- in the Dryden MNR District
- in the Zealand and Hartman Townships
- approximately 3.5 km north of Wabigoon
- approximately 15 km east of Dryden
- approximately 145 km east of Kenora
- approximately 2.5 km east of Aaron Provincial Park
- approximately 2.8 km southeast of Lola Lake Provincial Park
- approximately 1.5 km east of Thunder Lake.

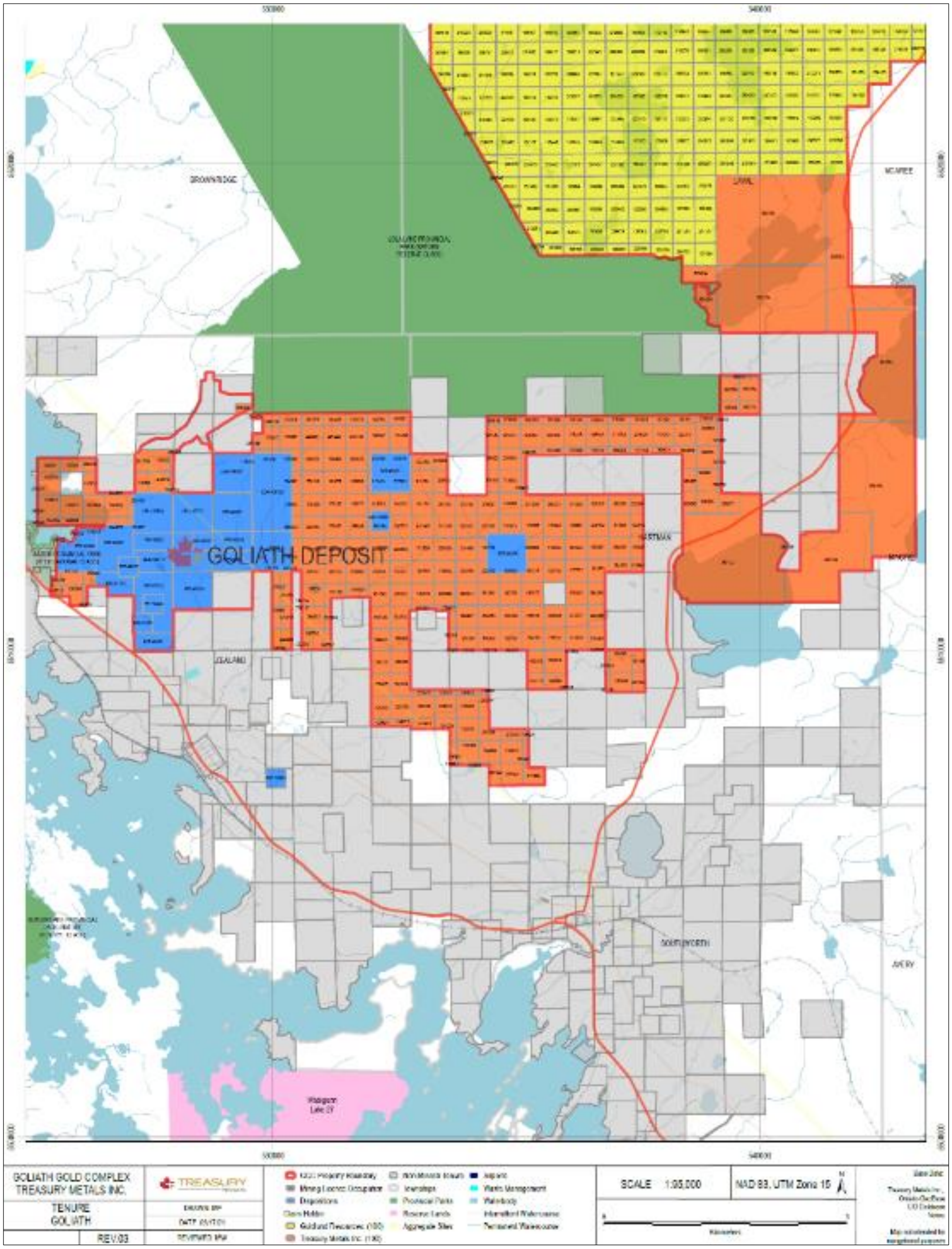
The Goliath property covers approximately 7,601 ha and consists of 284 mining claims totalling approximately 6,254 ha; four mining leases totalling 359.25 ha; and 28 land parcels (includes patented claims) totalling 1,347.189 ha. Of the 1,347.18 ha of the patents and leases, 90.2 ha are surface rights only from seven land parcels. Of the 284 mining claims, 267 are single-cell mining claims, eight are boundary cell mining claims, and nine are multi-cell mining claims. The mineral rights are 100% held by Treasury Metals and all mineral rights are in good standing.

The project is bounded by two provincial parks, as follows:

- The Lola Lake Nature Reserve is located at the northern boundary and was designated a nature reserve class park in 1985.
- Aaron Provincial Park is located at the western boundary of the property on the south shore of Thunder Lake. Aaron Provincial Park is a serviced recreation-class park established in 1958 and is operated in co-operation with the City of Dryden.

Figure 4-4 displays the Goliath property mineral and surface rights, which are summarized in Table 4-1.

Figure 4-4: Goliath Property Mineral Rights Map



Source: Treasury Metals (2020)

Table 4-1: Summary of Mineral Rights for the Goliath Property

Mineral Title	Count	Area (ha)
Mineral Claims	280	6,253.97
Patented Claims (and Tree Farm)	23	1,347.19
Leases	4	359.25
Total	307	7,960.41

Table 4-2: Goliath Mining Claims

Tenure ID	Township / Area	Tenure Type	Anniversary Date
100099	Zealand	Single Cell Mining Claim	2021-05-21
100140	Zealand	Single Cell Mining Claim	2021-02-26
100467	Hartman	Single Cell Mining Claim	2021-07-10
100483	Hartman	Single Cell Mining Claim	2021-08-21
100549	Zealand	Single Cell Mining Claim	2021-10-26
100562	Hartman	Single Cell Mining Claim	2021-07-10
100770	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
101188	Hartman	Single Cell Mining Claim	2021-02-28
101335	Hartman	Single Cell Mining Claim	2021-02-28
101428	Zealand	Single Cell Mining Claim	2021-02-26
101574	Hartman	Single Cell Mining Claim	2021-04-02
101679	Zealand	Single Cell Mining Claim	2021-05-21
101700	Hartman	Single Cell Mining Claim	2021-02-26
101742	Zealand	Single Cell Mining Claim	2021-02-26
101762	Hartman	Single Cell Mining Claim	2021-07-10
101763	Hartman	Single Cell Mining Claim	2021-08-21
101836	Hartman	Single Cell Mining Claim	2021-08-21
101838	Hartman	Single Cell Mining Claim	2021-08-21
101876	Hartman	Single Cell Mining Claim	2021-02-26
101878	Zealand	Single Cell Mining Claim	2021-10-26
101879	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
101992	Hartman	Single Cell Mining Claim	2021-02-26
103900	Zealand	Single Cell Mining Claim	2021-07-04
103904	Zealand	Boundary Cell Mining Claim	2021-07-04
115735	Zealand	Single Cell Mining Claim	2021-09-10
115838	Zealand	Single Cell Mining Claim	2021-10-26
115843	Hartman	Single Cell Mining Claim	2021-02-26
115974	Hartman	Single Cell Mining Claim	2021-02-26
115977	Zealand	Single Cell Mining Claim	2021-10-26
116125	Hartman	Single Cell Mining Claim	2021-08-21
116126	Hartman	Single Cell Mining Claim	2021-08-21
116189	Hartman	Single Cell Mining Claim	2021-02-28
116190	Hartman	Single Cell Mining Claim	2021-02-28
116250	Hartman	Single Cell Mining Claim	2021-07-10
116252	Zealand	Single Cell Mining Claim	2021-10-10
116253	Zealand	Single Cell Mining Claim	2021-10-10

Tenure ID	Township / Area	Tenure Type	Anniversary Date
116670	Hartman	Single Cell Mining Claim	2021-02-28
117149	Hartman	Single Cell Mining Claim	2021-08-21
117151	Zealand	Single Cell Mining Claim	2021-09-10
117702	Hartman	Single Cell Mining Claim	2021-07-10
117809	Hartman	Single Cell Mining Claim	2021-07-10
119174	Zealand	Single Cell Mining Claim	2021-07-04
119175	Zealand	Single Cell Mining Claim	2021-05-21
120432	Hartman	Single Cell Mining Claim	2021-02-28
120433	Hartman	Single Cell Mining Claim	2021-02-28
120537	Zealand	Single Cell Mining Claim	2021-05-21
121008	Zealand	Single Cell Mining Claim	2021-10-26
121756	Hartman	Single Cell Mining Claim	2021-04-02
121788	Hartman	Single Cell Mining Claim	2021-08-21
122427	Hartman	Single Cell Mining Claim	2021-07-10
122428	Hartman	Single Cell Mining Claim	2021-07-10
122429	Hartman	Single Cell Mining Claim	2021-07-10
123846	Hartman	Single Cell Mining Claim	2021-07-10
123847	Hartman	Single Cell Mining Claim	2021-07-10
123848	Hartman	Single Cell Mining Claim	2021-07-10
124944	Zealand	Single Cell Mining Claim	2021-10-26
128265	Hartman	Single Cell Mining Claim	2021-02-26
142114	Hartman	Single Cell Mining Claim	2021-02-26
142115	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
142700	Zealand	Single Cell Mining Claim	2021-02-26
142709	Hartman	Single Cell Mining Claim	2021-07-10
143486	Hartman	Single Cell Mining Claim	2021-04-02
145344	Hartman	Single Cell Mining Claim	2021-08-21
145345	Hartman	Single Cell Mining Claim	2021-08-21
145357	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
145372	Hartman	Single Cell Mining Claim	2021-02-26
152355	Hartman	Single Cell Mining Claim	2021-07-10
155460	Zealand	Single Cell Mining Claim	2021-10-26
155517	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
156887	Hartman	Single Cell Mining Claim	2021-08-21
156888	Hartman	Boundary Cell Mining Claim	2021-08-21
157591	Hartman	Single Cell Mining Claim	2021-08-21
157592	Hartman	Single Cell Mining Claim	2021-08-21
158237	Zealand	Single Cell Mining Claim	2021-09-06
158719	Zealand	Single Cell Mining Claim	2021-10-13
158848	Zealand	Single Cell Mining Claim	2021-10-26
159019	Zealand	Single Cell Mining Claim	2021-05-21
159020	Zealand	Single Cell Mining Claim	2021-05-21
159023	Zealand	Single Cell Mining Claim	2021-07-04
160968	Hartman	Boundary Cell Mining Claim	2021-02-28
162896	Hartman	Single Cell Mining Claim	2021-08-21
162897	Hartman	Single Cell Mining Claim	2021-08-21
162898	Hartman	Boundary Cell Mining Claim	2021-08-21

Tenure ID	Township / Area	Tenure Type	Anniversary Date
163600	Zealand	Single Cell Mining Claim	2021-10-13
163618	Hartman	Single Cell Mining Claim	2021-08-21
163620	Hartman	Single Cell Mining Claim	2021-08-21
163621	Hartman	Single Cell Mining Claim	2021-08-21
165122	Zealand	Single Cell Mining Claim	2021-05-21
166184	Zealand	Single Cell Mining Claim	2021-10-26
166860	Hartman	Single Cell Mining Claim	2021-02-26
166903	Hartman	Single Cell Mining Claim	2021-08-21
166956	Hartman	Single Cell Mining Claim	2021-02-28
168892	Hartman	Single Cell Mining Claim	2021-08-21
170773	Zealand	Single Cell Mining Claim	2021-10-26
170924	Zealand	Single Cell Mining Claim	2021-09-06
171448	Hartman	Single Cell Mining Claim	2021-02-26
171516	Zealand	Single Cell Mining Claim	2021-10-26
171530	Zealand	Single Cell Mining Claim	2021-10-26
171538	Hartman	Single Cell Mining Claim	2021-02-26
171539	Hartman	Single Cell Mining Claim	2021-02-26
178429	Hartman	Single Cell Mining Claim	2021-04-02
178444	Hartman	Single Cell Mining Claim	2021-02-28
178447	Hartman	Single Cell Mining Claim	2021-07-10
179643	Zealand	Single Cell Mining Claim	2021-10-26
179793	Zealand	Single Cell Mining Claim	2021-05-21
180381	Hartman	Single Cell Mining Claim	2021-07-10
180382	Hartman	Single Cell Mining Claim	2021-08-21
181126	Hartman	Single Cell Mining Claim	2021-02-28
181673	Hartman	Single Cell Mining Claim	2021-07-10
184571	Zealand	Single Cell Mining Claim	2021-05-21
194876	Hartman	Single Cell Mining Claim	2021-02-28
194877	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
196227	Hartman	Single Cell Mining Claim	2021-02-28
196284	Hartman	Single Cell Mining Claim	2021-07-10
198260	Hartman	Single Cell Mining Claim	2021-07-10
200046	Zealand	Single Cell Mining Claim	2021-10-26
200163	Zealand	Single Cell Mining Claim	2021-05-21
200790	Hartman	Single Cell Mining Claim	2021-02-26
202710	Hartman	Single Cell Mining Claim	2021-08-21
203359	Zealand	Single Cell Mining Claim	2021-10-26
203374	Hartman	Single Cell Mining Claim	2021-02-26
203386	Zealand	Single Cell Mining Claim	2021-10-13
203405	Hartman	Single Cell Mining Claim	2021-08-21
203406	Hartman	Single Cell Mining Claim	2021-08-21
203427	Hartman	Single Cell Mining Claim	2021-02-26
203493	Hartman	Single Cell Mining Claim	2021-02-26
204916	Hartman	Single Cell Mining Claim	2021-08-21
205715	Hartman	Single Cell Mining Claim	2021-08-21
208177	Zealand	Single Cell Mining Claim	2021-10-26
208830	Hartman	Single Cell Mining Claim	2021-02-26

Tenure ID	Township / Area	Tenure Type	Anniversary Date
208878	Hartman	Single Cell Mining Claim	2021-02-26
209519	Zealand	Single Cell Mining Claim	2021-10-26
211475	Zealand	Single Cell Mining Claim	2021-10-13
211495	Hartman	Single Cell Mining Claim	2021-02-26
211498	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
211510	Hartman	Single Cell Mining Claim	2021-02-26
211511	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
211536	Hartman	Single Cell Mining Claim	2021-04-02
212763	Hartman	Single Cell Mining Claim	2021-02-26
213494	Hartman	Single Cell Mining Claim	2021-04-02
213513	Hartman	Single Cell Mining Claim	2021-02-28
213514	Hartman	Single Cell Mining Claim	2021-02-28
213520	Zealand	Single Cell Mining Claim	2021-02-26
214844	Hartman	Single Cell Mining Claim	2021-08-21
214899	Zealand	Single Cell Mining Claim	2021-05-21
214921	Hartman	Single Cell Mining Claim	2021-02-28
214922	Hartman	Single Cell Mining Claim	2021-02-28
215649	Hartman	Single Cell Mining Claim	2021-02-28
215650	Hartman	Single Cell Mining Claim	2021-02-28
215651	Hartman	Single Cell Mining Claim	2021-02-28
215731	Hartman	Single Cell Mining Claim	2021-07-10
215732	Hartman	Single Cell Mining Claim	2021-07-10
215736	Zealand	Single Cell Mining Claim	2021-10-10
217007	Hartman	Single Cell Mining Claim	2021-02-28
219135	Zealand	Single Cell Mining Claim	2021-09-06
220280	Zealand	Single Cell Mining Claim	2021-09-10
220882	Zealand	Single Cell Mining Claim	2021-10-26
220897	Zealand	Single Cell Mining Claim	2021-02-26
220966	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
223002	Hartman	Single Cell Mining Claim	2021-04-02
223545	Hartman	Single Cell Mining Claim	2021-08-21
223546	Hartman	Single Cell Mining Claim	2021-08-21
223547	Hartman	Single Cell Mining Claim	2021-08-21
223551	Hartman	Single Cell Mining Claim	2021-02-26
223552	Zealand	Single Cell Mining Claim	2021-09-10
224392	Zealand	Single Cell Mining Claim	2021-05-21
225528	Hartman	Single Cell Mining Claim	2021-02-28
225529	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
225532	Hartman	Single Cell Mining Claim	2021-02-26
227552	Zealand	Single Cell Mining Claim	2021-10-26
227569	Zealand	Single Cell Mining Claim	2021-05-21
227611	Hartman, Zealand	Single Cell Mining Claim	2021-02-26
228203	Hartman	Single Cell Mining Claim	2021-07-10
228246	Zealand	Single Cell Mining Claim	2021-10-26
230308	Hartman	Single Cell Mining Claim	2021-08-21
230309	Hartman	Single Cell Mining Claim	2021-08-21
232298	Hartman	Single Cell Mining Claim	2021-02-26

Tenure ID	Township / Area	Tenure Type	Anniversary Date
232299	Hartman	Single Cell Mining Claim	2021-02-26
233657	Zealand	Single Cell Mining Claim	2021-05-21
234263	Hartman	Single Cell Mining Claim	2021-07-10
234264	Hartman	Single Cell Mining Claim	2021-07-10
235594	Hartman	Single Cell Mining Claim	2021-07-10
244573	Zealand	Single Cell Mining Claim	2021-05-21
244574	Zealand	Boundary Cell Mining Claim	2021-07-04
244575	Zealand	Single Cell Mining Claim	2021-05-21
244581	Zealand	Single Cell Mining Claim	2021-07-04
258276	Hartman	Single Cell Mining Claim	2021-08-21
258277	Hartman	Boundary Cell Mining Claim	2021-08-21
259461	Zealand	Single Cell Mining Claim	2021-10-13
259462	Hartman	Single Cell Mining Claim	2021-07-10
259479	Hartman	Single Cell Mining Claim	2021-08-21
259480	Hartman	Single Cell Mining Claim	2021-08-21
259609	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
261579	Zealand	Single Cell Mining Claim	2021-02-26
261732	Zealand	Single Cell Mining Claim	2021-07-04
262955	Hartman	Single Cell Mining Claim	2021-07-10
264269	Hartman	Single Cell Mining Claim	2021-07-10
264890	Hartman	Single Cell Mining Claim	2021-07-10
266791	Zealand	Single Cell Mining Claim	2021-05-21
266792	Zealand	Single Cell Mining Claim	2021-05-21
266823	Zealand	Single Cell Mining Claim	2021-02-26
268968	Hartman	Single Cell Mining Claim	2021-02-28
269068	Zealand	Single Cell Mining Claim	2021-05-21
269069	Zealand	Single Cell Mining Claim	2021-05-21
270316	Zealand	Single Cell Mining Claim	2021-05-21
270317	Zealand	Single Cell Mining Claim	2021-05-21
270918	Zealand	Single Cell Mining Claim	2021-10-10
272360	Hartman	Single Cell Mining Claim	2021-07-10
274210	Zealand	Single Cell Mining Claim	2021-09-10
274292	Zealand	Single Cell Mining Claim	2021-09-06
274756	Zealand	Single Cell Mining Claim	2021-02-26
275399	Hartman	Single Cell Mining Claim	2021-02-26
276115	Zealand	Single Cell Mining Claim	2021-10-26
277517	Zealand	Single Cell Mining Claim	2021-09-10
278095	Hartman	Single Cell Mining Claim	2021-07-10
278990	Zealand	Single Cell Mining Claim	2021-10-26
279027	Hartman	Single Cell Mining Claim	2021-04-02
279036	Hartman	Single Cell Mining Claim	2021-08-21
279038	Hartman	Single Cell Mining Claim	2021-08-21
279039	Hartman	Single Cell Mining Claim	2021-08-21
280381	Zealand	Boundary Cell Mining Claim	2021-05-21
281028	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
281029	Hartman	Single Cell Mining Claim	2021-02-28
282941	Hartman	Single Cell Mining Claim	2021-02-28

Tenure ID	Township / Area	Tenure Type	Anniversary Date
283008	Zealand	Single Cell Mining Claim	2021-10-10
283009	Zealand	Single Cell Mining Claim	2021-10-10
284291	Hartman	Single Cell Mining Claim	2021-02-28
284939	Hartman	Single Cell Mining Claim	2021-07-10
286386	Hartman, Zealand	Single Cell Mining Claim	2021-09-06
286872	Zealand	Single Cell Mining Claim	2021-05-21
287545	Zealand	Single Cell Mining Claim	2021-10-26
288175	Zealand	Single Cell Mining Claim	2021-10-26
288878	Hartman	Single Cell Mining Claim	2021-08-21
291656	Hartman	Single Cell Mining Claim	2021-07-10
293697	Zealand	Single Cell Mining Claim	2021-09-06
294225	Zealand	Single Cell Mining Claim	2021-02-26
294226	Zealand	Single Cell Mining Claim	2021-10-26
294231	Hartman	Single Cell Mining Claim	2021-02-26
294256	Hartman	Single Cell Mining Claim	2021-02-26
294962	Zealand	Single Cell Mining Claim	2021-10-26
296862	Hartman	Single Cell Mining Claim	2021-08-21
296863	Hartman	Single Cell Mining Claim	2021-08-21
298333	Hartman	Single Cell Mining Claim	2021-02-28
299048	Zealand	Single Cell Mining Claim	2021-07-04
310719	Zealand	Single Cell Mining Claim	2021-02-26
311313	Zealand	Single Cell Mining Claim	2021-02-26
311320	Hartman	Single Cell Mining Claim	2021-02-26
311331	Hartman	Single Cell Mining Claim	2021-02-26
312677	Hartman	Single Cell Mining Claim	2021-07-10
312746	Zealand	Single Cell Mining Claim	2021-07-10
314065	Hartman	Single Cell Mining Claim	2021-04-02
314095	Hartman	Single Cell Mining Claim	2021-08-21
314096	Hartman	Single Cell Mining Claim	2021-08-21
314097	Zealand	Single Cell Mining Claim	2021-09-10
314104	Hartman, Zealand	Single Cell Mining Claim	2021-07-10
320652	Zealand	Single Cell Mining Claim	2021-10-26
320898	Hartman	Single Cell Mining Claim	2021-08-21
323556	Hartman	Single Cell Mining Claim	2021-07-10
326092	Zealand	Single Cell Mining Claim	2021-10-26
326115	Hartman	Single Cell Mining Claim	2021-07-10
328110	Hartman	Single Cell Mining Claim	2021-04-02
329458	Hartman	Single Cell Mining Claim	2021-08-21
329515	Zealand	Single Cell Mining Claim	2021-05-21
329516	Zealand	Single Cell Mining Claim	2021-05-21
330119	Hartman	Single Cell Mining Claim	2021-07-10
330865	Hartman	Single Cell Mining Claim	2021-07-10
330866	Hartman	Single Cell Mining Claim	2021-02-28
330907	Hartman	Single Cell Mining Claim	2021-07-10
340035	Zealand	Single Cell Mining Claim	2021-05-21
341882	Hartman	Single Cell Mining Claim	2021-02-28
343265	Hartman	Single Cell Mining Claim	2021-07-10

Tenure ID	Township / Area	Tenure Type	Anniversary Date
343267	Zealand	Boundary Cell Mining Claim	2021-10-13
593754	Hartman	Single Cell Mining Claim	2022-06-03
593755	Hartman	Single Cell Mining Claim	2022-06-03
593756	Hartman	Single Cell Mining Claim	2022-06-03
593757	Hartman	Multi-cell Mining Claim	2022-06-03
593758	Hartman	Multi-cell Mining Claim	2022-06-03
593759	Hartman	Multi-cell Mining Claim	2022-06-03
593760	Hartman, MacFie	Multi-cell Mining Claim	2022-06-03
593761	Hartman, Laval, MacFie, McAree	Multi-cell Mining Claim	2022-06-03
593762	Laval, McAree	Multi-cell Mining Claim	2022-06-03
593763	Laval	Multi-cell Mining Claim	2022-06-03
593764	Laval	Multi-cell Mining Claim	2022-06-03
593824	Laval	Multi-cell Mining Claim	2022-06-04

4.2.6 Goldlund-Miller Property

4.2.6.1 Goldlund-Miller Property Location

The Goldlund-Miller property covers approximately 27,118 ha and is defined by mineral rights that are 100% held by Treasury Metals. Two deposits, Goldlund and Miller, comprise the Goldlund-Miller property, as detailed below.

The Goldlund deposit is located as follows:

- on the Goldlund-Miller property
- on 1:50,000 scale NTS Mapsheets 052F16 (Big Sandy Lake), 052K/01 (Hudson) and 052J/04 (Sioux Lookout)
- at approximately 49°54′ North and 92°20.5′ West
- at approximately 547000 E; 5527500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Patricia Mining Division
- in the Sioux Lookout MNR District
- in the Echo and Pickerel Townships
- approximately 40 km southeast of Sioux Lookout (42 km by road)
- approximately 40 km east of Dryden (62 km by road)
- approximately 12 km southeast of Ojibway Provincial Park
- approximately 1.2 km east of Crossecho Lake.

The Miller deposit is located as follows:

- on 1:50,000 scale NTS Mapsheet 052F16 (Big Sandy Lake)
- at approximately 49°57′ North and 92°15′ West

- at approximately 534000 E; 5534500 N, Zone 15U (NAD83 datum) UTM coordinates
- in the Pickerel Township
- approximately 27 km southwest of Sioux Lookout (35 km by road)
- approximately 47 km northeast of Dryden (65 km by road)
- approximately 1.7 km west of Ojibway Provincial Park
- approximately 1.2 km southwest of Little Vermilion Lake.

Refer to Figure 4-2 and Figure 4-3 for the location of the Goldlund and Miller deposits.

4.2.6.2 Goldlund-Miller Property Description

The Goldlund-Miller property consists of 1,349 mining claims totalling approximately 26,634 ha, 26 patented claims totalling 360.97 ha, one mining lease of 48.56 ha, and one licence of occupation of 74.84 ha.

The patented claims and mining lease allow for both mineral rights and surface rights, while the Licence of Occupation allows for mineral rights only.

Table 4-3 presents a summary of the mineral rights for the Goldlund-Miller property; the mining claims are shown in Table 4-4.

Table 4-3: Summary of Mineral Rights for the Goldlund-Miller Property

Minerals Rights	Count	Area (ha)
Mineral Claims	1349	26,633.89
Patented Claims	26	360.97
Mining Lease	1	48.56
Licence of Occupation	1	74.84
Total	N/A	27,118.26

Under the provincial system for mining claims, since January 2018, the 142 legacy claims have been converted into 1,342 single-cell mining claims, six boundary-cell mining claims, and one multi-cell mining claim. In 2022, many of the Goldlund single-cell claims were merged into multi-cell claims. While this reduces the total number of claims of the property, the effective area has remained unchanged as a result of this process.

All mineral rights are in good standing and have been granted extra time to allow for credit distributions due to the large number of claims involved.

The property was previously distributed into nine blocks to help manage exploration information. These divisions, which have been maintained by Treasury Metals, do not reflect any geological differences.The property is shown in Figure 4-5.

Table 4-4: Goldlund-Miller Property Mining Claims

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
100003	Pickerel	Single Cell Mining Claim	2021-04-05	234979	McAree	Single Cell Mining Claim	2020-08-05
100005	Echo	Single Cell Mining Claim	2021-04-26	235044	McAree	Single Cell Mining Claim	2020-09-28
100282	Echo	Single Cell Mining Claim	2021-01-13	235052	Echo	Single Cell Mining Claim	2020-08-11
100468	Pickerel	Single Cell Mining Claim	2020-08-11	235053	Echo	Single Cell Mining Claim	2020-08-11
100570	Pickerel	Single Cell Mining Claim	2022-01-13	235676	Laval	Single Cell Mining Claim	2020-09-30
100571	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	235677	Laval	Single Cell Mining Claim	2021-02-12
100832	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	235703	Vermilion	Single Cell Mining Claim	2021-01-13
100834	Drayton	Single Cell Mining Claim	2020-12-15	235727	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
100866	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	235728	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
100892	Jordan	Single Cell Mining Claim	2021-04-20	235740	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
100893	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	236625	McAree	Single Cell Mining Claim	2020-09-28
100896	Jordan, Kabik Lake Area	Single Cell Mining Claim	2020-12-15	236626	McAree	Single Cell Mining Claim	2020-09-28
100936	Jordan	Single Cell Mining Claim	2020-12-15	236636	McAree	Single Cell Mining Claim	2020-09-28
100937	Jordan	Single Cell Mining Claim	2020-12-15	238673	Laval	Single Cell Mining Claim	2021-02-12
100948	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	239380	Laval	Single Cell Mining Claim	2020-09-30
101003	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	240268	Laval	Single Cell Mining Claim	2020-09-30
101027	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2020-08-11	240310	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
101080	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	240311	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
101102	Echo	Single Cell Mining Claim	2020-08-11	240312	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
101103	Echo	Single Cell Mining Claim	2021-02-12	242208	Laval	Single Cell Mining Claim	2020-09-30
101126	Laval	Single Cell Mining Claim	2020-09-30	242217	Laval	Single Cell Mining Claim	2021-01-24
101127	Laval	Single Cell Mining Claim	2020-12-04	242696	Jordan	Single Cell Mining Claim	2021-04-20
101246	Echo	Single Cell Mining Claim	2021-02-12	242697	Jordan	Single Cell Mining Claim	2021-04-20
101268	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	243389	Jordan	Single Cell Mining Claim	2020-12-15
101332	McAree	Single Cell Mining Claim	2020-08-05	244113	Laval	Single Cell Mining Claim	2020-09-30
101336	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	245924	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
101359	Echo, McAree	Single Cell Mining Claim	2020-09-30	245927	Laval	Single Cell Mining Claim	2020-12-04
101380	Laval	Single Cell Mining Claim	2020-08-30	246994	Laval	Single Cell Mining Claim	2020-08-05
101407	Echo	Single Cell Mining Claim	2020-08-11	247547	Laval	Single Cell Mining Claim	2021-02-12
101408	Echo	Single Cell Mining Claim	2020-08-11	247548	Laval	Single Cell Mining Claim	2021-02-12
101498	Echo, Pickerel	Single Cell Mining Claim	2021-02-12	248253	Laval	Single Cell Mining Claim	2020-09-30
101593	Jordan	Single Cell Mining Claim	2020-12-15	248934	Laval	Single Cell Mining Claim	2020-09-30
101676	Laval	Single Cell Mining Claim	2020-09-30	249706	Laval	Single Cell Mining Claim	2020-09-30
101738	Laval	Single Cell Mining Claim	2021-01-24	249711	Laval	Single Cell Mining Claim	2021-01-24
101760	Jordan	Single Cell Mining Claim	2020-12-15	250924	Laval	Single Cell Mining Claim	2020-12-04
101761	Pickerel	Single Cell Mining Claim	2021-01-13	252057	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
101764	Echo	Single Cell Mining Claim	2021-02-12	252192	Laval	Single Cell Mining Claim	2020-09-30
101767	Laval	Single Cell Mining Claim	2020-10-31	253399	McAree	Single Cell Mining Claim	2020-09-28
101775	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	253503	Echo	Single Cell Mining Claim	2021-01-13
101776	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	253504	Echo	Single Cell Mining Claim	2021-01-13
101837	Pickerel	Single Cell Mining Claim	2021-01-13	255406	Laval	Single Cell Mining Claim	2021-02-12
101849	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	257839	Webb	Single Cell Mining Claim	2021-01-24
101850	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	258271	Echo	Single Cell Mining Claim	2021-01-13
101862	Jordan	Single Cell Mining Claim	2020-12-15	258272	Echo	Single Cell Mining Claim	2021-01-13
101863	Jordan	Single Cell Mining Claim	2020-12-15	258933	Jordan	Single Cell Mining Claim	2020-12-15
101864	Jordan	Single Cell Mining Claim	2020-12-15	258941	Echo	Single Cell Mining Claim	2021-02-12
101865	Jordan	Single Cell Mining Claim	2020-12-15	258942	Echo	Single Cell Mining Claim	2025-03-29
102027	Pickerel	Single Cell Mining Claim	2021-01-13	258943	Echo	Single Cell Mining Claim	2025-03-29
102028	Pickerel	Single Cell Mining Claim	2021-01-13	259483	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
102053	Jordan	Single Cell Mining Claim	2020-12-15	259484	Pickerel	Single Cell Mining Claim	2021-01-13
102054	Jordan	Single Cell Mining Claim	2020-12-15	259498	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
102055	Jordan	Single Cell Mining Claim	2020-12-15	259499	Pickerel	Single Cell Mining Claim	2021-01-13
102092	Webb	Single Cell Mining Claim	2021-01-24	259503	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
102093	Webb	Single Cell Mining Claim	2021-01-24	259504	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
102490	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	259517	Jordan	Single Cell Mining Claim	2020-12-15
102501	Vermilion	Single Cell Mining Claim	2021-01-13	259576	Jordan	Single Cell Mining Claim	2020-12-15
102506	Laval	Single Cell Mining Claim	2020-09-30	260150	Drayton	Single Cell Mining Claim	2020-12-15
102578	Laval	Single Cell Mining Claim	2020-09-30	260170	McAree	Single Cell Mining Claim	2020-09-28
102579	Laval	Single Cell Mining Claim	2020-09-30	260171	McAree	Single Cell Mining Claim	2020-09-28
102594	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260173	Echo, Pickerel	Single Cell Mining Claim	2020-08-11
102934	Pickerel	Single Cell Mining Claim	2021-01-13	260179	Jordan	Single Cell Mining Claim	2020-12-15
103716	McAree	Single Cell Mining Claim	2020-09-28	260180	Jordan	Single Cell Mining Claim	2020-12-15
104240	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260181	Echo, McAree	Single Cell Mining Claim	2021-04-05
104241	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	260189	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
105356	Laval	Single Cell Mining Claim	2021-02-12	260198	Pickerel	Single Cell Mining Claim	2021-01-13
105558	Laval	Single Cell Mining Claim	2020-09-30	260248	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
106443	Laval	Single Cell Mining Claim	2020-09-30	260249	Jordan	Single Cell Mining Claim	2020-11-10
106444	Laval	Single Cell Mining Claim	2020-09-30	260250	Jordan	Single Cell Mining Claim	2020-12-15
106667	Laval	Single Cell Mining Claim	2020-09-30	260800	Laval, McAree	Single Cell Mining Claim	2020-09-28
107263	Laval	Single Cell Mining Claim	2021-02-12	260848	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
107264	Laval	Single Cell Mining Claim	2021-02-12	260849	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
109467	Laval	Single Cell Mining Claim	2020-09-30	260852	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
111935	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	260884	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
114918	McAree	Single Cell Mining Claim	2020-09-30	260896	Jordan	Single Cell Mining Claim	2020-12-15
114971	McAree	Single Cell Mining Claim	2020-09-28	260941	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
115046	Jordan	Single Cell Mining Claim	2020-12-15	260942	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
115070	Echo	Single Cell Mining Claim	2021-04-26	260943	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
115091	Echo	Single Cell Mining Claim	2021-01-13	261485	McAree	Single Cell Mining Claim	2020-08-05
115111	Echo	Single Cell Mining Claim	2021-01-13	261486	McAree	Single Cell Mining Claim	2020-08-05
115600	Pickerel	Single Cell Mining Claim	2021-01-13	261487	McAree	Single Cell Mining Claim	2020-08-05
115601	Pickerel	Single Cell Mining Claim	2021-01-13	261493	Pickerel	Single Cell Mining Claim	2021-01-13
115831	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	261514	Echo, McAree	Single Cell Mining Claim	2020-09-30
115859	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2022-01-13	261545	Laval	Single Cell Mining Claim	2020-08-30
115860	Pickerel	Single Cell Mining Claim	2022-01-13	262161	Pickerel	Single Cell Mining Claim	2021-02-12
116038	Jordan	Single Cell Mining Claim	2020-12-15	262206	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
116042	McAree	Single Cell Mining Claim	2020-09-28	262207	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
116049	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	262257	Drayton	Single Cell Mining Claim	2020-12-15
116050	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	262279	Jordan	Single Cell Mining Claim	2020-12-15
116105	Webb	Single Cell Mining Claim	2021-01-24	262280	Jordan	Single Cell Mining Claim	2020-12-15
116169	Laval	Single Cell Mining Claim	2020-09-30	262281	Jordan	Single Cell Mining Claim	2020-12-15
116171	Echo	Single Cell Mining Claim	2020-09-28	262857	Laval, Webb	Single Cell Mining Claim	2020-09-30
116254	Laval	Single Cell Mining Claim	2020-10-31	262861	Laval	Single Cell Mining Claim	2020-09-30
116267	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	262926	Webb	Single Cell Mining Claim	2021-01-24
116268	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	262952	Jordan	Single Cell Mining Claim	2020-12-15

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
116272	Jordan	Single Cell Mining Claim	2020-12-15	262954	Pickerel	Single Cell Mining Claim	2021-01-31
116278	Echo	Single Cell Mining Claim	2021-02-09	262971	Echo	Single Cell Mining Claim	2021-02-09
116279	Echo	Single Cell Mining Claim	2021-02-09	263487	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
116344	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	263535	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
116350	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	263536	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
116368	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	263539	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
116404	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	263549	Echo	Single Cell Mining Claim	2021-04-15
116443	Laval, McAree	Single Cell Mining Claim	2020-08-05	263631	Echo, Pickerel	Single Cell Mining Claim	2021-02-12
116444	Laval	Single Cell Mining Claim	2020-09-30	263632	Echo, Pickerel	Single Cell Mining Claim	2021-02-12
116445	Laval	Single Cell Mining Claim	2020-09-30	264241	Echo	Single Cell Mining Claim	2020-08-11
116448	Webb	Single Cell Mining Claim	2021-01-24	264270	Echo	Single Cell Mining Claim	2020-09-28
116450	Jordan	Single Cell Mining Claim	2020-12-15	264286	Echo	Single Cell Mining Claim	2021-02-12
116489	McAree	Single Cell Mining Claim	2020-08-05	264872	Laval	Single Cell Mining Claim	2021-02-12
116490	Laval, McAree	Single Cell Mining Claim	2020-08-05	264873	Laval	Single Cell Mining Claim	2021-02-12
116544	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	264901	Laval	Single Cell Mining Claim	2020-09-30
116549	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	264902	Laval	Single Cell Mining Claim	2020-09-30
116594	Jordan	Single Cell Mining Claim	2021-04-20	264903	Laval	Single Cell Mining Claim	2020-09-30
116596	Jordan	Single Cell Mining Claim	2020-12-15	264928	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
116620	Jordan	Single Cell Mining Claim	2020-12-15	264929	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
116623	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	266146	Pickerel	Single Cell Mining Claim	2021-01-13
116725	Echo	Single Cell Mining Claim	2020-08-11	266147	Pickerel	Single Cell Mining Claim	2021-01-13
116791	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	266148	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
116826	Pickerel	Single Cell Mining Claim	2021-01-13	266165	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
116827	Pickerel	Single Cell Mining Claim	2021-02-12	267052	Laval	Single Cell Mining Claim	2021-01-24
116912	Drayton	Single Cell Mining Claim	2020-12-15	267152	Laval	Single Cell Mining Claim	2020-12-04
116937	Jordan	Single Cell Mining Claim	2020-12-15	267425	Pickerel	Single Cell Mining Claim	2022-01-13
116938	Jordan	Single Cell Mining Claim	2020-12-15	268200	Jordan	Single Cell Mining Claim	2020-12-15
116939	Jordan	Single Cell Mining Claim	2020-12-15	268207	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
117089	Jordan	Single Cell Mining Claim	2020-12-15	268208	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
117096	Echo	Single Cell Mining Claim	2020-08-11	268222	Pickerel	Single Cell Mining Claim	2021-01-13
117097	Echo	Single Cell Mining Claim	2021-02-12	268850	Drayton	Single Cell Mining Claim	2020-12-15
117098	Echo	Single Cell Mining Claim	2025-03-29	268851	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
117099	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	268904	Jordan	Single Cell Mining Claim	2020-12-15
117100	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	268955	Echo	Single Cell Mining Claim	2025-09-17
117148	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	268987	Echo	Single Cell Mining Claim	2020-09-30
117163	Pickerel	Single Cell Mining Claim	2021-01-13	269507	Laval	Single Cell Mining Claim	2020-08-30
117169	Pickerel	Single Cell Mining Claim	2021-01-13	269535	Echo	Single Cell Mining Claim	2020-08-11
117170	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	269536	Echo	Single Cell Mining Claim	2020-08-11
117190	Jordan	Single Cell Mining Claim	2020-12-15	269580	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
117672	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	269622	Pickerel	Single Cell Mining Claim	2021-01-13
117676	Echo	Single Cell Mining Claim	2020-08-11	269623	Pickerel	Single Cell Mining Claim	2021-01-13
117701	Pickerel	Single Cell Mining Claim	2021-01-13	269658	Drayton	Single Cell Mining Claim	2020-12-15
117754	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270190	Drayton	Single Cell Mining Claim	2020-12-15
117755	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270309	Laval	Single Cell Mining Claim	2020-09-30
117756	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	270313	Laval	Single Cell Mining Claim	2020-09-30
117802	Laval	Single Cell Mining Claim	2020-09-30	270318	Echo	Single Cell Mining Claim	2021-02-09
117810	Vermilion	Single Cell Mining Claim	2021-01-13	270435	Echo, McAree	Single Cell Mining Claim	2020-09-28
117811	Vermilion	Single Cell Mining Claim	2021-01-13	270436	McAree	Single Cell Mining Claim	2020-09-28
117817	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	270437	McAree	Single Cell Mining Claim	2020-09-28
117888	Laval	Single Cell Mining Claim	2021-01-24	270452	McAree	Single Cell Mining Claim	2020-09-28
117889	Laval	Single Cell Mining Claim	2020-09-30	270888	Webb	Single Cell Mining Claim	2021-01-24
118176	Jordan	Single Cell Mining Claim	2020-12-15	270914	Jordan	Single Cell Mining Claim	2020-12-15
118244	Pickerel	Single Cell Mining Claim	2021-01-13	270915	Echo	Single Cell Mining Claim	2020-12-15
120327	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	270916	Echo	Single Cell Mining Claim	2021-04-05
120349	Jordan	Single Cell Mining Claim	2020-12-15	270925	Jordan	Single Cell Mining Claim	2020-12-15
120350	Jordan	Single Cell Mining Claim	2020-12-15	270926	Jordan	Single Cell Mining Claim	2020-12-15
120381	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270935	Echo	Single Cell Mining Claim	2021-02-09
120382	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270990	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
120383	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	270991	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
120429	McAree	Single Cell Mining Claim	2020-08-05	270997	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
121009	Pickerel	Single Cell Mining Claim	2021-04-05	271020	Pickerel	Single Cell Mining Claim	2020-08-11
121010	Pickerel	Single Cell Mining Claim	2021-04-05	271024	Echo	Single Cell Mining Claim	2021-04-05
121075	McAree	Single Cell Mining Claim	2020-08-05	271132	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
121122	Pickerel	Single Cell Mining Claim	2021-02-12	271639	McAree	Single Cell Mining Claim	2020-08-05
121123	Pickerel	Single Cell Mining Claim	2021-02-12	272213	Echo	Single Cell Mining Claim	2020-08-11
121124	Pickerel	Single Cell Mining Claim	2020-08-11	272234	Pickerel	Single Cell Mining Claim	2021-01-13
121373	Laval	Single Cell Mining Claim	2020-09-30	272235	Echo	Single Cell Mining Claim	2020-09-28
121667	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	272236	Echo	Single Cell Mining Claim	2020-09-28
121746	Jordan	Single Cell Mining Claim	2020-12-15	272282	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
121823	Laval	Single Cell Mining Claim	2020-09-30	272283	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
121900	Laval	Single Cell Mining Claim	2020-12-04	272869	Jordan	Single Cell Mining Claim	2021-03-28
121901	Laval	Single Cell Mining Claim	2020-12-04	272874	Echo	Single Cell Mining Claim	2020-09-28
121902	Laval	Single Cell Mining Claim	2020-12-04	272875	Echo, McAree	Single Cell Mining Claim	2020-09-28
121903	Laval	Single Cell Mining Claim	2020-12-04	272885	Laval	Single Cell Mining Claim	2020-09-30
122325	Laval	Single Cell Mining Claim	2020-09-30	272888	Laval	Single Cell Mining Claim	2020-09-30
122326	Laval	Single Cell Mining Claim	2020-09-30	272897	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28
122327	Laval	Single Cell Mining Claim	2020-09-30	272907	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
122329	Echo	Single Cell Mining Claim	2021-01-13	273627	Pickerel	Single Cell Mining Claim	2021-01-13
122331	Echo	Single Cell Mining Claim	2025-04-05	274086	Pickerel	Single Cell Mining Claim	2021-01-13
122403	Laval	Single Cell Mining Claim	2021-01-24	274824	Echo	Single Cell Mining Claim	2021-01-13
122431	Laval	Single Cell Mining Claim	2020-09-30	274839	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
122448	Jordan	Single Cell Mining Claim	2020-12-15	274865	Pickerel	Single Cell Mining Claim	2021-01-13
123023	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	276144	McAree	Single Cell Mining Claim	2020-09-28
123024	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	276145	McAree	Single Cell Mining Claim	2020-09-28
123025	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	276743	Jordan	Single Cell Mining Claim	2020-12-15
123030	Jordan	Single Cell Mining Claim	2021-03-28	276744	Jordan	Single Cell Mining Claim	2020-12-15
123100	Echo	Single Cell Mining Claim	2021-01-13	276761	Echo	Single Cell Mining Claim	2021-04-26
123145	Laval	Single Cell Mining Claim	2020-12-04	277472	Echo	Single Cell Mining Claim	2021-02-12
123738	Echo	Single Cell Mining Claim	2020-09-28	277473	Echo	Single Cell Mining Claim	2020-08-11
123826	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	277474	Echo	Single Cell Mining Claim	2020-08-08
123827	Laval	Single Cell Mining Claim	2021-02-12	277476	Pickerel	Single Cell Mining Claim	2021-04-05
123828	Laval	Single Cell Mining Claim	2021-02-12	277530	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
124215	Laval	Single Cell Mining Claim	2021-01-24	277531	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
124385	Laval	Single Cell Mining Claim	2020-09-30	278124	Jordan	Single Cell Mining Claim	2020-12-15
124401	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	278175	Pickerel	Single Cell Mining Claim	2021-04-05
124402	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	278208	Drayton	Boundary Cell Mining Claim	2020-12-15
124937	Pickerel	Single Cell Mining Claim	2021-01-13	278209	Drayton	Single Cell Mining Claim	2020-12-15

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
124938	Pickerel	Single Cell Mining Claim	2021-01-13	278569	Jordan	Single Cell Mining Claim	2021-04-20
124942	Echo, Webb	Single Cell Mining Claim	2021-04-26	278757	Laval	Single Cell Mining Claim	2020-09-30
124943	Echo	Single Cell Mining Claim	2021-04-26	278763	Laval	Single Cell Mining Claim	2021-01-24
125260	Laval	Single Cell Mining Claim	2021-02-12	278995	Jordan	Single Cell Mining Claim	2020-12-15
125261	Laval	Single Cell Mining Claim	2021-02-12	278996	Jordan	Single Cell Mining Claim	2020-12-15
125687	Pickerel	Single Cell Mining Claim	2021-01-13	279001	Echo	Single Cell Mining Claim	2021-02-12
126858	Laval	Single Cell Mining Claim	2021-02-12	279002	Echo	Single Cell Mining Claim	2021-02-12
126884	Echo, McAree	Single Cell Mining Claim	2025-09-30	279003	Echo	Single Cell Mining Claim	2025-09-30
126885	McAree	Single Cell Mining Claim	2020-09-28	279004	Echo	Single Cell Mining Claim	2025-09-30
126961	McAree	Single Cell Mining Claim	2020-09-28	279005	Pickerel	Single Cell Mining Claim	2021-04-05
127543	Jordan	Single Cell Mining Claim	2020-12-15	279006	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
127544	Jordan	Single Cell Mining Claim	2020-12-15	279560	Pickerel	Single Cell Mining Claim	2021-01-13
127545	Jordan	Single Cell Mining Claim	2020-12-15	279561	Pickerel	Single Cell Mining Claim	2021-01-31
127597	Echo	Single Cell Mining Claim	2021-01-13	279562	Pickerel	Single Cell Mining Claim	2021-01-31
127598	Echo	Single Cell Mining Claim	2021-01-13	279564	Pickerel	Single Cell Mining Claim	2021-01-13
127599	Echo	Single Cell Mining Claim	2020-08-11	279565	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
128305	Pickerel	Single Cell Mining Claim	2021-01-13	279579	Jordan	Single Cell Mining Claim	2020-12-15
128306	Pickerel	Single Cell Mining Claim	2021-01-13	279664	Echo	Single Cell Mining Claim	2020-09-30
128335	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	279692	Pickerel	Single Cell Mining Claim	2021-01-13
128915	Jordan	Single Cell Mining Claim	2020-12-15	280225	Drayton	Single Cell Mining Claim	2020-12-15
128977	Pickerel	Single Cell Mining Claim	2021-01-13	280255	Echo	Single Cell Mining Claim	2020-08-11
129011	Drayton	Single Cell Mining Claim	2020-12-15	280257	Jordan	Single Cell Mining Claim	2020-12-15
129012	Drayton	Single Cell Mining Claim	2020-12-15	280261	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
129508	Laval	Single Cell Mining Claim	2020-09-30	280262	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
129554	McAree	Single Cell Mining Claim	2020-09-30	280274	Jordan	Single Cell Mining Claim	2020-12-15
129555	McAree	Single Cell Mining Claim	2020-09-28	280275	Jordan	Single Cell Mining Claim	2020-12-15
129557	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	280312	Webb	Single Cell Mining Claim	2021-01-24
129564	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	280316	Jordan	Single Cell Mining Claim	2020-12-15
129581	Jordan	Single Cell Mining Claim	2020-12-15	280349	McAree	Single Cell Mining Claim	2020-08-05
129609	Webb	Single Cell Mining Claim	2021-01-24	280892	Drayton	Single Cell Mining Claim	2020-12-15
129612	Jordan	Single Cell Mining Claim	2020-12-15	280953	Jordan	Single Cell Mining Claim	2020-12-15
129646	Laval, McAree	Single Cell Mining Claim	2021-01-24	280954	Jordan	Single Cell Mining Claim	2021-03-28
129691	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	280986	Jordan	Single Cell Mining Claim	2020-12-15
130020	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	281620	Pickerel	Single Cell Mining Claim	2021-04-05
130021	Jordan	Single Cell Mining Claim	2020-12-15	281682	McAree	Single Cell Mining Claim	2020-08-05
130296	Laval	Single Cell Mining Claim	2020-09-30	282013	Echo	Single Cell Mining Claim	2021-01-13
130305	Laval	Single Cell Mining Claim	2020-09-30	282227	Pickerel	Single Cell Mining Claim	2021-01-13
130309	Laval	Single Cell Mining Claim	2021-01-24	282233	Laval	Single Cell Mining Claim	2020-09-30
130712	Jordan	Single Cell Mining Claim	2021-04-20	282333	Jordan	Single Cell Mining Claim	2020-12-15
130981	Laval	Single Cell Mining Claim	2020-09-30	282334	Jordan	Single Cell Mining Claim	2020-12-15
130982	Laval	Single Cell Mining Claim	2020-09-30	282335	Jordan	Single Cell Mining Claim	2020-12-15
130983	Laval	Single Cell Mining Claim	2020-12-04	282916	Laval	Single Cell Mining Claim	2020-09-30
131407	Jordan	Single Cell Mining Claim	2020-12-15	282917	Echo	Single Cell Mining Claim	2021-01-13
131408	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	282918	Echo	Single Cell Mining Claim	2021-02-09
134204	Echo	Single Cell Mining Claim	2021-01-13	282919	Echo	Single Cell Mining Claim	2021-01-13
135251	McAree	Single Cell Mining Claim	2020-09-28	283028	Jordan	Single Cell Mining Claim	2020-12-15
135273	McAree	Single Cell Mining Claim	2020-09-28	283029	Jordan	Single Cell Mining Claim	2020-12-15
136994	Laval, Webb	Single Cell Mining Claim	2021-01-24	283036	Echo	Single Cell Mining Claim	2021-02-09
137949	Laval	Single Cell Mining Claim	2020-09-30	283040	Echo	Single Cell Mining Claim	2021-02-09
137950	Laval	Single Cell Mining Claim	2020-09-30	283055	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
137951	Laval	Single Cell Mining Claim	2020-09-30	283056	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
137952	Laval	Single Cell Mining Claim	2020-09-30	283057	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
138858	Laval	Single Cell Mining Claim	2020-08-05	283612	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
138905	Laval	Single Cell Mining Claim	2021-02-12	283617	Jordan	Single Cell Mining Claim	2021-03-28
139221	Jordan	Single Cell Mining Claim	2021-03-28	283631	Jordan	Single Cell Mining Claim	2020-12-15
139598	Laval	Single Cell Mining Claim	2020-09-30	283644	Echo	Single Cell Mining Claim	2020-09-28
141432	Pickerel	Single Cell Mining Claim	2021-01-13	283709	Echo	Single Cell Mining Claim	2021-02-12
141433	Pickerel	Single Cell Mining Claim	2021-01-13	283743	Laval, McAree	Single Cell Mining Claim	2020-08-05
141435	Echo	Single Cell Mining Claim	2020-09-28	283744	Laval	Single Cell Mining Claim	2020-09-30
141436	Echo	Single Cell Mining Claim	2020-09-28	283745	Laval	Single Cell Mining Claim	2020-09-30
141714	Laval	Single Cell Mining Claim	2021-01-24	284329	Pickerel	Single Cell Mining Claim	2021-01-13
142420	Laval	Single Cell Mining Claim	2021-02-12	284331	Echo, McAree	Single Cell Mining Claim	2020-09-28
142682	Echo	Single Cell Mining Claim	2020-08-11	284945	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
143033	Laval, Webb	Single Cell Mining Claim	2021-01-24	286226	Pickerel	Single Cell Mining Claim	2021-01-13
143456	Jordan	Single Cell Mining Claim	2020-12-15	286228	Echo, Webb	Single Cell Mining Claim	2021-04-26
143464	Echo	Single Cell Mining Claim	2020-08-11	286229	Echo, Webb	Single Cell Mining Claim	2021-04-26
143465	Echo	Single Cell Mining Claim	2021-02-12	286230	Echo	Single Cell Mining Claim	2020-09-28
143466	Echo	Single Cell Mining Claim	2021-02-12	286231	Echo, Webb	Single Cell Mining Claim	2020-09-28
143467	Echo	Single Cell Mining Claim	2021-02-12	286247	Echo	Single Cell Mining Claim	2025-09-17
143468	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	287391	Laval	Single Cell Mining Claim	2020-09-30
144756	Echo	Single Cell Mining Claim	2021-04-26	287483	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
144781	Echo	Single Cell Mining Claim	2021-01-13	287505	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
145341	Pickerel	Single Cell Mining Claim	2021-01-13	288853	Echo	Single Cell Mining Claim	2020-09-28
145342	Pickerel	Single Cell Mining Claim	2021-01-13	288871	Echo	Single Cell Mining Claim	2021-01-13
145343	Pickerel	Single Cell Mining Claim	2021-01-13	289610	Laval	Single Cell Mining Claim	2020-12-15
145371	Pickerel	Single Cell Mining Claim	2021-01-13	289648	Drayton	Single Cell Mining Claim	2020-12-15
145395	Jordan	Single Cell Mining Claim	2020-12-15	289668	Drayton	Single Cell Mining Claim	2020-12-15
145396	Jordan	Single Cell Mining Claim	2020-12-15	289761	Laval	Single Cell Mining Claim	2020-09-30
145492	Jordan, Kabik Lake Area, Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	289793	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
145493	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	290334	Webb	Single Cell Mining Claim	2021-01-24
145500	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2020-12-15	290345	Drayton	Boundary Cell Mining Claim	2020-12-15
148834	Laval	Single Cell Mining Claim	2020-09-30	290360	Echo	Single Cell Mining Claim	2020-09-28
148835	Laval	Single Cell Mining Claim	2020-09-30	290368	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
150149	Echo	Single Cell Mining Claim	2021-01-13	290369	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
151621	Echo	Single Cell Mining Claim	2021-01-13	290377	Echo	Single Cell Mining Claim	2021-02-09
151622	Echo	Single Cell Mining Claim	2021-01-13	290439	Jordan	Single Cell Mining Claim	2020-12-15
151623	Echo	Single Cell Mining Claim	2021-01-13	290451	Pickerel	Single Cell Mining Claim	2020-08-11
151646	Echo	Single Cell Mining Claim	2021-02-12	290604	McAree	Single Cell Mining Claim	2020-09-28
151670	Laval, McAree	Single Cell Mining Claim	2020-08-05	290965	Echo	Single Cell Mining Claim	2021-04-05
151671	McAree	Single Cell Mining Claim	2020-08-05	291010	Echo, Pickerel	Single Cell Mining Claim	2021-01-13
151721	McAree	Single Cell Mining Claim	2020-09-28	291030	Echo	Single Cell Mining Claim	2021-02-12
151742	Pickerel	Single Cell Mining Claim	2021-01-13	291056	McAree	Single Cell Mining Claim	2020-08-05
152294	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	291057	Laval	Single Cell Mining Claim	2020-09-30
152345	Webb	Single Cell Mining Claim	2021-01-24	291058	Laval	Single Cell Mining Claim	2020-12-04
152356	Vermilion	Single Cell Mining Claim	2021-01-13	291059	Laval	Single Cell Mining Claim	2020-09-30
152357	Vermilion	Single Cell Mining Claim	2021-01-13	291060	Laval	Single Cell Mining Claim	2020-09-30

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
152371	Vermilion	Single Cell Mining Claim	2021-01-13	291114	McAree	Single Cell Mining Claim	2020-09-28
152375	Laval	Single Cell Mining Claim	2020-09-30	291696	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
152378	Laval	Single Cell Mining Claim	2020-10-31	291737	Laval	Single Cell Mining Claim	2021-02-12
152403	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	291744	Webb	Single Cell Mining Claim	2021-01-24
153623	Pickerel	Single Cell Mining Claim	2021-01-13	291753	Laval	Single Cell Mining Claim	2020-09-30
153871	Laval	Single Cell Mining Claim	2020-09-30	291786	Laval	Single Cell Mining Claim	2020-09-30
154210	Pickerel	Single Cell Mining Claim	2021-04-05	291787	Laval	Single Cell Mining Claim	2020-09-30
154232	Echo	Single Cell Mining Claim	2025-09-17	291788	Laval	Single Cell Mining Claim	2020-10-31
155481	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	291802	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
156254	McAree	Single Cell Mining Claim	2020-09-28	292633	Laval	Single Cell Mining Claim	2021-02-12
156838	Jordan	Single Cell Mining Claim	2020-12-15	292859	Laval	Single Cell Mining Claim	2020-12-04
156857	Echo	Single Cell Mining Claim	2021-04-26	293016	Pickerel	Single Cell Mining Claim	2021-01-13
157589	Pickerel	Single Cell Mining Claim	2021-01-13	293894	Laval	Single Cell Mining Claim	2020-09-30
157590	Pickerel	Single Cell Mining Claim	2021-01-13	294169	Laval, McAree	Single Cell Mining Claim	2020-08-05
157604	Pickerel	Single Cell Mining Claim	2021-01-31	294208	Echo	Single Cell Mining Claim	2020-08-11
158107	Pickerel	Single Cell Mining Claim	2021-01-13	294705	Laval	Single Cell Mining Claim	2021-02-12
158118	Jordan	Single Cell Mining Claim	2020-12-15	294927	Echo, McAree	Single Cell Mining Claim	2020-09-30
158119	Jordan	Single Cell Mining Claim	2020-12-15	295593	Jordan	Single Cell Mining Claim	2020-12-15
158246	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	295611	Echo, Webb	Single Cell Mining Claim	2021-04-26
158789	Jordan	Single Cell Mining Claim	2020-12-15	295612	Echo	Single Cell Mining Claim	2020-09-28
158790	Jordan	Single Cell Mining Claim	2020-12-15	295613	Echo	Single Cell Mining Claim	2020-09-28
158795	Drayton	Single Cell Mining Claim	2020-12-15	296322	Echo	Single Cell Mining Claim	2021-02-12
158817	McAree	Single Cell Mining Claim	2020-09-28	296323	Echo	Single Cell Mining Claim	2021-02-12
158818	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	296324	Echo	Single Cell Mining Claim	2020-08-11
158824	Jordan	Single Cell Mining Claim	2020-12-15	296325	Echo	Single Cell Mining Claim	2025-03-29
158828	McAree	Single Cell Mining Claim	2020-08-05	296326	Echo	Single Cell Mining Claim	2025-09-30
158829	McAree	Single Cell Mining Claim	2020-08-05	296327	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
158849	Pickerel	Single Cell Mining Claim	2021-01-13	296871	Pickerel	Single Cell Mining Claim	2021-01-13
158854	Jordan	Single Cell Mining Claim	2020-12-15	296872	Pickerel	Single Cell Mining Claim	2021-01-13
158888	Webb	Single Cell Mining Claim	2021-01-24	296875	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
158890	Jordan	Single Cell Mining Claim	2020-12-15	296880	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18
158891	Drayton, Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	296881	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
159148	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	296887	Jordan	Single Cell Mining Claim	2020-12-15
159469	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	296888	Jordan	Single Cell Mining Claim	2020-12-15
159502	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	296892	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
159503	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	296984	Pickerel	Single Cell Mining Claim	2021-04-05
159518	Jordan	Single Cell Mining Claim	2020-12-15	296991	Pickerel	Single Cell Mining Claim	2021-01-13
159528	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	297229	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20
159564	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	297230	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20
159595	McAree	Single Cell Mining Claim	2020-08-05	297358	Laval	Single Cell Mining Claim	2021-01-24
160125	Echo	Single Cell Mining Claim	2020-09-30	297528	Jordan	Single Cell Mining Claim	2020-12-15
160149	Laval	Single Cell Mining Claim	2020-08-30	297551	McAree	Single Cell Mining Claim	2020-09-30
160166	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	297552	McAree	Single Cell Mining Claim	2020-09-28
160212	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	297553	McAree	Single Cell Mining Claim	2020-09-28
160256	Pickerel	Single Cell Mining Claim	2021-01-13	297554	Echo, Pickerel	Single Cell Mining Claim	2020-08-11
160257	Echo, Pickerel	Single Cell Mining Claim	2021-02-12	297586	Pickerel	Single Cell Mining Claim	2021-01-13
160265	Laval	Single Cell Mining Claim	2020-08-05	297623	Drayton, Jordan	Single Cell Mining Claim	2020-11-10
160271	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	297624	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
160272	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	297625	Jordan	Single Cell Mining Claim	2020-12-15
160273	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	298199	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
160377	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	298252	Jordan	Single Cell Mining Claim	2021-04-20
160381	Laval	Single Cell Mining Claim	2020-12-04	298253	Jordan, Kabik Lake Area	Single Cell Mining Claim	2020-12-15
160382	Laval	Single Cell Mining Claim	2020-12-04	298288	Jordan	Single Cell Mining Claim	2020-12-15
160816	Drayton	Single Cell Mining Claim	2020-12-15	298294	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
160945	Echo	Single Cell Mining Claim	2021-02-09	298327	McAree	Single Cell Mining Claim	2020-08-05
161516	Laval	Single Cell Mining Claim	2021-01-24	298335	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
161537	Laval	Single Cell Mining Claim	2020-09-30	298646	Laval	Single Cell Mining Claim	2020-09-30
161538	Laval	Single Cell Mining Claim	2020-09-30	298909	Echo	Single Cell Mining Claim	2020-08-11
161542	Jordan	Single Cell Mining Claim	2020-12-15	298910	Echo	Single Cell Mining Claim	2020-08-11
161549	Echo	Single Cell Mining Claim	2020-09-28	301437	Laval	Single Cell Mining Claim	2020-09-30
161561	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	304062	Pickerel	Single Cell Mining Claim	2021-01-13
161562	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	304063	Pickerel	Single Cell Mining Claim	2021-01-13
161563	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	304064	Pickerel	Single Cell Mining Claim	2021-01-13
161564	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	304247	Laval	Single Cell Mining Claim	2020-12-04
161616	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	304248	Laval	Single Cell Mining Claim	2020-12-04
161622	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	304390	Laval	Single Cell Mining Claim	2021-02-12
161623	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	305016	Webb	Single Cell Mining Claim	2021-01-24
161630	Echo	Single Cell Mining Claim	2021-04-15	305303	Laval	Single Cell Mining Claim	2021-02-12
162268	McAree	Single Cell Mining Claim	2020-09-28	306006	Laval	Single Cell Mining Claim	2020-09-30
162269	McAree	Single Cell Mining Claim	2020-09-28	309427	Jordan	Single Cell Mining Claim	2021-04-20
162853	Jordan	Single Cell Mining Claim	2020-12-15	309568	Laval	Single Cell Mining Claim	2020-09-30
162872	Echo	Single Cell Mining Claim	2021-04-26	309569	Laval	Single Cell Mining Claim	2021-01-24
163283	Jordan	Single Cell Mining Claim	2021-04-20	310275	Laval	Single Cell Mining Claim	2020-12-04
163585	Echo	Single Cell Mining Claim	2021-02-12	310998	Laval	Single Cell Mining Claim	2021-01-24
163586	Pickerel	Single Cell Mining Claim	2021-04-05	311329	Pickerel	Single Cell Mining Claim	2022-01-13
163631	Pickerel	Single Cell Mining Claim	2021-01-31	311330	Pickerel	Single Cell Mining Claim	2021-01-13
163635	Pickerel	Single Cell Mining Claim	2021-01-13	311698	Laval	Single Cell Mining Claim	2021-02-12
163639	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	312332	Webb	Single Cell Mining Claim	2021-01-24
163646	Jordan	Single Cell Mining Claim	2020-12-15	312784	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
163654	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	313382	Echo	Single Cell Mining Claim	2021-01-13
163963	Jordan	Single Cell Mining Claim	2020-12-15	314060	Echo	Single Cell Mining Claim	2021-02-12
164240	Echo, McAree	Single Cell Mining Claim	2020-09-28	314061	Echo	Single Cell Mining Claim	2021-02-12
164269	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	314062	Echo	Single Cell Mining Claim	2025-02-12
164282	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	314063	Echo	Single Cell Mining Claim	2025-02-12
164305	Drayton	Single Cell Mining Claim	2020-12-15	314064	Echo, McAree	Single Cell Mining Claim	2025-09-30
164306	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	314066	Pickerel	Single Cell Mining Claim	2021-04-05
164829	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	314122	Jordan	Single Cell Mining Claim	2020-12-15
164835	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	314371	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
164836	Jordan	Single Cell Mining Claim	2020-12-15	314654	McAree	Single Cell Mining Claim	2020-09-28
164847	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	314659	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
164891	Webb	Single Cell Mining Claim	2021-01-24	314660	Jordan	Single Cell Mining Claim	2020-12-15
164892	Jordan	Single Cell Mining Claim	2020-11-10	314661	Jordan	Single Cell Mining Claim	2020-12-15
164958	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18	314666	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
165508	Jordan	Single Cell Mining Claim	2021-04-20	314678	Pickerel	Single Cell Mining Claim	2021-01-13
165653	Laval	Single Cell Mining Claim	2020-08-30	314679	Pickerel	Single Cell Mining Claim	2021-01-13
165856	Laval	Single Cell Mining Claim	2020-09-30	314686	Jordan	Single Cell Mining Claim	2020-12-15

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
166159	Laval	Single Cell Mining Claim	2020-08-30	314687	Jordan	Single Cell Mining Claim	2020-12-15
166160	Laval	Single Cell Mining Claim	2020-08-30	314720	Drayton, Jordan	Single Cell Mining Claim	2020-11-10
166161	Laval	Single Cell Mining Claim	2020-08-30	314721	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
166185	Echo	Single Cell Mining Claim	2020-08-11	314796	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
166186	Echo	Single Cell Mining Claim	2020-08-11	315449	Jordan	Single Cell Mining Claim	2021-04-20
166274	Pickerel	Single Cell Mining Claim	2021-02-12	315450	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20
166438	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	316298	Laval, Webb	Single Cell Mining Claim	2020-09-30
166439	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	316843	Laval	Single Cell Mining Claim	2020-09-30
166441	Laval	Single Cell Mining Claim	2020-12-04	320646	Pickerel	Single Cell Mining Claim	2021-01-13
166854	Drayton	Single Cell Mining Claim	2020-12-15	320647	Pickerel	Single Cell Mining Claim	2021-01-13
166855	Drayton	Single Cell Mining Claim	2020-12-15	320651	Echo	Single Cell Mining Claim	2020-09-28
166879	Jordan	Single Cell Mining Claim	2020-12-15	320953	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
166938	Laval	Single Cell Mining Claim	2021-01-24	320954	Laval	Single Cell Mining Claim	2021-02-12
166943	Echo	Single Cell Mining Claim	2020-09-28	320955	Laval	Single Cell Mining Claim	2021-02-12
167515	Drayton	Boundary Cell Mining Claim	2020-12-15	320968	Webb	Single Cell Mining Claim	2021-01-24
167528	Jordan	Single Cell Mining Claim	2020-12-15	321013	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
167529	Jordan	Single Cell Mining Claim	2020-12-15	321047	Webb	Single Cell Mining Claim	2021-01-24
167534	Laval	Single Cell Mining Claim	2020-09-30	321574	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
167546	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	322261	Pickerel	Single Cell Mining Claim	2021-01-13
167556	Echo	Single Cell Mining Claim	2021-02-09	322262	Pickerel	Single Cell Mining Claim	2021-01-13
167557	Echo	Single Cell Mining Claim	2021-02-09	322263	Pickerel	Single Cell Mining Claim	2021-01-13
167627	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	322335	Jordan	Single Cell Mining Claim	2020-12-15
167663	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	322809	Pickerel	Single Cell Mining Claim	2021-01-13
168213	Echo, Pickerel	Single Cell Mining Claim	2021-01-13	322810	Pickerel	Single Cell Mining Claim	2021-01-13
168240	Echo	Single Cell Mining Claim	2021-02-12	322814	Echo, Webb	Single Cell Mining Claim	2020-09-28
168241	Echo	Single Cell Mining Claim	2021-02-12	322827	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
168271	Laval	Single Cell Mining Claim	2020-12-04	322828	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
168313	Echo, McAree	Single Cell Mining Claim	2020-09-28	322829	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
168355	Echo	Single Cell Mining Claim	2021-02-12	323553	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
168896	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	323554	Kabik Lake Area	Single Cell Mining Claim	2021-01-13
168941	Laval	Single Cell Mining Claim	2020-09-30	323555	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
168953	Webb	Single Cell Mining Claim	2021-01-24	325434	Echo	Single Cell Mining Claim	2021-01-13
168971	Jordan	Single Cell Mining Claim	2021-03-28	326101	Echo	Single Cell Mining Claim	2021-02-12
168984	Echo, Laval, McAree, Webb	Single Cell Mining Claim	2020-09-28	326103	Pickerel	Single Cell Mining Claim	2020-08-11
168993	Laval	Single Cell Mining Claim	2020-09-30	326136	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
169567	Laval, Webb	Single Cell Mining Claim	2020-09-30	326137	Pickerel	Single Cell Mining Claim	2021-01-13
169766	Laval	Single Cell Mining Claim	2021-02-12	326145	Pickerel	Single Cell Mining Claim	2021-01-13
170273	Pickerel	Single Cell Mining Claim	2021-01-13	326154	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18
170274	Pickerel	Single Cell Mining Claim	2021-01-13	326747	Echo	Single Cell Mining Claim	2020-09-28
170339	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	326780	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
170770	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	326781	Pickerel	Single Cell Mining Claim	2021-01-13
170772	Echo	Single Cell Mining Claim	2020-09-28	326816	Jordan	Single Cell Mining Claim	2020-12-15
170784	Kabik Lake Area	Single Cell Mining Claim	2021-01-13	326819	Drayton, Parnes Lake Area	Boundary Cell Mining Claim	2020-12-15
170790	Echo	Single Cell Mining Claim	2025-09-17	326820	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
170791	Echo	Single Cell Mining Claim	2025-09-17	326858	McAree	Single Cell Mining Claim	2020-09-28
171510	Echo	Single Cell Mining Claim	2020-08-11	326859	McAree	Single Cell Mining Claim	2020-09-28
171520	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	326860	Pickerel	Single Cell Mining Claim	2021-01-13
171546	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	326865	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
171547	Pickerel	Single Cell Mining Claim	2021-01-13	326866	Jordan	Single Cell Mining Claim	2020-12-15
173418	Laval	Single Cell Mining Claim	2020-08-05	326871	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
173419	Laval	Single Cell Mining Claim	2020-08-05	327420	Webb	Single Cell Mining Claim	2021-01-24
173634	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	327421	Webb	Single Cell Mining Claim	2021-01-24
174143	Laval	Single Cell Mining Claim	2020-09-30	327422	Webb	Single Cell Mining Claim	2021-01-24
174817	Laval	Single Cell Mining Claim	2020-09-30	327424	Drayton, Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
175970	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20	327425	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
176109	Laval	Single Cell Mining Claim	2020-09-30	327536	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
176110	Laval	Single Cell Mining Claim	2020-09-30	328084	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
176113	Laval	Single Cell Mining Claim	2021-01-24	328158	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
176801	Laval	Single Cell Mining Claim	2020-12-04	328170	Laval	Single Cell Mining Claim	2020-08-30
177364	Laval	Single Cell Mining Claim	2020-09-30	328192	Echo	Single Cell Mining Claim	2020-08-11
177626	Jordan	Single Cell Mining Claim	2020-12-15	328757	McAree	Single Cell Mining Claim	2020-08-05
177654	Jordan	Single Cell Mining Claim	2020-12-15	328800	Echo, Pickerel	Single Cell Mining Claim	2021-02-12
177658	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	328810	Laval	Single Cell Mining Claim	2020-12-15
177659	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	328814	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
177671	Echo	Single Cell Mining Claim	2020-09-30	328839	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
177673	Pickerel	Single Cell Mining Claim	2021-01-13	328878	Drayton	Single Cell Mining Claim	2020-12-15
177674	Pickerel	Single Cell Mining Claim	2021-01-13	328976	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
177679	Jordan	Single Cell Mining Claim	2020-12-15	328977	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
177717	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	328978	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
178320	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	328983	Laval	Single Cell Mining Claim	2020-12-04
178364	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	329507	Laval	Single Cell Mining Claim	2020-09-30
178365	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	329511	Echo	Single Cell Mining Claim	2021-01-13
178394	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	329586	Laval, Webb	Single Cell Mining Claim	2021-01-24
178408	Jordan	Single Cell Mining Claim	2021-04-20	329587	Laval, Webb	Single Cell Mining Claim	2021-01-24
178416	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	329588	Laval, Webb	Single Cell Mining Claim	2021-01-24
178982	Laval	Single Cell Mining Claim	2020-08-30	330115	Jordan	Single Cell Mining Claim	2020-12-15
179008	Echo	Single Cell Mining Claim	2020-08-11	330117	Pickerel	Single Cell Mining Claim	2021-01-31
179069	McAree	Single Cell Mining Claim	2020-08-05	330118	Pickerel	Single Cell Mining Claim	2021-01-13
179120	Pickerel	Single Cell Mining Claim	2021-01-13	330121	Echo	Single Cell Mining Claim	2021-02-12
179121	Pickerel	Single Cell Mining Claim	2021-01-13	330122	Echo	Single Cell Mining Claim	2020-09-28
179665	Drayton	Single Cell Mining Claim	2020-12-15	330123	Laval	Single Cell Mining Claim	2020-09-30
179721	Jordan	Single Cell Mining Claim	2020-12-15	330133	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
179791	Echo	Single Cell Mining Claim	2021-01-13	330134	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
179792	Echo	Single Cell Mining Claim	2021-01-13	330206	Echo	Single Cell Mining Claim	2021-04-15
179872	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	330255	Echo	Single Cell Mining Claim	2021-01-13
179874	Laval	Single Cell Mining Claim	2020-12-04	330792	Echo	Single Cell Mining Claim	2021-02-12
179875	Laval	Single Cell Mining Claim	2020-12-04	330871	McAree	Single Cell Mining Claim	2020-09-28
180269	Laval	Single Cell Mining Claim	2021-01-24	330908	Echo	Single Cell Mining Claim	2020-09-28
180364	Webb	Single Cell Mining Claim	2021-01-24	334998	Echo	Single Cell Mining Claim	2021-02-12
180365	Laval, Webb	Single Cell Mining Claim	2021-01-24	337508	Jordan	Single Cell Mining Claim	2021-04-20
180371	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	337509	Jordan	Single Cell Mining Claim	2021-04-20
180380	Pickerel	Single Cell Mining Claim	2021-01-13	338373	Laval	Single Cell Mining Claim	2020-12-04
180383	Echo	Single Cell Mining Claim	2020-12-15	338374	Laval	Single Cell Mining Claim	2020-12-04
180395	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	338375	Laval	Single Cell Mining Claim	2020-12-04
180396	Jordan	Single Cell Mining Claim	2020-12-15	338931	Laval	Single Cell Mining Claim	2020-09-30
180413	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	338932	Laval	Single Cell Mining Claim	2020-09-30

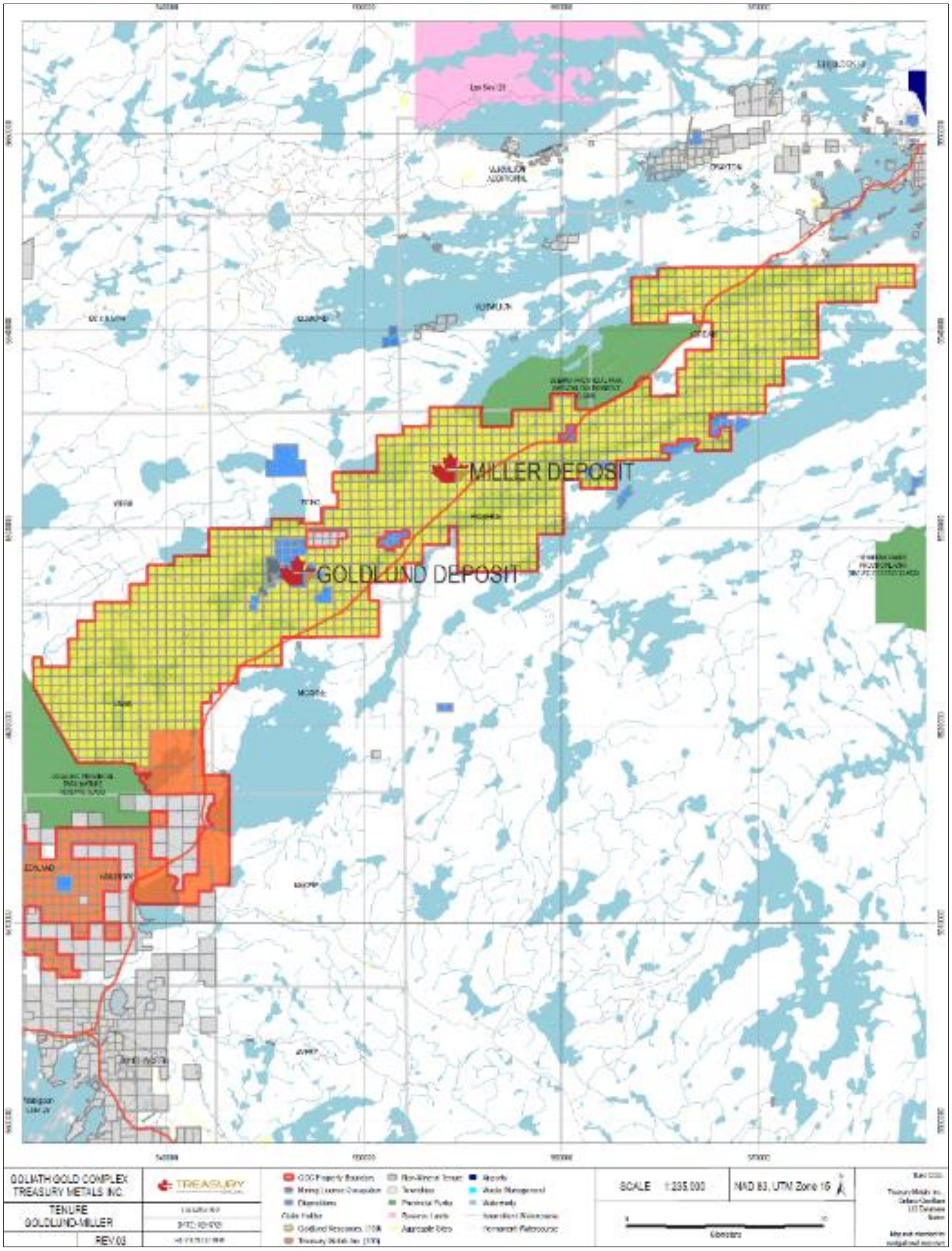
Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
180457	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	339893	Jordan	Single Cell Mining Claim	2020-12-15
180480	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	339921	Jordan	Single Cell Mining Claim	2021-04-20
180991	Echo, Pickerel	Single Cell Mining Claim	2020-08-11	339967	Pickerel	Single Cell Mining Claim	2021-01-13
180993	Echo	Single Cell Mining Claim	2020-09-28	339984	Echo	Single Cell Mining Claim	2020-08-08
181072	Laval	Single Cell Mining Claim	2020-09-30	340019	Laval	Single Cell Mining Claim	2020-08-30
181133	McAree	Single Cell Mining Claim	2020-09-28	340020	Laval	Single Cell Mining Claim	2020-08-30
181143	Echo	Single Cell Mining Claim	2020-08-11	340551	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
181671	Pickerel	Single Cell Mining Claim	2021-01-13	340611	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
181672	Pickerel	Single Cell Mining Claim	2021-01-13	340612	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
181674	Echo	Single Cell Mining Claim	2020-09-28	340615	McAree	Single Cell Mining Claim	2020-08-05
181715	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	340659	Pickerel	Single Cell Mining Claim	2021-01-13
181757	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	340660	Pickerel	Single Cell Mining Claim	2021-02-12
181779	Jordan	Single Cell Mining Claim	2021-03-28	340677	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
181801	Laval	Single Cell Mining Claim	2020-09-30	340678	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
181802	Laval	Single Cell Mining Claim	2020-09-30	340860	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
181816	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	340863	Laval	Single Cell Mining Claim	2020-12-04
182377	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	340961	Echo	Single Cell Mining Claim	2021-01-13
186191	Echo	Single Cell Mining Claim	2021-01-13	341341	Laval, Webb	Single Cell Mining Claim	2021-01-24
187731	Laval	Single Cell Mining Claim	2021-01-24	341349	Laval	Single Cell Mining Claim	2020-09-30
187732	Laval	Single Cell Mining Claim	2021-01-24	341372	Echo	Single Cell Mining Claim	2020-09-28
188977	Laval	Single Cell Mining Claim	2021-02-12	341923	Laval	Single Cell Mining Claim	2021-01-24
189616	Laval, Webb	Single Cell Mining Claim	2021-01-24	341949	Jordan	Single Cell Mining Claim	2020-12-15
189979	Laval	Single Cell Mining Claim	2020-09-30	341954	Echo	Single Cell Mining Claim	2020-08-11
190830	Laval	Single Cell Mining Claim	2020-08-05	341963	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
191676	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	341994	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
192278	Laval	Single Cell Mining Claim	2020-09-30	341995	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
193322	Jordan	Single Cell Mining Claim	2021-04-20	342031	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
193323	Jordan	Single Cell Mining Claim	2021-04-20	342045	Jordan	Single Cell Mining Claim	2021-03-28
193324	Jordan	Single Cell Mining Claim	2021-04-20	342209	McAree	Single Cell Mining Claim	2020-09-28
193567	Laval	Single Cell Mining Claim	2021-01-24	342410	Laval	Single Cell Mining Claim	2021-01-24
193568	Laval	Single Cell Mining Claim	2021-01-24	342426	Laval	Single Cell Mining Claim	2021-02-12
194214	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	342427	Laval	Single Cell Mining Claim	2021-02-12
194256	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	342553	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
194280	Jordan	Single Cell Mining Claim	2020-12-15	342620	Echo, Pickerel	Single Cell Mining Claim	2021-01-13
194292	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	343233	McAree	Single Cell Mining Claim	2020-09-28
194316	Laval	Single Cell Mining Claim	2020-09-30	343240	Echo	Single Cell Mining Claim	2020-08-11
194317	Laval	Single Cell Mining Claim	2020-09-30	343310	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
194318	Laval	Single Cell Mining Claim	2020-09-30	343345	Laval	Single Cell Mining Claim	2021-02-12
194818	Jordan	Single Cell Mining Claim	2020-12-15	343375	Jordan	Single Cell Mining Claim	2021-03-28
194819	Jordan	Single Cell Mining Claim	2020-12-15	343924	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
194820	Jordan	Single Cell Mining Claim	2021-04-20	343964	Webb	Single Cell Mining Claim	2021-01-24
194825	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	343965	Laval	Single Cell Mining Claim	2021-01-24
194872	McAree	Single Cell Mining Claim	2020-08-05	344643	Pickerel	Single Cell Mining Claim	2021-01-13
194873	McAree	Single Cell Mining Claim	2020-08-05	345434	Laval	Single Cell Mining Claim	2020-09-30
194923	Laval	Single Cell Mining Claim	2020-08-30	545974	Kabik Lake Area	Multi-cell Mining Claim	2021-03-19
195114	Laval	Single Cell Mining Claim	2020-09-30				
195115	Laval	Single Cell Mining Claim	2021-01-24	100003	Pickerel	Single Cell Mining Claim	2021-04-05
195116	Laval	Single Cell Mining Claim	2021-01-24	100005	Echo	Single Cell Mining Claim	2021-04-26
195528	Pickerel	Single Cell Mining Claim	2021-01-13	100282	Echo	Single Cell Mining Claim	2021-01-13
195529	Pickerel	Single Cell Mining Claim	2021-02-12	100468	Pickerel	Single Cell Mining Claim	2020-08-11
195532	Laval	Single Cell Mining Claim	2020-09-30	100570	Pickerel	Single Cell Mining Claim	2022-01-13
195533	Laval	Single Cell Mining Claim	2020-09-30	100571	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
195534	Laval	Single Cell Mining Claim	2020-09-30	100832	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
195543	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	100834	Drayton	Single Cell Mining Claim	2020-12-15
195584	Drayton	Single Cell Mining Claim	2020-12-15	100866	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
196208	Laval	Single Cell Mining Claim	2020-09-30	100892	Jordan	Single Cell Mining Claim	2021-04-20
196209	Laval	Single Cell Mining Claim	2020-09-30	100893	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18
196210	Laval	Single Cell Mining Claim	2020-09-30	100896	Jordan, Kabik Lake Area	Single Cell Mining Claim	2020-12-15
196211	Laval	Single Cell Mining Claim	2020-09-30	100936	Jordan	Single Cell Mining Claim	2020-12-15
196269	Laval	Single Cell Mining Claim	2020-09-30	100937	Jordan	Single Cell Mining Claim	2020-12-15
196270	Drayton	Boundary Cell Mining Claim	2020-12-15	100948	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
196280	Jordan	Single Cell Mining Claim	2020-12-15	101003	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28
196283	Pickerel	Single Cell Mining Claim	2021-01-13	101027	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2020-08-11
196298	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	101080	Echo, Pickerel	Single Cell Mining Claim	2021-01-13
196301	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	101102	Echo	Single Cell Mining Claim	2020-08-11
196307	Echo	Single Cell Mining Claim	2021-02-09	101103	Echo	Single Cell Mining Claim	2021-02-12
196308	Echo	Single Cell Mining Claim	2021-02-09	101126	Laval	Single Cell Mining Claim	2020-09-30
196309	Echo	Single Cell Mining Claim	2021-02-09	101127	Laval	Single Cell Mining Claim	2020-12-04
196319	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101246	Echo	Single Cell Mining Claim	2021-02-12
196320	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101268	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
196833	Webb	Single Cell Mining Claim	2021-01-24	101332	McAree	Single Cell Mining Claim	2020-08-05
196858	Laval	Single Cell Mining Claim	2020-09-30	101336	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
196861	Laval	Single Cell Mining Claim	2020-09-30	101359	Echo, McAree	Single Cell Mining Claim	2020-09-30
196862	Laval	Single Cell Mining Claim	2020-09-30	101380	Laval	Single Cell Mining Claim	2020-08-30
197506	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101407	Echo	Single Cell Mining Claim	2020-08-11
197507	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	101408	Echo	Single Cell Mining Claim	2020-08-11
197558	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	101498	Echo, Pickerel	Single Cell Mining Claim	2021-02-12
197567	Echo	Single Cell Mining Claim	2025-04-15	101593	Jordan	Single Cell Mining Claim	2020-12-15
197572	Jordan	Single Cell Mining Claim	2020-12-15	101676	Laval	Single Cell Mining Claim	2020-09-30
197583	Echo	Single Cell Mining Claim	2021-04-05	101738	Laval	Single Cell Mining Claim	2021-01-24
197662	Laval, McAree	Single Cell Mining Claim	2020-08-05	101760	Jordan	Single Cell Mining Claim	2020-12-15
198227	Echo, McAree	Single Cell Mining Claim	2020-09-28	101761	Pickerel	Single Cell Mining Claim	2021-01-13
198228	McAree	Single Cell Mining Claim	2020-09-28	101764	Echo	Single Cell Mining Claim	2021-02-12
198259	Pickerel	Single Cell Mining Claim	2021-01-13	101767	Laval	Single Cell Mining Claim	2020-10-31
198261	Echo	Single Cell Mining Claim	2020-09-28	101775	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
198262	Echo, McAree	Single Cell Mining Claim	2020-09-28	101776	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
198289	Echo	Single Cell Mining Claim	2021-02-12	101837	Pickerel	Single Cell Mining Claim	2021-01-13
198353	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	101849	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
198896	Echo	Single Cell Mining Claim	2021-01-13	101850	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
200042	Echo	Single Cell Mining Claim	2020-09-28	101862	Jordan	Single Cell Mining Claim	2020-12-15
200043	Echo	Single Cell Mining Claim	2021-04-26	101863	Jordan	Single Cell Mining Claim	2020-12-15
200044	Echo	Single Cell Mining Claim	2021-04-26	101864	Jordan	Single Cell Mining Claim	2020-12-15
200045	Echo, Webb	Single Cell Mining Claim	2021-04-26	101865	Jordan	Single Cell Mining Claim	2020-12-15
200446	Laval	Single Cell Mining Claim	2021-01-24	102027	Pickerel	Single Cell Mining Claim	2021-01-13
200489	Laval	Single Cell Mining Claim	2021-02-12	102028	Pickerel	Single Cell Mining Claim	2021-01-13
200797	Kabik Lake Area	Single Cell Mining Claim	2021-01-13	102053	Jordan	Single Cell Mining Claim	2020-12-15
202071	McAree	Single Cell Mining Claim	2020-09-28	102054	Jordan	Single Cell Mining Claim	2020-12-15

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
202162	Jordan	Single Cell Mining Claim	2020-12-15	102055	Jordan	Single Cell Mining Claim	2020-12-15
202729	Echo	Single Cell Mining Claim	2021-01-13	102092	Webb	Single Cell Mining Claim	2021-01-24
203025	Laval	Single Cell Mining Claim	2021-02-12	102093	Webb	Single Cell Mining Claim	2021-01-24
203026	Laval	Single Cell Mining Claim	2021-02-12	102490	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28
203371	Echo	Single Cell Mining Claim	2021-02-12	102501	Vermilion	Single Cell Mining Claim	2021-01-13
203372	Echo	Single Cell Mining Claim	2025-09-17	102506	Laval	Single Cell Mining Claim	2020-09-30
203373	Echo, McAree	Single Cell Mining Claim	2025-09-30	102578	Laval	Single Cell Mining Claim	2020-09-30
203407	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	102579	Laval	Single Cell Mining Claim	2020-09-30
203439	Jordan	Single Cell Mining Claim	2020-12-15	102594	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
203440	Jordan	Single Cell Mining Claim	2020-12-15	102934	Pickerel	Single Cell Mining Claim	2021-01-13
204077	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	103716	McAree	Single Cell Mining Claim	2020-09-28
204100	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	104240	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
204119	Echo	Single Cell Mining Claim	2020-08-11	104241	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
204126	Echo, McAree	Single Cell Mining Claim	2021-04-05	105356	Laval	Single Cell Mining Claim	2021-02-12
204127	Echo, McAree	Single Cell Mining Claim	2020-09-28	105558	Laval	Single Cell Mining Claim	2020-09-30
204137	Pickerel	Single Cell Mining Claim	2021-01-13	106443	Laval	Single Cell Mining Claim	2020-09-30
204178	Webb	Single Cell Mining Claim	2021-01-24	106444	Laval	Single Cell Mining Claim	2020-09-30
204893	Webb	Single Cell Mining Claim	2021-01-24	106667	Laval	Single Cell Mining Claim	2020-09-30
204914	Pickerel	Single Cell Mining Claim	2021-01-13	107263	Laval	Single Cell Mining Claim	2021-02-12
204915	Pickerel	Single Cell Mining Claim	2021-01-13	107264	Laval	Single Cell Mining Claim	2021-02-12
204943	Echo	Single Cell Mining Claim	2021-02-09	109467	Laval	Single Cell Mining Claim	2020-09-30
204952	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	111935	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
204989	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	114918	McAree	Single Cell Mining Claim	2020-09-30
204990	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	114971	McAree	Single Cell Mining Claim	2020-09-28
205005	Echo	Single Cell Mining Claim	2025-04-15	115046	Jordan	Single Cell Mining Claim	2020-12-15
205023	Echo	Single Cell Mining Claim	2020-09-28	115070	Echo	Single Cell Mining Claim	2021-04-26
205578	Echo	Single Cell Mining Claim	2021-01-13	115091	Echo	Single Cell Mining Claim	2021-01-13
205579	Echo	Single Cell Mining Claim	2021-01-13	115111	Echo	Single Cell Mining Claim	2021-01-13
205612	McAree	Single Cell Mining Claim	2020-08-05	115600	Pickerel	Single Cell Mining Claim	2021-01-13
205613	Laval, McAree	Single Cell Mining Claim	2020-08-05	115601	Pickerel	Single Cell Mining Claim	2021-01-13
205614	Laval	Single Cell Mining Claim	2020-09-30	115831	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
205615	Laval	Single Cell Mining Claim	2020-12-04	115859	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2022-01-13
205616	Laval	Single Cell Mining Claim	2020-12-04	115860	Pickerel	Single Cell Mining Claim	2022-01-13
205662	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	116038	Jordan	Single Cell Mining Claim	2020-12-15
206222	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	116042	McAree	Single Cell Mining Claim	2020-09-28
206267	Laval	Single Cell Mining Claim	2021-02-12	116049	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
206273	Webb	Single Cell Mining Claim	2021-01-24	116050	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
206284	Jordan	Single Cell Mining Claim	2021-03-28	116105	Webb	Single Cell Mining Claim	2021-01-24
206290	Vermilion	Single Cell Mining Claim	2021-01-13	116169	Laval	Single Cell Mining Claim	2020-09-30
206298	Laval	Single Cell Mining Claim	2020-09-30	116171	Echo	Single Cell Mining Claim	2020-09-28
206299	Laval	Single Cell Mining Claim	2020-09-30	116254	Laval	Single Cell Mining Claim	2020-10-31
206319	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	116267	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
207561	Pickerel	Single Cell Mining Claim	2021-01-13	116268	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
208797	Echo	Single Cell Mining Claim	2020-08-11	116272	Jordan	Single Cell Mining Claim	2020-12-15
208798	Echo	Single Cell Mining Claim	2021-01-13	116278	Echo	Single Cell Mining Claim	2021-02-09
208816	Pickerel	Single Cell Mining Claim	2021-04-05	116279	Echo	Single Cell Mining Claim	2021-02-09
208840	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116344	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
208841	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116350	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
208842	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116368	Echo, Pickerel	Single Cell Mining Claim	2020-08-11
209121	Laval	Single Cell Mining Claim	2021-01-24	116404	Echo, Pickerel	Single Cell Mining Claim	2021-01-13
210118	McAree	Single Cell Mining Claim	2020-09-28	116443	Laval, McAree	Single Cell Mining Claim	2020-08-05
210220	Jordan	Single Cell Mining Claim	2020-12-15	116444	Laval	Single Cell Mining Claim	2020-09-30
210768	Echo	Single Cell Mining Claim	2021-01-13	116445	Laval	Single Cell Mining Claim	2020-09-30
211032	Laval	Single Cell Mining Claim	2020-09-30	116448	Webb	Single Cell Mining Claim	2021-01-24
211457	Echo	Single Cell Mining Claim	2021-02-12	116450	Jordan	Single Cell Mining Claim	2020-12-15
211458	Echo	Single Cell Mining Claim	2026-08-02	116489	McAree	Single Cell Mining Claim	2020-08-05
211494	Pickerel	Single Cell Mining Claim	2021-01-13	116490	Laval, McAree	Single Cell Mining Claim	2020-08-05
211509	Pickerel	Single Cell Mining Claim	2021-01-13	116544	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18
211516	Pickerel	Single Cell Mining Claim	2021-01-13	116549	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
211527	Jordan	Single Cell Mining Claim	2020-12-15	116594	Jordan	Single Cell Mining Claim	2021-04-20
211528	Jordan	Single Cell Mining Claim	2020-12-15	116596	Jordan	Single Cell Mining Claim	2020-12-15
211534	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116620	Jordan	Single Cell Mining Claim	2020-12-15
211535	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	116623	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
212170	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116725	Echo	Single Cell Mining Claim	2020-08-11
212171	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116791	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-18
212201	Drayton	Single Cell Mining Claim	2020-12-15	116826	Pickerel	Single Cell Mining Claim	2021-01-13
212231	Jordan	Single Cell Mining Claim	2020-12-15	116827	Pickerel	Single Cell Mining Claim	2021-02-12
212241	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	116912	Drayton	Single Cell Mining Claim	2020-12-15
212301	Laval	Single Cell Mining Claim	2020-09-30	116937	Jordan	Single Cell Mining Claim	2020-12-15
212759	Pickerel	Single Cell Mining Claim	2021-01-13	116938	Jordan	Single Cell Mining Claim	2020-12-15
212760	Pickerel	Single Cell Mining Claim	2021-01-13	116939	Jordan	Single Cell Mining Claim	2020-12-15
212761	Pickerel	Single Cell Mining Claim	2021-01-13	117089	Jordan	Single Cell Mining Claim	2020-12-15
212764	Jordan	Single Cell Mining Claim	2020-12-15	117096	Echo	Single Cell Mining Claim	2020-08-11
212803	Webb	Single Cell Mining Claim	2021-01-24	117097	Echo	Single Cell Mining Claim	2021-02-12
212804	Webb	Single Cell Mining Claim	2021-01-24	117098	Echo	Single Cell Mining Claim	2025-03-29
212875	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	117099	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
212876	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	117100	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
212877	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	117148	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
213428	Jordan	Single Cell Mining Claim	2020-12-15	117163	Pickerel	Single Cell Mining Claim	2021-01-13
213459	Jordan	Single Cell Mining Claim	2020-12-15	117169	Pickerel	Single Cell Mining Claim	2021-01-13
213507	McAree	Single Cell Mining Claim	2020-08-05	117170	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
213518	Pickerel	Single Cell Mining Claim	2021-01-13	117190	Jordan	Single Cell Mining Claim	2020-12-15
213519	Pickerel	Single Cell Mining Claim	2021-01-13	117672	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
213570	Laval	Single Cell Mining Claim	2020-08-30	117676	Echo	Single Cell Mining Claim	2020-08-11
214104	Pickerel	Single Cell Mining Claim	2021-04-05	117701	Pickerel	Single Cell Mining Claim	2021-01-13
214173	McAree	Single Cell Mining Claim	2020-08-05	117754	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
214209	Pickerel	Single Cell Mining Claim	2021-01-13	117755	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
214890	Laval	Single Cell Mining Claim	2021-01-24	117756	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
214895	Laval	Single Cell Mining Claim	2020-09-30	117802	Laval	Single Cell Mining Claim	2020-09-30
214896	Echo	Single Cell Mining Claim	2021-01-13	117810	Vermilion	Single Cell Mining Claim	2021-01-13
214897	Echo	Single Cell Mining Claim	2021-01-13	117811	Vermilion	Single Cell Mining Claim	2021-01-13
214900	Echo	Single Cell Mining Claim	2021-02-09	117817	Kabik Lake Area	Single Cell Mining Claim	2021-03-28
214901	Echo	Single Cell Mining Claim	2025-04-05	117888	Laval	Single Cell Mining Claim	2021-01-24
214902	Echo	Single Cell Mining Claim	2025-11-13	117889	Laval	Single Cell Mining Claim	2020-09-30
214920	Echo	Single Cell Mining Claim	2020-09-28	118176	Jordan	Single Cell Mining Claim	2020-12-15
214975	Pickerel	Single Cell Mining Claim	2021-02-12	118244	Pickerel	Single Cell Mining Claim	2021-01-13
214982	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	120327	Kabik Lake Area	Single Cell Mining Claim	2020-12-15

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
214983	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	120349	Jordan	Single Cell Mining Claim	2020-12-15
215059	Jordan	Single Cell Mining Claim	2020-12-15	120350	Jordan	Single Cell Mining Claim	2020-12-15
215060	Jordan	Single Cell Mining Claim	2020-12-15	120381	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
215187	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	120382	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
215190	Laval	Single Cell Mining Claim	2020-12-04	120383	Parnes Lake Area	Single Cell Mining Claim	2020-12-15
215191	Laval	Single Cell Mining Claim	2020-12-04	120429	McAree	Single Cell Mining Claim	2020-08-05
215620	Laval, Webb	Single Cell Mining Claim	2021-01-24	121009	Pickerel	Single Cell Mining Claim	2021-04-05
215621	Laval	Single Cell Mining Claim	2020-09-30	121010	Pickerel	Single Cell Mining Claim	2021-04-05
215628	Laval	Single Cell Mining Claim	2020-09-30	121075	McAree	Single Cell Mining Claim	2020-08-05
215629	Laval	Single Cell Mining Claim	2020-09-30	121122	Pickerel	Single Cell Mining Claim	2021-02-12
215630	Echo	Single Cell Mining Claim	2021-01-13	121123	Pickerel	Single Cell Mining Claim	2021-02-12
215631	Echo	Single Cell Mining Claim	2021-01-13	121124	Pickerel	Single Cell Mining Claim	2020-08-11
215634	Echo	Single Cell Mining Claim	2020-09-28	121373	Laval	Single Cell Mining Claim	2020-09-30
215704	Webb	Single Cell Mining Claim	2021-01-24	121667	Drayton, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
215705	Webb	Single Cell Mining Claim	2021-01-24	121746	Jordan	Single Cell Mining Claim	2020-12-15
215706	Webb	Single Cell Mining Claim	2021-01-24	121823	Laval	Single Cell Mining Claim	2020-09-30
215707	Webb	Single Cell Mining Claim	2021-01-24	121900	Laval	Single Cell Mining Claim	2020-12-04
215726	Jordan	Single Cell Mining Claim	2020-12-15	121901	Laval	Single Cell Mining Claim	2020-12-04
215730	Pickerel	Single Cell Mining Claim	2021-01-13	121902	Laval	Single Cell Mining Claim	2020-12-04
215733	Echo	Single Cell Mining Claim	2020-09-28	121903	Laval	Single Cell Mining Claim	2020-12-04
215745	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	122325	Laval	Single Cell Mining Claim	2020-09-30
215760	Echo	Single Cell Mining Claim	2021-02-09	122326	Laval	Single Cell Mining Claim	2020-09-30
215772	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	122327	Laval	Single Cell Mining Claim	2020-09-30
216315	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	122329	Echo	Single Cell Mining Claim	2021-01-13
216316	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	122331	Echo	Single Cell Mining Claim	2025-04-05
216322	Jordan	Single Cell Mining Claim	2021-03-28	122403	Laval	Single Cell Mining Claim	2021-01-24
216323	Jordan	Single Cell Mining Claim	2021-03-28	122431	Laval	Single Cell Mining Claim	2020-09-30
216324	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	122448	Jordan	Single Cell Mining Claim	2020-12-15
216340	Jordan	Single Cell Mining Claim	2020-12-15	123023	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
216347	Kabik Lake Area	Single Cell Mining Claim	2021-03-28	123024	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
216357	Pickerel	Single Cell Mining Claim	2020-08-11	123025	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
216358	Pickerel	Single Cell Mining Claim	2020-08-11	123030	Jordan	Single Cell Mining Claim	2021-03-28
216399	Echo	Single Cell Mining Claim	2021-01-13	123100	Echo	Single Cell Mining Claim	2021-01-13
216400	Echo	Single Cell Mining Claim	2021-01-13	123145	Laval	Single Cell Mining Claim	2020-12-04
216421	Echo	Single Cell Mining Claim	2021-02-12	123738	Echo	Single Cell Mining Claim	2020-09-28
216459	McAree	Single Cell Mining Claim	2020-08-05	123826	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
216460	McAree	Single Cell Mining Claim	2020-08-05	123827	Laval	Single Cell Mining Claim	2021-02-12
216461	Laval	Single Cell Mining Claim	2020-09-30	123828	Laval	Single Cell Mining Claim	2021-02-12
216462	Laval	Single Cell Mining Claim	2020-12-04	124215	Laval	Single Cell Mining Claim	2021-01-24
216463	Laval	Single Cell Mining Claim	2020-09-30	124385	Laval	Single Cell Mining Claim	2020-09-30
217013	Echo, McAree	Single Cell Mining Claim	2020-09-28	124401	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
217014	McAree	Single Cell Mining Claim	2020-09-28	124402	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
217015	McAree	Single Cell Mining Claim	2020-09-28	124937	Pickerel	Single Cell Mining Claim	2021-01-13
217046	Pickerel	Single Cell Mining Claim	2021-01-13	124938	Pickerel	Single Cell Mining Claim	2021-01-13
217047	Pickerel	Single Cell Mining Claim	2021-01-13	124942	Echo, Webb	Single Cell Mining Claim	2021-04-26
217049	Echo, McAree	Single Cell Mining Claim	2020-09-28	124943	Echo	Single Cell Mining Claim	2021-04-26
217065	Echo	Single Cell Mining Claim	2021-02-12	125260	Laval	Single Cell Mining Claim	2021-02-12
217091	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	125261	Laval	Single Cell Mining Claim	2021-02-12
217135	Laval	Single Cell Mining Claim	2021-02-12	125687	Pickerel	Single Cell Mining Claim	2021-01-13
217136	Laval	Single Cell Mining Claim	2021-02-12	126858	Laval	Single Cell Mining Claim	2021-02-12
217656	Laval	Single Cell Mining Claim	2020-09-30	126884	Echo, McAree	Single Cell Mining Claim	2025-09-30
217699	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	126885	McAree	Single Cell Mining Claim	2020-09-28
217700	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	126961	McAree	Single Cell Mining Claim	2020-09-28
217701	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	127543	Jordan	Single Cell Mining Claim	2020-12-15
217748	Laval	Single Cell Mining Claim	2020-09-30	127544	Jordan	Single Cell Mining Claim	2020-12-15
219031	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	127545	Jordan	Single Cell Mining Claim	2020-12-15
219661	Pickerel	Single Cell Mining Claim	2021-01-13	127597	Echo	Single Cell Mining Claim	2021-01-13
219662	Pickerel	Single Cell Mining Claim	2021-01-13	127598	Echo	Single Cell Mining Claim	2021-01-13
219663	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	127599	Echo	Single Cell Mining Claim	2020-08-11
219666	Echo	Single Cell Mining Claim	2020-09-28	128305	Pickerel	Single Cell Mining Claim	2021-01-13
220907	Kabik Lake Area	Single Cell Mining Claim	2021-01-13	128306	Pickerel	Single Cell Mining Claim	2021-01-13
220908	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	128335	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
221671	McAree	Single Cell Mining Claim	2020-09-28	128915	Jordan	Single Cell Mining Claim	2020-12-15
222299	Echo	Single Cell Mining Claim	2021-04-26	128977	Pickerel	Single Cell Mining Claim	2021-01-13
222300	Echo, Webb	Single Cell Mining Claim	2021-04-26	129011	Drayton	Single Cell Mining Claim	2020-12-15
222301	Echo	Single Cell Mining Claim	2021-04-26	129012	Drayton	Single Cell Mining Claim	2020-12-15
222327	Echo	Single Cell Mining Claim	2021-01-13	129508	Laval	Single Cell Mining Claim	2020-09-30
222328	Echo	Single Cell Mining Claim	2021-01-13	129554	McAree	Single Cell Mining Claim	2020-09-30
222992	Jordan	Single Cell Mining Claim	2020-12-15	129555	McAree	Single Cell Mining Claim	2020-09-28
223234	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15	129557	Drayton, Jordan	Single Cell Mining Claim	2020-12-15
223564	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	129564	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
223565	Pickerel	Single Cell Mining Claim	2021-01-13	129581	Jordan	Single Cell Mining Claim	2020-12-15
223569	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	129609	Webb	Single Cell Mining Claim	2021-01-24
223570	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	129612	Jordan	Single Cell Mining Claim	2020-12-15
223579	Jordan	Single Cell Mining Claim	2020-12-15	129646	Laval, McAree	Single Cell Mining Claim	2021-01-24
223927	Laval	Single Cell Mining Claim	2020-09-30	129691	Kabik Lake Area	Single Cell Mining Claim	2021-04-18
223928	Laval	Single Cell Mining Claim	2020-09-30	130020	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
224215	Jordan	Single Cell Mining Claim	2020-12-15	130021	Jordan	Single Cell Mining Claim	2020-12-15
224217	Drayton	Single Cell Mining Claim	2020-12-15	130296	Laval	Single Cell Mining Claim	2020-09-30
224241	Echo	Single Cell Mining Claim	2020-08-11	130305	Laval	Single Cell Mining Claim	2020-09-30
224242	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	130309	Laval	Single Cell Mining Claim	2021-01-24
224243	Pickerel	Single Cell Mining Claim	2021-01-13	130712	Jordan	Single Cell Mining Claim	2021-04-20
224244	Drayton, Jordan	Single Cell Mining Claim	2020-12-15	130981	Laval	Single Cell Mining Claim	2020-09-30
224248	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	130982	Laval	Single Cell Mining Claim	2020-09-30
224259	Pickerel	Single Cell Mining Claim	2021-01-13	130983	Laval	Single Cell Mining Claim	2020-12-04
224666	Laval	Single Cell Mining Claim	2020-09-30	131407	Jordan	Single Cell Mining Claim	2020-12-15
224944	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	131408	Jordan, Parnes Lake Area	Single Cell Mining Claim	2020-12-15
225523	McAree	Single Cell Mining Claim	2020-08-05	134204	Echo	Single Cell Mining Claim	2021-01-13
225573	Laval	Single Cell Mining Claim	2020-08-30	135251	McAree	Single Cell Mining Claim	2020-09-28
225600	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	135273	McAree	Single Cell Mining Claim	2020-09-28
225663	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	136994	Laval, Webb	Single Cell Mining Claim	2021-01-24
225712	Laval	Single Cell Mining Claim	2020-09-30	137949	Laval	Single Cell Mining Claim	2020-09-30
225713	Laval	Single Cell Mining Claim	2021-01-24	137950	Laval	Single Cell Mining Claim	2020-09-30
225714	Laval	Single Cell Mining Claim	2020-09-30	137951	Laval	Single Cell Mining Claim	2020-09-30
226548	Laval	Single Cell Mining Claim	2021-02-12	137952	Laval	Single Cell Mining Claim	2020-09-30
226982	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13	138858	Laval	Single Cell Mining Claim	2020-08-05
227058	Pickerel	Single Cell Mining Claim	2021-01-13	138905	Laval	Single Cell Mining Claim	2021-02-12

Tenure ID	Township / Area	Tenure Type	Anniversary Date	Tenure ID	Township / Area	Tenure Type	Anniversary Date
227663	Echo	Single Cell Mining Claim	2021-01-13	139221	Jordan	Single Cell Mining Claim	2021-03-28
227664	Echo	Single Cell Mining Claim	2021-01-13	139598	Laval	Single Cell Mining Claim	2020-09-30
228073	Laval	Single Cell Mining Claim	2020-09-30	141432	Pickerel	Single Cell Mining Claim	2021-01-13
228960	McAree	Single Cell Mining Claim	2020-09-28	141433	Pickerel	Single Cell Mining Claim	2021-01-13
228961	McAree	Single Cell Mining Claim	2020-09-28	141435	Echo	Single Cell Mining Claim	2020-09-28
229389	Laval	Single Cell Mining Claim	2020-09-30	141436	Echo	Single Cell Mining Claim	2020-09-28
229543	Jordan	Single Cell Mining Claim	2020-12-15	141714	Laval	Single Cell Mining Claim	2021-01-24
229564	Echo	Single Cell Mining Claim	2021-04-26	142420	Laval	Single Cell Mining Claim	2021-02-12
229565	Echo	Single Cell Mining Claim	2021-04-26	142682	Echo	Single Cell Mining Claim	2020-08-11
230169	Laval	Single Cell Mining Claim	2020-09-30	143033	Laval, Webb	Single Cell Mining Claim	2021-01-24
230170	Laval	Single Cell Mining Claim	2021-01-24	143456	Jordan	Single Cell Mining Claim	2020-12-15
230286	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	143464	Echo	Single Cell Mining Claim	2020-08-11
230310	Pickerel	Single Cell Mining Claim	2021-01-13	143465	Echo	Single Cell Mining Claim	2021-02-12
230321	Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13	143466	Echo	Single Cell Mining Claim	2021-02-12
230548	Jordan	Single Cell Mining Claim	2021-04-20	143467	Echo	Single Cell Mining Claim	2021-02-12
230900	Jordan	Single Cell Mining Claim	2020-12-15	143468	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05
230990	Drayton	Single Cell Mining Claim	2020-12-15	144756	Echo	Single Cell Mining Claim	2021-04-26
231009	McAree	Single Cell Mining Claim	2020-09-28	144781	Echo	Single Cell Mining Claim	2021-01-13
231018	Jordan	Single Cell Mining Claim	2020-12-15	145341	Pickerel	Single Cell Mining Claim	2021-01-13
231546	Jordan	Single Cell Mining Claim	2020-12-15	145342	Pickerel	Single Cell Mining Claim	2021-01-13
231583	Webb	Single Cell Mining Claim	2021-01-24	145343	Pickerel	Single Cell Mining Claim	2021-01-13
231584	Webb	Single Cell Mining Claim	2021-01-24	145371	Pickerel	Single Cell Mining Claim	2021-01-13
231633	McAree	Single Cell Mining Claim	2020-09-28	145395	Jordan	Single Cell Mining Claim	2020-12-15
231961	Laval	Single Cell Mining Claim	2020-09-30	145396	Jordan	Single Cell Mining Claim	2020-12-15
232188	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	145492	Jordan, Kabik Lake Area, Pickerel, Vermilion	Single Cell Mining Claim	2021-01-13
232221	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	145493	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
232237	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-04-20	145500	Kabik Lake Area	Single Cell Mining Claim	2020-12-15
232238	Jordan, Kabik Lake Area	Single Cell Mining Claim	2021-03-28	148834	Laval	Single Cell Mining Claim	2020-09-30
232239	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	148835	Laval	Single Cell Mining Claim	2020-09-30
232240	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	150149	Echo	Single Cell Mining Claim	2021-01-13
232271	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	151621	Echo	Single Cell Mining Claim	2021-01-13
232272	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	151622	Echo	Single Cell Mining Claim	2021-01-13
232312	McAree	Single Cell Mining Claim	2020-08-05	151623	Echo	Single Cell Mining Claim	2021-01-13
232875	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	151646	Echo	Single Cell Mining Claim	2021-02-12
232876	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-04-05	151670	Laval, McAree	Single Cell Mining Claim	2020-08-05
232942	Kabik Lake Area	Single Cell Mining Claim	2021-04-18	151671	McAree	Single Cell Mining Claim	2020-08-05
232946	McAree	Single Cell Mining Claim	2020-08-05	151721	McAree	Single Cell Mining Claim	2020-09-28
232977	Pickerel	Single Cell Mining Claim	2021-02-12	151742	Pickerel	Single Cell Mining Claim	2021-01-13
232990	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	152294	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
233653	Laval	Single Cell Mining Claim	2020-09-30	152345	Webb	Single Cell Mining Claim	2021-01-24
233658	Echo	Single Cell Mining Claim	2021-02-09	152356	Vermilion	Single Cell Mining Claim	2021-01-13
233727	Parnes Lake Area	Single Cell Mining Claim	2020-12-15	152357	Vermilion	Single Cell Mining Claim	2021-01-13
233728	Laval	Single Cell Mining Claim	2020-12-04	152371	Vermilion	Single Cell Mining Claim	2021-01-13
233983	Echo	Single Cell Mining Claim	2021-01-13	152375	Laval	Single Cell Mining Claim	2020-09-30
234234	Laval, Webb	Single Cell Mining Claim	2021-01-24	152378	Laval	Single Cell Mining Claim	2020-10-31
234235	Laval	Single Cell Mining Claim	2021-01-24	152403	Kabik Lake Area	Single Cell Mining Claim	2021-04-05
234249	Laval	Single Cell Mining Claim	2020-09-30	153623	Pickerel	Single Cell Mining Claim	2021-01-13
234250	Drayton	Boundary Cell Mining Claim	2020-12-15	153871	Laval	Single Cell Mining Claim	2020-09-30
234267	Laval	Single Cell Mining Claim	2020-09-30	154210	Pickerel	Single Cell Mining Claim	2021-04-05
234272	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	154232	Echo	Single Cell Mining Claim	2025-09-17
234276	Jordan	Single Cell Mining Claim	2020-12-15	155481	Kabik Lake Area, Pickerel	Single Cell Mining Claim	2021-01-13
234277	Jordan	Single Cell Mining Claim	2020-12-15	156254	McAree	Single Cell Mining Claim	2020-09-28
234285	Echo	Single Cell Mining Claim	2021-02-09	156838	Jordan	Single Cell Mining Claim	2020-12-15
234297	Kabik Lake Area	Single Cell Mining Claim	2021-04-05	156857	Echo	Single Cell Mining Claim	2021-04-26
234345	Kabik Lake Area	Single Cell Mining Claim	2020-12-15	157589	Pickerel	Single Cell Mining Claim	2021-01-13

Figure 4-5: Goldlund-Miller Property Mineral Rights Map



Source: Treasury Metals (2021)

4.3 Project Ownership

4.3.1 Goliath Property

Treasury Metals, a former subsidiary of Laramide Resources Ltd. (Laramide), was spun out of Laramide as a dividend to Laramide's shareholders. Treasury Metals was listed and began trading on the Toronto Stock Exchange (TSX) exchange on August 19, 2008, under the trade symbol "TML".

The Goliath property consists of two historic properties that were consolidated into one: the larger Thunder Lake property, purchased from Teck and Corona, and the Laramide property.

4.3.2 Goldlund-Miller Property

On June 3, 2020, Treasury Metals entered into a definitive share purchase agreement with First Mining to acquire the Goldlund-Miller property through the acquisition of Tamaka. The mineral rights to the Goldlund-Miller property are held by Goldlund Resources Inc., a wholly owned subsidiary of Tamaka. On August 7, 2020, the acquisition was completed whereby Treasury Metals acquired all the issued and outstanding shares of Tamaka.

4.4 Property Agreements

4.4.1 Goliath Property

4.4.1.1 Thunder Lake Property Acquisition Timeline

Laramide closed its purchase transaction of the Thunder Lake property as of October 2007 (Laramide Press Release: October 4, 2007). Laramide purchased, through its former wholly owned subsidiary, Divine Lake Exploration Corp. (now "Treasury Metals Inc."), 100% of Corona's (82%) and Teck's (18%) respective interests in the Thunder Lake property. On closing, Corona received from Laramide a cash consideration of $5 million and under the terms of the agreement Corona received from Laramide aggregate cash payments of $10 million and a 10% interest in Treasury Metals after it became a public company. Teck received cash consideration of approximately $1,137,299 at closing and received from Laramide aggregate cash payment of $2,274,598 and a 2.27% interest in Treasury Metals. The balance of consideration for the properties was payable as follows:

- cash payment of $6,137,229 – 60 days after the closing date
- cash payment of $6,137,229 – 120 days after the closing date
- 12.27% of the common shares of Treasury Metals issued and outstanding on completion of a transaction pursuant to which Treasury Metals becomes a public company.

Treasury Metals announced in a press release (August 26, 2008) that it had completed the final instalment of the purchase price to Corona and Teck pursuant to the purchase agreement. In accordance with the 2007 Purchase Agreement, Corona and Teck shall receive, for no additional consideration, that number of common shares sufficient for each of Corona and Teck to maintain their respective percentage interest in the Company of 10% and 2.27% until the Company receives aggregate proceeds from the insurance of common shares of $7.5 million. This threshold has been reached. Laramide and Treasury Metals have met all of the obligations to Teck and Corona.

4.4.1.2 Laramide Property

As part of the spin-out of Treasury Metals, Laramide transferred to Treasury Metals its Goliath property (herein referred to as the Laramide property) and certain of Laramide's other non-uranium assets. As of May 2010, Laramide held approximately 13.7% of the issued and outstanding Treasury Metals common shares. Treasury Metals owns 100% of the Laramide property subject to royalties as detailed in Section 4.7.

4.4.1.3 2009 Property Expansion

In 2009, the Goliath property was expanded from its original size through the combined staking and acquisition of 18 unpatented mining claims and the signing of an option agreement pursuant to which Treasury Metals has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson property) contiguous to the Goliath project.

4.4.1.4 Unpatented Mining Claims

In 2009, the Company acquired and/or staked 18 additional unpatented mining claims (111 units) totalling 1,776 ha. These 18 additional claims are in the Hartman and Zealand townships.

4.4.1.4.1 2009 Brisson Property

On December 11, 2009, the Company entered into an option agreement to acquire a 100% interest in the mining rights (only) of certain patented lands (40.8711 ha) from Edward Henry Brisson (the Brisson property) located immediately west and contiguous to the Goliath project. Under the terms of the agreement, the Company made option payments totalling $100,000 and issued common shares of the Company equal to $100,000 based on the market price of the date issue. The property purchase (surface rights) was completed on March 31, 2011.

4.4.1.5 2010-2011 Property Expansion & Dryden Tree Nursery area

In 2010 and 2011 the Goliath property was further expanded by (1) acquiring the Dryden Tree Nursery; (2) staking three unpatented mining claims; and (3) making a final option payment. These expansions are described below.

On November 5, 2010, the Company acquired a 100% interest in two private land parcels consisting of mineral and surface rights (PIN 42089-0066, 100.62 ha) and the surface rights (PIN 42089-0065, 26.20 ha) formerly known as the Dryden Tree Nursery. The Dryden Tree Nursery is situated immediately northwest and contiguous to the Goliath property and covers 126.82 ha.

In 2011, the Company staked three additional unpatented mining claims (20 units) totalling 320 ha in Hartman Township.

On April 12, 2011, the Company completed the final payment on the option to purchase the LeClerc surface rights (only) patent (Parcel 34303, 16.59 ha) located immediately east of the Thunder Lake deposit within the Goliath project area.

4.4.1.6 2014 Mining Leases

Effective October 1, 2014, 11 Treasury Metals unpatented mining claims were converted to three 21-year mining leases which expire on September 30, 2035.

Mining lease 109532 has mining and surface rights covering 131.523 ha in N1/2 Lot 4, Concession 4 and S1/2 Lot 4, Concession 5 of Zealand Township and comprises all of mining claims K1119541, 1119542, K1119547, K1119548, K1119549, K1119550, K1119559 and K1119560, being all that land and land under water.

Mining lease 109533 has mining rights only covering 65.559 ha in Lot 5, Concession 5 of Zealand Township and comprises all of mining claims K1145301 and K3017938, being all that land and land under water.

Mining lease 109534 has mining rights only covering 63.940 ha in Lot 7, Concession 4 of Zealand Township and comprises of all of mining claim K1145300, being all that land and land under water.

4.4.1.7 Application for Mining Leases (Application)

In 2019, Treasury Metals made a request for a lease on 38 mining claims (in the Zealand and Hartman townships). As of the date or this report, the leasing process was still in progress.

4.4.2 Goldlund-Miller Property

4.4.2.1 Tamaka

Thirty-six claim units totalling 576 ha were optioned from an arm's-length vendor (the Vendors) through Goldlund Resources Inc. The terms of the agreement with the Vendors stated that Tamaka Gold Corporation (Tamaka) must spend $1 million by September 5, 2009 to earn a 100% interest in the claims subject to a 1% NSR. The $1 million commitment was fulfilled, and the title of the claims was transferred by the vendors to Goldlund Resources Inc. in 2009.

4.4.2.2 First Mining

On June 17, 2016, First Mining Gold Corp. (First Mining) announced the completion of the amalgamation with Tamaka. The amalgamation resulted in Tamaka becoming a wholly owned subsidiary of First Mining. First Mining issued 92.5 million common shares of First Mining to the shareholders of Tamaka as part of the transaction.

4.4.2.3 Treasury Metals

On June 3, 2020, Treasury Metals announced it had entered into a definitive share purchase agreement with First Mining to acquire the Goldlund-Miller property through the acquisition of Tamaka. The mineral rights to the Goldlund-Miller property are held by Goldlund Resources Inc., a wholly owned subsidiary of Tamaka.

On August 7, 2020, the acquisition was completed whereby Treasury Metals acquired all of the issued and outstanding shares of Tamaka. Under the terms of the agreement, First Mining shall receive the following:

- 130 million common shares (Common Shares) of Treasury Metals (the Share Consideration).
- 35 million Common Share purchase warrants of Treasury Metals (the Warrants), with each Warrant entitling the holder thereof to purchase one Common Share at an exercise price of $0.50 for a period of 36 months following the closing of the Transaction (the Warrant Consideration).

- A 1.5% net smelter returns royalty covering all the Goldlund claims (the Goldlund Royalty), with the option for Treasury Metals to buy-back 0.5% of the Goldlund Royalty for $5 million.
- A milestone cash payment of $5 million, with 50% payable upon receipt of a final and binding mining lease under the Mining Act (Ontario) to extract “ore” from an open pit mine at Goldlund, and the remaining 50% payable upon the extraction of 300,000 tonnes of “ore” from a mine at Goldlund.

4.5 Surface Rights

The surface rights controlled by Treasury Metals are sufficient to support the proposed mining operation and the access to power, water, mining personnel are also sufficient.

4.5.1 Goliath Property

Treasury Metals holds the surface rights on 10 patents, a portion of one additional patent (PAT-46017), six land parcels, and four mining leases on the Goliath property.

4.5.2 Goldlund-Miller Property

Treasury Metals holds the surface rights on the 27 patents and one mining lease on the Goldlund-Miller property. However, for the Licence of Occupation, only mineral rights have been granted.

4.6 Royalties and Encumbrances

4.6.1 Goliath Property

The Goliath property is held 100% by Treasury Metals, subject to certain underlying royalties and payment obligations on 13 of the 21 land parcels, totalling approximately $103,500 per year (see Table 4-5 for details).

Treasury Metals also has an option agreement pursuant to which Treasury Metals has the right to acquire a 100% interest in the mining rights (only) of certain patented lands (the Brisson property – 40.8711 ha) located immediately west and contiguous to the Goliath project.

The option on one patented land parcel to earn in 100% as described for the Brisson property was completed in March 2011.

The Goliath and Goldlund properties are subject to a royalty with an affiliate of Sprott Resources Streaming and Royalty Corp. (Sprott) whereby Sprott will receive a 2.2% NSR on all minerals produced on the Goliath Gold Complex for the life of the project. The Company has the right to repurchase 50% of this royalty until December 31, 2028 for various purchase prices, at the Company’s sole discretion, and the royalty also reduces by 50% upon the production of 1.5 million ounces of gold.

Table 4-5: Options and Royalty Obligations, Patented Land Parcels – Goliath Property

Party	Parcel ID	Advance Royalty (Per Year)	Due Date	Option Amount	NSR (%)	Comments
Lundmark[1]	41941	C$50,000 **	January 1	-	2.0	
Collins[1]	17395	-	-	-	2.0	
Sheridan[1]	21374	-	-	-	1.0	
Johnson[1]	15401	-	-	-	2.0	
Hudak[1]	21609	US$3,500 *	January 1	-	2.0	
Fraser[1]	15395	C$50,000	January 1	-	2.0	
Delk[2]	24724	-	-	-	2.5	
Davenport[2]	19088	-	-	-	2.0	
Jones[3]	41215	-	-	-	2.5	
Nemeth[2]	6556	-	-	-	2.0	
Sterling[4]	4822	-	-	-	2.0	
Medlee [4]	21553	-	-	-	2.5	
Schultz[4]	13492	-	-	-	2.0	Includes 3 patents
Brisson[5]	23R2434	-	-	-	-	
Total C$		$100,000				
Total US$		$3,500				

Notes: *subject to withholding tax. (1) Thunder Lake West; (2) Thunder Lake East; (3) Jones property; (4) Laramide property; (5) surface rights.

4.6.2 Goldlund-Miller Property

Royalties pertaining to the Goldlund-Miller property as defined in this document are as follows:

- The Goldlund Mines Limited Royalty Agreement, dated December 10, 2003, consists of six patented claims as well as the three patented claims covered by the Mining Lease. Goldlund Mines will receive a 1% NSR on any ore mined above 50 m below the existing shaft collar as of the date of the agreement. Goldlund Resources is entitled to a right of first refusal in the event Goldlund Mines wishes to dispose of its interest in the NSR. Goldlund Resources has the right but not the obligation to purchase one-half of the NSR for $500,000 at any time within three years from the date of the royalty agreement. This right has now expired.

- The Rio Algom Limited Option Agreement, dated August 28, 2014, consists of 21 patented claims. Goldlund Resources will pay a 2.5% NSR and will have the right but not the obligation to purchase the NSR in its entirety for a one-time payment of $2.5 million with a 10-day notification of intent to exercise the purchase right. Goldlund Resources is entitled to a right of first refusal if Rio Algom Limited wishes to sell the NSR.

- As part of the purchase agreement of Goldlund from First Mining, First Mining was provided a 1.5% NSR royalty (later purchased by an affiliate of Sprott Resources Streaming and Royalty Corp. (Sprott)) covering all of the Goldlund claims (the "Goldlund Royalty"), with the option for Treasury to buy back 0.5% of the Goldlund Royalty for $5.0 million.

Royalties pertaining to areas outside the resource as defined in this document:

- The 1074127 Ontario Limited Agreement, dated October 18, 2011, consists of 13 mining claims located in the Patricia and Kenora Mining districts of the Province of Ontario. 1074127 Ontario Limited (the 'Vendor') retains a 2% NSR in accordance with industry practice on the sale of all minerals from the property. Goldlund Resources has the sole and exclusive option to purchase 100% of the 2% NSR at any time for the sum of $1.5 million and has a right of first refusal in the event that the Vendor wishes to dispose of its interest in the NSR.

4.7 Permitting Considerations

4.7.1 Goliath Property

Treasury Metals warrants that it possesses all permits required to execute the exploration activities it has undertaken to date on the property.

4.7.2 Goldlund-Miller Property

Treasury Metals warrants that it possesses all permits required to execute exploration activities on the Goldlund-Miller project.

4.8 Environmental Considerations

4.8.1 Goliath Property

There are no known environmental liabilities associated with the Goliath property, other than those normally expected due to historical exploration and mining activities and associated historical mine workings.

All closure works associated with the former bulk sample workings conducted by Teck have been completed in accordance with the Mine Rehabilitation Code and the Mine Closure Plan. As detailed, all mine hazards observed on site have been addressed in the Closure Plan and the site is consistent with the Closure Plan. Rehabilitation is proceeding as per the Closure Plan and in accordance with Part VII of the Mining Act, O. Reg. 240/00, and the Mine Rehabilitation Code.

4.8.2 Goldlund-Miller Property

The QPs are unaware of any environmental liabilities associated with the Goldlund-Miller property related to the historical operation that are the responsibility of Treasury Metals. The QPs are unaware of any additional environmental liabilities or other factors and risks that may affect access, title, or ability that would prevent Treasury Metals from conducting exploration activities on the property.

The Goldlund project has two historic shafts that have been capped, an underground portal that has been blocked, a small open pit that is partially flooded, a waste rock stockpile, a mineralized material stockpile, a building housing the original mill on the property, and a small tailing containment facility. All have been overgrown with vegetation.

Treasury Metals will continue to evaluate and work collaboratively with regulators to ensure that all aspects of historical workings and their long-term implications are addressed as part of the development of the Goldlund-Miller Property.

4.9 Social License Considerations

4.9.1 Lac Seul First Nation

On September 1, 2011, Tamaka entered into a negotiation protocol with the Ojibway of Lac Seul First Nation (LSFN). The negotiation protocol establishes a committee through which Tamaka and LSFN will negotiate exploration activities on certain lands in the District of Sioux over which the LSFN asserts traditional territory rights. Under the negotiation protocol, Tamaka must also consult with LSFN from time to time regarding its exploration activities, includingeconomic and business opportunities, environmental matters, and training, employment and retention programs for LSFN members mutually beneficial to the Company and LSFN and the rights, if any, asserted by other First Nations over the subject area. As consideration for LSFN's consultations, advice and assistance, Tamaka shall pay to LSFN, in connection with each drillhole conducted by Tamaka, $200 per drillhole setup and $1.50/m of drilling, and a one-time payment of 71,433 units (each unit being one Tamaka share and one warrant with an agreed value of $1.05 per unit or $75,005 in the aggregate), which were issued on execution of the agreement. As a result of the Amalgamation, these units were converted into units of First Mining. The negotiation protocol contemplates that an agreement to cover exploration will be entered into once the Goldlund project is further advanced. Treasury has not entered into an exploration agreement with LSFN at this time. The negotiation protocol contemplates that an agreement to cover exploration will be entered into once the Goldlund project is further advanced. Treasury has not entered into an exploration agreement with LSFN at this time.

4.9.2 Wabigoon Lake Ojibway Nation

On September 13, 2011, Tamaka entered into a Memorandum of Understanding (MOU) with Wabigoon Lake Ojibway Nation (WLON) and a community relations services agreement with Wabigoon Lake Development Corporation (WLDC). The MOU governs the Company's conduct with respect to the exploration activities it undertakes in respect of the Goldlund project on land over which WLON asserts traditional territory rights. Pursuant to the MOU, Tamaka must notify WLDC of anticipated exploration activities, provide certain training, employment and business opportunities to the WLDC, and cover costs incurred in connection with the monthly meetings of a working group established under the MOU and any community meetings held in connection with the MOU.

WLDC provides ongoing advisory and consultation services with respect to Tamaka's obligations under the MOU based on the community relations services agreement, which confirms the financial commitments to cover the costs described in the MOU, including for WLON's capacity for the implementation of the agreement. As consideration for WLDC's services, Tamaka shall pay to WLDC, in connection with each drillhole conducted by Tamaka, $200 per drillhole setup and $1.50/m of drilling, and a one-time payment of 71,433 units (each unit being one Tamaka share and one warrant with an agreed value of $1.05 per unit or $75,005 in the aggregate) which were issued on execution of the agreement. As a result of the Amalgamation, these units were converted into units of First Mining shares.

The MOU contemplates that an exploration agreement, and them Impact and Benefits Agreements will be entered into once the Goldlund project is further advanced. Treasury Metals has not entered into any exploration agreement with WLON at this time.

4.10 QP Opinion

There are no other known significant factors or risks that may affect access, title, or the right or ability to perform work on the Goliath Complex Project at this time.

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

5.1 Accessibility

The Goliath project is located in the Kenora Mining Division in northwestern Ontario, approximately 4 km northwest of the Village of Wabigoon, 20 km east of Dryden, and 2 km north of the Trans-Canada Highway 17. The Goldlund and Miller projects are located between Dryden and Sioux Lookout, about 30 km northeast of the Goliath project, off Highway 72. Aerial imagery of the Goliath project and the Goldlund project is provided in Figure 5-1 and Figure 5-2, respectively.

Access to the Goliath project is north from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Anderson and Tree Nursery roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals. Access to the Goldlund site is east off Highway 72 via Goldlund Mine Road. The Miller project site is accessed via forestry road east off Highway 72. Access roads for the Goldlund and Miller sites are maintained by the Sustainable Forest Licence Holder (Domtar) for the area.

Figure 5-1: Goliath Project Office

Source: Treasury Metals, (2021).

Figure 5-2: Goldlund Project Site

Source: Treasury Metals, (2021).

5.2 Climate

Located in the west-central portion of the Boreal Shield Ecozone, the Goliath Gold Complex area experiences a continental climate generally characterized by short, mild summers and long, cold winters with relatively low precipitation. The terrain is generally flat and absent of orographic features that can block air masses or produce localized increases in precipitation. Annual temperatures range from 27°C to -26°C with an average rainfall between 60 and 80 cm and average snowfall between 1.3 and 2.3 m.

5.3 Local Resources and Infrastructure

All major industrial services and supplies are available in Dryden and Sioux Lookout and the area is serviced by both the Dryden Airport and Sioux Lookout Airport. The Goliath project is located 20 km from Dryden, which has a population of 5,586 according to the Statistics Canada 2016 census. The Goldlund and Miller projects are located 43 km and 35 km, respectively, south of Sioux Lookout, which has a population of 5,272. The Goliath Gold Complex is located about 300 km northwest of the City of Thunder Bay, a major economic centre along the Trans-Canada Highway and port at the northwest head of the St. Lawrence Seaway on Lake Superior.

The Complex is located in an area used by the public for recreational fishing, hunting, boating, and commercial activities, including tourism. Traditional land and resource use is also practiced by a number of Indigenous communities. The local economy is largely based on forestry and tourism.

Major and minor hydro transmission lines cross portions of the Goliath project area. The Canadian Pacific Railway line is located approximately 2 km to the southwest, parallel to Highway 17. The Trans-Canada natural gas pipeline crosses portions of the Goliath property. The closest centre of active mining operations is in the Red Lake area, approximately 155 km northwest of the project; however, northwestern Ontario generally possesses the necessary labour and infrastructure to support new exploration and mining operations.

At this time, Treasury Metals holds the sufficient surface rights necessary for any potential future mining operations including tailings storage areas, waste disposal areas, and a processing plant.

5.4 Physiography

The area is typical of glaciated terrain of the Canadian Shield. The topography overall is gently rolling, with glaciated high points seldom exceeding 50 m above local lake levels. Elevations across the Goliath Gold Complex area are generally between 370 and 430 masl. The localized topography levels range from of 390 to 400 masl in the principal deposit area at the Goliath property, from 380 masl to upwards of 430 masl at the Goldlund property, and from 390 masl to 400 masl at the Miller property.

The Goldlund deposit area contains a number of glaciated bedrock intrusions opposed to the flat till of the Goliath area. Low ground is covered by deep glacial till and frequent small lakes and/or swamps.

The Complex is located within the Ontario Shield Ecozone, which is characterized by extensive wetlands and boreal forests. Typical tree species include trembling aspen (Populus tremuloides), balsam poplar (Populus balsamifera), white and black spruces (Picea glauca, Picea marina), white birch (Betula papyrifera) and willow (Salix spp.).

5.5 Seismicity

Seismic activity in the Dryden and Goliath area is generally low. The Canadian Hazards Information Service (CHIS), a part of the Geological Survey of Canada (GSC) conducted seismic monitoring programs in the northern Ontario and eastern Manitoba portions of the Canadian Shield. The number of earthquakes documented in northern Ontario represented one of the lower densities in eastern Canada (Adams et al, 2015).

6 HISTORY

Text for this section of the report was extracted from the NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex prepared by Ausenco in 2021.

6.1 Exploration History

The Goliath Complex now includes both the Goliath and Goldlund-Miller Properties. These properties were developed independent of each other and have their own unique history.

6.1.1 Goliath Property – Early History

The first gold mining on record in the region was in Van Horne Township in the early 1900s with very limited gold production from auriferous veining in biotite schist within the regional Wabigoon fault system. Sporadic exploration was carried out along the belt throughout the 1900s with only limited documentation of exploration activity conducted on the property.

The earliest known government report covering the larger Dryden-Sioux Lookout Belt is the Ontario Department of Mines Report and Geology Map by Satterly (1941). Ministry of Northern Development and Mines (MNDM) geologist Gary Beakhouse has written a number of reports covering the geology of the region and the Western Superior Province (Beakhouse, 2010, 2002, 2001, 2000 and Beakhouse et al1995). Reconnaissance lake sediment geochemistry and detailed airborne geophysical surveys are also available for Thunder Lake and surrounding areas (Hornbrook and Fisk 1989, and Ontario Geological Survey, 1987).

According to Page (1991), the first reference to exploration work conducted on the property describes an "interesting contact between amphibolite, laminated grey gneisses, and beds of mica-tourmaline schists on Sheridan Option legacy claim SV200". There is no record of further work on the property until the mid-1950s.

In 1956-57, Compton-Wabigoon conducted geological mapping, magnetometer surveys, and the completion of two diamond drillholes totalling 458 m to explore the mineral potential of the major iron formation unit located in Lots 1-4, Concession V and VI, along the northern boundary of the property. Also in 1956, G. L. Pidgeon completed surface work and one shallow drillhole (drilled south) testing a sphalerite showing in the south half of Lot 6, Concession 4 (Fraser Option legacy claim 0134). The showing and drill collar was located in the field by Teck, but subsequent surface sampling of sphalerite-rich mineralization did not return any significant gold values (best 10 ppb). Teck determined the drillhole attempted to test the showing down-dip on the mineralization. This showing had been previously sampled by Satterly in 1941 with similar negative results (Page, 1991).

From 1966 to 1968, Algoma Steel Corp. Ltd. conducted geological mapping and drilled five holes totalling 304 m. This program was concentrated on the main iron formation focused in the same area as Compton-Wabigoon's work 10 years earlier (Page, 1991). Inco completed ground surveys and one drillhole (52 m) in the vicinity of Teck grid coordinates L18E, 4+00E. Teck could not locate the drill site in the field and no assays were reported in the drill log; however, the hole is located within 50 m of a strong linear (>1,000 m) VLF-EM conductor which Teck believes was the probable drill target.

6.1.2 Goliath Property – 1989 to 1999

The exploration history on the property is described in a number of technical reports prepared for Treasury Metals which is summarized below (Roy, 2010, Roy et al., 2012; Roy and Trinder, 2011; Roy and Trinder, 2008; Wetherup and Kelso, 2008).

Three major mining companies conducted exploration work on the Thunder Lake gold deposit (Goliath deposit) from 1989 to 1999 (last field work 1998): Teck, Corona, and Laramide. At that time, the property held by all three companies covered more than 1,300 ha. Teck held the majority of the property and all of the surface exposure.

Exploration and resource development work at Goliath was undertaken by Teck from 1989 to 1999 on what was then called the "Thunder Lake property". During this period, the property was divided into two properties called "Thunder Lake East" and "Thunder Lake West". The property was optioned to Corona, previously called Continental Caretech Corporation (CCC), by which CCC could earn an interest in the project under terms of an initial agreement dated January 3, 1994. Corona funded the exploration work from 1994 to 1999, but Teck remained the project operator both designing and running all field exploration activities.

The total exploration expenditures spent on the property from 1989 to 1999 by Teck and Teck-Corona was approximately $9.7 million (Page, 1995a; Page, 1995b, Page et al., 1999a and b; Page and Waqué, 1999; Page and Waqué, 1998).

6.1.3 Goliath Property – Teck Exploration 1989 to 1993

It was not until 1989 that reconnaissance exploration work by Teck, in search of Hemlo-type gold mineralization in the region as part of their Quest project, identified a large weakly altered felsic rock unit containing sporadic anomalous values in gold, silver, zinc, and lead extending through parts of Lots 3 through 8 of Concession 4 in Zealand Township. Grab assays averaging 2.98 g/t Au, 24.7 g/t Ag, 1.20% Zn, and 0.43% Pb were reported by Page (1991). Weakly altered quartz-eye felsic rock (muscovite-sericite schist unit?) returned an assay of 630 ppb Au. This discovery was followed by land acquisition and exploratory work by Teck.

The exploration program during that period consisted of establishing a 104.7 line-km exploration grid across the property, geological mapping, prospecting, sampling, and geophysical surveying consisting of ground magnetic, induced polarization (IP) surveys and VLF-EM surveys. Eleven samples were submitted for petrographic analyses and one outcrop was stripped using a bulldozer (on line L15+80W, 2+25N).

A short, seven-hole diamond drill program was completed to test chargeability anomalies. It is during this program that the Goliath deposit (Main Zone) Hole TL1 was discovered by Teck in the fall of 1990, which prompted resource definition and exploration work on the property throughout the 1990s.

It was determined there was a positive correlation between gold content and the presence of sphalerite and galena, but the highest gold assays were generally associated with siliceous intervals containing only 1% to 3% zinc, and 0.1% to 1.5% lead.

The whole rock geochemistry indicated the felsic schists (muscovite-sericite schist) generally represented the altered equivalents of massive to gneissic felsic (volcanic?) rocks and are moderately enriched in silica and potassium, moderately to strongly depleted in sodium, and strongly depleted in calcium and magnesium.

Drilling programs were subsequently conducted in each of the next three years (1991, 1992, 1993) with the completion of an additional 49 drillholes focused on evaluating the resource potential of the main gold deposit.

In 1993, the property was optioned to Corona. Table 6-1 summarizes the exploration activities conducted by Teck from 1990 to 1993.

Table 6-1: Teck Exploration Summary from 1990 to 1993

Year	Company & Work Locations	Work Completed
1990	Thunder Lake West	Reconnaissance exploration
	Thunder Lake West	Line cutting (104.7 line-km), mapping of exploration grid
	Thunder Lake West	Geological mapping and prospecting
	Thunder Lake West	122 grab and chip samples collected (32 sent for whole rock)
	Thunder Lake West	11 petrographic samples completed; one outcrop stripped
	Independent Exploration Services	Ground magnetic, VLF-EM survey (entire grid), 31.8 line-km IP
	SAGAX Geophysique Inc.	31.8 line-km of IP
	SAGAX Geophysique Inc.	Diamond drilling program – 7 holes (TL1 to TL7) TL1 Goliath discovery hole
1991	Thunder Lake West	Diamond drilling program – 17 holes (TL8 to TL24)
1992	Thunder Lake West	Diamond drilling program – 22 holes (TL25 to TL37)
1993	Thunder Lake West	Diamond drilling program – 10 holes (TC-1 to TC-10)
	Thunder Lake West	Property optioned to Corona (funding exploration)

6.1.4 Goliath Property – Teck-Corona Exploration 1994 to 1999

Exploration activities conducted from 1994 to 1999 consisted of seven diamond drilling programs, re-logging and sampling of previously drillholes, mechanical stripping (22 trenches), chip and channel sampling and mapping, geological mapping (1:5,000 scale), baseline environmental studies, underground development work, bulk sampling, metallurgical testing, site remediation work, custom mill testing, and mineral resource estimation(s) (see Table 6-2 for details).

A suite of 10 litho-geochemical rock samples was collected in September 1995 on legacy claims 1106349 and 1106351 in the southwestern portion of the property. None of the rock samples were found to have been subjected to significant alteration as there was no evidence of sodium, potassium, or calcium enrichment or depletions and none contained any significant gold or base metal values.

In August 1996, some mechanical stripping and sampling was completed in the northern part of legacy claim K1106349 east of East Thunder Lake Road to expose the source of an IP anomaly identified by previous Teck ground geophysical surveys (Waqué, 1996). The new exposure was chipped, channel sampled, and geologically mapped. No significant gold mineralization or alteration was identified from the sampling and mapping program.

Table 6-2: Teck-Corona Exploration Summary from 1994 to 1999

Year	Company & Work Locations	Work Completed
1994	Teck-Corona (Teck Operator)	Diamond drill program – 69 holes (TL44 to TL110, 5 wedges)
	Teck-Corona	Re-logging core of previous holes, 12 whole rock samples
	Teck-Corona	Re-examination of existing surface exposures
1995	Teck-Corona	Diamond drilling program – 25 holes (TL-111 to TL127, 8 wedges)
	Teck-Corona	Litho-geochemical survey (10 rock samples)
1996	Teck-Corona	Diamond drilling program – re-logging 3 holes + 51 new holes (TL128 to TL142, 13 wedges; TLE11 to TLE33)
	Teck-Corona	Resource estimate completed
	Teck-Corona	Mechanical stripping, chip and channel sampling, mapping
	Teck-Corona	August (1 outcrop area, legacy claim K1106349)
	Teck-Corona	Geological mapping (1:5,000), 22 trenches/sampling No. 1 shoot (Main Zone)
	Teck-Corona	No. 1 shoot - 200 kg bulk sample (preliminary metallurgical testing)
	Teck-Corona	Prepared first resource estimate
	Teck-Corona	Geochemical analyses of core and surface samples
1997	Teck-Corona	Diamond drilling program – 65 holes (TL143 to TL206, 1 wedge)
	Teck-Corona	Baseline environmental studies, updated the 1996 Resource Estimate
	Teck-Corona	Preliminary underground program (No.1 and No. 2 shoots) designed
1998	Teck-Corona	Diamond drilling program – 71 holes (TL207 to TL277)
	J.S. Redpath Limited	Underground development – ramp and drifting
	Lakefield Research Ltd., Stock Mine Mill	Exploration, face sampling, bulk sampling, metallurgical testing
	NAR Environmental Consultants	Portal remediation work
	NAR Environmental Consultants	Updated inferred resource estimate
	Corona Gold Corporation (Jones Lot)	Diamond drilling program – 12 holes (Main Zone)
1999	St. Andrews Goldfields for Teck	2,226 t bulk sample sent by Teck to stock mill – custom mill testing

Teck completed a program of geological mapping, trenching, channel sampling, and the completion of 6,596 m of diamond drilling from May 14, 1996, to November 4, 1996 (Stewart et al., 1997). This program was undertaken to better define the alteration corridor east of the resource area, to trench the Main Zone in the No. 1 shoot area to determine controls on the gold mineralization and obtain a bulk sample, to drill test the Main Zone at depths below previous drilling, and to test footwall zones by deepening selected holes.

Geological mapping at a scale of 1:5000 was concentrated mainly in the eastern portion of the property and 15 of the existing trenches were re-examined and chip/channel sampled. Geological mapping and sampling identified new favourable target areas for gold mineralization in the eastern half of the property. The geology of the area was re-interpreted, and the existing geology map was updated.

A trench located on grid line L8+50W was excavated exposing the bedrock over the Main Zone No. 1 shoot. The trench was mapped and a total of 48 channel samples and two chip samples were collected and analysed for gold and multi-elements. A bulk sample of approximately 200 kg was also blasted from the No. 1 shoot for preliminary metallurgical testing (Stewart et al., 1997).

A total of 115 samples from 60 drillholes were collected primarily from the Main Zone for geochemical analyses. Additional samples were also collected from surface outcrops enlarging the surface sample database to include 500 samples in total (Stewart et al., 1997). Overall, this work indicated that higher gold values correlate with increases in lead, zinc, silver, mercury, SiO_2, and SiO_2/Al_2O_3 concentrations in the Main Zone. It was also determined that zinc and lead concentrations decrease across the zone from west to east and that mercury is a good indicator to define the alteration corridor and that the alteration zone remained untested east of the deposit for an additional strike length of at least 2,800 m.

In 1997, a baseline environmental study (water, flora, and fauna) was commissioned by Teck and preliminary engineering plans and cost estimates for an underground program, including permitting, were completed. The environmental work was completed by NAR Environmental Consultants (Sudbury, Ontario). Initial baseline water quality and biological surveys were completed in 1997 and water sampling was continued in 1998 (Page et al., 1999b).

6.1.4.1 Underground Development and Bulk Sampling Program

In 1998 Teck completed an underground exploration and bulk sampling program at Goliath. This entire underground program, from surface site preparation through final closure plan, was completed between May 15 and September 15, 1998. This program was initiated for the following reasons (Page et al., 1999b; Emdin, 1998):

- to determine the nature and continuity of gold mineralization in the Main Zone
- to obtain a bulk sample of the Main Zone mineralization for gold and metallurgical analyses
- to determine what structures controlled the high-grade shoots within the Main Zone by geological mapping
- to establish the true grade of the gold mineralization

The underground work contract was awarded to J. S. Redpath Limited of North Bay, Ontario. A 27 m long inclined trench provided a 9 m high outcrop face suitable for the construction of a portal collar. A decline was prepared at a grade of 15% with the portal located just north of Norman Road and the north boundary of the Laramide property (Figure 6-1). The decline was 4.0 m high by 4.5 m wide and approximately 275 m in length extending 25 m past the Main Zone mineralized structure (Roy et al., 2012). A total of 220 m of drifting (3.0 m by 3.0 m cross-section) was completed along the Main Zone (exposing shoots 1 and 2) extending both east and west of the decline at an approximate vertical depth of 35 m (-38 m floor elevation) for a total of 496 m of underground development. The lateral development followed units of altered schists with weak to strong sulphide mineralization. A total of 23,035 tonnes of rock was excavated.

Geological mapping was undertaken of all drift, slash faces, and backs. Chip sampling of all drift and slash faces was completed at two elevations (Page et al., 1999b). Muck and slash round samples were collected and analysed for gold.

Four bulk sample areas from the Main Zone (No. 1 and No. 2 shoots) totalling 2,375 tonnes were excavated consisting of blasted muck from drift rounds and slashed and material from a 400 tonne take-down-back (TDB) test mining area grading in excess of 3 g/t Au. The bulk sample was processed through a crushing plant, reduced in volume through a sampling tower, and representative splits were processed and analysed for gold content at Lakefield Research Ltd.

Figure 6-1: 1998 Decline Portal for Main Zone Access at Goliath

Source: Historic photo circa 1998. Supplied by Treasury Metals (2015).

Teck concluded that in general, rock and alteration units defined from surface mapping and surface drilling were effective for the underground mapping program. The strongest gold mineralization was found to be localized in siliceous quartz-sericite schists containing disseminated sulphides, sulphide veins, and sulphide-mineralized quartz veins with rare coarse gold/electrum. The more significant mineralized areas are in contact with units of dark-coloured intermediate quartz porphyry. While the general distribution of alteration and mineralization outlined by surface drilling correlated reasonably well with the results of the underground program, Teck reported there was a marked decrease in both the strike length (50 to 65 m expected down to 22 m) and gold grade (15.2 g/t Au expected down to 9.05 g/t Au) of significant mineralization. The grade of the bulk sample (2,336 tonnes @ 9.05 g/t Au) was found to be lower than what was calculated from face and muck samples. Both the grade and the tonnage of the bulk sample were lower than what was anticipated from surface drillhole information. Teck also commented that nugget effects, while present, did not significantly increase the grade of large tonnages of mineralization.

The QP notes that the comparison between the anticipated grade and continuity was made against the 1997 resource which was estimated via a polygonal method on a longitudinal section. Polygonal resource estimation was a common method used in the 1990s. The expected strike length of the zone would have been driven solely by the spacing between the drill intercepts and the grade would be continuous up to the edge of the adjoining polygon where it would abruptly change to the grade of the next drillhole intercept. The disappointing results may just be a reflection of the resource estimation method used. The deposit was re-estimated in 1998 and included the underground bulk sampling and new drilling using an ordinary kriging method for grade interpolation.

After the underground work was completed, the portal was sealed and the area re-contoured, reseeded, and fully reclaimed in late 1999.

6.1.4.2 Custom Milling of Bulk Samples

A 2,355 tonne bulk sample was shipped to the St. Andrews Goldfields' mill near Timmins, Ontario for custom milling in the fall of 1999 (Jobin-Bevans, 2007). The custom milling sample returned average recoveries of 5.63 g/t Au and 15.28 g/t Ag as calculated by St. Andrew Goldfields. The gold recovery was calculated at 96.83% and silver at 38.0%. According to Jobin-Bevans (2007), there was some disagreement as to the total recovery reported by St. Andrew Goldfields and at that time, assays of the mill feed were being reviewed by the Corona-Teck Joint Venture. Initial evaluation of the mill feed samples by an independent umpire laboratory apparently indicated the number of ounces would increase. The resolution of this dispute remains unknown at this time.

Following the bulk sampling and custom milling program, work was suspended on the Goliath project, largely due to the gold grade and tonnage being lower than expected when compared to the resource estimate, and also due to a downturn in the mining industry when gold prices dropped below US$300/oz.

The property was put on care and maintenance until economic circumstances changed to justify additional work to upgrade the inferred gold resource to possible minable reserve categories (Page et al., 1999a).

6.1.5 Laramide Resources Ltd. Exploration Program

Mineralization found at Goliath was projected to extend on the adjacent Laramide property at an approximate depth of 800 m below surface.

During 1994, the historic Laramide property (then consisting of parcels 4822 and 21553 covering an area of 109.5 ha south of the Goliath deposit) was geologically mapped and a ground magnetic/IP survey was completed. Teck/Corona's work had already established zones associated with gold mineralization on their property were responsive to IP survey methods.

These exploration activities have been described in detail by Hogg (2002, 1996). To facilitate this work, a north-south exploration grid was cut with a baseline established along Norman Road (formally Nelson Road) and north-south oriented gridlines were cut at a line spacing of 100 m. The baseline was established along the same road used for Teck's baseline.

The near-surface ground geophysical survey completed by Rayan Exploration Ltd. identified three zones of high to moderate chargeability, as follows:

- northern property boundary anomaly
- eastern property anomaly, 250 m south of the baseline
- southern anomaly located approximately 400 m south of the baseline

In 1996, nine trenches and ten pits were excavated, and some surface sampling was completed. Trench No. 2 and trench No. 4 exposed weakly mineralized zones hosted in biotite schist. In trench No. 2, a narrow zone of quartz veined and pyritized biotite schist returned 480 ppb Au.

A graphitic shear identified at the contact between biotite schist and mafic volcanic rocks was mapped in trench No. 8 explaining the high IP chargeability anomaly that extends across the property 400 m south of the baseline. Eight diamond drillholes were also completed; seven of these holes being collared along the north boundary of the property.

According to Hogg (2002), the exploration work indicated that the degree of silicification and frequency of occurrence of gold mineralization on the property increased to the north. However, no economically significant gold grades were reported.

In June 2002, Laramide acquired a third parcel of land (13492) covering 57 ha to the south, giving them a contiguous land package totalling 166.5 ha in Zealand Township. During the following period of depressed gold prices, no further work was carried out, although the option agreements were kept in place and claims maintained in good standing. The Teck property was later acquired by Laramide in which Treasury Metals was originally a subsidiary company until becoming its own publicly listed company on the TSX on August 19, 2008.

A summary of exploration activities on the Laramide property is provided in Table 6-3.

Table 6-3: Laramide Property Exploration Summary

Year	Company & Work Locations	Work Completed
1994	Laramide Resources Ltd.	Exploration Grid, Geological Mapping
	Laramide Resources Ltd.	Ground Geophysics (Magnetic/IP)
1996	Laramide Resources Ltd.	9 Trenches and 10 pits (mapping and sampling)
	Laramide Resources Ltd.	Diamond Drilling – 8 holes (G1 to G8) testing the Main Zone at depth

6.1.6 Historical Drilling

6.1.6.1 Teck-Corona

Teck-Corona drilling between 1990 and 1999 support a good portion of the mineral resource estimate described in Section 14 of this report. Information on this historical drilling is described in Section 10 and analytical procedures are described in Section 11.

6.1.6.2 Laramide Resources Ltd.

Eight exploratory diamond drillholes totalling 1,622 m were completed on the Laramide property in October 1996 (Hogg, 2002). These NQ holes, numbered G-1 to G-8, were all drilled due north (grid north) at a collar inclination of -45°. Holes G-1 to G-6 were drilled on land parcel 4822, Treasury Metals patented claims PA3900 and PA8429. Drillholes G-7 and G-8 were collared on land parcel 21553, Treasury Metals patented claim PA9074. All holes were drilled on patented land acquired by Laramide in 1996 with seven of the holes collared along the north boundary of the property.

These holes tested the depth extension of the Thunder Lake gold deposit (Goliath deposit) at vertical depths ranging from 105 to 223 m from surface and were collared both south of the deposit and south of Norman Road where the exploration baseline had been established.

According to Hogg (2002), some narrow intersections of biotite schist (BMS?) and felsic tuff (MSS?) were reported to contain anomalous gold and silver values. Hole G-2 returned the best intersection of 675 ppb Au over a core length of 6.0 m. Anomalous gold values were also reported from the same horizon of silicified biotite schist for Holes G-1 and G-3 located 100 m to the east and west of Hole G-2.

Hole G-5 was collared further south to test a moderate to high chargeability ground IP anomaly. A weakly pyritized biotite schist containing possible graphitic mineralization was interpreted to be the source of the geophysical anomaly.

6.2 Historical Mineral Resource Estimates

The mineral resource estimate described in this section are now considered historical in nature. They are provided here for historical context only. Treasury Metals is not treating these historical estimates as current mineral resources or reserves and the QP has not undertaken any independent investigation of the resource estimates; therefore, the resources described below should not be relied upon but they are relevant in that they attest to the historical development of the mineral resources for the project. These historical resource estimates are no longer current and have been superseded by the resource estimate described in Section 14 of this report.

Three historical gold resource estimates were reported on the Thunder Lake gold deposit from 1996 to 1998 using the results from surface and annual exploration diamond drilling programs (see Table 6-4).

Table 6-4: Historical Mineral Resource Estimate by Teck-Corona

Year	Gold (oz)	"Inferred" Historical Resource Estimate
1996	854,000	3.65 Mt grading 7.28 g/t Au
1997	853,000	3.78 Mt grading 7.02 g/t Au
1998	618,700	2.974 Mt grading 6.47 g/t Au

Note: Resources are based on a cut-off grade of 3.0 g/t Au and minimum thickness of 3.0 m. Source: Wetherup and Kelso (2008).

According to Stewart (1996), all the drilling completed to the end of February 1996 was used to prepare a preliminary inferred resource estimate of the deposit totalling 2.8 Mt averaging 9.13 g/t Au for a total of 822,000 oz Au. This resource was estimated based on 56 diamond drillholes and one wedge hole covering a strike length of 1,000 m of the deposit to a vertical depth of 500 m using a minimum horizontal thickness of 3.0 m and block cut-off grade of 3.0 g/t Au.

At the completion of the 1996 drilling campaign, an inferred resource estimate of 3.65 Mt grading 7.28 g/t Au for a total of 854,000 oz Au was estimated (see Table 6-4). In 1997, a new inferred resource estimate was completed based on diamond drilling at 25 m spacing's totalling 3.78 Mt grading 7.02 g/t Au for a total of 853,000 oz Au, as follows (Wetherup et al., 2007):

- Main Zone: 2.87 Mt, 744,000 oz Au, at 2.87 g/t Au
- C Zone: 0.91 Mt, 109,000 oz Au, at 3.75 g/t Au.

According to Wetherup and Kelso (2008), these resource estimates were carried out using the polygonal method (polygons obtained by half-distances between drillholes) and were based on a cut-off grade of 3.0 g/t Au, a specific gravity of 2.7 gm/cm^3, and a minimum thickness of 3.0 m.

A final resource estimate was prepared based on all diamond drilling and surface work, including underground bulk sampling and drilling, completed to 1998 (see Table 6-4). This estimate included 678 underground samples and 219 diamond drillholes from within the resource area (Wetherup et al., 2007). This resource was estimated using computer generated three-dimensional (3D) solid models of the Main Zone and C Zone muscovite-sericite-schist (MSS) units using blocks measuring 3.0 m (thickness) x 10.0 m (height) x 10.0 m (strike length) and using the ordinary kriging method for grade interpolation.

The new inferred resource estimate prepared by Teck geologists in 1998 was 2.974 Mt grading at 6.47 g/t Au (approximately 618,700 oz Au). According to Wetherup and Kelso (2007), this estimate included 2.95 Mt of 6.52 g/t Au present in the Main Zone and 49 kt grading 3.71 g/t Au in the C Zone.

Since 2008, a number of resource estimates were completed on the Goliath deposit by various consultants. These conform to the CIM best practice guidelines in effect at the time the resources were completed. Table 6-5 summarizes these historical estimates along with the Teck-Corona estimates that have now been superseded by the resource estimate discussed in Section 14 of this report.

Table 6-5: Summary of Historical Resource Estimates

Company	Year	Cut-off	Measured			Indicated			Inferred			Estimation Method
			Tonnes (kt)	Au g/t	Ounces (koz)	Tonnes (kt)	Au (g/t)	Ounces (koz)	Tonnes (kt)	Au g/t	Ounces (koz)	
		0 g/t Au							3650	7.25	854	Polygon
Teck-Corona	1997	3.0 g/t Au							3780	7.02	853	Polygon
Teck-Corona	1998	3.0 g/t Au							2974	6.47	619	OK
A.C.A Howe International	2008	3.0 g/t Au				560	5.9	110	3,300	5.9	625	OK
A.C.A Howe International	2012	0.3 g/t Au (OP)				6,002	1.8	326	11,093	1.0	352	OK
		1.5 g/t Au (UG)				3,136	4.3	433	4,789	3.3	514	
P&E Mining Consultants	2015	0.35 g/t AuEq (OP)	1,015	1.90	62	17,174	1.22	676	1,315	1.0	43	ID³
		1.9 g/t AuEq (UG)	103	7.32	24	2,264	4.84	352	2,120	4.2	287	
P&E Mining Consultants	2019	0.40 g/t AuEq (OP)	762	1.91	47	11,849	1.37	522	595	1.1	20	ID³
		1.9 g/t AuEq (UG)	163	6.42	34	3,429	5.34	589	1,414	4.4	201	

Notes: (OP) = amenable to open pit extraction, (UG) = amenable to underground extraction, OK = ordinary kriging, ID³ = inverse distance cubed.

6.3 Goldlund-Miller Property

6.3.1 Ownership

The ownership history of the Goldlund project is complex, dating back to the 1940s. Table 6-6 shows a summary of the past ownership and the exploration and development work completed by the various companies on the property.

Table 6-6: Summary of Past Exploration & Development Work on the Property

Year	Company	Geology	Geophysics	Trenching	Surface Sampling	Diamond Drilling	Underground Development
1941-47	Lundward Gold Mines Ltd.					X	
1945, 47	Windward Gold Mines Ltd					X	
1950	Conecho Mines Ltd.					X	
1946-50	East Lund Gold Mines		X			X	X
1951-52	Newland Mines Limited						X
1971	Windfall Oil & Mines	X				X	
1976-80	Goldlund Mines Ltd.					X	
1980	Windfall Oils & Mines						
1984	Goldlund Mines Ltd.					X	
1987	Camreco Inc.		X	X	X	X	
1988	Camreco Inc.					X	X
1991-92	Noranda Exploration Ltd	X	X	X		X	
1992	Camreco Inc.						
2003	Atikwa			X	X		
2003	Quartz Crystal Dryden Inc.			X	X		
2007	Tamaka Holdings					X	
2011	Tamaka Gold	X	X		X	X	
2012	Tamaka Gold			X			
2013	Tamaka Gold					X	
2017	First Mining					X	
2018	First Mining				X	X	

6.3.2 Exploration

Exploration activities on the Goldlund project date from the 1940s, where in 1941, A. Ward and R. Lundmark (two prospectors working for the Mosher group) discovered gold mineralization in the southwestern part of Echo Township (Page, 1984). From 1946 to 1952 there were significant exploration activities carried out on the Newlund Mines Limited and Windward Gold Mines prospects. The Newlund prospect was extensively explored by 4,570 m of underground drifts and crosscuts on four levels (200 ft, 350 ft, 500 ft, and 800 ft), and 6,220 m of core drilling from a 255 m deep vertical shaft. The 200 ft level on the Newlund prospect was extended more than 3.2 km to the west to connect with the 68 m vertical shaft on the Windward prospect, crossing the entire Windward claim block (Page, 1984). From 1952 to 1973, there was only limited exploration activities carried out on the Echo Township gold prospects.

In 1974, Goldlund Mines Limited and Rayrock Mines Limited entered into an agreement and rehabilitated the surface facilities including the installation of a new headframe and hoist and dewatering the underground workings to the second level (350 ft). A program of bulk sampling, underground chip sampling, and core drilling of 41 holes totalling 4,932 ft (approximately 1,500 m) was carried out. No further activities were carried out, as the prospect was deemed uneconomic given the gold price at that time (Page, 1984).

In total, approximately 143,825 m of drilling has been completed in 808 surface drillholes, and approximately 18,624 m of drilling has been completed in 480 underground holes. Table 6-7 shows a summary of the surface drilling and Table 6-8 shows a summary of the underground drilling.

Table 6-7: Summary of Past Surface Drilling on Goldlund-Miller

Year	Company	No. Holes	Amount (m)
1941	Lunward Gold Mines Ltd.	5	459
1942	Lunward Gold Mines Ltd.	27	2,076
1945	Lunward Gold Mines Ltd.	45	1,106
1946	Lunward Gold Mines Ltd.	81	9,197
1947	Lunward Gold Mines Ltd.	10	1,151
1947	Windward Gold Mines	18	2,528
1950	Conecho Mines	15	3,054
1950	North Denison Mines	1	273
1976	Goldlund Mines Limited	11	1,233
1976	Selco Mining Corp	1	125
1977	Goldlund Mines Limited	3	281
1979	Goldlund Mines Limited	70	3,897
1980	Goldlund Mines Limited	21	1,152
1980	Windfall Oils and Mines	46	6,344
1982	Donald Wilkonson	1	152
1983	Goldlund Mines Limited	4	165
1984	Goldlund Mines Limited	25	3700
1987	Camreco Inc. (GML)	24	7,230
1988	Camreco Inc. (GML)	62	7,303
1989	Camreco Inc. (GML)	33	941
1991	Noranda Exploration Co Ltd	3	219
2007	Tamaka Holdings	43	10,082
2008	Tamaka Gold	66	19,177
2011	Tamaka Gold	31	12,782
2013	Tamaka Gold	14	5,205
2014	Tamaka Gold	10	3,797
2017	First Mining Gold Corp.	124	35,487
2018	First Mining Gold Corp.	14	4,711
Total		808	143,827

Table 6-8: Summary of Past Underground Drilling on the Property

Year	Company	Level (ft)	No. Holes	Amount (m)
1950	Newlund Mines Limited	200	40	1,882
1951	Newlund Mines Limited	200	8	514
1951	Windward Gold Mines	200	10	556
1952	Newlund Mines Limited	200	20	693
1952	Windward Gold Mines	200	6	312
1973	Rayrock Mines Ltd. (NEWL)	200	22	655
1979	Goldlund Mines Ltd.	200	91	3,479
1980	Goldlund Mines Ltd.	200	78	2754
1951	Newlund Mines Limited	350	15	641
1952	Newlund Mines Limited	350	3	60
1973	Rayrock Mines Ltd. (NEWL)	350	19	848
1980	Goldlund Mines Ltd.	350	58	2,119
1951	Newlund Mines Limited	500	20	744
1952	Newlund Mines Limited	500	13	343
1980	Goldlund Mines Ltd.	500	44	1,869
1952	Newlund Mines Limited	800	33	1,155
Total			480	18,624

In addition to drilling, Tamaka carried out a trenching program in 2012 that included the excavation, stripping, mapping, channel sampling and a detailed structural analysis. The structural analysis was carried out by Mr. N. Pettigrew of Fladgate Exploration Consulting Services (Pettigrew, 2012). In total, 13 trenches were excavated covering approximately 7,733.35 m^2 and a total of 1,601 channel samples were collected and submitted for assay.

Table 6-9 presents a summary compilation of the historical exploration activities conducted by various companies on the remaining portions of the project, outside of the immediate Goldlund deposit area.

Table 6-9: Summary Compilation of Historical Work on the Property Outside of the Goldlund Deposit

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Beartrack	Laval	1950	Graham Bousquet Gold Mines	Diamond drilling (12 holes - 366 m)	Bousquet North
Beartrack	Laval	1970	Canadian Nickel Company	Diamond drilling (1 hole - 56 m)	-
Beartrack	Laval	1977	Hollinger Mines	Geological mapping	-
Beartrack	Laval	1978	Hollinger Mines	Magnetic and EM surveys	-
Beartrack	Laval	1978	Selco Mining	Diamond drilling (1 hole - 73 m)	-
Beartrack	Laval	1985	Mistango Consolidated Resources	Airborne magnetic and VLF- EM surveys	-
Beartrack	Laval	1987	Camreco Inc.	Magnetic and VLF survey	-
Beartrack	Laval	1989	Robert J. Service	Trenching	Bousquet South
Beartrack	Laval	1990	A Glatz	Magnetic survey	Bousquet South
Beartrack	Laval	1991	Champion Bear Resources	Geological mapping, trenching, magnetic and VLF surveys	-
Beartrack	Laval	1992	Champion Bear Resources	Magnetic and VLF survey	-
Beartrack	Laval	1992	Champion Bear Resources	Diamond drilling (11 holes - 1,129 m)	Bousquet South
Beartrack	Laval	1996	Corona Gold	Magnetic and VLF survey	-
Beartrack	Laval	1997	Corona Gold	Diamond drilling (12 holes - 3,158 m)	Bousquet South & North
Franciscan	Echo	1950	El Pen Rey Mines	Diamond drilling (3 holes - 415 m)	El Pen Rey
Franciscan	Echo	1950	North Denison Mines	Diamond drilling (3 holes - 824 m)	El Pen Rey
Franciscan	Echo	1973	Goldlund Mines	Diamond drilling (3 holes - 110 m)	El Pen Rey
Franciscan	Echo	1979	Goldlund Mines	Diamond drilling (1 hole - 42 m)	Tarbush
Franciscan	Echo	1980	Goldlund Mines	Magnetic survey and diamond drilling (3 holes - 188 m)	Tarbush
Franciscan	Echo	1981	Goldlund Mines	Diamond drilling (2 holes - 196 m)	Tarbush
Franciscan	Echo	1984	Loydex Resources	Geological mapping	-
Franciscan	Echo	1987	Norad Resources	Magnetic survey	-
Franciscan	Echo	1988	Norad Resources	EM survey, Geological sampling	El Pen Rey
Franciscan	Echo	1995	Tri Origin Exploration	Geological mapping and prospecting	-
Franciscan	Echo	1996	Tri Origin Exploration	Magnetic survey and diamond drilling (8 holes - 1,353 m)	-
Franciscan	Echo	1997	Tri Origin Exploration	Trenching and soil survey	-
Franciscan	Pickerel	1952	Kenwell Oil & Mines	Geological mapping and prospecting	-
Franciscan	Pickerel	1980	Cadre Corporation	Geological review	-
Franciscan	Pickerel	1982	Tarbush Lode Mining	Magnetic survey and diamond drilling (8 holes - 660 m)	Tarbush
Goldlund	Echo	1945	Lundward Gold Mines	Diamond drilling (12 holes - no drill logs available)	Goldlund
Goldlund	Echo	1947	Lundward Gold Mines	Diamond drilling (38 holes - 4,863 m)	Goldlund
Goldlund	Echo	1950	East Lund Gold Mines	Diamond drilling (2 holes - 38 m)	-
Goldlund	Echo	1950	Glenecho Mines	Diamond drilling (1 hole - 294 m)	-
Goldlund	Echo	1953	McCombe Mining & Exploration	Diamond drilling (1 hole - 109 m)	-
Goldlund	Echo	1970	Dryden Project	Diamond drilling (1 hole - 86 m) - assayed for Cu - Ni	-
Goldlund	Echo	1980	Goldlund Mines	Magnetic survey	-
Goldlund	Echo	1983	Tarbush Lode Mining	Diamond drilling (3 holes - 396 m)	-
Goldlund	Echo	1976-1979	Goldlund Mines	Diamond drilling (5 holes - 484 m)	Not Much
Goldlund	McAree	1950	Conwest Exploration	Diamond drilling (4 holes - 699 m)	Tablerock
Goldlund	McAree	1950	Porcupine Peninsular Gold Mines	Diamond drilling (8 holes - 1,718 m)	-
Goldlund	McAree	1951	Orlac Red Lake Mines	Magnetic survey	-
Goldlund	McAree	1951	Pacemaker Petroleum	Magnetic survey	-
Goldlund	McAree	1976	Donald Wilkinson	Diamond drilling (1 hole - 151 m)	-
Goldlund	McAree	1980	Tarbush Lode Mining	Diamond drilling	Tablerock
Goldlund	McAree	1981	Sulpetro Minerals	Magnetic and horizontal loop electromagnetic field (HLEM) survey	-
Goldlund	McAree	1982	Tarbush Lode Mining	Diamond drilling (3 holes - 425 m)	Tablerock
Goldlund	McAree	1982	Tarbush Lode Mining	Diamond drilling (4 holes - 370 m)	-
Goldlund	McAree	1985	Tarbush Lode Mining	Airborne magnetic and VLF- EM surveys	-
Goldlund	McAree	1988	Norad Resources	Magnetic survey	-
Goldlund	McAree	1988	Norad Resources	Geological sampling	-
Goldlund	McAree	1988	Norad Resources	EM survey	-
Goldlund	McAree	1989	Norad Resources	Geological sampling	-
Goldlund	McAree	1991	Noranda Exploration Co Ltd.	Diamond drilling (3 holes - 201 m)	
Goldlund	McAree	2001	Tamaka Gold	Diamond drilling (27 holes - 10,667 m)	-
Goldlund	McAree	2007	Tamaka Gold	Diamond drilling (43 holes - 10,242 m)	-

Exploration Block	Township	Year	Company	Activity	Prospect/ Occurrence
Goldlund	McAree	2008	Tamaka Gold	Diamond drilling (66 holes - 18,974 m)	-
Goldlund	McAree	2013	Tamaka Gold	Diamond drilling (24 holes - 9,001 m)	-
Goldlund	McAree	2017	First Mining Gold	Diamond drilling (100 holes - 24,299 m)	
Laval	Laval	1952	Eclund Gold Mines	Diamond drilling (6 holes - 269 m)	-
Laval	Laval	1952	Floregold Red Lake Mines	Diamond drilling (2 holes - 292 m)	-
Laval	Laval	1956	Canadian Pacific Railway Company	Prospecting	-
Laval	Laval	1970	Canadian Nickel Company	Diamond drilling (2 holes - 292 m)	Troutfly
Laval	Laval	1972	Canadian Nickel Company	Diamond drilling (1 hole - 152 m)	-
Laval	Laval	1984	Mistango Consolidated Resources	Magnetic survey	-
Laval	Laval	1985	Mistango Consolidated Resources	Airborne magnetic and VLF- EM surveys	-
Laval	Laval	1986	Mistango Consolidated Resources	Diamond drilling (4 holes - 449 m)	Troutfly
Laval	Laval	1987	Camreco Inc.	Magnetic and VLF survey	-
Laval	Laval	1987	Mistango Consolidated Resources	Trenching, magnetic survey and diamond drilling (8 holes - 759 m)	Troutfly
Laval	Laval	1989	Camreco Inc.	Soil survey	-
Laval	Laval	1996	Corona Gold	Geological mapping and prospecting	-
Laval	Laval	1997	Corona Gold	Magnetic and VLF survey	-
Laval	Laval	1998	Corona Gold	Diamond drilling (40 holes - 3,826 m)	Troutfly
Laval	Laval	????	Amant Gold Mines	Diamond drilling (4 holes - 269 m)	-
Laval	McAree	1950	Porcupine Peninsular Gold Mines	Diamond drilling (2 holes - 389 m)	-
Miles	Pickerel	1950	Conwest Exploration	Geological mapping, trenching and diamond drilling (5 holes - 950 m)	Nova & Scotia
Miles	Pickerel	1950	Macho River Gold Mines	Line cutting - geological mapping	-
Miles	Pickerel	1951	Lake Fortune Gold Mines	Resistivity survey	-
Miles	Pickerel	1981	Nahanni Mines	Diamond drilling (2 holes - 349 m)	Scotia
Miles	Pickerel	1983	Tarbush Lode Mining	VLF-EM survey and soil sampling	-
Miles	Pickerel	1984	Tarbush Lode Mining	Outcrop stripping and magnetic survey	Miles
Miles	Pickerel	1985	Tarbush Lode Mining	Outcrop stripping and diamond drilling (7 holes - 620 m)	Eaglelund
Miles	Pickerel	1985	Tarbush Lode Mining	Airborne magnetic and VLF- EM surveys	-
Miles	Pickerel	1996	Nufort Resources	Diamond drilling (2 holes - 397 m)	Scotia
Miles	Pickerel	1947-1948	Clinger Gold Mines	Line cutting, magnetic survey and geological mapping	-
Quyta	Pickerel	1950	Eagle Lund Mines & Gold Eagle Mines	Geological mapping and diamond drilling (9 holes - 707 m)	Eaglelund
Quyta	Pickerel	1950	Batch River Gold Mines	Diamond drilling (4 holes - 309 m)	Batch River
Quyta	Pickerel	1976	Albert Carruthers	Diamond drilling (3 holes - 116 m)	-
Quyta	Pickerel	1980	Nahanni Mines	Geological mapping	-
Quyta	Pickerel	1981	Nahanni Mines	Diamond drilling (10 holes - 1,930 m)	Quyta
Quyta	Pickerel	1982	Nahanni Mines	Geological mapping	-
Quyta	Pickerel	1985	Nahanni Mines	Magnetic survey	-
Quyta	Pickerel	1988	Concentrated Rare Earth Minerals	Geological mapping, electro- magnetic (EM) survey, magnetic survey	-
Quyta	Pickerel	1990	Nahanni Mines	Very low frequency- electro- magnetic (VLF-EM) and magnetic surveys	-
Quyta	Pickerel	1992	Nufort Resources	Line cutting and geological mapping	-
Quyta	Pickerel	1996	Nufort Resources	Diamond drilling (5 holes - 950 m)	Quyta
Quyta	Pickerel	1997	D. Brown & T. Darling	Prospecting and geological mapping	-
Quyta	Pickerel	1998	D. Brown & T. Darling	Prospecting and geological mapping	-

6.4 Historical Production

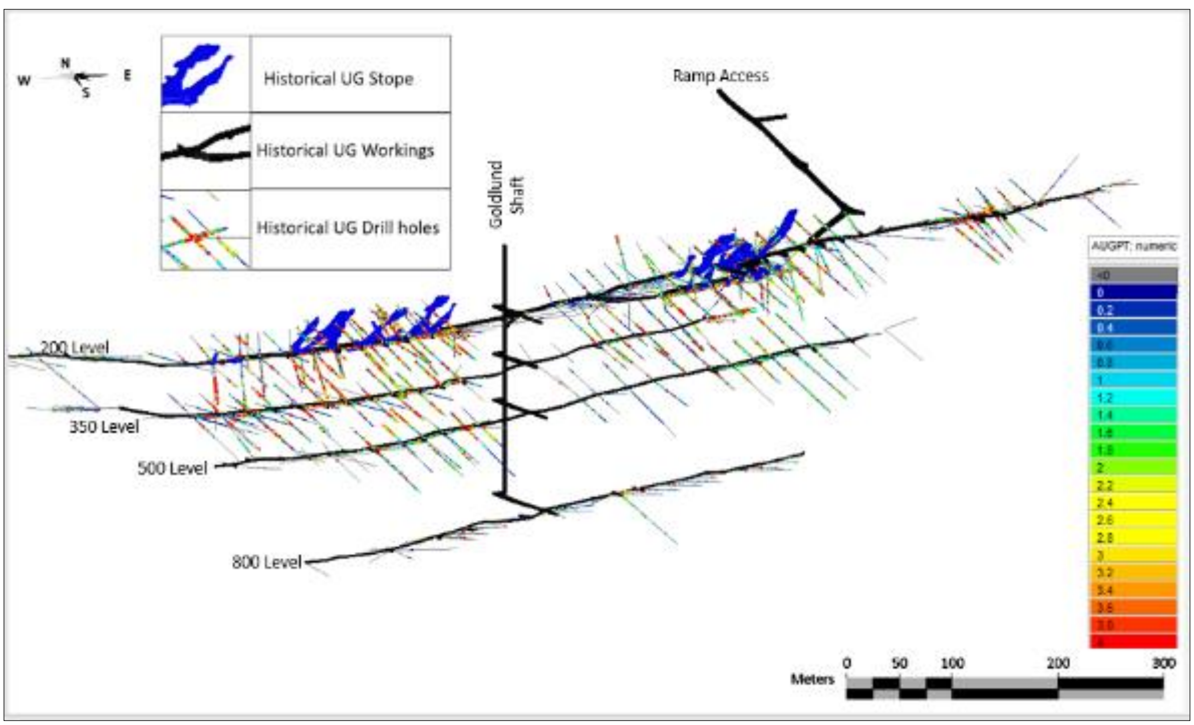
6.4.1 Description

From mid 1982 to early 1985, Campbell Resources Inc. (Campbell Chibougamau), through its wholly owned subsidiary Goldlund Mines Limited, operated an underground mine and an open pit mine and processed material through the mill at the site. Pieterse (2005) compiled the production records that show underground mine production of 100,000 tons (approximately 90,700 t) at an estimated grade of 0.15 oz/ton Au (approximately 5.14 g/t Au) and open pit production of 43,000 tons (approximately 39,000 t) at an estimated grade of 0.17 oz/ton Au (approximately 5.83 g/t Au).

Plant records show that some 132,000 tons (120,000 t) were processed, from which some 18,000 oz of gold were recovered. The head grade was 0.15 oz/ton Au (approximately 5.14 g/t Au) and mill recovery of the gold was reported to be 86.6% (Pieterse, 2005). In total, some 1,050 ft (approximately 320 m) of shaft sinking, 1,385 ft (approximately 420 m) of ramp driving and 19,600 ft (approximately 6,000 m) of drifting and cross cuts were developed for the production.

Figure 6-2 displays an isometric view of the Goldlund shaft and associated underground workings and underground drilling. The historical stopes mined at Goldund are shown in blue.

Figure 6-2: Isometric View (NE) of Historical Underground Workings at Goldlund



Source: CGK (2020).

Table 6-10 summarizes the total production records for the Goliath Complex.

Table 6-10: Production History

Deposit	Year	Tonnes Mined	Ounces Produced
Goliath	1999 (bulk sample)	2,375	415
Goldlund	1982-1985	120,000	18,000
Miller	Not mined	0	0

6.4.2 Historical Mineral Resource Estimates

No historical mineral resources estimates are known prior to Tamaka's ownership of the project. There were several previous mineral resources estimates completed by Tamaka prior to Goldlund being acquired by First Mining Gold Corp. There are also previous mineral resources estimates that were completed by First Mining prior to the purchase of Goldlund by Treasury Metals.

All the previous mineral resources estimates are based on prior data and reports obtained and prepared by Tamaka and First Mining and so are relevant in that they attest to the historical development of the mineral resources for the project. The qualified person has not undertaken the work required to verify these previous mineral resources estimates. Therefore, Treasury Metals is not treating any of these previous mineral resources estimates as current mineral resources estimates that should be relied upon. Table 6-11 presents a summary of the previous mineral resources estimates.

Table 6-11: Historical Mineral Resource Estimates for Goldlund Deposit

Company	Year	Classification	Tonnes	Au (g/t)	Ounces
Tamaka/Goldlund	2012	Measured	3,928,950	1.86	233,690
		Indicated	2,839,200	1.57	143,355
		Measured & Indicated	6,768,150	1.73	377,045
		Inferred	18,905,000	1.03	627,790
Tamaka/Goldlund	2013	Measured	11,333,000	1.55	564,575
		Indicated	7,623,000	0.92	226,036
		Measured & Indicated	18,956,000	1.3	790,611
		Inferred	42,542,000	0.78	1,070,223
Tamaka/Goldlund	2014	Measured	8,459,000	2.1	571,450
		Indicated	10,643,000	1.82	622,800
		Measured & Indicated	19,102,000	1.94	1,940,250
		Inferred	25,845,000	2.51	2,085,000
FMCG/Goldlund	2017	Measured	-	-	-
		Indicated	9,324,100	1.87	560,497
		Measured & Indicated	9,324,100	1.87	560,497
		Inferred	40,895,000	1.33	1,754,092
FMCG/Goldlund	2019	Measured	-	-	-
		Indicated	12,860,000	1.96	809,200
		Measured & Indicated	12,860,000	1.96	809,200
		Inferred	18,362,000	1.49	876,954

6.5 Miller Property

There has been no historical exploration or drilling activities on the Miller deposit prior to 2018. In 2018 and 2019, First Mining completed two drill programs on Miller, as described in Section 10 of this report.

7 GEOLOGICAL SETTING AND MINERALIZATION

Text for this section of the report was extracted from the NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex prepared by Ausenco in 2021.

7.1 Regional Geology

The Goliath, Goldlund and Miller projects are located in the Eagle-Wabigoon-Manitou greenstone belt situated in the northeasterly projecting arm of the Wabigoon Subprovince of the Archean Age Superior Province (see Figure 7-1). This belt is situated in a 150 km wide volcano-plutonic domain with an exposed strike extent of 700 km and extends an unknown distance beneath Palaeozoic strata at either end (Beakhouse et al., 1995).

South of the property, and just north of the Village of Wabigoon, is the “Wabigoon Fault” which is a major regional fault structure. It separates a northern domain characterized by generally southward-facing alternating panels of metavolcanic and metasedimentary rocks, from a southern domain of generally northward-facing metavolcanic rocks (Beakhouse, 2000).

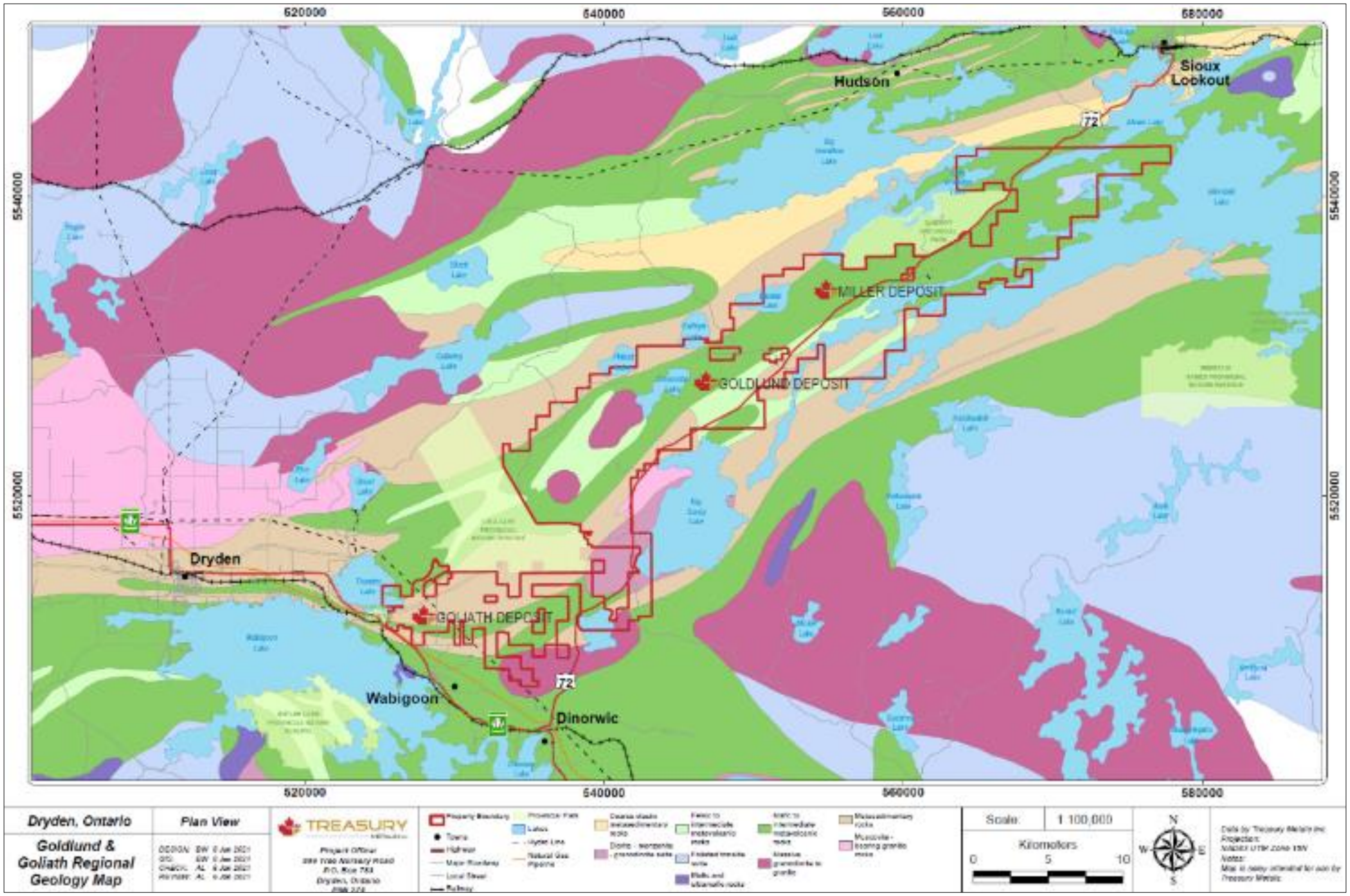
The stratigraphic assemblage has been subdivided into five principal rock groups: the Northern Volcanic Belt, the Northern Sedimentary Belt (Abram Group), the Central Volcanic Belt (Neepawa Group), the Southern Sedimentary Belt (Minnitaki Group), and the Southern Volcanic Belt. The Goliath, Goldlund and Miller projects are located within the Central Volcanic Belt (Figure 7-2).

The greenstone belt is a volcano-plutonic complex and is one of the four-types of lithotectonic domains within the Superior Province intruded by syn-volcanic to post-tectonic granitoid plutons. The magmatic components of the greenstone belts include ultramafic to intermediate volcanics and more felsic volcanic and pyroclastic rocks.

The sedimentary component of greenstone belts includes both clastic and chemical deposits. Plutonic rocks in these domains include synvolcanic tonalitic, quartz dioritic, and granodioritic plutons, the emplacement of which is thought to have deformed the greenstone belts into arc forms. Metamorphic grade is generally green schist or sub-green schist grade except for narrow belts or the margins of larger belts which commonly display mineral assemblages typical of low-pressure amphibolite grade rocks (Percival and Easton, 2007a and 2007b).

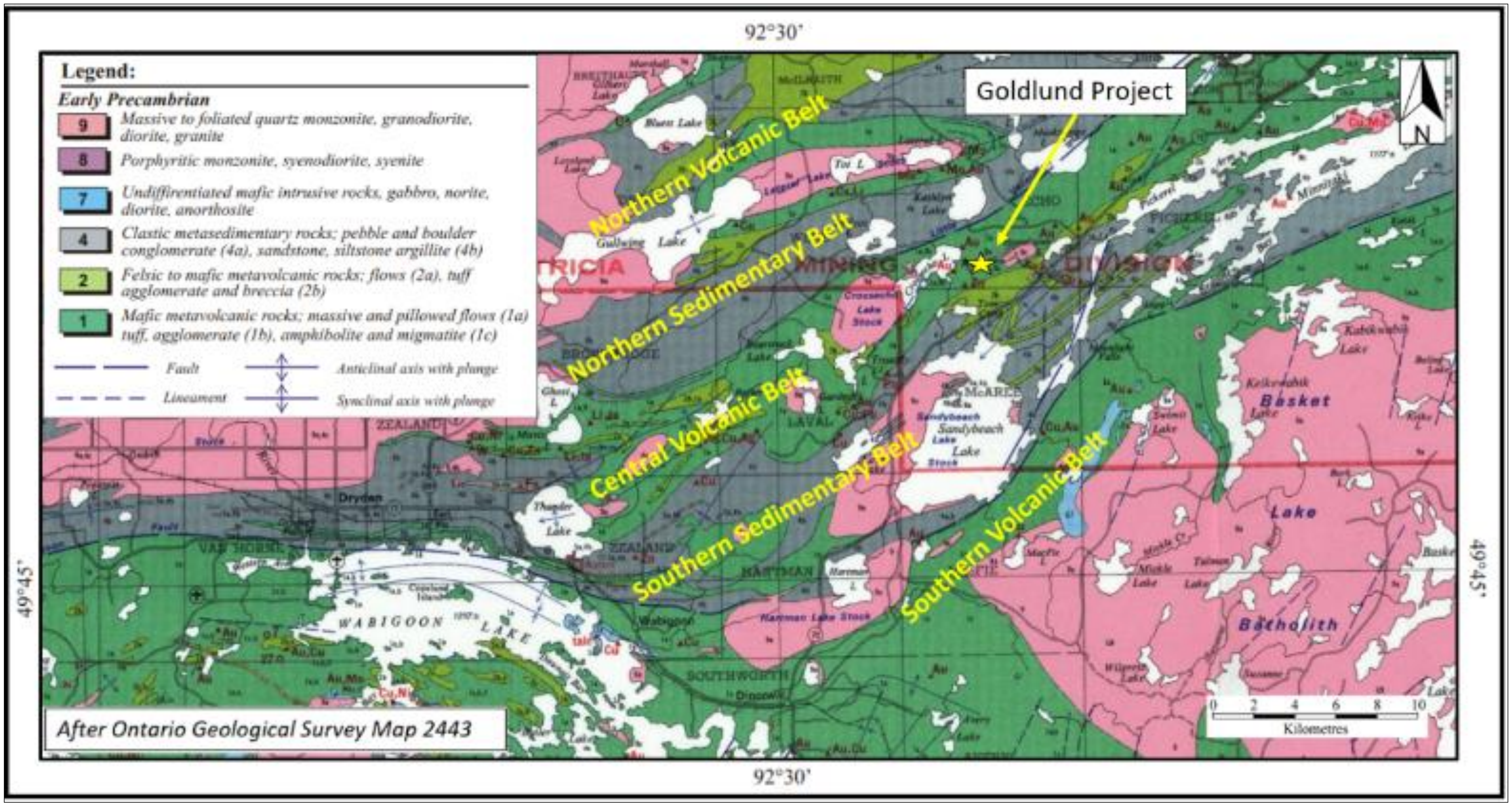
The Central Volcanic Belt (Neepawa Group) has been subdivided into a lower tholeiitic and an upper andesite-basalt division near the Goliath, Goldlund and Miller deposits. The lower division consists of tholeiitic mafic and felsic volcanic rocks with associated subvolcanic intrusions. The upper division consists of calc-alkaline, tholeiitic mafic to felsic volcanic units that crop out around the Beartrack, Troutfly, and Gardner Lakes. The Central Volcanic Belt (Neepawa Group) and the Southern Volcanic Belt are comprised of metavolcanic and metasedimentary rocks, while the Southern Sedimentary Belt (Minnitaki Group) forms an intervening belt of sedimentary units.

Figure 7-1: Regional Geology Plan



Source: Treasury Metals (2021).

Figure 7-2: Regional Geology Map showing Volcanic & Sedimentary Belts



Source: Treasury Metals

The rocks of the Southern Sedimentary Belt (Minnitaki Group) are mainly greywacke and quartzo-feldspathic greywacke, with subordinate argillite and chert, with minor mafic and felsic volcanic units. A distinctive banded chert-iron formation marks the base of the group throughout a large part of the area and displays a complex outcrop pattern, which defines the nature of the structural patterns.

7.2 Goliath Project Geology

The earliest descriptions of the local geology were carried out by Satterly (1941) for the Ontario Department of Mines. These were later expanded with the updating of geological maps by the Ontario Geological Survey from 1995 to 2002 (Beakhouse, 2002; 2001; 2000; Beakhouse et al., 1995). A detailed geology map covering Zealand Township was published by Beakhouse and Pigeon (2003). Geology maps and descriptions of Laval and Hartman Townships were completed by Berger (1990).

The property area geology described below integrates all of the geological mapping, diamond drilling programs, and structural studies completed by Teck, Corona, CCIC and Treasury geological staff from 2008 to present (Roy et al., 2012; Roy and Trinder, 2011; Magyarosi and Peshkepia, 2011; Ilieva, 2008). The rocks have been grouped into the "Thunder Lake Assemblage" of predominantly meta-sedimentary rocks, and the "Thunder River Mafic Metavolcanic Rocks" (Figure 7-3).

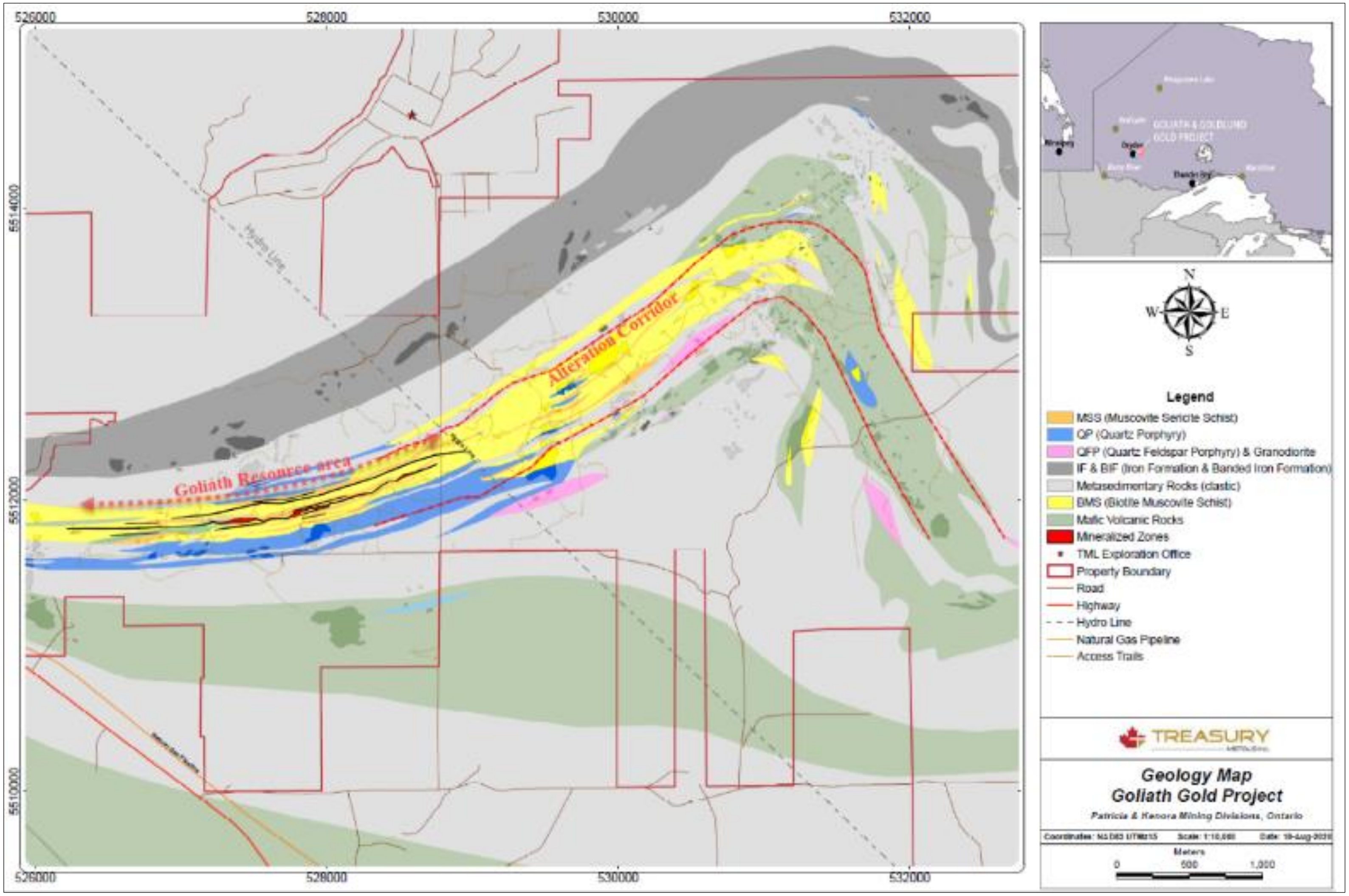
7.2.1 Thunder Lake Assemblage

The Thunder Lake Assemblage, an upper greenschist to lower amphibolite metamorphic grade volcanogenic-sedimentary complex, is typically separated into the "Thunder Lake Sediments" and "Thunder Lake Volcanics" (Beakhouse 2000). Underlying much of the project area, the assemblage comprises quartz-porphyritic felsic to intermediate metavolcanic rocks represented by biotite gneiss, mica schist, quartz-porphyritic mica schist, a variety of metasedimentary rocks and minor amphibolite rocks (Figure 7-3 and Figure 7-1).

Beakhouse (2001) described the Thunder Lake Sediments to be a package of rocks separated into two panels along its strike length by the Thunder Lake Volcanics. These metasedimentary rocks are dominated by biotite-muscovite and biotite schist (greywackes) with subordinate inter-layered metasedimentary rocks (probably pyroclastic siltstone and arkosic sandstone) which exhibit well-preserved primary sedimentary structures such as graded bedding, scour, and rip-up clasts unlike the nearby Zealand Sediments adjacent to the Wabigoon Fault whose primary features are contorted by a high degree of strain (Beakhouse, 2001).

The northern panel of Thunder Lake Sediments include ink blue coloured magnetite layers that are closely associated with distinctive garnet-rich layers and calc-silicate rock, described in earlier publications as iron formation (Satterly, 1941). Iron formation can be locally banded as "banded iron formation" (BIF) consisting of alternating layers of chert and magnetite. These iron formation units are the source of the prominent aeromagnetic anomaly that is folded across the western half of the property.

Figure 7-3: Local Geology, Goliath Project, Northwestern Ontario



Source: Treasury Metals (2020).

Table 7-1 : Thunder Lake Assemblage Rock Description

Rock Type	Description
Biotite Muscovite Schist (BMS)	Dark grey to grey, fine- to medium-grained mica schist. Usually, it consists of intercalated leucocratic and melanocratic bands. This unit contains a high number of grey to milky white quartz veins. Most of the veins are 1-15 cm wide, parallel, or crosscutting the foliation. Some veins are associated with highly chloritized and silicified intervals with tourmaline and sulphides.
Muscovite Sericite Schist (MSS) Interpreted as Altered Felsic Metavolcanic Rocks	Light grey to beige grey, fine- to medium-grained quartz- sericite schist. It is variably siliceous, commonly contains interbedded, dark grey biotite-muscovite bands and grey to milky white quartz veins. It is characterized by the presence of moderate to strong pervasive sericite alteration and gold- and silver-bearing disseminated sulphides.
Iron Formation (IF)	Dark greenish grey calc-silicate metamorphic rocks, which include coarse- to medium-grained gneiss, biotite schist, 10 to 15 cm wide distinctive layers enriched with garnet, chlorite, and narrow ink blue magnetite bands. The rock unit is magnetic and contains disseminated pyrite.
Metasedimentary Rocks (MSED)	Grey to dark grey-green medium-grained massive unit, which consists of biotite, feldspar, quartz, muscovite with a weak patchy potassium and sericite alteration and rare hematite (rusty brown) alteration. Foliation is poorly developed but more prominent in contact and altered areas. Quartz veins, parallel or crosscutting the foliation are very common. This unit can be distinguished by the presence of numerous “quartz eyes” or quartz porphyroblast (identified as “arkose metasediments” or “quartz feldspar porphyry” in Teck/Corona drill logs and historic reports). This unit may contain 1-5% bleb-finely disseminated pyrite and chalcopyrite.
Biotite Schist (BS)	Dark grey to black, fine- to medium-grained, slightly to well-foliated schist. Locally contains disseminated pyrite in the foliation planes and fractures. It was referred to as pelites or greywackes in the historical reports
Chloritic-Biotite Schist (Chl-BS)	Dark grey to greenish grey medium-grained, slightly to well-foliated schist. Locally it contains disseminated pyrite along foliation planes and fractures. Referred to as pelites or greywackes in the historical reports.

Source: Roy and Trinder (2011).

Compositional layering in metasedimentary rocks strike 90° in the western portion of the property around the Goliath deposit and dip from 70° to 80° south-southeast. The rock formational units strike northeast, east of the deposit. Schistosity is commonly developed within both the metasedimentary rocks and metavolcanic rocks and exhibits a similar orientation (Hogg, 2002). In general, the foliation and schistosity is parallel to stratigraphy.

Sandwiched between the sediments are the Thunder Lake Volcanics, a unit dominated by felsic metavolcanic rocks conformably inter-layered with wacke-siltstone. These rocks host the majority of gold mineralization at Goliath. The lenses of metasedimentary rocks that occur within the felsic unit are similar to those making up the main sedimentary unit. All of the rocks have been subjected to folding and moderate to intense shearing with local hydrothermal alteration, quartz veining and sulphide mineralization.

7.2.2 Thunder River Mafic Metavolcanic rocks

The Thunder River Mafic Metavolcanic rocks underlie the southern part of the property between the southern panel of the Thunder Lake Sediments and the Zealand Sediments north of the Wabigoon Fault (Table 7-2 and Figure 7-2). The mafic rocks are generally massive but are pillowed locally and include amphibolite and mafic dykes which are characterized as chlorite schists (Beakhouse, 2000). Some rocks have been described as ultramafic in character (Hogg, 2002). These ultramafic rocks have been mapped locally as soapstone.

Table 7-2: Thunder River Mafic Metavolcanic Rocks

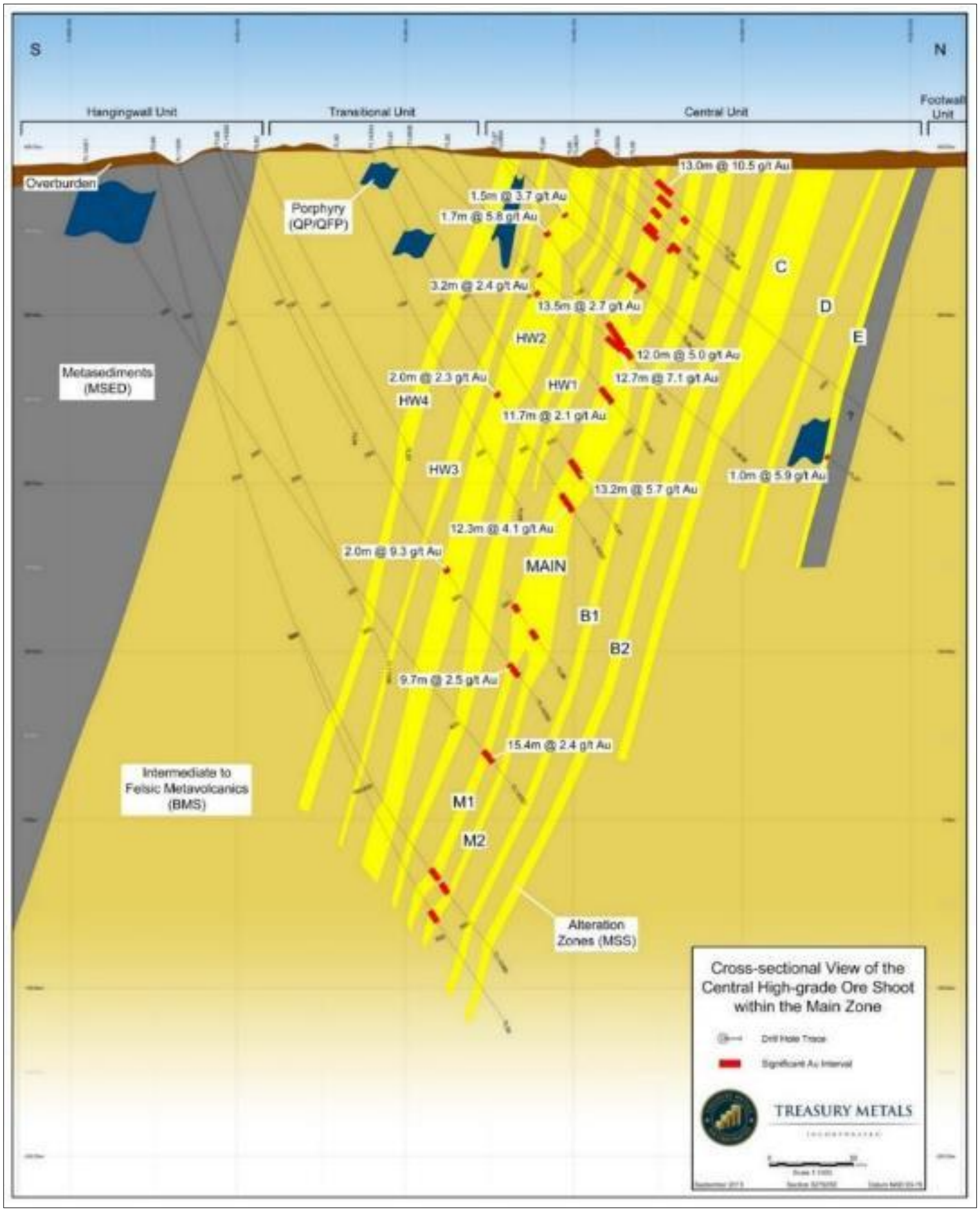
Rock Type	Description
Mafic Dyke (MD)	Usually narrow dark green to almost black massive or slightly foliated fine- to medium-grained biotite-chlorite schist. The width of the layers can reach up to 5 m. The dykes can be either parallel to or crosscut the foliation.
Amphibolite (AMP)	Coarse- to medium-grained, dark green to black to green units, which consist mainly of 30-50% amphibole (hornblende and actinolite), 30% to 40% feldspar and pyroxene with rare post genetic quartz veins and layers of chlorite schist. It has typical "salt and pepper" appearance and nematoblastic texture.
Greenschist	Usually dark green to almost black foliated fine- to medium-grained schist, which consists mainly of chlorite, biotite, feldspar, amphibole. The width of the layers can reach up to 5 m.

7.3 Goliath Deposit Description

For the purpose of the exploration and development, the following four groupings are consistently recognized from south to north at the Goliath deposit (modified after Page, 1994; see Figure 7-4):

- "Hanging Wall Unit" of metasedimentary rocks (MSED) which share a sharp contact or may gradually grade to a biotite-muscovite schist (BMS) that have been intruded by quartz ± feldspar-porphyry intrusive rocks which may appear periodically along the strike length of the deposit.
- "Transitional Unit" BMS occasionally intruded by porphyry rocks.
- "Central Unit" that consists of:
    - BMS, occasionally intruded by porphyry rocks, interlayered with up to four hanging wall alteration zones (HW1 to HW4) consisting of muscovite-sericite schist (MSS) that can have significant gold mineralization that are often silicified
    - a core section of rocks, approximately 100 to 150 m true thickness, that hosts the most significant gold concentrations in the deposit (the Main and C Zones) and consists of intensely deformed and variably altered felsic, fine- to medium-grained, MSS and BMS with minor metasedimentary rocks
    - Rocks similar to those that hosts the D and E Zones in silicified MSS rocks surrounded by BMS.
- "Footwall Unit" of predominantly metasedimentary rocks (MSED, BMS and weak iron formation) with some porphyritic intrusive bodies and minor felsic gneiss and schist rocks.

Figure 7-4: Geology of the Goliath Deposit



Source: Treasury Metals (2015).

Considering that the host rocks of the Goliath deposit are extremely altered and are now schists held together by fine-grained quartz which gives them their competency, Treasury Metals devised a system of grouping the altered schists into two distinct geological units that could be mapped across the deposit: the MSS and BMS units. These units are differentiated based on the relative modal abundance of biotite rich versus sericite rich layers, quartz (silicification) and sulphide mineral content. In general, the most altered rocks containing greater than 60% quartz-sericite felsic bands, are silicified and often contain base metal mineralization, have been mapped as MSS (light coloured) units. Those units containing less than 60% white mica have been mapped as BMS (dark coloured). Figure 7-5 visually illustrates the difference between the two rock units. It should be noted that contacts are almost always gradational. Gold is usually associated with the MSS units in association with sphalerite and galena or occurs in smaller MSS bands hosted within the BMS units.

Figure 7-5: Dimond Drill Core Photograph showing BMS (top) and MSS (bottom)

Source: Treasury Metals (2015).

7.3.1 Structural Geology

Page (1994), Beakhouse (2001), Ravnaas et al. (2007) and Wetherup (2008) have described and interpreted the key structural features on the property identifying three deformation events and three related generations of fold axes. Geological and trench mapping programs, as well as structural studies of bedrock and drill core, have been undertaken by Treasury Metals to obtain a better understanding of the structural geology of the property. Structures and veins observed in the area of the Goliath deposit have been interpreted within, and relative to, this basic framework.

7.3.1.1 D_o Pre-Deformation Structures

The D_0 pre-deformation structures developed during the rock formation and are a result of possibly transposed bedding and/or alteration zones. They can be observed in core and bedrock as alternating leucocratic quartz-sericite and melanocratic biotite-feldspar layers and represent compositional layering within felsic metavolcanic and metasedimentary rocks. The width of the layers varies from 0.5 to 10 cm, but locally forms larger units interbedded with layers of metasedimentary rocks. Larger zones (< 40 m wide) of dominantly quartz-sericite schist locally contain greyish, very fine-grained layers or "ribbons" of quartz V0 veins which are usually associated with sulphide (pyrite-sphalerite-galena-chalcopyrite) mineralization and have the potential to host coarse gold. The association of almost pure, very fine-grained quartz layers within the centre of a larger zone of quartz-sericite schist could represent transposed and metamorphosed sericite alteration around quartz veins within the felsic metavolcanic rocks. Sulphide minerals observed in drill core commonly occur along S1 foliation planes and appear to have been remobilized.

Contacts between the lithostratigraphic units were measured in the outcrops and in the core. Within the felsic volcanic rocks the contacts between the MSS and BMS units can range from transitional to sharp. More noticeable is the contact between the felsic volcanic rocks and the metasedimentary rocks that is usually marked by a very small angular discordance and is almost parallel to the primary bedding. The strike and dip are approximately 090°/70°S, but can change from 068°/72°S to 090°/80°S. It is interpreted that the primary syngenetic gold and silver mineralization was deposited during this event because the mineralization is mostly contained within the sericite schist and/or biotite-muscovite schist. Isolated concentrations of gold lying outside of these units may be related to later remobilization or alteration and gold deposition at other parallel but different stratigraphic horizons as zones of mineralization are all parallel to one another parallel to stratigraphy.

7.3.1.2 D1 Deformation

The D1 deformation is represented by well-developed foliation S1 and isoclinal folds F1 within the felsic metavolcanic rocks (BMS, MSS) and metasedimentary rocks (biotite schist or “BS”) and iron formation). The foliation and the axes of the folds were measured in the outcrops, in the trenches and during the orientation drilling of holes TL0822 to TL0837. The foliation is approximately 074°/70°S, but it can vary from 064°/62°S to 090°/80°S. The mafic metavolcanic rock unit texture tends to be more massive as the foliation is suppressed.

F1 folds were observed in the outcrops and in the core. The folds are isoclinal, and the fold axes are parallel to the F1 foliation. The dip and strike of the axial planes are approximately 090°/70° but it can change from 080°/68°S to 100°/78°S. In most cases, the hinges/fold noses display evidence of distension where continuing compressional deformation has stretched the hinge and its limbs are highly attenuated and thinned. These fold noses are often completely decapitated from their limbs and generally only hook-shaped or quartz lenses remain, which suggests that some of the boudinage or quartz lenses observed in the felsic metavolcanic rocks may be related to F1 structures. Deformed, white, coarse-grained quartz veins ± tourmaline, ± stringers or porphyroblasts of sulphides, 1 to 10 cm wide, occur dispersed throughout the felsic metavolcanic and metasedimentary rocks. White, coarse-grained quartz veins are not localized to certain pre-deformational stratigraphy and are interpreted to be syn-tectonic (V1) as they are affected by D1 deformation and occur in all rock types. They typically crosscut the foliation but may be parallel in some instances. The assay results show no direct correlation between the quartz veins and elevated gold and silver concentrations.

7.3.1.3 D_2 Deformation

The D_2 deformation is observed as zones of disturbed foliation related to closed F_2 folds and V_2 quartz veins. Rare F_2 fold hinges are observed in the outcrops. They are several centimeters in scale and affect the position of the felsic volcanic package that hosts mineralization on the Goliath project. Where F_2 fold axes and fold noses occur within the gold-silver mineralized zones in the felsic metavolcanic rocks, gold and silver values are commonly 10 to 100 times higher than in the adjacent intervals (Roy et al., 2012). In some cases, they contain coarse-grained visible gold (VG) or electrum, but even the very fine-grained mineralization returns higher gold or silver concentrations. Throughout the 2008 mapping program the orientation of the F_2 fold axes were measured in the outcropping rocks. The strike of the F_2 plane is approximately 220° to 230° and dips 85° to 90° southward. In addition, the F_2 fold axes are almost vertical and the intersections of these fold axes and the mineralization plunge steeply westward. Overall, discrete F_2 fold zones are narrow (up to 10 to 15 cm wide), widely spaced (5 to 25 m) and locally carry significant gold mineralization. Determining where F_2 folds are likely to be located will identify areas of potential high-grade mineralization. S and Z folded F_1 foliation, V_0 and V_1 quartz veins, and non-deformed crosscutting V_2 veins are all features attributed to the D_2 deformational event. The veins are differentiated on the basis of mineralogy, texture, and amount of strain.

7.3.1.4 D_3 Deformational Event & Northwest Fault

The D_3 deformational event is represented by brittle faults and fractures filled in with quartz, chlorite, feldspar, carbonate and/or fault gouge. Local shear zones and faults are exposed in outcrops and old trenches.

The first fault system is almost vertical and strikes 220° to 240°. The system consists of almost parallel micro-faults with dextral displacement on a centimeter scale. Very often it is accompanied with a 1.0 to 1.5 m wide sericite alteration.

The second fault system, exposed in the outcrops, has almost a north-south orientation. The azimuth bearing ranges from 352° to 008° and the dips from 85° to 90°. Usually, the fault zone consists of 2 to 3 micro-faults located within an interval with widths ranging from 0.5 to 1.0 m. These faults can be found in all rock types including clastic metasedimentary, felsic volcanic and mafic volcanic rocks. Commonly the rocks adjacent to the faults are highly fractured.

The most significant feature found in the drillholes that can be related to D3 deformation is what Teck-Corona described as the Northwest Fault. This is a brittle structure which strikes west to west-northwest and dips shallowly northward and was observed in most of the deeper holes. Drill section interpretation by Teck-Corona shows very little dip-slip movement along this structure (approximately 5 to 10 m, hanging wall up). Most shallow dipping structures are dip-slip in nature, but since this is such a prevalent feature there may be a significant component of strike-slip motion, since dip-slip offset is minor.

A third generation of white, coarse-grained quartz veins (V_3) are formed during the D_3 event. These veins occur in all rock units and typically crosscut the foliation obliquely with sharp margins. No deformation appears to have occurred in these veins, which can also cut D_2 structures. V_3 veins are hematized on the surface, have been previously sampled, and do not return any significant gold or silver values. D_3 deformation is not related to the gold-silver mineralization emplacement. However, the Northwest Fault appears to offset the mineralized zone towards the northeast of the main deposit. Wetherup (2008) demonstrated that high-grade mineralization occurs along the steeply southwest plunging intersections of F_1-F_2 fold axes and that these shoots are offset by the northwest Fault.

7.3.2 Mineralization

The Goliath deposit is located 250 to 300 m north of Norman Road and since 1990 the main resource area has been defined by extensive diamond drilling efforts concentrated over a strike length of over 2.0 km. To date, 12 zones containing gold and silver mineralization have been identified within the Central Unit of the main deposit. From south to north, they are the:

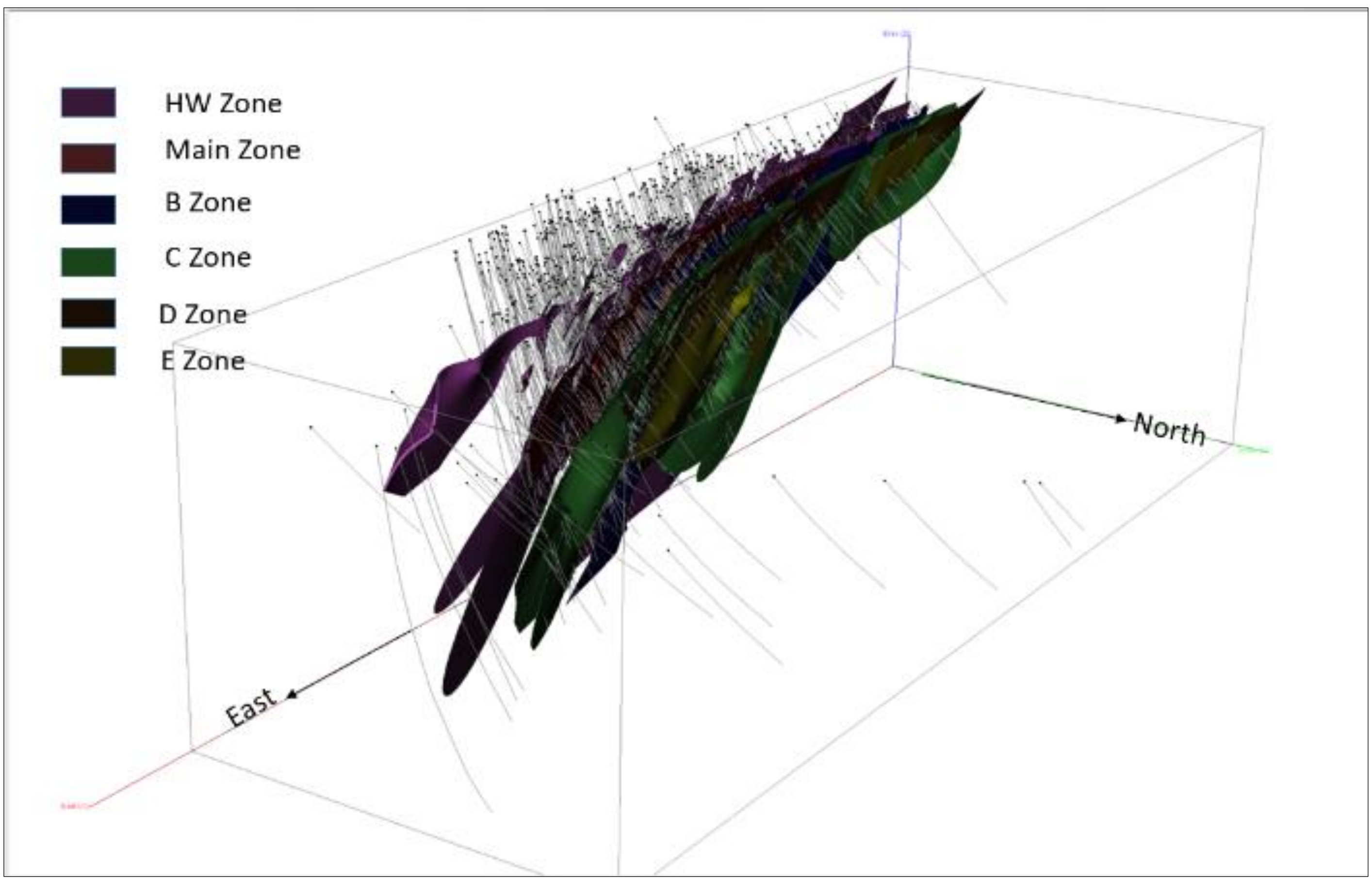
- Hanging Wall Zones (HW1 to HW5 subzones), hosted in mostly BMS rock units and small amounts of metasedimentary and porphyry intrusive rocks.
- Main Zone (M1 and M2 subzones), which is 5 to 40 m wide and occurs principally in silicified and sulphide mineralized (sphalerite, galena, and pyrite) MSS rocks.
- B Zone, hosted in BMS rocks residing between the Main and C Zones.
- C Zone (C1 and C2 subzones), hosted in silicified and sulphide-bearing (sphalerite and galena) MSS rocks.

- D and E Zones, hosted in mostly a mixture of MSS and BMS rocks surrounded by significant amounts of metasedimentary rocks and minor porphyry intrusive rocks.

It is noted that the BMS rocks located between the M1 and M2 and the C1 and C2 subzones often display lower grade mineralization, which is largely due to smaller MSS bands hosted within the BMS units.

The majority of the historical gold and silver resource estimates reside in the Main Zone and C Zone (Figure 7-6). At Goliath, the gold-bearing zones all strike from 090° to 072° with dips that are consistently 72° to 78° toward the south or southeast. The main area of gold, silver and sulphide mineralization and alteration occurs up to a maximum drill-tested vertical depth of approximately 805 m (TL135) below the surface, over a drill-tested strike-length of approximately 3,000 m within the current defined resource area. Gold mineralized zones remain open at depth. The historic Teck-Corona drilling confirmed that anomalous gold mineralization occurs over a strike length of at least 3,500 m (Corona, 1998). Exploration work by Treasury has shown alteration zones containing intersections of gold mineralization extend over a strike length of at least 5,000 m. Overall, rocks surrounding the principal defined target zones are often anomalous in gold mineralization (background gold concentrations).

Figure 7-6: Perspective View of the Goliath Deposit showing Interpretated Mineralized Zones



Source: SRK, 2023

The mineralized zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. Stratigraphically, gold mineralization is concentrated in an approximately 100 to 200 m wide Central Unit composed of intensely altered felsic metavolcanic rocks (quartz-sericite and biotite-muscovite schist) with minor argillaceous metasedimentary rocks. Higher-grade gold within the central unit is concentrated in a pyritic alteration zone consisting of MSS, quartz-eye gneiss and quartz-feldspar gneiss with lower grade gold in BMS.

Historically, drilling had focused primarily on targeting the Main and C Zones. Mineralogical studies have determined that native gold and silver (electrum) are associated with finely disseminated sulphides, coarse-grained pyrite, and very narrow light grey translucent “ribbon” quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite in decreasing order of abundance. The sulphide content ranges from 3% to 5% but is locally up to 15%.

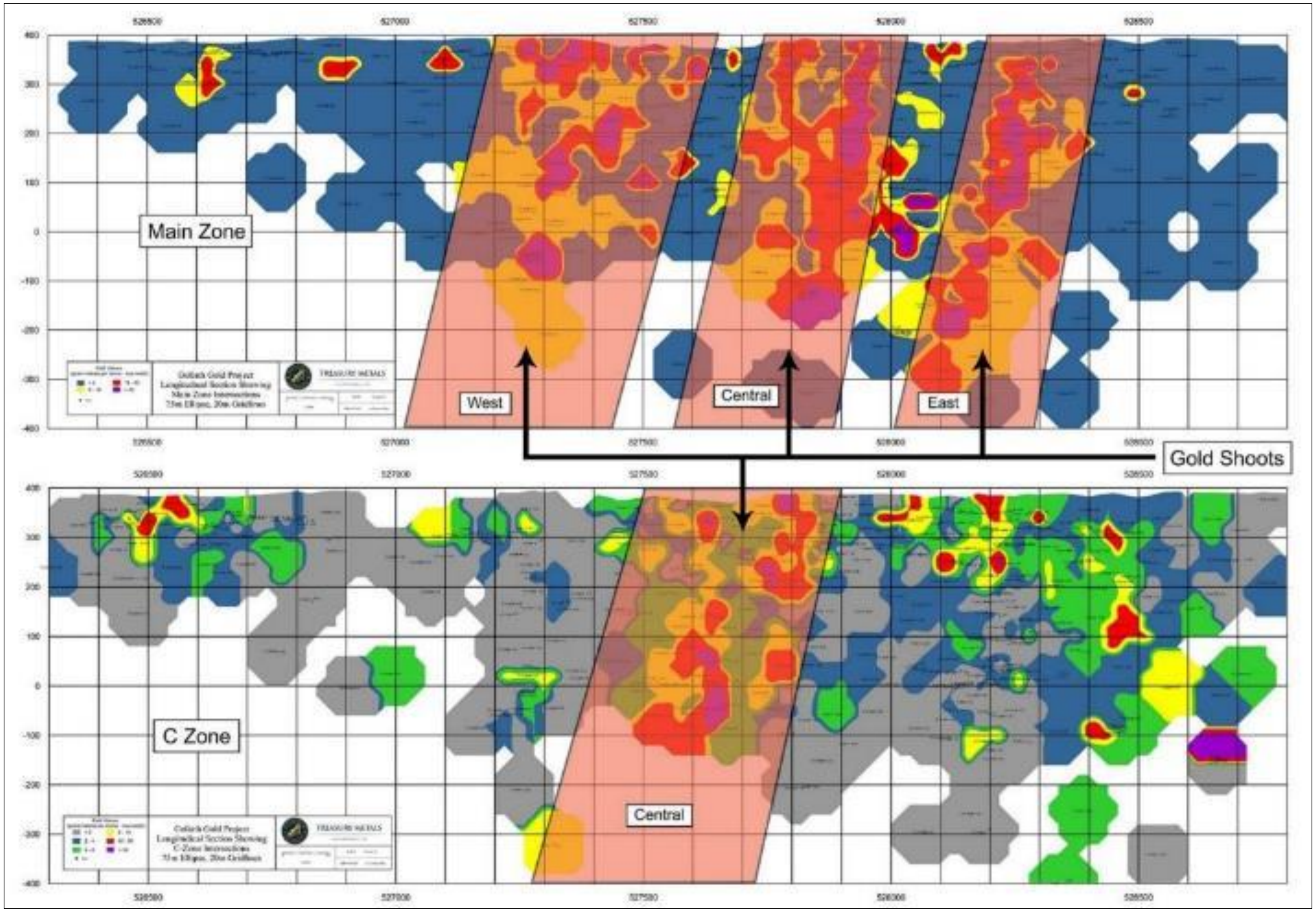
Visible gold and/or electrum are rare and occur mainly within the leucocratic bands of MSS but can also in the melanocratic bands enriched with biotite and chlorite. In general, the highest gold and silver values occur in association with very strong pervasive quartz-sericite alteration. An increase in gold and silver correlates with an increase in pyrite and more specifically an increase in sphalerite content. The modal abundance of sphalerite usually exceeds that of galena and pyrite. Although the presence of elevated sphalerite and galena have been used as an indicator of the potential presence of gold with the deposit, there are some instances when gold is not present even through the base metals are clearly visible in drill core. In addition, an increase in chalcopyrite and galena content has a lower correlation to an increase in gold values.

Two distinct types of pyrite are recognized: disseminated fine-grained cubic euhedral crystals occurring in the foliation planes; and disseminated subhedral to irregular grains and stringers, with inclusions of galena, occurring in quartz veins and along the margins of the veins. The second type is commonly associated with other base metal sulphides. Pyrite can occur as fine-grained disseminations in the foliation planes, disseminations in the matrix, blebs, stringers and or veinlets. The base metals sulphides can be concentrated in blebs and stringers of sphalerite, cubic fine-grained galena and on occasion as chalcopyrite.

Silver-to-gold ratios are generally unpredictable and have a substantial range. Possibly during the syngenetic mineralization event, more silver than the gold was contained in the hydrothermal solutions (ratio Ag/Au>1), but during the epigenetic mineralization event, some of the gold was redistributed and there was enrichment in structurally induced zones of enhanced porosity and permeability. A similar relationship of gold to base metals is observed.

In the Goliath deposit, high-grade gold mineralization and silver occur in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west (Figure 7-7). In the Main Zone, three shoots have been well defined named the “East”, “Central” and “West” shoots and a central shoot has been delineated along the C Zone. Corona (1998) interpreted the high-grade shoots to be the result of tight folding of the mineralized horizon (gold concentrated in fold noses) that appear to occur at regular intervals (Figure 7-7). The shoots have considerable down-plunge continuity and are all open and untested down dip at depth. Treasury has interpreted that these zones may be connected through a large folded anticlinal feature with a fold axis that strikes down the centre of the deposit and plunges around 10° to 20° east.

Figure 7-7: Longitudinal Section of Main Zone (top) and C Zone (bottom)



Source: Treasury Metals (2015).

The Main Zone is comprised of one larger well-defined pyritic, and often silicified, MSS Zone or is bifurcated into two sub-Zones (M1 and M2) separated by less-altered BMS rocks. C Zone gold mineralization always occurs in the C1 and C2 subzones hosted in sulphide mineralized and silicified MSS that demonstrate excellent on strike and down dip continuity throughout the deposit.

The portion of the Central Unit of the deposit that hosts the B, C Zone and D and E Zones ranges in thickness from 75 to 150 m but is often lower in grade than the Main Zone. It should be noted that the D and E Zones have often only been sporadically drill tested since many holes historically end before intersecting them. Since the 2011 technical report, Treasury Metals has re-entered 30 historical Teck and Treasury Metals drillholes to extend the holes in order to intersect the C, D and E Zones and have conducted an extensive infill sampling program of existing core to provide B Zone assay data to add to the mineral resource.

The Hanging Wall Zones (HW1 to HW5) are located 10 to 50 m south of the Main Zone. These zones are often narrow in width (1 to 3 m) and remain open along strike and at depth. Many of the historic Teck intersections of these zones were not consistently sampled because they were not significantly mineralized or contained no visible base metal minerals (sulphide content ranges from 3% to 5%). Gold and silver are probably included in the pyrite or around the pyrite micro grains. Only a few flakes of coarse-grained gold or electrum were visible in the core or in the grab samples. Most of the sulphides are located mainly in blebs or stringers parallel to the foliation planes. Usually blebs, stringers and veinlets of pyrite are associated with the stringers of sphalerite, cubic fine-grained galena, chalcopyrite and pyrrhotite. Very often they infill small fractures in the host rock or occur along margins of quartz veins.

7.3.3 Alteration

The Goliath deposit consists of hydrothermally altered felsic metavolcanic and metasedimentary rocks. Alteration has been traced through drilling and geological mapping for an approximate strike length of at least 5 km. The alteration consists of primarily sericitization and silicification in association with the gold mineralization. Chloritization is visible in metamorphosed and altered mafic rocks in the area. Very rare flakes of aquamarine green mica (fuchsite: Cr muscovite) occur in the strongly altered sericite alteration and will sometime appear within the vicinity of high-grade gold.

Page (1995a) correlated the sericitic alteration of MSS with moderate potassium enrichment and significant sodium depletion. CCIC made the following observations from the analyses of 756 whole rock samples collected from holes TL0801, TL0802, TL0807, TL0808, and TL0823:

- The intervals with significant gold and silver mineralization are very strongly altered.
- Very often extensive pervasive hydrothermal alteration obscures primary textural and structural features to the extent that it is not possible to identify the original rock type.
- The hydrothermal alteration commonly involves massive depletion of CaO and Na2O and addition of H2O, K, silica and sulphur as quartz ribbons and sericite.
- The feldspar and biotite are totally replaced by sericite, quartz and disseminated pyrite.
- Most of the mineralized zones are hosted by fine to medium-grained MSS or in patches of sericite alteration in BMS.
- The chlorite alteration is more intense in zones of fractured and brecciated host rocks. As a result of the depletion of CaO and Na2O from the feldspar and addition of MgO and Fe_2O_3, sulphur and silica, quartz-pyrite-chlorite-tourmaline veins were formed.

- Complex, overprinting alteration and metamorphic assemblages and diverse metal associations are interpreted to be the result of an overprinting of hydrothermal and metamorphic fluids, which were focused in the zones of structurally induced porosity/permeability.

7.4 Goldlund Project

7.4.1 Property Geology

The Goldlund project is characterized by a 3 km wide belt of Precambrian mafic metavolcanic rocks that strike northeast across the project area. These mafic metavolcanic rocks are bounded by Precambrian metasedimentary rocks to the north and to the south, with a wedge of Precambrian felsic metavolcanic rocks that occur at the southern contact between the Precambrian mafic metavolcanic rocks and the Precambrian metasedimentary rocks (see Figure 7-8).

The mafic metavolcanic rocks have a 1.5 km wide tuffaceous member to the south and a series of spherulitic basaltic flows interlayered with basaltic pillow lavas and some tuffs to the north. The basaltic metavolcanic rocks are dark green, massive in texture and weakly to strongly foliated. Other textures have also been observed, including amygdular flows, pillowed flows, lapilli tuff, feldspar crystal flows, and variolitic (or "spherulitic") flows.

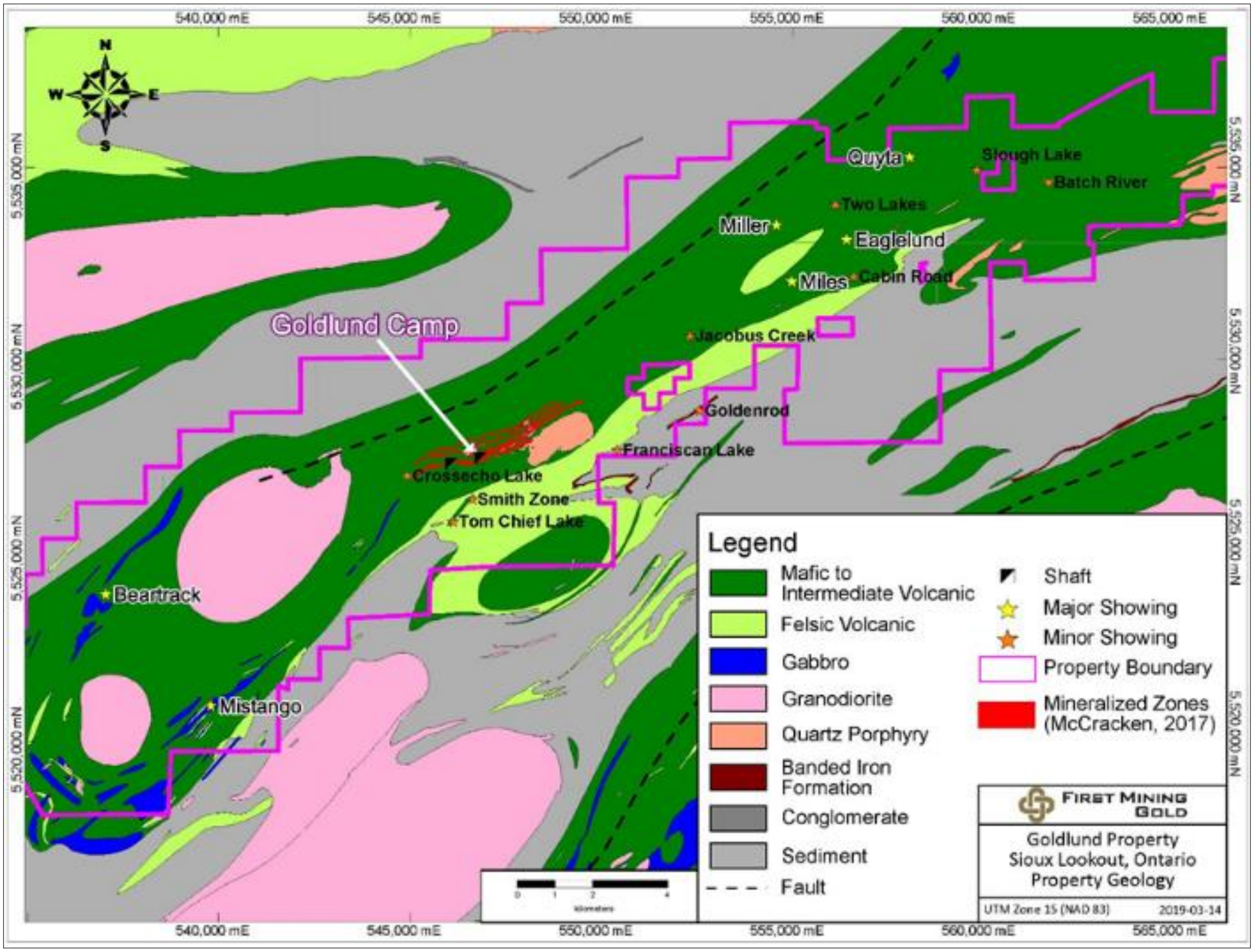
The mafic metavolcanic rocks are commonly magnetic, although significant variation in the strength of magnetism has been observed from outcrop to outcrop. In some cases, coarse magnetite crystals were observed and magnetite content up to several percent was observed. In contrast, very little pyrite or carbonate has been observed in the basaltic metavolcanic rocks in the Goldlund area. The metavolcanic rocks in the Goldlund area also lack the iron (Fe)-carbonate/sericite altered shear zones that are commonly observed in other greenstone belts.

Veining is relatively common within the mafic metavolcanic rocks. The most commonly observed veins are single, thin, sharp-walled, irregular quartz veins, containing minor chlorite and trace pyrite mineralization. Larger veins and veinlets with minor carbonate, biotite, and chalcopyrite have also been observed and occasionally sampled. In particular, large (sometimes more than 20 cm) irregular quartz veins have been observed to form within the mafic metavolcanic rocks in close proximity to the mineralized felsic metavolcanic rocks in some places. It is unknown whether these veins carry gold. "Transverse" style veining is also observed occasionally within the mafic metavolcanic rocks, suggesting that the competency contrast between different mafic metavolcanic rock phases may be sufficient to localize veining and potentially, gold mineralization.

Albite-trondhjemite to diorite sills ("granodiorite" in mine terminology) have intruded near the contact between the mafic metavolcanic tuffaceous rocks to the south and the spherulitic mafic metavolcanic rocks to the north. These strata-parallel sills dip from vertical to -80° southward and range from 14 m to 60 m in thickness. A subsidiary suite of sills intrudes the narrow tuffaceous metavolcanic rocks that are interbedded with the spherulitic mafic metavolcanic rocks. These strata-parallel intrusions are known to extend north-eastward well beyond the Goldlund deposit, toward the Miller deposit, and south-westward beyond Crossecho Lake where they re-appear just south of Troutfly Lake. It has been postulated that this series of intrusions may occur intermittently over a strike length of 15 km.

The albite-trondhjemite to diorite sills that host the most important zones of mineralization at the Goldlund project have been referred to as "grey granodiorite" due to their light colour and significant amounts of biotite and free quartz (Armstrong, 1951). Meta-gabbroic or meta-dioritic rocks in both transitional and intrusive contact with the "granodiorite", as well as crosscutting feldspar and quartz-feldspar porphyry dykes, were at times themselves referred to as "granodiorite", causing the terminology to become confused. The sills of granodiorite and/or its gabbroic counterparts to the northeast and southwest of the mineral deposit at the Goldlund project have been considered primary exploration targets in the past.

Figure 7-8: Goldlund Property Geology Map



Source: WSP (2019).

7.4.2 Structural Geology

Chorlton (1991) interpreted four-stages of deformation in the Sandy Beach Lake – Sioux Lookout area, based on the overprinting of individual structures and fabrics. These are described below.

The Stage 1 deformation is expressed by a locally preserved foliation, sub-parallel to bedding. The relatively shallow angle between bedding and foliation may be an indication of thrusting.

Stage 2 deformation is associated with the emplacement of the granitoid bodies throughout the area.

Stage 3 deformation is largely responsible for the northeast-trending structural grain of the belt. Northwest-southeast compression and sinistral rotation generated large-amplitude upright folds with steep, northeasterly trending axial planes, together with steep northeasterly trending shear zones. Shear zones northwest of the Beartrack–Crossecho Lakes area and southeast of the Sandy Beach Lake area tend to be sinistral-oblique, southeast-side-up, while those in the central portion of the belt tend to be sinistral and sub-horizontal.

Stage 4 deformation reflects the final phase of convergence in the belt. Large- to small-scale folds with steep, north-northeasterly striking axial planes overprint the Stage 3 folds. Irregular belt boundaries and rigid internal stocks restricted further lateral extension and resulted in vertical displacements along the core of pre-existing shears.

7.4.3 Mineralization

Gold occurs in essentially two types of deposits in the Goldlund area. The most important gold mineralization is associated with quartz vein and stock-work structures, which are found in albite-trondhjemite sills, as well as in porphyry sills and mafic metavolcanic rocks (Page, 1984). Trace to minor quantities of gold (and silver) are found in disseminated and massive sulphide deposits (copper-nickel, copper-zinc) in metavolcanic rocks.

Gold mineralization is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. These stockwork zones form bands within the sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite.

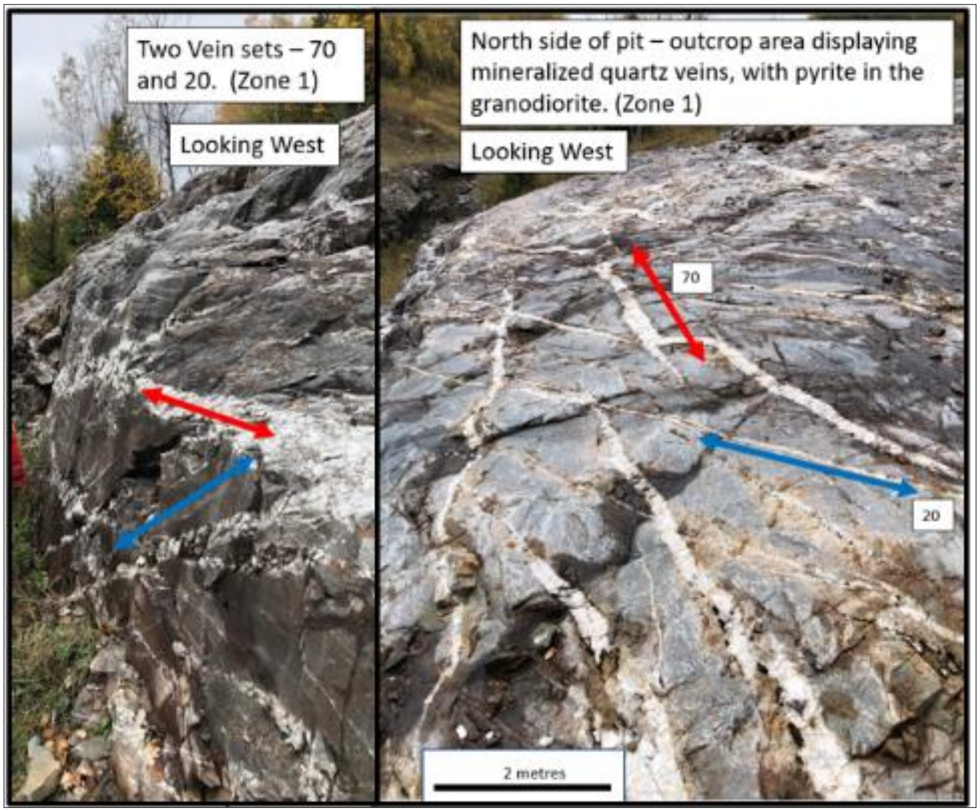
The mineralized sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the project. Although locally they may differ by up to ± 20° in strike and dip, overall, they are a very consistent 239°/58°N (70 set) and 189°/53°W (20 set) orientation. Figure 7-9 displays photographs of the quartz stockwork veins south of the historical open pit.

Figure 7-10 illustrates the planes and poles of the principal vein sets at Goldlund. The left-hand stereonet in Figure 7-10 (A), displays planes and contoured poles to gold-bearing 20 set (blue) and 70 set (red) veins for all 128 measurements. The planes of the two vein-sets have an intersection lineation of 294°/53°NW (Pettigrew 2012).

The right-hand stereonet in Figure 7-10 (B) displays the contoured poles with cylindrical best fit and resulting average planes of 20 set (blue) and 70 set (red) veins including the average intersection lineation between the two vein-sets. The actual angle between the average two vein sets is 42°.

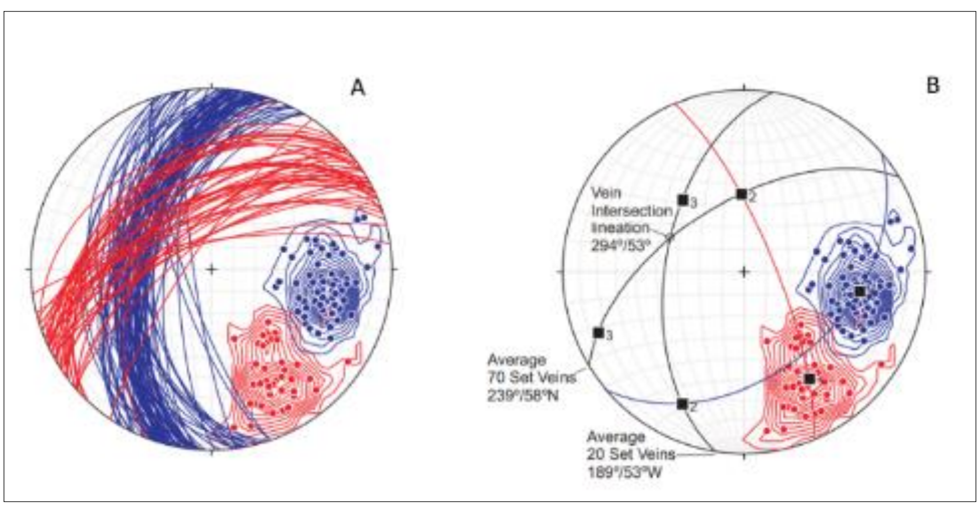
The 20 and 70 set of veins are synchronous and have often been described as conjugate in their formation. This is borne out by their orientation as the acute angle between planes of the two veins sets range from ~25° to 50° with the average of all measured veins being 42°, right-hand stereonet (Figure 7-10).

Figure 7-9: Goldlund Project Zone 1 Quartz Stockwork Mineralization



Source: Ausenco (2021)

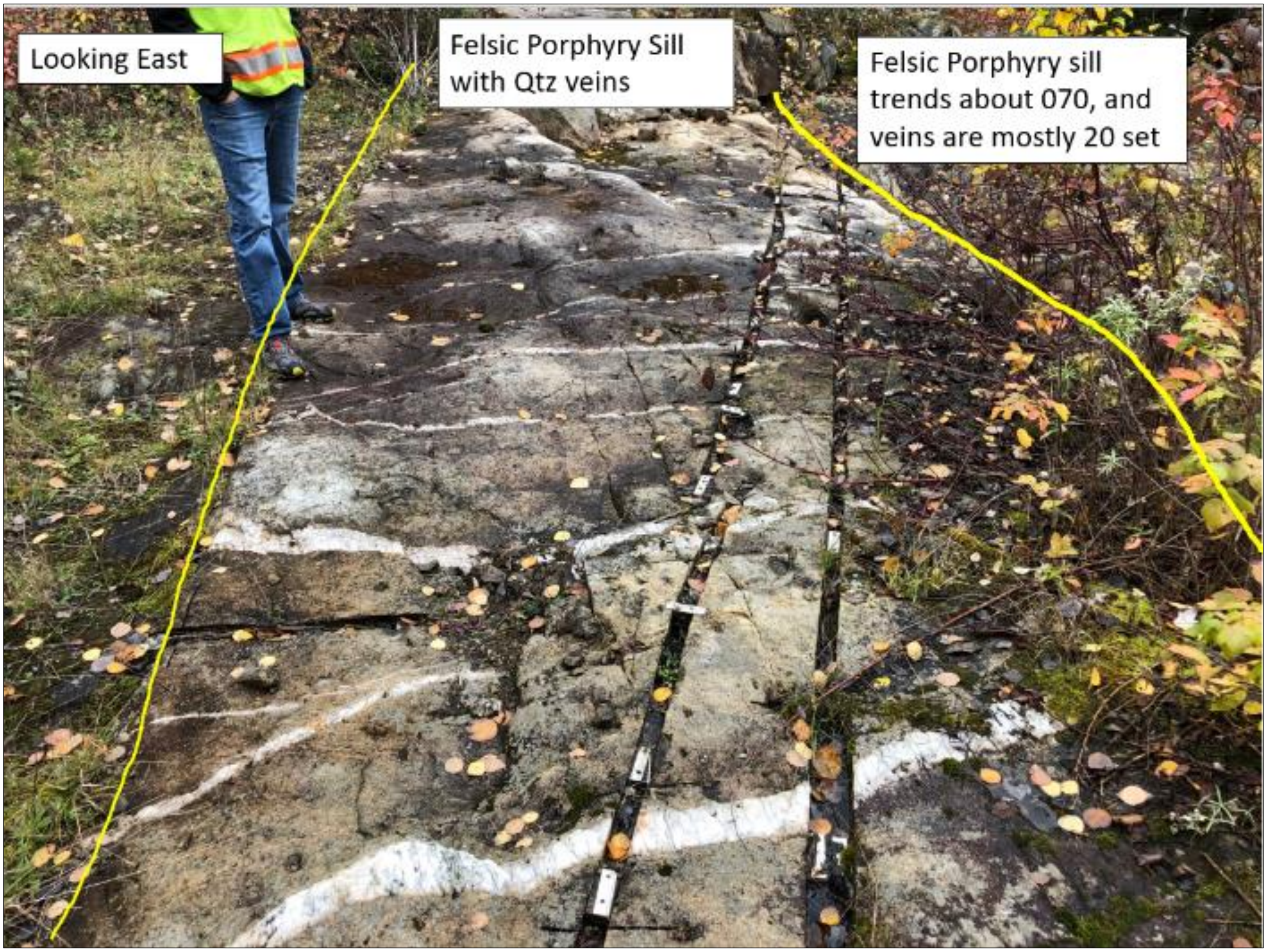
Figure 7-10: Stereonet of the Planes & Poles to Goldlund Gold-Bearing Veins



Source: Pettigrew (2000)

Figure 7-11 displays transverse veins (20 set) developed in a felsic porphyry sill observed in trench GDA-12-01.

Figure 7-11: Transverse Quartz Veins developed in Felsic Porphyry Sill in Trench TR-12-01



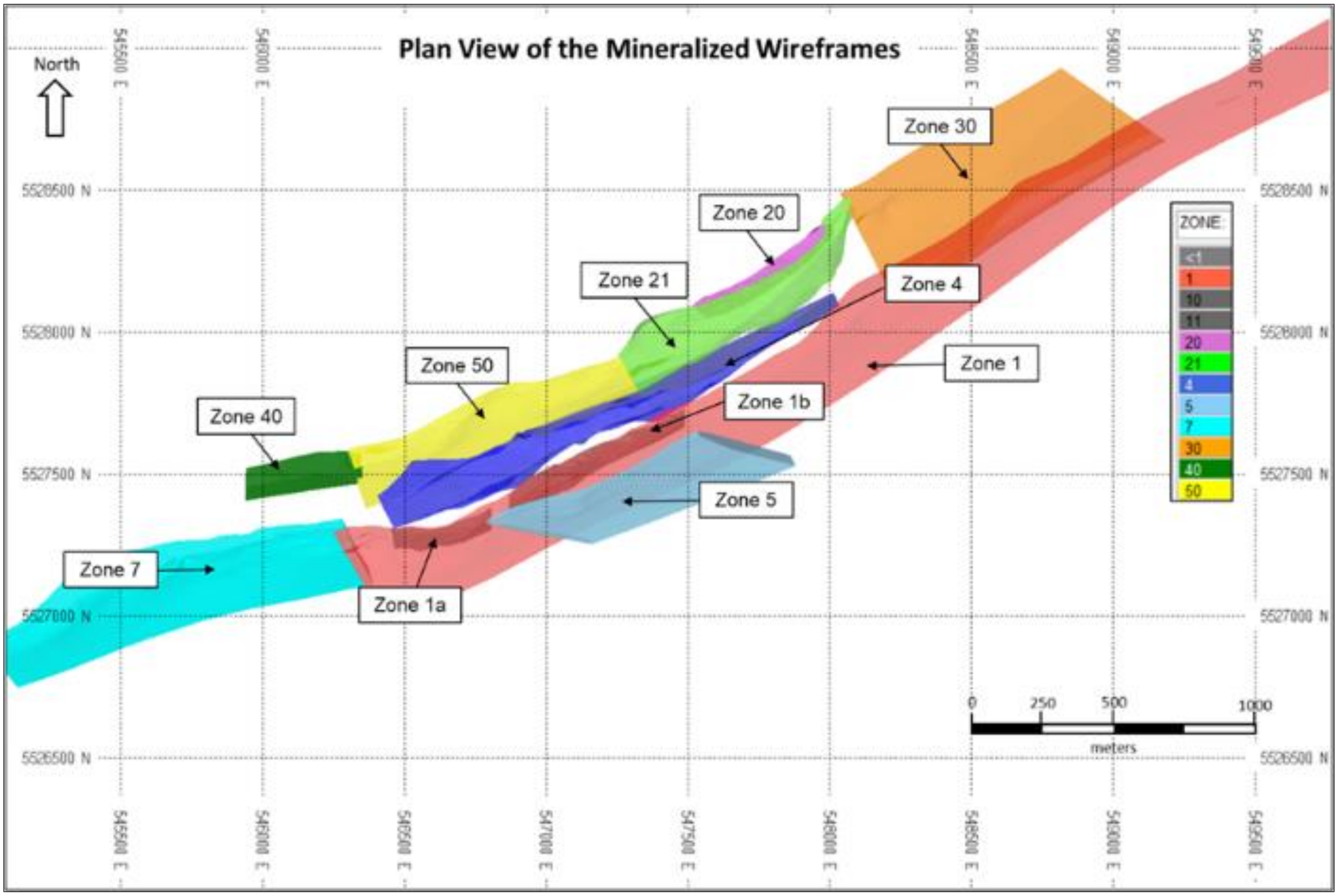
Source: Ausenco (2021)

The gold mineralization has been interpreted by Miro Mytry P. Geo., of First Mining, as a series of nine northeast-trending sub-parallel zones, using a nominal 0.1 g/t Au threshold, as shown in Figure 7-12. This interpretation was prepared prior to the acquisition of Goldlund by Treasury Metals and is considered appropriate for this style of mineralization.

Zones 1, 7, and 5 consist principally of gold mineralization associated with the stockwork veins in the large granodiorite sills.

Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralization that is hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only minor contribution from the granodiorite sills.

Figure 7-12: Plan View of the Goldlund Project showing Interpreted Mineralized Zones



Source: Treasury Metals

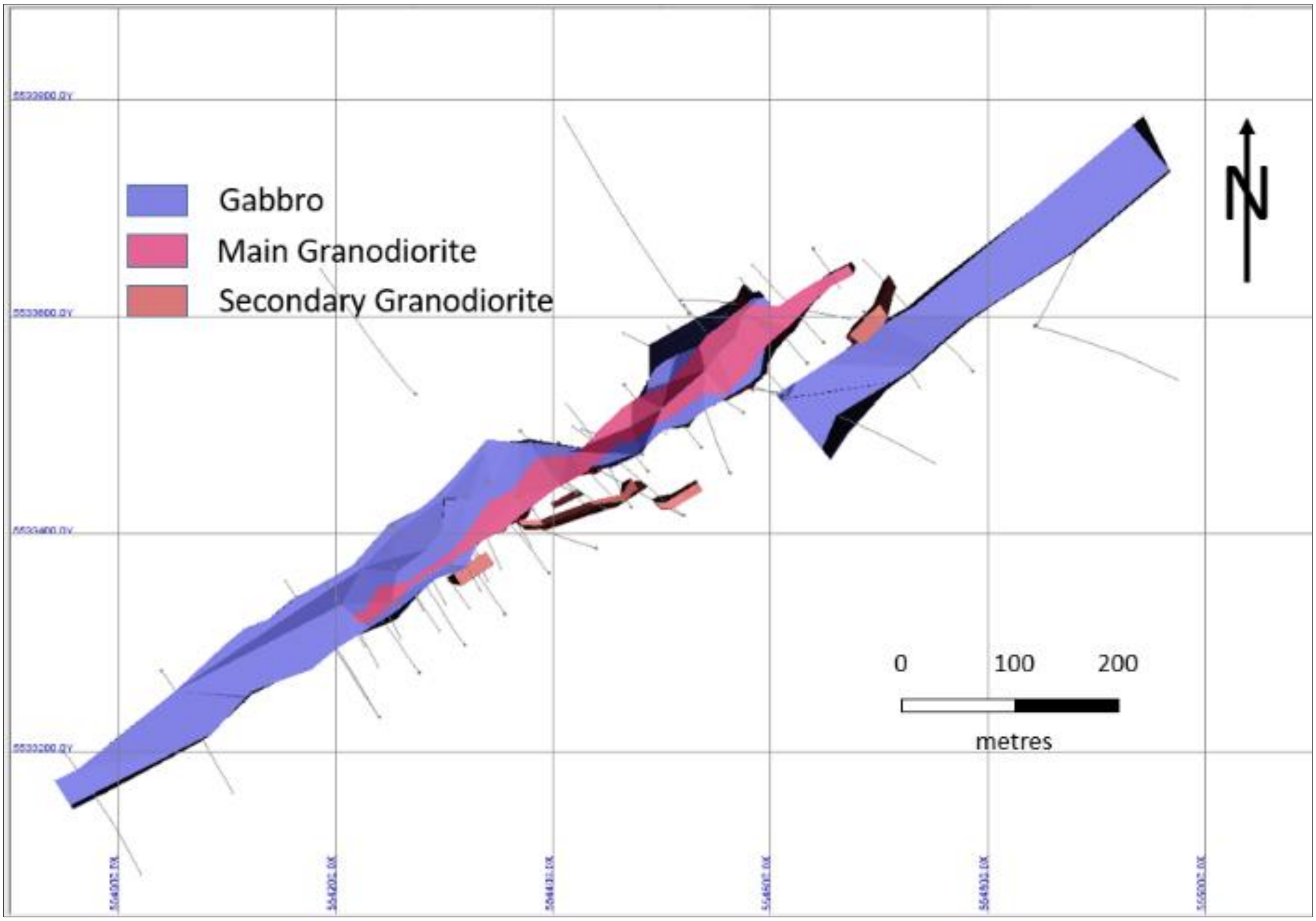
7.5 Miller Project

The Miller project is situated approximately 8 km northeast and along strike of the Goldlund project. The geology and gold mineralization are similar to those of the Goldlund project, as described in Section 7.4

Figure 7-13 presents a plan view of the interpreted geology.

Similar to the Goldlund deposit, the gold mineralization at the Miller deposit "is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprising of numerous quartz veinlets less than 1 to 20 cm thick. These stockwork zones form bands within the sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite" (WSP, 2020).

Figure 7-13: Plan View of the Miller Project showing Interpreted Geology



Source: SRK

8 DEPOSIT TYPES

8.1 Goliath Deposit Model

In 2001, Teck-Corona originally described the Goliath deposit as a shear-hosted mesothermal gold deposit with structurally controlled gold mineralization related to local silica and sulphide replacements, and widespread, small, discordant to concordant quartz and sulphide veins. However, the deposit is not hosted within a shear Zone and is missing most of the critical attributes of these types of deposits. The host rocks do not contain typical iron-carbonate alteration mineral assemblages and gold is not commonly hosted by quartz veins in association with silicification (Beakhouse, 2002). Furthermore, the gold mineralization is generally associated with highly elevated silver (locally >100 g/t Ag but varies significantly across the deposit), zinc and lead in the form of stringers and layers within felsic volcanic schist which is not common in shear-hosted mesothermal gold deposits (Page, 1995a).

Page (1995b) describes the alteration of the host rocks in the area of the deposit as being enriched in potassium and depleted in sodium, which is a diagnostic feature peculiar to volcanogenic massive sulphide (VMS) deposits. Wetherup (2008) suggested that the deposit may be part of a VMS system within a bimodal package of folded volcanic strata on the basis of this classic K-Na alteration signature along with the close association of gold with silver, zinc, and lead. No massive sulphide cap has been found to date. However, in 2012 isolated lenses of massive sulphides consisting of pyrrhotite and pyrite (no base metals) were intersected in drillholes TL12245 and TL12247 in the nose of the northeast regional fold. Although this model does not fit perfectly, it should not be dismissed as a possible mechanism in which the gold was originally introduced into the system. In addition, future exploration work should also not dismiss the possibility of perhaps finding a gold-zinc VMS deposit near surface or at depth elsewhere on the property.

Treasury favours a hybrid deposit-type model, also known as a "pre-orogenic atypical greenstone belt gold model" as a promising genetic model to explain the geology, structures and mineralization observed within the Goliath deposit. In this model, early gold-rich volcanogenic sulphide mineralization is overprinted by subsequent deformation and alteration events which can contribute to further concentration and/or remobilizing of both precious and base metals. This model also integrates potential VMS and magmatic hydrothermal Archean lode gold deposit (magmatic hydrothermal) models in the formation of the deposit. It is likely that the Goliath deposit does not fit into any one idealized model and neither should be discounted.

Hardie et al. (2012) suggested "the gold mineralization at the Rainy River gold deposit can be interpreted as a hybrid deposit-type consisting of an early gold-rich volcanogenic sulphide mineralization [pre-orogenic] overprinted by shear-hosted mesothermal [post-orogenic] gold mineralization. Both styles of gold mineralization have been progressively overprinted by deformation, whereby auriferous quartz veins post-date the sulphide stringers and veins and were emplaced during active deformation". The presence of isoclinal folding of the pyrite-sphalerite-chalcopyrite-galena stringer veinlets gives the mineralization a relative timing of pre- to syn-deformational. Folded mineralized stringers are found within the quartz-sericite- schist at the main deposit.

Treasury believes that there are similarities between the Rainy River deposit and Goliath and have integrated the hybrid deposit-type model into a final simplified four-stage hybrid model for the genesis of the Goliath deposit. The four stages are described below.

Stage 1: Pre-Orogenic Event. Anomalous gold, silver, zinc, and lead mineralization is introduced as part of a VMS and/or magmatic hydrothermal system along a pre-orogenic structure consisting of stratigraphically sheared felsic volcanic (or volcanoclastic) and sedimentary rocks. If it is a VMS system, potassic alteration accompanies the mineralization event or the felsic rocks are altered by the hydrothermal solutions moving through this conduit. Quartz and quartz-feldspar porphyries may be the heat engine, or remnants of the heat source, that drove the hydrothermal solutions as these intrusive rocks are early-stage and are folded and deformed with the rest of the rocks in subsequent deformation events. At this stage, the sericite altered weakly mineralized zone may have been several 100 m in width.

Stage 2: D_1 Deformation Event. The stratigraphic units within the deposit are isoclinally folded into an anticlinal (anticlinorium) structure whose fold axis runs east-west along the entire strike length of the deposit and plunges 10° to 20° to the east following the altered felsic volcanic rocks, which are sheared and foliated (axial planar S_1 and F_1). V_1 quartz veins are formed parallel to stratigraphy.

Stage 3: D_2 Deformation Event. Northeast-striking (060°) F_2 structures intersect F_1 structures accompanied by later magmatic hydrothermal solutions which remobilize the gold, silver and base metals and re-concentrate and upgrade them within steeply west-dipping shoots that now host the "high-grade" gold and silver mineralization. Silicification accompanies this event and V_2 quartz veins are developed. The relative abundance of base metals varies along strike depending on the original concentrations at different locations along the initial shear structure.

Stage 4: D_3 Deformation Event. Brittle faults, fractures and white non-mineralized V_3 quartz veins form (dip moderately north-northeast) and crosscut or follow local foliation.

8.2 Goldlund Deposit Model

The Goldlund project hosts Archean, shear zone-hosted quartz vein mineralization (Archean lode-gold), occurring as extensional quartz vein systems, particularly between rocks with high competency contrast. Archean lode-gold deposits occur in a broad range of structural settings, and at different crustal levels, but they share a similarity in ore fluid characteristics. Mineralization is typically late tectonic, occurring after the main phases of regional thrusting and folding, and generally late-syn to post-peak metamorphism with most of the significant deposits in areas of greenschist facies. Many deposits are related to the reactivation of earlier structures.

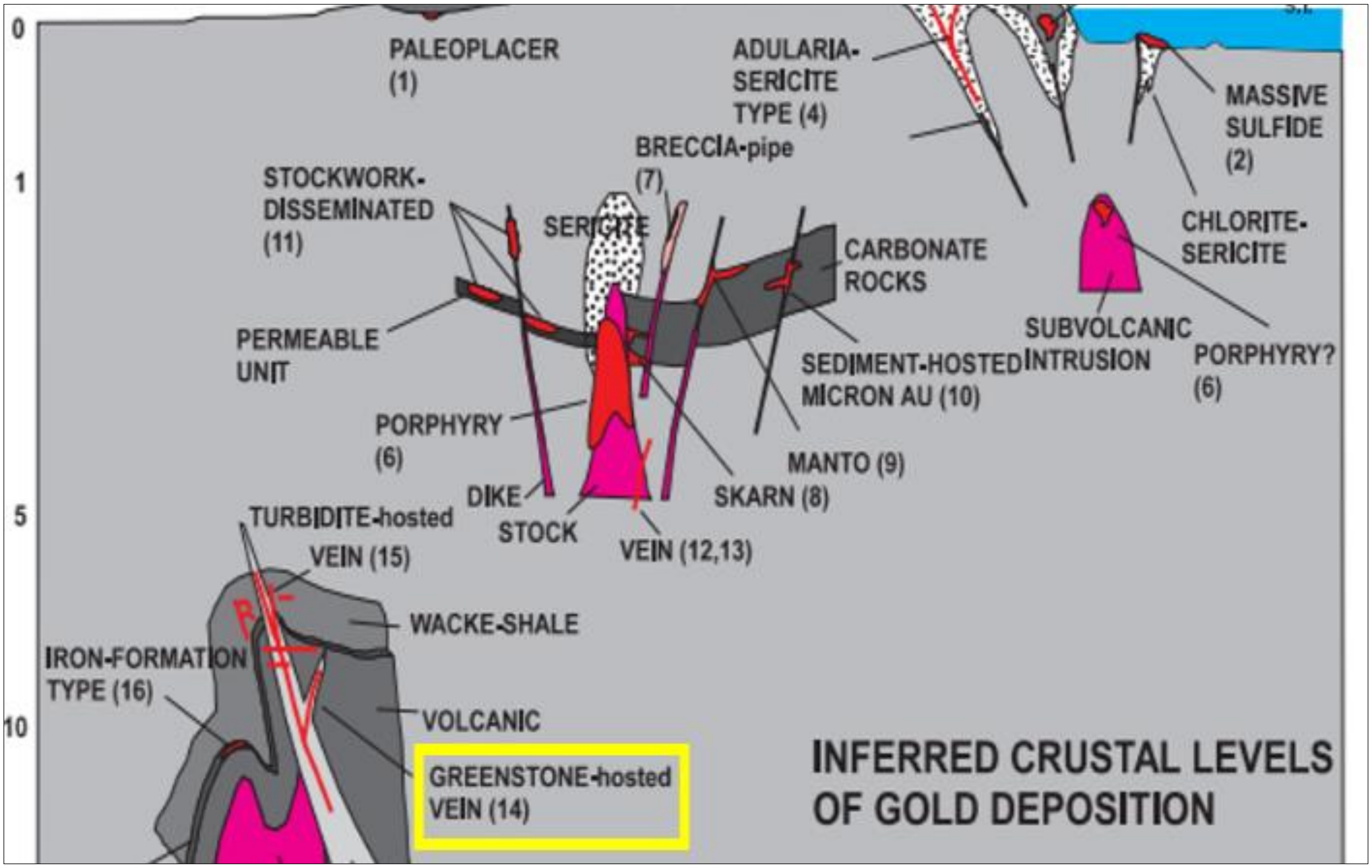
Archean lode-gold occurrences are common in the Sandybeach Lake – Sioux Lookout area and are concentrated in the Southern and Central volcanic belts. Vein systems in both belts are the product of Stage 3 deformation and are related to the northeast-southwest extension associated with northwest-southeast compression and shortening; the brittle-ductile deformation near the steep, northeast-trending shear zones; and the tightening of the Stage 3 folds.

Gold-bearing vein systems in the Southern Volcanic Belt are typically controlled by the steep, Stage 3 northeasterly trending shears. The host mafic metavolcanic rocks are typically chlorite-ankerite schists up to several meters in width. Pyrite, with subordinate chalcopyrite, sphalerite, and galena, are the main sulphide minerals in the auriferous veins.

A few shear zone hosted gold occurrences are also present in the Central Volcanic Belt, but the dominant and economically most significant type are the transverse vein systems within competent rocks, particularly in the intermediate to mafic meta-subvolcanic intrusive sills. Vein systems occupy tensional fractures related to internal deformation of the competent units as folds tightened during Stage 3 deformation. Vein arrays could be expected to develop near fold hinges, within fold limbs, and along axial planar foliations. The orientations of individual veins within the arrays are affected by their locations within the folds.

The gold mineralization at Goldlund has similarities to the Buffalo Gold deposit in Red Lake, Ontario and the Sigma Mine in Val-d'Or, Quebec (Pettigrew, 2012). In 1997, Robert, Poulsen, and Dube' classified the Sigma Mine as a greenstone-hosted quartz-carbonate vein deposit, that occurs within greenstone-belts spatially associated with major fault zones. The quartz-carbonate veins are associated with brittle-ductile shear zones. Figure 8-1 shows a schematic representation of the crustal levels inferred for gold deposition for the commonly recognized deposit types. The depth scale (left-hand side of the drawing) is approximate and logarithmic. The greenstone-hosted quartz-carbonate vein deposit is labelled as 14 and highlighted with a yellow box. This gold deposit type forms at a depth of approximately 10 km.

Figure 8-1: Schematic of Representation of Gold Deposit Models



Source: Robert, Poulsen & Dube (1997)

8.3 Miller Deposit Model

Gold mineralization at Miller is similar to Archean shear Zone hosted quartz vein model (Archean lode gold). The Archean lode gold occurrences are common in the Sandy Beach Lake – Sioux Lookout area and are concentrated in the Southern and Central volcanic belts.

Vein systems in both belts are the product of Stage 3 deformation and are related to:

- northeast-southwest extension associated with northwest-southeast compression and shortening
- ductile-brittle deformation near steep northeast-trending shear zones
- tightening of Stage 3 folds.

Vein systems in the Southern Volcanic Belt are typically controlled by the steep, Stage 3 northeasterly trending shears. Host mafic rocks are chlorite-ankerite schists up to several meters in width. Pyrite, with subordinate chalcopyrite, sphalerite, and galena are the main sulphide minerals in auriferous veins.

A few shear zone hosted occurrences are also present in the Central Volcanic Belt, but the dominant, and economically most significant type, are transverse vein arrays within competent rocks and particularly the intermediate to mafic sub-volcanic intrusive sheets. Vein systems occupy tensional fractures related to internal deformation of the competent units as folds tightened during Stage 3 deformation. Vein arrays could be expected to develop near fold hinges, within fold limbs, and along axial planar foliations. The orientations of individual veins within the arrays are affected by their locations within folds.

9 EXPLORATION

9.1 Goliath Deposit

Since 2008, Treasury Metals has focused its exploration work on the western half of the property to evaluate the gold potential of the Goliath deposit. During this 12-year period, exploration activities consisted of re-establishing the former Teck exploration grid, geological mapping and sampling, prospecting, the completion of structural studies, trenching and channel sampling, the completion of a ground IP geophysical survey and two airborne geophysical surveys, downhole IP and tomography surveys, metallurgical testing, mineral resource estimations of the main deposit (including Preliminary Economic Analyses in 2012, 2017 and 2020) and the completion of 18 diamond drilling programs (see Table 9-1).

The 2008, 2009 and 2010 exploration programs were conducted and managed by Caracle Creek International Consulting Inc. (CCIC) of Toronto, Ontario (Ilieva, 2008, Ilieva 2009, Palich, 2010). Treasury Metals personnel assumed field management all exploration activities as of February 2011.

The exploration work completed on the property has been documented in a number of independent technical reports prepared for the Company and is summarized below (P&E, 2015; Roy et al., 2012; Roy and Trinder, 2011; Roy and Trinder, 2008). Assessment reports filed with the Ministry of Northern Development and Mines ("MNDM") provides additional information on their exploration activities. The reader is directed to Section 10 for details regarding the diamond drilling programs completed by Treasury Metals from 2008 to 2020. Table 9-1 provides a summary of the exploration work conducted by Treasury Metals from 2008 to 2021.

Table 9-1: Exploration Activities from 2008 to 2021

Year	Company	Work Completed
2008	CCIC	Core reclamation: exploration grid cut (65.9 line-km)
	CCIC	Geological mapping (1:5,000 Scale), 32 grab samples collected including 17 whole rock and REE analyses
	CCIC	Diamond drilling program – 55 holes (TL0801 to TL0855)
2008	CCIC	Structural study on 2008 drill core
	CCIC	One Main Zone trench, 10 Channels, 29 samples, channel sampling iron formation (3 channels, 25 samples) + mapping
	Firefly Aviation Ltd.	Aeromagnetic (HRAM) survey, 309 line-km covering 3,064 ha
	JVX Geophysical Surveys & Consulting	Ground IP/resistivity survey, 29.6 line-km covering 230 ha
	A.C.A. Howe International Limited	Mineral resource estimate (NI 43-101 compliant)
2009	CCIC	Prospecting, sampling and mapping program covering nine legacy claims; outcrop sampling (5 grabs) and channel sampling (34 channels, 115 channel samples)
	CCIC	Diamond drilling program – 31 holes (TL0956 to TL0986)
2010	CCIC	Downhole DCIP/resistivity EarthProbe survey; 60 holes profiled; 94 hole-to-hole tomography imaging; 4-line, 21 surface-to-hole tomography pairings; petrographic/ SEM Study (Beakhouse, 2010); SCIP core testing
	CCIC	3 phase diamond drilling program – 32 holes (TL1087 to TL10118)
	CCIC	Trenching of Main Zone, mapped and channel sampled, 47 channel samples, 2 duplicate channels, 4 geological units mapped
	A.C.A. Howe International Limited	Updated resource estimate & preliminary economic analyses
	Ministry of Northern Development and Mines	Petrographic and scanning electron microscopy
2011	Treasury Metals	Diamond drilling program – 111 holes (TL11119 to TL11229)
	G & T Metallurgical Services Limited, B.C.	Preliminary metallurgical test program, 59 kg composite sample; grindability, gravity and cyanidation testing
	Fugro Airborne Surveys	DIGHEM EM & magnetic survey (July), helicopter, 582.62 line-km
	A.C.A. Howe International Limited	Updated resource estimate (NI 43-101)
2012	G & T Metallurgical Services Limited, B.C.	2 Tests: gravity + cyanidation and just cyanidation (48 hours); Sample size 398.5 kg, ½ diamond core, 163 samples
	Treasury Metals	2 phase diamond drilling program – 81 holes (TL12278 to TL12295; 15 re-entry holes)
	Treasury	Goliath 3D inversion study (Ellis, 2012); petrographic work
	A.C.A. Howe International Limited	Preliminary economic analyses (using 2011 Resource Estimate)
	Vancouver Petrographic	This section study on mineralized drill core
2013	Treasury Metals	Diamond drilling program – 48 holes (TL13296 to TL13336; 7 Re-entry holes)
2014	Treasury Metals	2 Phase diamond drilling program – 48 holes (TL14337 to TL14377; 5 re-entry holes, 3 wedges and 1 abandoned hole)
	Treasury Metals	Soil mobile metal ion survey (MMI) – property-wide survey
	Gekko Systems Pty Ltd (Australia)	Leach optimization testwork and bulk concentrate production; cyanide detox testwork; high-grade and medium-grade ore testwork (gravity, flotation, cyanide leach recovery)
2015	Treasury Metals	Diamond drilling program – 27 holes (TL14378B, TL15379 to TL15402; 2 re-entry holes); infill core sampling program (95 holes, 2,091 samples); cyanide bottle roll testing program (19 holes, 391 samples).
	P & E Mining Consultants Inc.	Updated mineral resource estimate (NI 43-101)
2016	Treasury Metals	Diamond drill program – 19 holes (TL16403 to TL16420), 1 wedge hole (TL16-415W1)
	Treasury Metals	Condemnation field mapping program – 146 grab samples (G156001 to G156146), 15 channel samples (C156351 to C156365), 7 coarse blanks and 7 standards (CDN-CM-26) were used during the sampling. Covers an area of approximately 1.4 km^2.
	Treasury Metals	Eastern alteration corridor mapping and sampling program
	Treasury Metals	Gossan showing mapping and sampling program
2017	Treasury Metals	2 phase diamond drill program – 43 holes (TL17421A to TL17463)
	Treasury Metals	Iron formation mapping program – 36 grab samples, in addition to 2 coarse blanks and 2 standards (CDN-CM-26 & CDN-GS-1P5K) were used during the sampling. Covers an area of approximately 5 km^2
	Treasury Metals	Outcrop mapping program (western map area, northwest map area, Central map area, Eastern map area)
	Treasury Metals	Infill sampling program – 5256 Samples (across 142 drillholes), including 525 blanks and standards.
2018	Treasury Metals	Diamond drill program – 38 holes (TL18464 to TL18501)
	Treasury Metals	Soil gas hydrocarbon sampling program – 845 soil samples. Covers an area of approximately 9.88 km^2
	Knight Piésold Consulting Ltd.	Geotechnical drill program – 20 holes. Covers an area of approximately 2 km^2.
2019	Golden Mallar Corp	Hole to hole spectral induced polarization/resistivity survey
	Treasury Metals	Soil gas hydrocarbon sampling program – 1,040 soil samples. Covers an area of approximately 10.25 km^2
	Treasury Metals	Diamond drill program – 12 holes (TL19502 to TL19513)
2020	Treasury Metals	Diamond drill program – 15 holes (TL20514 to TL20528)
	Axiom Exploration	Soil gas hydrocarbon sampling program – 1,260 Soil Samples. Covers an area of approximately 12.50 km^2
2021	Treasury Metals	Diamond drill program – 73 holes (TL20529B to TL20601)

9.1.1 2008 Exploration Activities

9.1.1.1 Geological Mapping Program

An exploration grid was cut in January 2008 to facilitate geological mapping, sampling, ground geophysical surveys, trenching and diamond drilling programs. A total of 69.5 line-km were cut with the base line established along Norman Road which represented the former border between the old Laramide and Teck properties. Grid lines were cut at 50 m intervals perpendicular to the baseline in an attempt to establish or mimic the former Teck grid. The grid consisted of 30 lines at approximately 1,500 m length, 11 lines at 1,225 m, and five lines at 1,025 m.

Geological mapping, at a scale of 1:50,000, was completed between June and August 2008. Major lithological units were identified, structures interpreted, and a new geological map of the property was completed (Figure 9-1). Thirty-two representative and grab samples were taken (Ilieva, 2009), and 17 samples were sent to Accurassay Laboratory in Thunder Bay for fire assay, whole rock and rare-earth element (REE) analyses. None of the samples returned any significant precious or base metal assays.

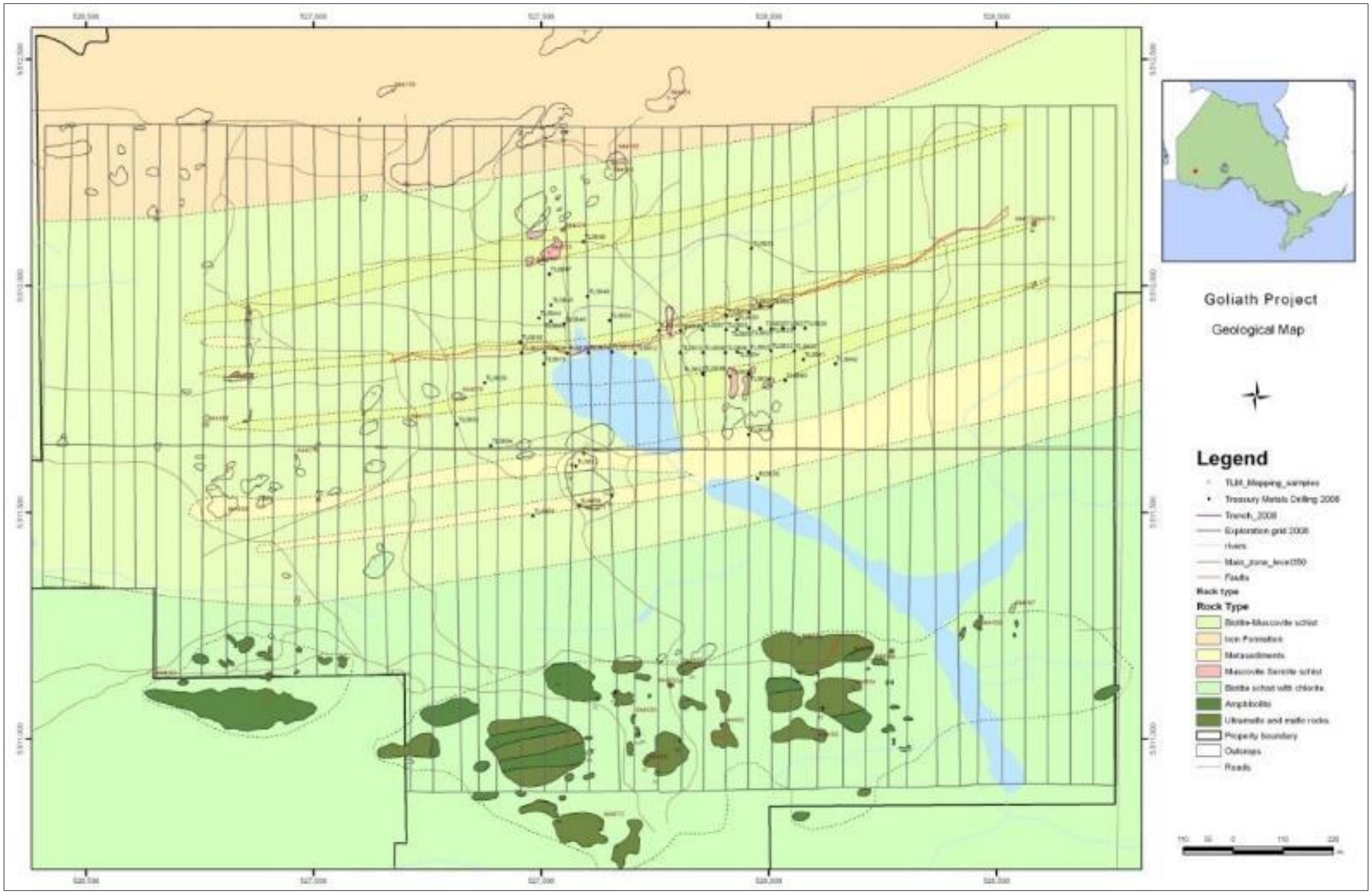
9.1.1.2 Structural Geology Study

CCIC was retained by Treasury Metals to review both the geological and structural data on its Thunder Lake property (now the Goliath property) and prepared a report containing a structural description and interpretation of the geology (Wetherup, 2008). Three different generations of folds and deformational events were described (see Table 9-2).

Oriented core was used during the 2008 diamond drilling program for the first time to collect additional structural data (Roy et al., 2012). Core from drillholes TL0822 to TL0837 was used for this study. Foliation, geological contacts, fault lines and fold axes were measured using an Ezy-Mark™ core orientation tool provided by BoreInfo Ltd. (BoreInfo). The purpose of this program was to clarify the spatial relationships between the structural features and their influence on the mineralization.

CCIC observed that the F_2 folds (axial planes) upgrade gold mineralization within the Main Zone and that gold is focused in shoots where F_1 and F_2 structures intersect and where F_2 structures are concentrated (in the shoots). Shoot structures are steeply plunging (west as observed on current Treasury Metals longitudinal sections of the Main and C Zones). In addition, it should be noted that the zones of alteration and gold mineralization strike more northerly and assume a northeast strike east of the deposit and are nearly parallel to the strike of the F_2 axial planes. Therefore, it was concluded that it might be more difficult for exploration drilling to locate and intersect gold-bearing shoots in this region.

Figure 9-1: Geological Grid Map (Goliath Deposit Outlined in Red)



Source: Treasury Metals (2015)

Table 9-2: Summary of Structural Features Observed at the Goliath Deposit

Event	Structure	Description	Veins	Description
D_0	S_0	Compositional layering of meta-volcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	Greyish, highly deformed, S_1 foliation parallel quartz-sulphide ribbons and silicification surrounded by quartz-sericite schist
D_1	F_1 S_1	Isoclinal folding F_1 axial planar and layer parallel foliation/schistosity	V_1	White deformed, locally crosscutting quartz+/-tourmaline+/-sulphide veins
D_2	F_2	Closed (60°) folds; axial planes ~045/90°; discrete, 50-40 m spaced, axial planes	V_2	Weakly deformed white quartz+/-sulphide veins along F_2 axial planes & at 45° to F_2 axial planes.
D_3	NW Fault	Brittle faults/fractures dip moderately NNE	V_3	Un-deformed white, non-planar quartz veins dip moderately NNE and follow foliation locally

Source: Treasury Metals (2015)

9.1.1.3 Trench and Channel Sampling

A 1,005 m long trench, oriented north-south, was excavated in September 2008 to expose auriferous "Main Zone" mineralization intersected by diamond drilling within the Goliath deposit (Ilieva, 2009). The trench, located at UTM 527782E, 5511893N (NAD 83, Zone 15N), is an elongated oval shape and measured at surface 46 m in length, 14 to 15 m wide and 5 m deep. A decline was added at the southern end of the trench for easier access.

Two outcrops were exposed and geologically mapped at a scale of 1:200 and channel sampled perpendicular to strike. The bedrock geology was described as strongly altered (sericitized) volcanic rocks. Ten channel samples (designated Channel 1 to 10) were cut across the two exposures and 29 samples were collected. Each channel is approximately 4 to 5 cm wide and 5 to 6 cm deep (Roy and Trinder, 2008). A blank or standard was inserted in alternating order at every tenth sample. All samples were dispatched to Accurassay for gold and base metal analyses.

Two zones of mineralization were exposed in Channel 3 and Channel 5 located about 2.5 m to the south. Channel 3 (Sample 644112) returned the highest gold value of 27.55 g/t Au and 2.19 g/t Ag over a sample length of 0.65 m (see Table 9-3). A 1.5 m lower grade mineralized interval was also sampled in Channel 5 where samples 644115, 644116 and 644117, each 0.5 m in length, returned 1.75 g/t Au, 2.74 g/t Au and 1.03 g/t Au, respectively.

Table 9-3: Significant Assay Results from 2008 Channel Sampling

Channel	Sample Number	Length (m)	Au (g/t)	Ag (g/t)	Cu (ppm)	Pb (ppm)	Zn (ppm)
3	644112	0.65	27.55	2.19	43	98	34
5	644115	0.50	1.75	3.70	145	280	351
5	644116	0.50	2.74	3.78	48	346	386
5	644117	0.50	1.03	1.97	39	92	87

Source: Treasury Metals (2015).

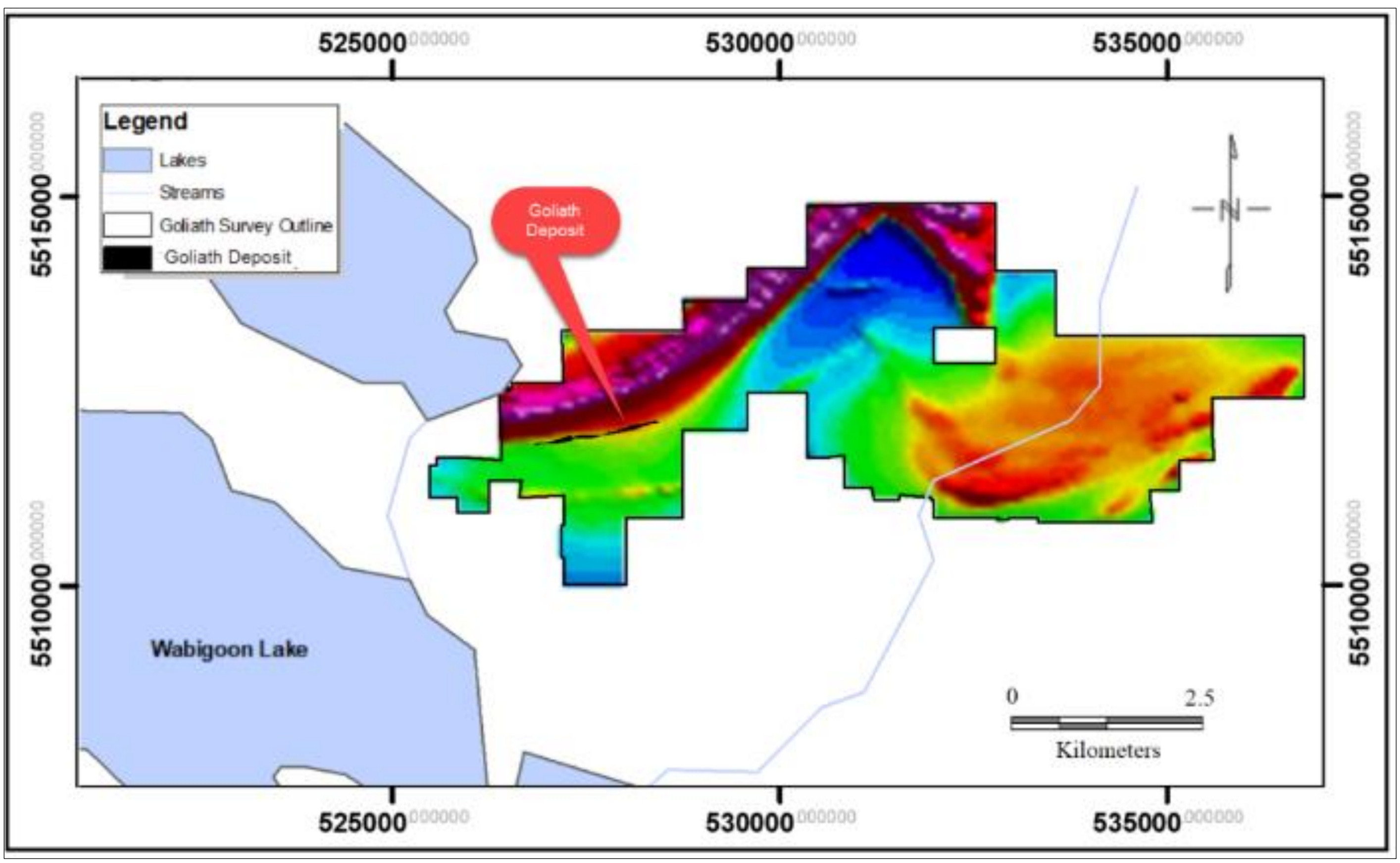
Three channels were cut across a bedrock exposure of an iron formation that outcrops on either side of Tree Nursery Road located at in Zealand Township (UTM 528767E, 5513144N; UTM 528803E, 5513165N; UTM 528802E 5513155N, NAD83, Zone15N). Twenty-five channel samples were collected and dispatched to Accurassay in Thunder Bay for analyses for gold, base metals, and trace element geochemistry (31 element package). Only one sample returned gold value in excess of 0.2 g/t Au.

9.1.1.4 Geophysical Surveys

9.1.1.4.1 Aeromagnetic (HRAM) Survey

A high-resolution aeromagnetic survey (HRAM) was completed by Firefly Aviation Ltd. (Firefly) during March 2008. A total of 2,165-line km were flown by fixed wing aircraft covering an area of 180 km^2 (see Figure 9-2) North-south survey lines were flown at 100 m spacing and east-west tie lines flown every 500 m covering a large area of Zealand and Hartman Townships and the southern portions of Brownridge and Laval Townships (Evans, 2008). Standard and enhanced gridding filters were applied to the Goliath survey data based on the calculated international geomagnetic reference model (IGRF). This survey was conducted using a NAD83, Zone 15 projection and datum.

Figure 9-2 : Firefly Geophysical Total Magnetic Field Intensity Map



Source: Modified from McKenzie (2008)

According to McKenzie (2008), the data was subsequently interpreted by Balch Exploration Consulting Inc. (BECI). The bedrock underlying the survey area reflects the typical magnetic signature of a regional greenstone belt which is expressed as a large arcuate high/low sequence reflecting the magnetite precipitated during and after formation along with subsequent tectonic deformation. However, the Goliath deposit is not detected on the airborne magnetic survey and occurs in a magnetic low. The property is underlain by large-scale synclinal and anticlinal folded structures, and it was concluded that the magnetic data provides a better understanding of the F1 fold architecture. Secondary F2 structures, believed to be responsible for upgrading concentrations of both gold and silver at Goliath, are not identified by the survey results. A regional thrust fault is mapped throughout the southern extent of the survey. This is coincident with a string of discrete magnetic bodies occurring along the trace of the fault.

9.1.1.4.2 Ground Induced Polarization/Resistivity Survey

JVX Geophysical Surveys and Consulting (JVX) was contracted by Treasury Metals to conduct 29.6 line-km spectral IP/resistivity survey on the Goliath project grid from March 31 to May 1, 2008. The maximum vertical depth of penetration of this survey was approximately 60 m (Palich, 2010b). This grid covered the main resource area for a strike length of approximately 2.0 km. The exploration grid consisted of 21 north-south oriented lines at 100 m spacing plus two line segments from stations 750S to 750N. The survey instrumentation consisted of a Scintrex IPC-7 (2.5 kW) transmitter and Scintrex IPR-12 receivers. Surveys were completed in time domain with a pole-dipole array ('a' =25 m, n=1 to 8).

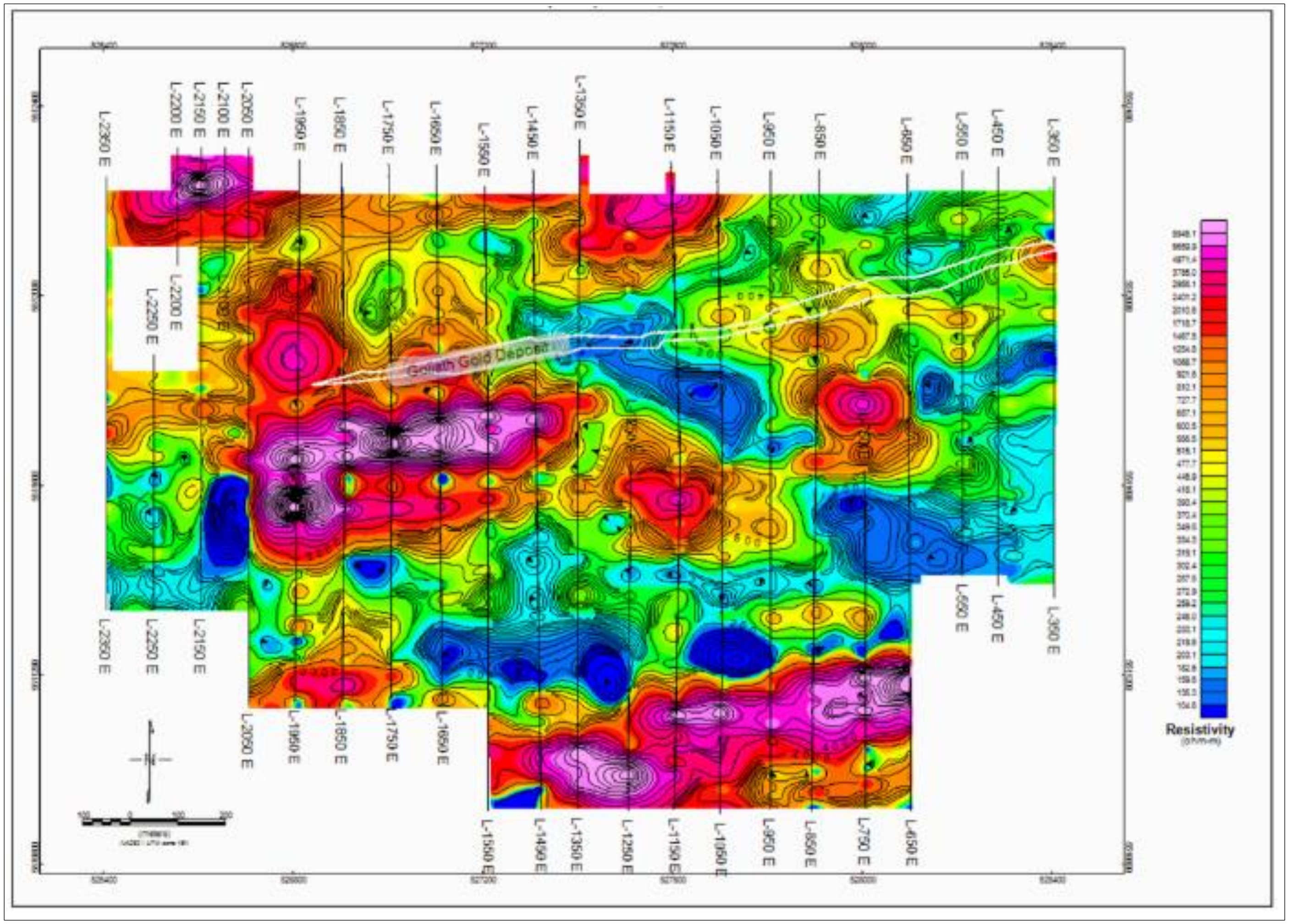
The contract stated that ground magnetic data would also be collected. However, due to time constraints, including poor weather, the deep IP and ground magnetic surveys were not completed (McKenzie, 2008). Plan maps at the scale of 1:5,000 resistivity (n=2) are presented in Figure 9-3.

It was determined that much of the survey area is covered by extensive surficial overburden with 43% of the survey area at 250 Ωm or less. Conductive overburden can mask chargeable bodies thus requiring a high percentage of sulphide mineralization to overcome this problem. However, JVX noted that despite the presence of conductive overburden, the conductivity was not as high as initially anticipated (Johnson and Webster, 2008).

The Goliath deposit is marked by weak resistivity highs in an area of predominantly low resistivity. Overall, the main gold deposit has a weak and uncertain IP/resistivity expression. It appears to be defined by three marginal IP anomalies associated with relative resistivity highs. This signature does not improve to the east or west of the deposit. South of the deposit, there is a coinciding chargeability and resistivity anomaly in the western portion of the deposit between lines L1950 to L450. A possible northwest-trending fault was also identified by the survey.

A series of pseudo-sections were also generated at the scale of 1:2,500 and can be found in the JVX report. Examination of these sections identified a possible northwest-trending fault and low values of chargeability which was interpreted to possibly displace the mineralization in a west-northwest direction (Ilieva, 2009). Seven IP anomalies were defined for possible follow-up exploration work and CCIC recommended that the data be inverted for proper 3D interpretation of the IP survey results (see Table 9-4).

Figure 9-3: 2008 JVX Ltd. Resistivity (n=2) Map, Goliath Property



Source: Ilieva (2009)

Table 9-4: 2008 IP Survey Targets Selected for Further Investigation

Anomaly ID	Easting	Northing	Comments
TL_0001	526661	5512237	Cluster of strong IP anomalies at north end of lines 2050W, 2200W; Shallow; N1 resistivities are moderate to high; Short time constants - response of fine-grained disseminated sulphides (+gold)
TL_0002	526908	5511224	Very strong, shallow IP anomalies 0 part of 300 m long IP zone with weaker end members that may define an east/west IP zone that crosses entire grid; Coincident lower resistivities at depth may indicate a partial cause by bedrock conductors; Strong IP anomalies noted - masked by conductive cover - short time constants upgrading for gold target
TL_0003	527010	5511629	Stronger of two IP anomalies - lower resistivity at depth - possible bedrock conductor - time constant uniformly long
TL_0004	527009	5511705	Part of 400 m long IP zone - may be on strike with Thunder Lake gold deposit; Moderate resistivity noted - possible bedrock conductor
TL_0005	527507	5512155	Two nearby strong, shallow IP anomalies 250 m north of Thunder Lake. N1 resistivities are moderate. Some outcrop/subcrop and a prospecting history are likely. Time constants are long or mixed
TL-0006	528006	5511247	One of two strong IP anomalies south of the Thunder Lake deposit; Part of east-west-trending IP/resistivity zones; Interpreted as probable bedrock conductors; This anomaly portion has short time constants and high resistivities - interesting for gold; N1 resistivity is high suggesting thin overburden
TL_0007	528006	5511021	One of two strong IP anomalies south of the Thunder Lake deposit; Part of east-west-trending IP/resistivity zones; Interpreted as probable bedrock conductors

9.1.2 2009 Exploration Activities

In 2009, general reconnaissance prospecting and some focused stripping and channel sampling, was completed by CCIC from July 6, 2009 to August 4, 2009. A small grid was cut and geologically mapped on the Collins Patent and the remaining work was concentrated on the Jones, Johnson Patent and 12 legacy claims.

Five grab samples were collected during the prospecting exercise, 22 channel samples collected from three stripped outcrops on legacy claim 1119562 and 93 channels collected from two stripped outcrops located just east of Tree Nursery Road near the power lines on the Johnson Patent (Parcel 15401) in Zealand Township, Lot 5, Concession 4.

Three samples returned significant gold assays from this program. The best gold assay was obtained from sample 59109 that assayed 20.519 g/t Au over a channel sample length of 1.0 m on the Johnson Patent. The host rock is described as a biotite-muscovite schist containing 1% to 2% sulphides and is identified by Treasury Metals as an outcrop exposure of Zone D just east of Tree Nursery Road, west of the hydro line. A second channel was cut directly adjacent to sample 59109 over a sample length of 1.0 m. That sample was subsequently cut into five 20 cm samples to isolate where the gold was concentrated. One of these samples C59139 returned 3.296 g/t Au over a sample length of 0.20 m. One grab sample from the reconnaissance prospecting program returned 2.14 g/t Au. However, the location of this sample was not disclosed in the memo-style report.

During the month of July, three and a half days were spent completing general reconnaissance prospecting, outcrop sampling and a channel sampling program to generate future exploration targets for geological mapping and sampling on legacy claim 4211252 (CCIC 2009b). Work was focused in Lot 1, Concession II within the southern portion of Zealand Township.

A detailed grid was set up over an outcrop area where five outcrops were exposed, and a 100 m long east oriented baseline and cross lines were established, and the lines were mapped at a scale of 1:500. The area was found to be underlain by predominantly meta-sedimentary rocks with lesser amounts of felsic volcanic/quartz porphyry rocks. A total of 24 channel samples, ranging from 0.3 to 1.0 m in length, were taken from five distinct outcrops with interesting mineralization (quartz veins with elevated pyrite) and dispatched to Accurassay for gold analyses. There are no individual sample descriptions of the mineralization. None of the samples returned any significant gold assays (best 5 ppb Au).

In July 2009, a reconnaissance prospecting program was conducted to ascertain the geology underlying legacy claim 4211250 (CCIC, 2009c). A total of 1.5 line-km were traversed in Lot 9, Concession II within the southern portion of Zealand Township. Only one large outcrop ridge was encountered on the traverse which appeared to be an unmineralized granitoid intrusive rock. No samples were taken.

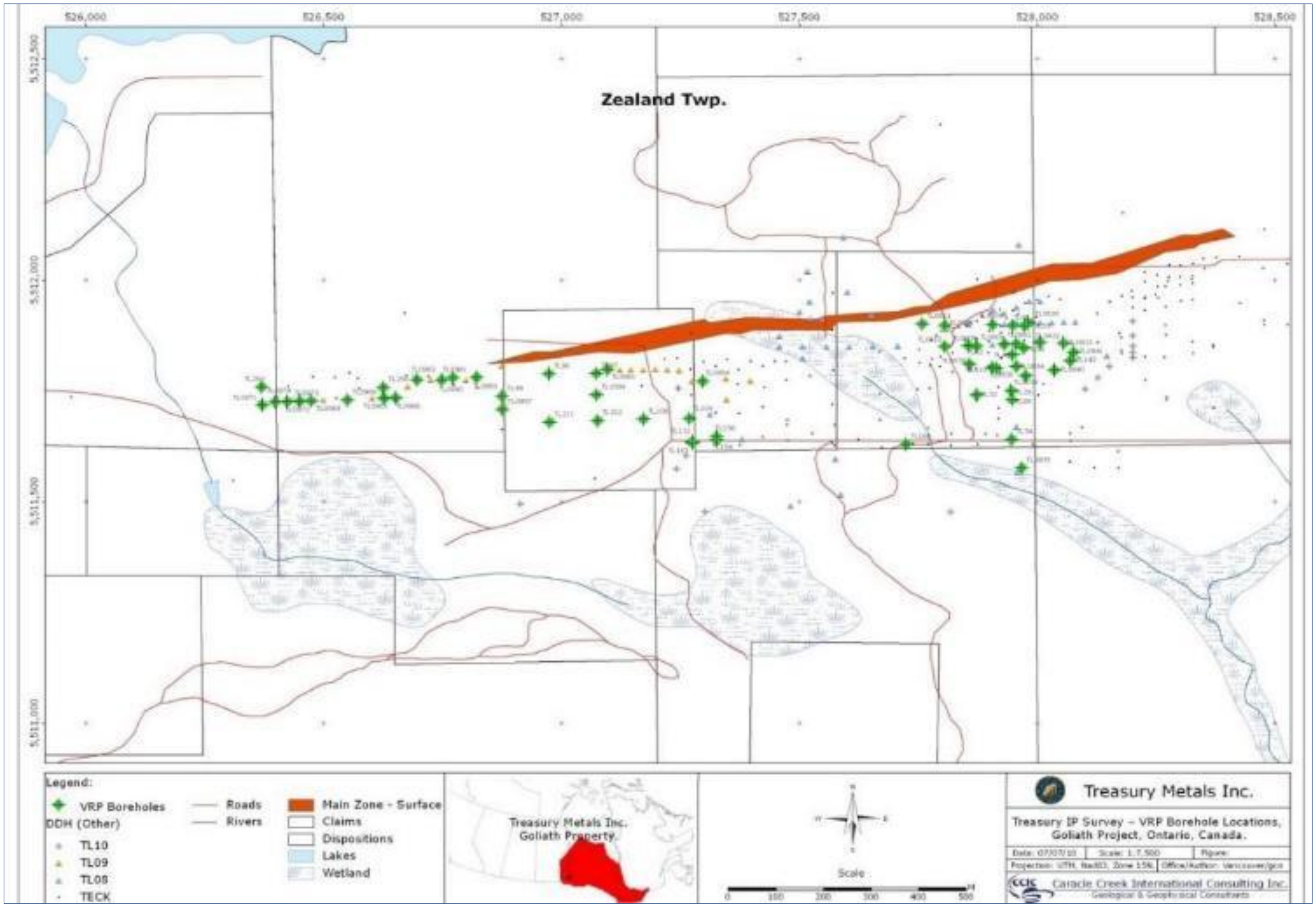
9.1.3 2010 Exploration Activities

9.1.3.1 Ground Geophysical Surveys

A downhole direct current induced polarization (DCIP/resistivity) survey was completed by CCIC over a 24-day period in the spring of 2010 (Palich, 2010). The survey consisted of 60 holes profiled for vertical resistivity/chargeability and 94 hole-to-hole tomography images between holes up to 150 m separation (see Figure 9-4). Four surface lines with 21 surface-to hole tomography pairings were also completed. The survey was designed to:

- characterize the resistivity/chargeability signatures of rock types and ore zones.
- determine if zones containing significant concentrations of gold can be isolated with distinct geophysical signatures.
- test if a new CCIC IP/resistivity technology called EarthProbeTM was capable of imaging between drillholes.

Figure 9-4: Vertical Resistivity Probe & Tomography Drillhole Locations



Source: Treasury Metals (2010)

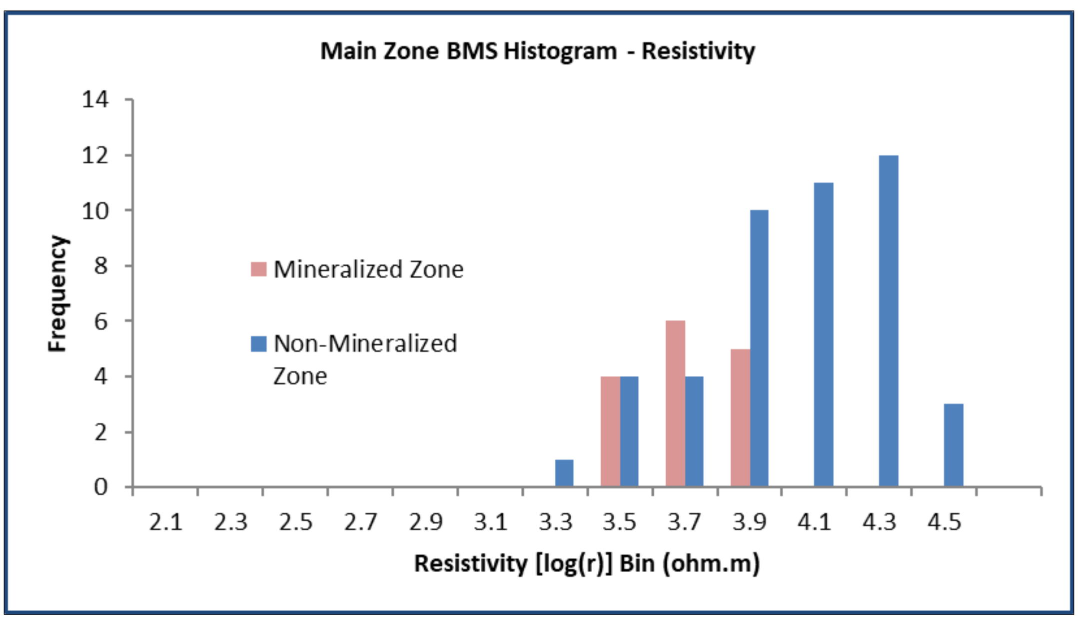
The EarthProbeTM survey method utilizes closely spaced electrode at 5 m separation distances to a centralized data acquisition system that enables arbitrary selection of current and potential electrodes through relays (Roy and Trinder, 2011). Rapid data acquisition and signal processing techniques allow for efficient use of conventional and non-conventional arrays and the removal of natural and cultural noise. The result is a high resolution DCIP system able to delineate both large resistivity/chargeability anomalies and narrow structural features down to depths of approximately 240 m (Roy et al., 2012).

9.1.3.2 Resistivity/Chargeability Corrections

CCIC identified seven distinct resistivity/chargeability correlations from the DCIP survey (Palich, 2010), as follows:

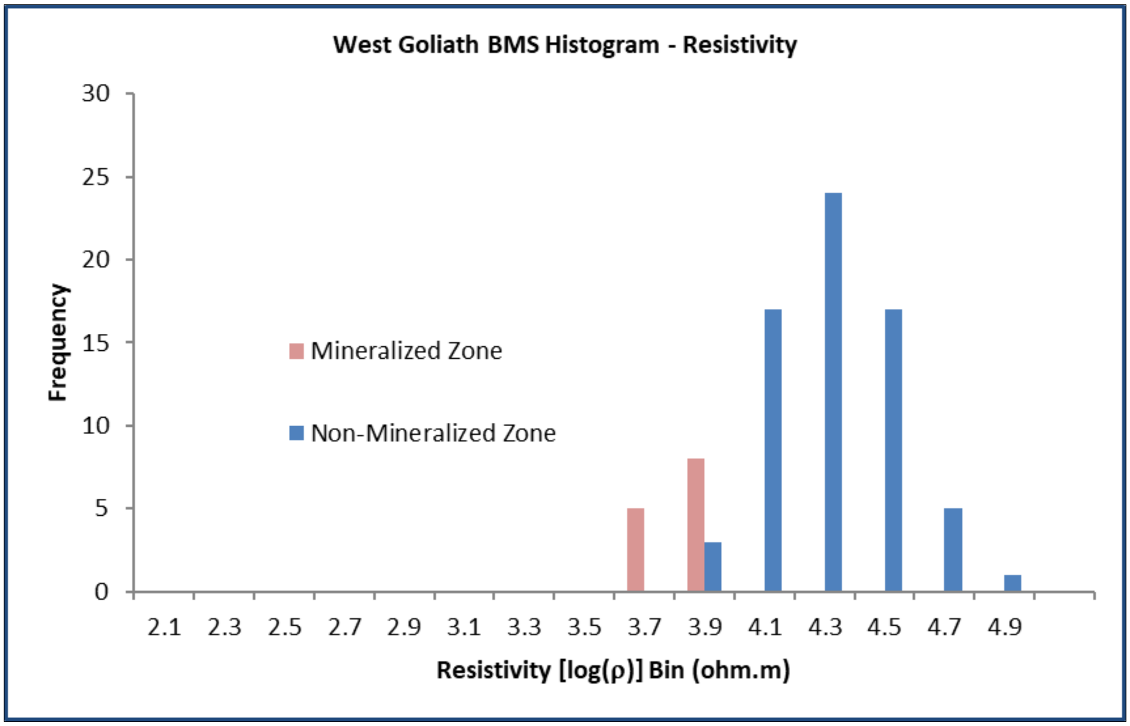
- mineralized zones exhibit low resistivity and high chargeability
- different DCIP signatures between Main Zone and West Goliath extensional area
- resistivity responses greater than 7,900 Ωm (3.9 log Ωm) reflect non-mineralized zones (see Figures 9.5 and 9.6)
- resistivity responses less than 5,000 Ωm (3.7 log Ωm) reflect mineralized zones (Figure 9-5 and Figure 9-6)
- chargeability responses less than 30 mV/V in the Main Zone and less than 50 mV/V in the West Goliath extensional area reflect non-mineralized zones
- chargeability responses greater than 50 mV/V reflect mineralized zones
- there is overlap of resistivity and chargeability response between the mineralized and non-mineralized zones in the Main Zone, suggesting that the occurrence of gold may be controlled by multiple factors (e.g., several alteration types) each having a unique IP signature.

Figure 9-5: Mineralized vs. Non-Mineralized Resistivity Response, Main Zone



Source: Treasury Metals (2015)

Figure 9-6: Mineralized vs. Non-Mineralized Resistivity Response, West Goliath Extension



Source: Treasury Metals (2015)

9.1.3.3 Mineralization Response Signature

CCIC characterized the following three mineralization responses from the survey (Palich, 2010):

- Anomalous resistivity responses occur in association with mineralized zones that are greater than 4.0 m thick and exhibit a gold grade greater than 2 ppm.
- An anomalous resistivity response does not occur if the thickness of the mineralized zone is less than 2.0 m unless the intersection is close (less than 5.0 m) to a thicker mineralized zone.
- An anomalous resistivity response typically does not occur if the thickness of the mineralized zone is less than 4 m unless the gold grade exceeds 2 ppm and zinc exceeds 2,000 ppm.

9.1.3.4 Anomaly Summary

CCIC summarized the anomaly findings as follows (Palich, 2010):

- Numerous in-hole and off-hole low resistivity responses were identified.
- Main Zone: A high level of electrical continuity existed between known gold intersections suggesting that mineralization is continuous.

- West Goliath extensional exploration area: Vertical resistivity probe and tomography results were well correlated with known mineralization zones showing limited additional extent from previously drilled intersections. A shallow conductor (50-70 m) was identified near drillholes TL0965, TL0966, TL0968, TL0969 and TL0972.

- Four low resistivity anomalies were identified from the surface survey. At least one of these anomalies is beyond the western extent of existing drilling.

The DCIP survey was not correlated to the sericite alteration zones. CCIC recommended completing that correlation as well as characterizing the bulk resistivity/chargeability using the entire vertical resistivity probe and drillhole assay dataset (Palich, 2010). They also recommended compiling the special resolution of the resistivity responses into a format that could be overlain with the existing 3D model of the deposit and drilling four IP anomalies identified in the West Goliath extensional exploration area.

9.1.3.5 SCIP Core Testing

CCIC collected 79 sample core induced polarization (SCIP) readings on limited intervals of mineralized core form three 2008 drillholes in early August 2010 (Palich, 2010b). They also compared the results of the 2010 EarthProbeTM IP survey to the 2008 JVX traditional IP survey. The results of this work are summarized below. SCIP core test readings were collected using a GDD SCIP Rx 8-32 unit as follows:

- Hole TL0802: 38 reading were taken of mineralized BMS between 121.1 and 128.9 m

- Hole TL0803: 26 readings were collected in mineralized MSS between 62.0 to 70.2 m

- Hole TL0836A: 15 readings were taken from mineralized MSS occurring from 165.07 to 168.08 m.

The SCIP could not identify any clear correlations between chargeability and resistivity with gold mineralization or gold assays observed in these drill cores. However, both resistivity and chargeability values within the mineralized zones were consistent with the bulk resistivity and chargeability values obtained in the mineralized zones during the EarthProbeTM drillhole surveys.

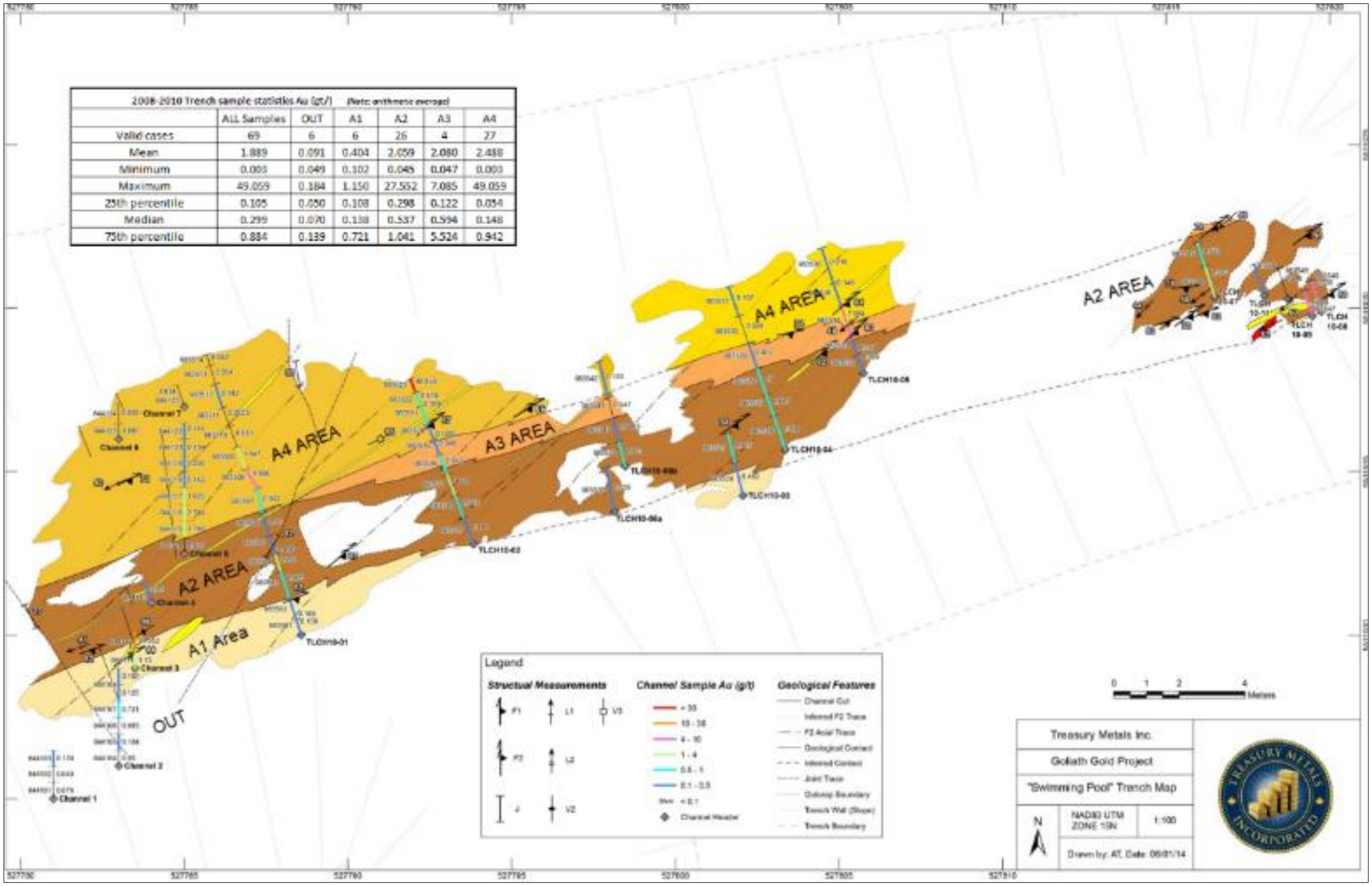
Although the vertical depth of penetration for the EarthProbeTM survey is deeper (250 m), compared to the JVX survey, which could only reach a vertical depth of around 60 m, CCIC was not able to define any new ground geophysical anomalies that were not already identified by the 2008 JVX IP survey.

9.1.3.6 Trenching Program

The 2008 trench was extended by CCIC to expose mineralized bedrock of the Main Zone for an approximate strike length of 42 m in the summer of 2010. This trench exposes the central shoot of the Main Zone and is located around drill section 527800E. It was geologically and structurally mapped at a scale of 1:100 and then systematically channel sampled (see Figure 9-7). Table 9-5 summarizes the structures mapped in the Main Zone trench.

Overall, CCIC concluded that the best potential for the highest gold concentrations is likely to occur near the F_1-F_2 intersections and in areas where there is an increased intensity of F_2 structures in the formation of high-grade shoots. It was also noted that concentrations of sulphide minerals also increased where F_2 fold hinges cut the Main Zone. They also recommended that future drilling programs should be focused along these westward plunging shoots.

Figure 9-7: Geology & Structural Map of the Main Zone Trench with Gold Channel Sample Assay Results (2008-2010)



Source: Treasury Metals (2014)

Table 9-5: Summary of Structures Mapped in the 2010 Main Zone Trench

Event	Structure	Description	Veins	Description
D_0	S_0	Compositional layering of meta-volcanic and meta-sedimentary rocks; argillic alteration zones (?)	V_0	White to grey, highly deformed, S_1 foliation parallel very fine-grained quartz-sulphide ribbons and silicification with narrow sericite lamellae
D_1	F_1	Isoclinal folding	V_1	White coarse-grained deformed, foliation parallel distended quartz lenses (rare)
	S_1	F_1 axial planar and layer parallel foliation/schistosity ~073/80°		
	L_1	Stretching lineation, axis to isoclinal fold hinges; trend ~248 °, plunge 52°		
D_2	F_2	Closed (interlimb angle 60 °) folds; axial planes ~052/83°; discrete, 20 cm to 1.5 m spacing	V_2	Weakly deformed white quartz+/-sulphide lenses along F_2 axial planes.
	L_2	F_2 fold axes trend 228° and plunge 49°		
D_3	J (?)	Brittle joints oriented ~162/81° and 032/82°; possibly related to NW Fault	V_3	White un-deformed, planar crosscutting quartz-tourmaline+/-sulphide veins near vertical WSW striking.

Source: Wetherup (2010)

A total of 47 channel samples plus two duplicates was collected for the trench covering all four geological units (see Table 9-6). Six of the samples collected assayed in excess of 3.0 g/t Au. Table 9-6 lists the samples assaying above 1.0 g/t Au.

Table 9-6: 2010 Channel Samples Greater Than 1.0 g/t Au

Channel	Sample Number	Length (m)	Unit	Au (uncapped) (g/t)	Ag (g/t)
TLCH10-02	983523	0.55	4	49.059	
TLCH10-01	983508	0.75	4	6.686	
TLCH10-05	644115	0.50	4	1.748	3.70
TLCH10-01	983509	0.85	4	1.647	
TLCH10-05	983534	1.00	3	7.084	217.14
TLCH10-03	644112	0.65	2	27.552	2.19
TLCH10-08	983546	1.00	2	5.556	
TLCH10-09	983547	0.60	2	4.989	133.43
TLCH10-01	983504	0.50	2	2.281	
TLCH10-07	983544	0.90	2	1.373	
TLCH10-02	983517	0.65	2	1.117	

Source: Treasury Metals (2015)

Overall, samples from Unit 1 (three samples taken) were generally low with the highest of 1.15 g/t Au over a channel sample length of 0.5 m (sample 644111). Unit 2 (22 samples), which contained the most sulphide mineralization, returned three high-grade samples of 27.55 g/t Au over a sample length of 0.65 m, 5.56 g/t Au over 1.0 m and 4.99 g/t Au over 0.60 m. The latter sample also returned 133.43 g/t Ag over the 0.6 m channel length. A metallic screen fire assay of sample 644112 returned 12.98 g/t Au. Unit 3, with a total of five samples, averaged 2.11 g/t Au with a high of 7.08 g/t Au and 217.14 g/t Ag over a sample length of 1.0 m (sample 983534). Seventeen samples were collected from Unit 4 and averaged 2.99 g/t and returned the highest gold assay grade of the program of 49.06 g/t over a sample length of 0.55 m hosted in the MSS rocks.

Overall, the 69 trench samples from the 2008 and 2010 work program returned an average of 1.889 g/t Au with a median grade of 0.299 g/t Au. High-grade assays in excess of 3.0 g/t Au are sporadic and do not form a continuous zone at that location.

9.1.3.7 Petrographic & Scanning Electron Microscope Study

Two polished sections of two samples collected from diamond drillhole TL0814 for petrographic examination (Beakhouse, 2010). The samples were analysed by Gary Beakhouse of the Ministry of Northern Development of Mines (MNDM) under plan polarized, cross-polarized and reflected light as well as on the OGL scanning electron microscope (SEM). The following observations were reported:

- Minor amounts of gold were present in both thin sections; small grains infilling pyrite in association with galena, between sphalerite grains or between larger pyrite crystals.
- It was unclear if the gold occurred in the sulphides or whether the association observed is representative and accounts for the high gold assay results (38.63 g/t Au and 44.62 g/t Ag).
- Gold is spatially associated with galena and sphalerite and appears to be paragenetically late.
- Galena and sphalerite exhibit a paragenetically late timing relative to other sulphides occurring as overgrowths around, and veins within, pyrite and minor amounts of arsenopyrite.
- The timing relationship of chalcopyrite is unclear.
- Silicate mineralogy consists of quartz, feldspar, white mica, and calcium alumosilicate (stilpnomelane?).

Mineralogical observations are supported by 14 photomicrographs identifying the various mineral phases and relationships.

9.1.4 2011 Exploration Activities

A DIGHEM electromagnetic and magnetic helicopter supported airborne geophysical was carried out for Treasury Metals over the Goliath property between July 14 and July 16, 2011 (Fugro Airborne Surveys, 2011). A total of 531.46 line-km of traverse lines (oriented north-south) were flown with a spacing of 100 m and 54.16 km of tie lines with a spacing of 1,000 m for a total of 585.6 km for the complete survey.

Fugro created the following set of maps: (1) horizontal gradient enhanced total magnetic intensity; (2) calculated vertical magnetic gradient; (3) apparent resistivity (56,000 Hz); (4) apparent resistivity (7,200 MHz); and (5) DIGHEM EM anomaly maps. All final maps were created at a scale of 1:20,000 with the Universal Transverse Mercator (UTM Zone 15N) coordinate system, NAD83 Datum. The results of the Fugro airborne survey are summarized below from the technical report by Roy et al. (2012):

- Magnetic calculated vertical gradient (CVG) and horizontal gradient enhanced total magnetic intensity maps clearly define geological rock contacts throughout the property.

- An iron formation with high magnetic responses (BIF) is defined in the western part of the property.
- The Thunder Lake Assemblage of meta-volcanic and meta-sedimentary rocks also show strong magnetic intensity in the southern parts of the property.
- A combination of magnetic and resistivity parameters has outlined a few interesting magnetic lows that coincide with resistivity highs that might reflect alteration zones or siliceous caps warranting further investigation.
- Deep conductive units are potentially capped by superficial resistive units.
- Several low resistivity zones where values are less than 100 ohm-m likely represent conductive clays or graphitic shales which some of the more discrete responses might be caused by conductive sulphide content or clay-altered shears.
- The survey identified 987 EM anomalies with nearly 69% of those linked to conductive overburden or metasedimentary rocks, about 7.5% are due to cultural sources and approximately 23.5% are due to possible or probably bedrock sources.

9.1.5 2012 Exploration Activities

9.1.5.1 Goliath 3D Inversion Study of Aeromagnetic Survey Data

In 2012, a 3D inversion modelling study was completed by Ellis (2012) using the Fugro airborne magnetic survey data (assuming they used the Fugro dataset). This study was initiated to (1) attempt to identify the aeromagnetic signature of the Goliath deposit, (2) determine the possible explanation for the apparent termination of the zone east of the main deposit and (3) define possible easterly extensions of the gold-bearing zone again east and northeast of the deposit.

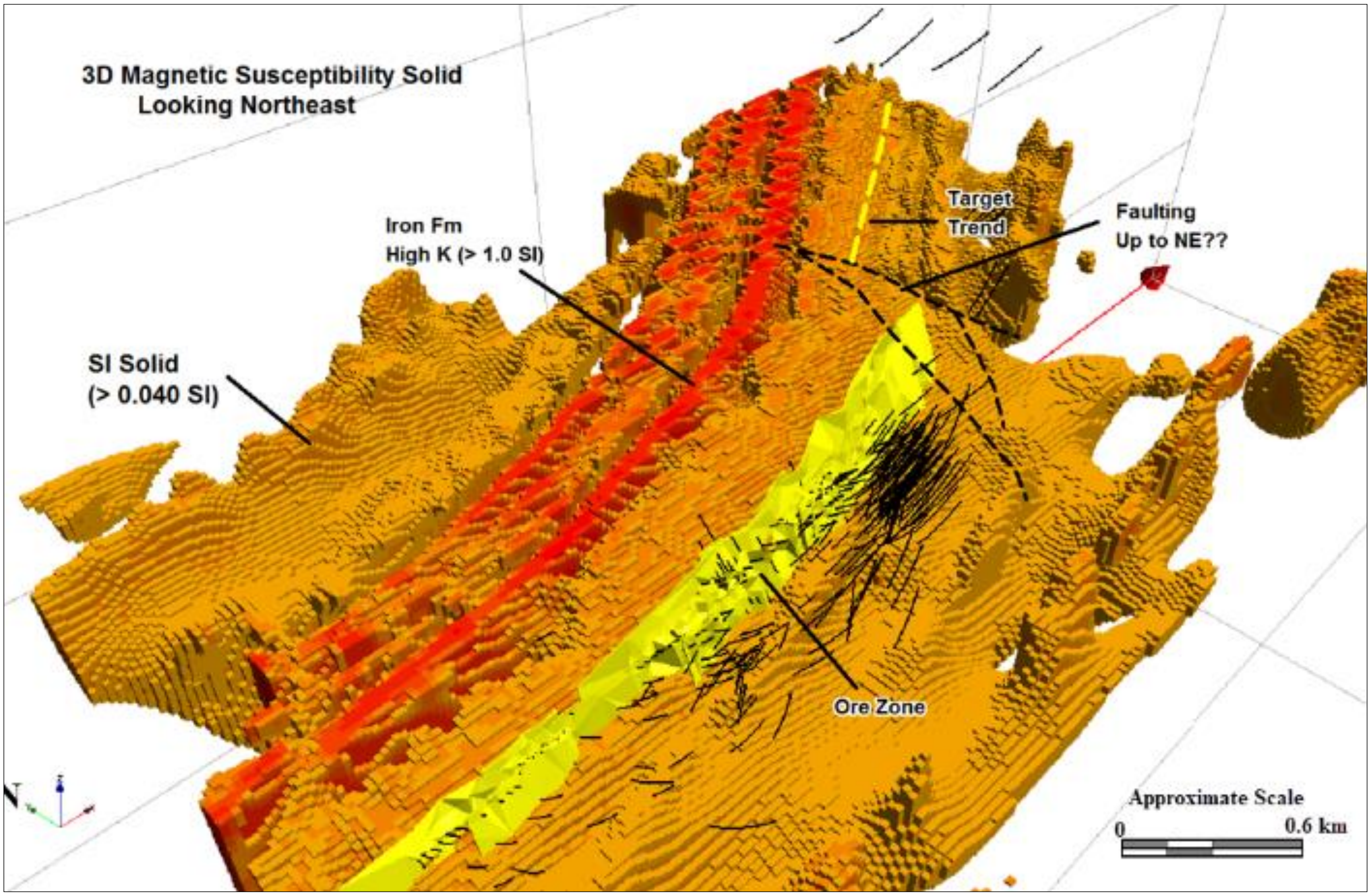
The 3D inversion modelling of the aeromagnetic data generates a solid of magnetic susceptibility that will fit the raw magnetic data within a predefined error tolerance. A series of 3D susceptibility solid maps at 350 m elevation, including cross-sections of the model, were prepared to compare with both known zones of gold mineralization at Goliath and local geology.

The models clearly define a north-trending normal fault that displays left lateral motion disrupting the main deposit in the east and shifting the main zone north of its present location (Figure 9-8). The red areas on the map in the north represent the iron formation. Ellis (2012) also had the following additional observations:

- There is a bend in the iron formation to the north that is consistent with shifting of the target trend of mineralization to the north by the fault.
- The stratigraphy hosting the gold mineralization is not always concordant with mineralization (structurally controlled).

The 3D inversion modelling was able to demonstrate where the gold mineralized zone resumes east of the fault for future drill targeting of the Main Zone (eastern alteration corridor, east of the fault).

Figure 9-8: 3D Magnetic Susceptibility Solid Map, 350 m Elevation



Source: Treasury Metals (2019).

9.1.5.2 Thin Section Study of Mineralized Drill Core

Eighteen samples collected from nine diamond drillholes were submitted to Vancouver Petrographics Ltd. in Langley, BC, in 2012 for petrographic thin section work (Table 9-7). An examination of these samples concluded that:

- Seven samples likely represented either felsic to intermediate meta-volcanic rocks (samples TLTS-3 to 7, TLTS-10 and TLTS-12).
- Seven samples represented exhalative rocks containing massive or semi-massive sulphides with some local significant occurrences of visible gold (samples TLTS-11, TLTS-14 to 18, and TLTS-13).
- Two samples likely represented mafic meta-volcanic rocks (samples TLTS-2, TLTS-9).
- One possibly a meta-microdiorite rock (sample TLTS-8).
- One was a possible anhydrite-quartz-amphibole-green biotite vein hosted in felsic to intermediate meta-volcanic rock (sample TLTS-1).

Detailed petrographic descriptions and photomicrographs were included with the final report.

Table 9-7: 2012 Petrographic Study Results

Sample Number	Drillhole	Depth (m)	Comments
TLTS-1	TL11229	224.75	F_l/chlorite veining
TLTS-2	TL11229	148.60	Orange porphyroblasts and cordierite (?)
TLTS-3	TL11223	527.00	Green silicate band with silicification and some sulphide mineralization
TLTS-4	TL11229	234.42	MSS (east), northeast exploration area, no mineralization
TLTS-5	TL11135	321.20	MSS (west), silicified, no mineralization
TLTS-6	TL11222	358.00	BMS (east), northeast exploration area
TLTS-7	TL11209A	129.15	BMS (west) from western zone
TLTS-8	TL11222	363.15	Massive, less foliated BMS with quartz eyes
TLTS-9	TL11187	179.95	Mafic dyke
TLTS-10	TL11209A	129.80	F2 fold
TLTS-11	TL11148	55.35	Massive fuchsite/chlorite with black tourmaline/amphibole
TLTS-12	TL11193	377.10	Mineralized zone with coarse pyrite, chalcopyrite, sphalerite, and garnet; sample no.1076645 (0.25 g/t)
TLTS-13	TL11121	266.70	Semi-massive sulphide band; sample #981132 (19.63 g/t)
TLTS-14	TL11121	268.15	Deformed quartz veins (no VG); Scattered sulphides, no VG, sample #981135 (Trace)
TLTS-15	TL11122	270.70	Low grade (1-2 g/t); sample #981248 (1.24 g/t)
TLTS-16	TL11152	239.20	Medium to high grade; stringers adjacent to quartz veins, sample #1007597 (18.6 g/t)
TLTS-17	TL11135	325.95	Medium to high grade; edge of semi-massive sulphide band, increased Pb, sample #983067 (10.3 g/t)
TLTS-18	TL11130	341.30	Deformed & boudinage quartz veins (with VG); several VG flecks with quartz, sample#981797 (89.2 g/t)

9.1.6 2014 Exploration Activities

A mobile metal ion (MMI) soil sampling program was conducted on selected target area throughout the Goliath project area from July to October 2014. A total of 1,850 samples were collected during this period by two Treasury Metals field sampling teams. Target grids were located over numerous areas targeting airborne EM, magnetic, ground IP and geological units of interest including iron formation to the north and the strike extension of the Goliath deposit. All samples were collected following sampling procedures outlined by SGS Minerals Services (SGS, 2013a, 2013b).

An orientation survey identified the optimal sampling depth of 10 to 25 cm below the surface. No grid lines were physically cut, but samples were collected using a GPS at line spacings of 200 m and samples taken at 25 m stations. Additional infill lines at 100 m were added to higher priority target areas after the survey results were made available.

Samples were analysed at SGS and response ratios for gold and multi-elements, including base metals copper, lead, and zinc were calculated by Treasury Metals and the results plotted on a regional plan map of the property (Figure 9-9). Five high-priority targets for ground-truthing and further field investigation were identified from this survey, as follows:

- Anomaly P – Iron formation possibly intercepted by F2 gold-bearing structures northeast of the Goliath deposit; moderate to strong linear Au/Cu/Sb/W and weak Ag and as response ratios (RRs); highest Au RR of 60. This anomaly was drill tested in 2015 by holes TL15401 and TL15402 with no significant gold intersections.

- Anomaly N – Nose of regional fold structure (iron formation and eastern strike extension zone of the Goliath deposit). High magnetic anomaly, moderate to strong Au/Ag/Cu, weak Pb and Zn RRs in close proximity to historical Teck holes that intersected some significant gold mineralization.

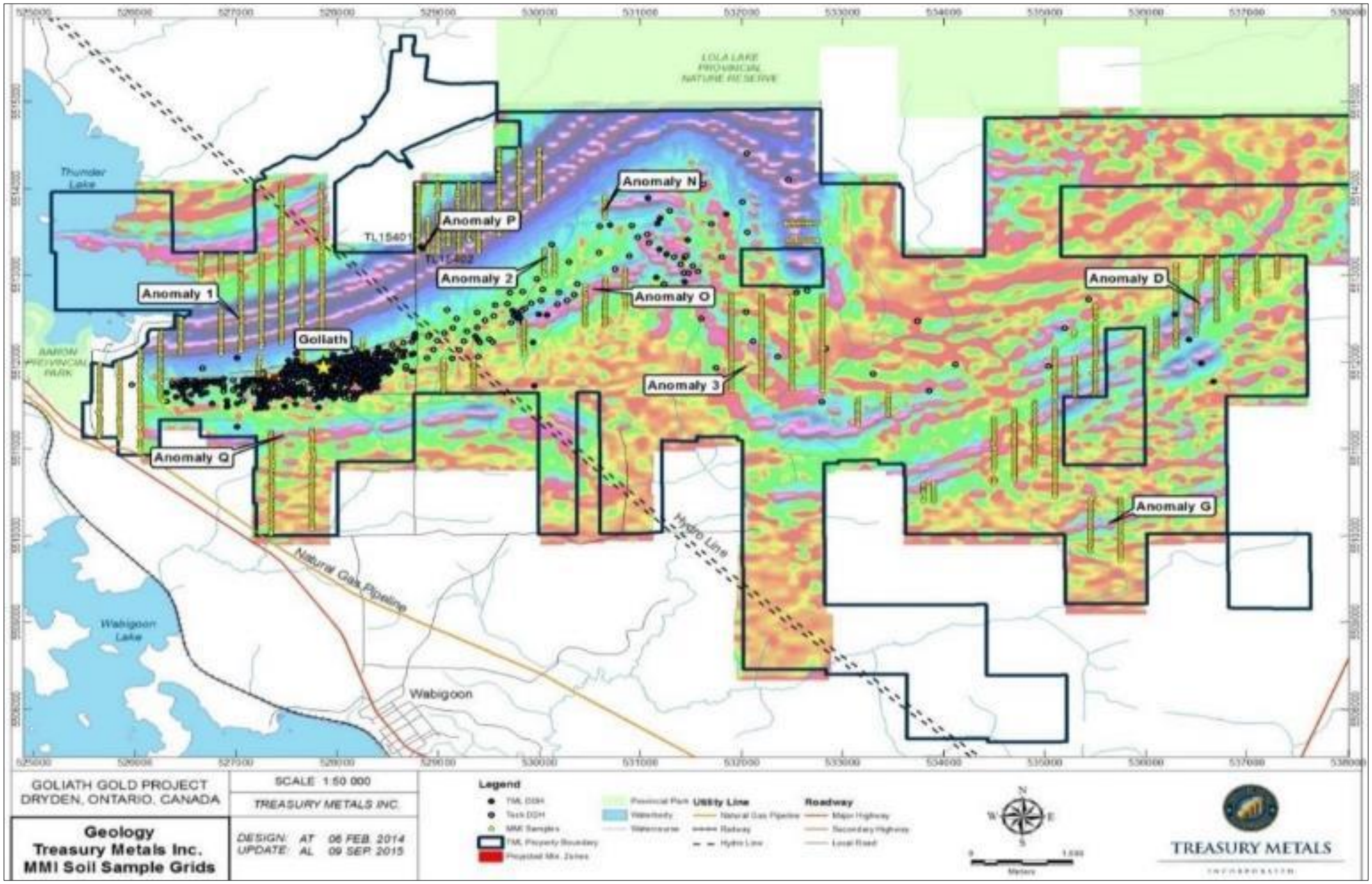
- Anomaly O – Corresponding magnetic and EM linear anomaly, moderate to strong Ag/As/Pb/As and weak Au/Bi/Cu/Sb/W RRs.

- Anomaly G – EM anomaly following a magnetic trend. Moderate to strong Cu/Pb/Zn and weak Sb/W RR.

- Anomaly D – Strong tungsten/zinc, moderate to strong Ag/Cu/Sb, weak As in close proximity to 2012 Treasury Metals drilling fence where one hole intersected 2.0 g/t Au over a core length of 2.0 m (hole TL12266) in a 70 m wide MSS unit located in the far east of the property (represents extreme east extension of the Goliath gold zone).

With the exception of Anomaly P, which was drill tested during the 2015 drilling program, the remaining anomalies need to be investigated in the field to determine if the source of the anomalies can be explained at surface.

Figure 9-9: 2014 MMI Sample Grid Location Map



Source: Treasury Metals (2019)

9.1.7 2015 Exploration Activities

An infill core sampling program was completed at the conclusion of the 2015 diamond drilling program to further evaluate the gold potential of the B Zone and test other zones throughout the deposit known to contain significant gold mineralization but were never previously sampled or assayed (Treasury Metals drill core only). This program covered untested areas of either extensions or potential new zones of previously unsampled drill core focusing on identifying zones that would reside in a potential open pit located from surface to a vertical depth of around 200 m. The boxes containing the target intervals of drill core were retrieved from the core farm located on site, examined, and logged by the geologist, and samples were marked up for splitting. Canadian Standards and blanks were submitted for each hole. Split core samples were then dispatched to Accurassay Laboratories for gold analyses.

A total of 2,090 new split core samples were collected from 95 drillholes. The program was successful in identifying new zones of gold mineralization in half (56) of the 110 new target zones that were identified for inspection. Gold assay intersections in excess of 1.0 g/t are summarized on Table 9-8. A near-surface hole and a newly tested Hanging Wall Zone both reported significant results: hole TL10116 returned 6.08 g/t Au over 6.0 m at a vertical depth of 17 m from surface and TL0853 returned 4.53 g/t Au over a sample length of 5.0 m at a depth of 160 m. Visible gold was observed in some of the drill core. Table 9-8 lists gold assay intersections above 1 g/t Au.

Table 9-8: Gold Assay Intersections above 1 g/t Gold – Infill Sampling Program

Hole Number	Section	From (m)	To (m)	Length (m)	Au (g/t)	Target Zone Description
TL13301	528300	105.00	106.00	1.00	7.15	D Zone, Visible Gold
TL10116	527825	14.00	20.00	6.00	6.08	Hanging Wall 1 (In-Pit)
TL11184	527225	203.00	204.00	1.00	5.77	B2 Zone
TL11210	527700	360.00	361.00	1.00	5.62	B1 Zone
TL0853	527300	177.00	182.00	5.00	4.53	Main Zone (In-Pit)
TL11206A	527225	411.00	412.03	1.03	3.37	Main Zone
TL12278	527600	306.00	307.00	1.00	2.81	B1 Zone
TL0852-12RE	527575	354.00	355.00	1.00	2.58	Main Zone
TL12283	527325	433.00	434.00	1.00	2.52	B1 Zone
TL15385B	527675	375.00	377.00	2.00	2.25	B1 Zone
TL14358	528000	180.00	183.00	3.00	2.29	Main Zone
TL11128	528150	443.00	444.00	1.00	1.90	C Zone

9.1.8 2016 Exploration Activities

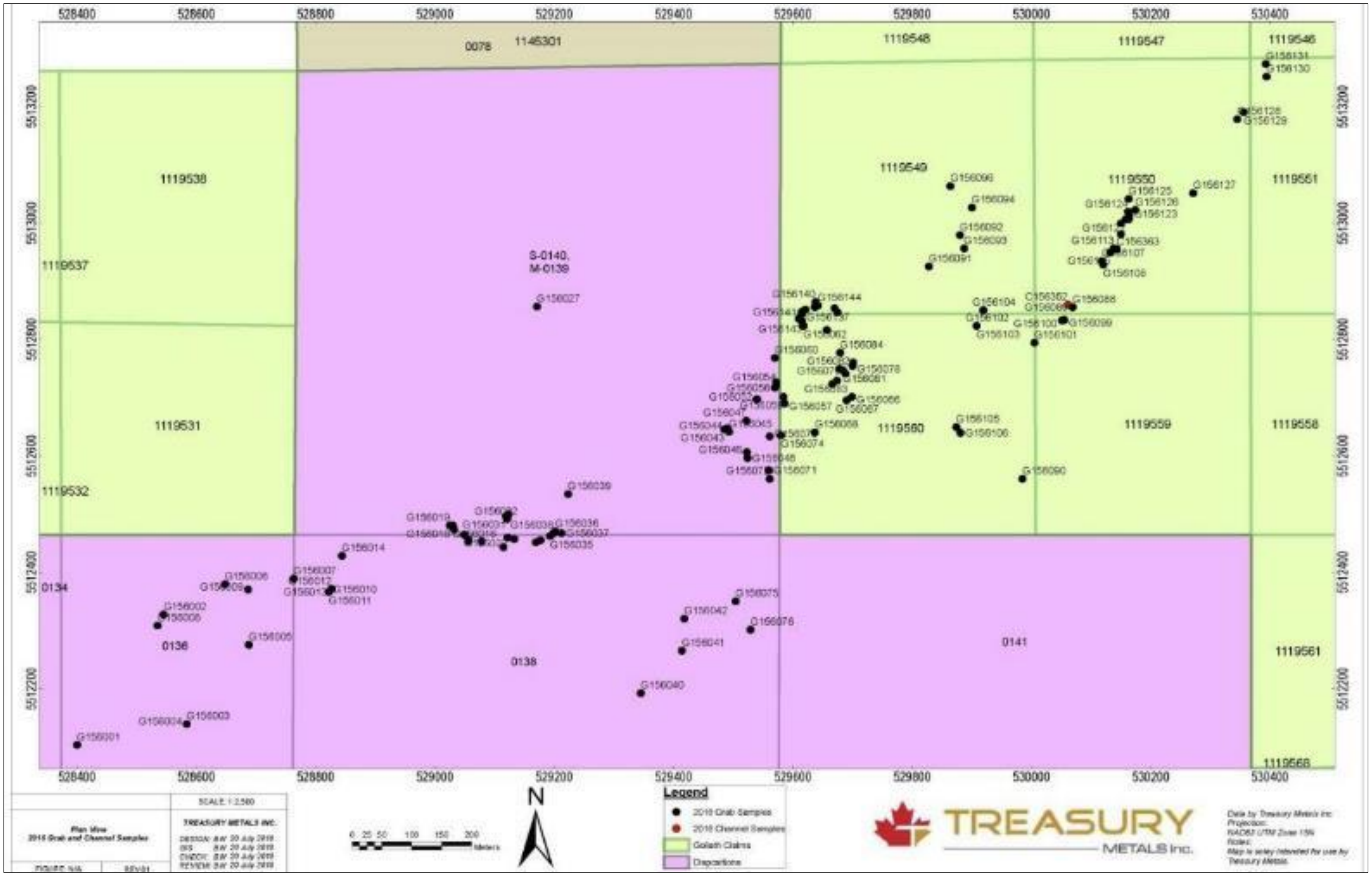
9.1.8.1 Field Mapping & Sampling Program

In 2016, Treasury Metals completed a field mapping and sampling program conducted by geologist Cheyenne Sica. This program consisted of a total of 134 grab samples and 13 channel samples (not including seven coarse blanks and seven standards CDN-CM-26) that were collected and dispatched to ActLabs in Dryden, Ontario, for gold assay and multi-element analysis. A total of 65 samples were taken over three separate patented claims and an additional 69 samples were taken from five unpatented legacy claims (see Figure 9-10). The samples were mainly located along strike of the known resource over approximately 2.4 km and covering an area of approximately 1.4 km^2.

The purpose of this program was to:

- map the terrain and geology of the proposed mine infrastructure sites
- locate and GPS survey historical drill collar locations
- ground-truth the surface mineralization locations of gold chutes interpreted by Exploration Manager Paul Dunbar from historical drillhole compilation and newly prepared longitudinal sections of the eastern alteration corridor (EAC)
- map and sample the eastern strike extension of Goliath deposit Main Zone, C Zone, and parallel zones (D-G) along the EAC
- further investigate, prospect, and sample the Gossan showing
- follow up on MMI anomalies observed from the previous year's MMI sampling program
- identify new exploration drill targets to potentially increase gold ounces outside of the currently defined resource area.

Figure 9-10: 2016 Grab & Channel Sampling Program



Source: Treasury Metals (2019)

9.1.8.2 Proposed Mine Infrastructure Sites Mapping & Sampling

From September 17, 2016, to October 26, 2016 the locations of the proposed mine infrastructure sites were surveyed to explore for outcrops with potential gold mineralization.

Infrastructure to the west of Tree Nursery Road, north of the proposed open pit, is located in old slash from previous logging activities with new growth of small alder trees. Several outcrops of MSS and BMS were mapped and sampled in this area with no detectable gold mineralization.

The location of the tailings pond is dominantly in muskeg lowland. In the northern portion of the proposed tailings pond location, scattered outcrops of iron formation are present with no evidence of alteration, deformation, or mineralization. The southeastern portion of the tailings pond covers the strike extension of the Goliath deposit Main Zone and C Zone extensions. In this area mineralized BMS and MSS rocks were sampled and mapped returning assays from 0.42 g/t Au to 1.42 g/t Au (Table 9-9).

The proposed site of the polishing pond is in a mixture of old slash with new growth alders, muskeg lowland, Jackpine forest high ground with sandy soil, and a swamp surrounding a small creek. Outcrops of mineralized BMS and MSS rocks were mapped in this area returning assays of 0.37 g/t Au to 0.41 g/t Au (Table 9-9).

Figure 9-11 shows the surface terrain and comparison of locations of resurveyed Teck diamond drillholes. The red units denote the surface projection of mineralized gold chutes interpreted by Paul Dunbar using the best intersections from historical Teck and Treasury Metals drillhole data.

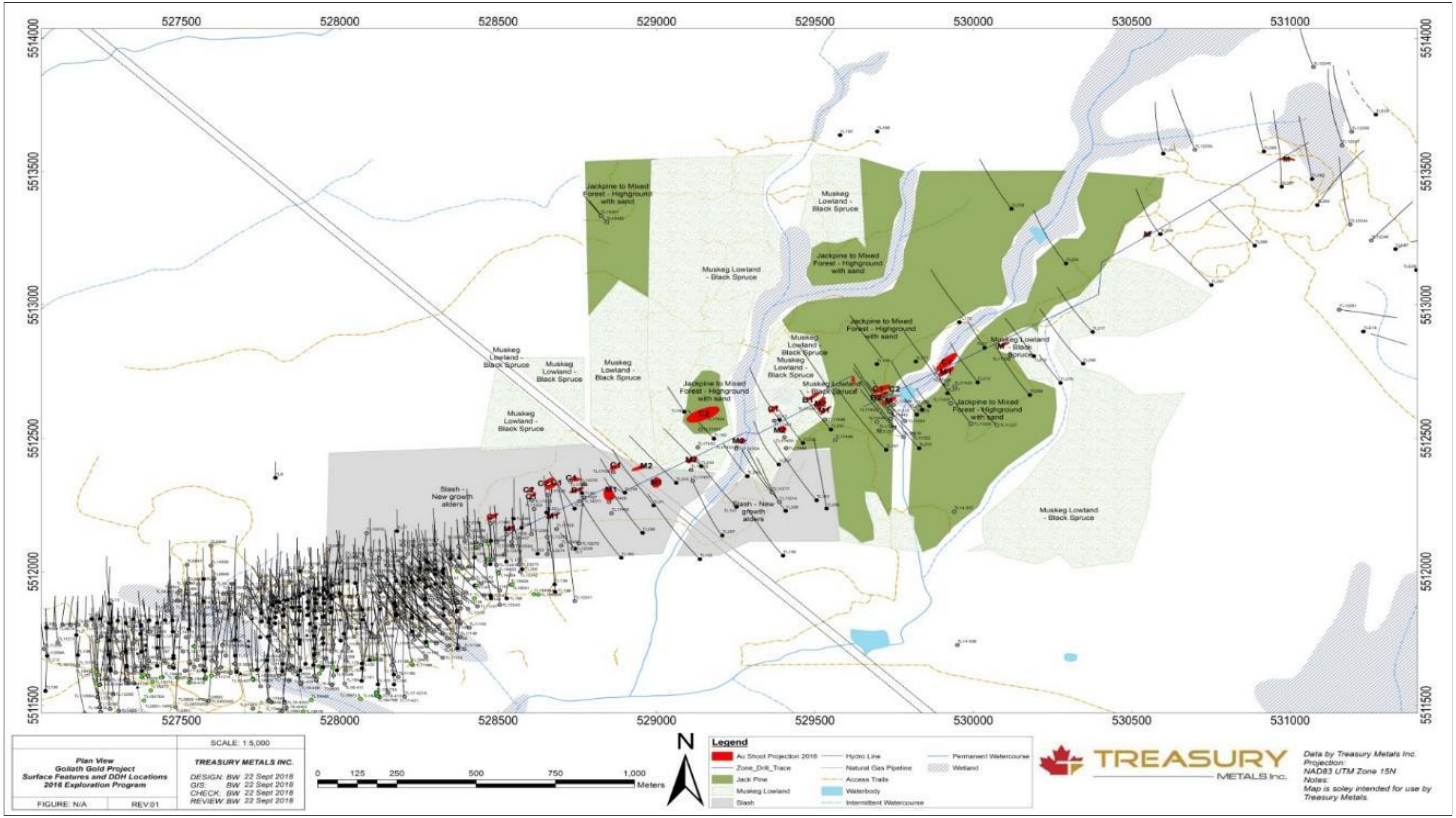
The QP notes that some of the samples from the condemnation mapping program returned grades above the open pit cut-off presented in this study (Table 9-9).

Table 9-9: Significant Assay Results from 2016 Condemnation Field Mapping Program

Sample Number	Easting	Northing	Sample Description	Au (ppm)	Ag (ppm)	Cu (ppm)	Pb (ppm)	Zn (ppm)
G156061	529615	5512847	BMS: Strongly silica flooded with weak to moderate shear fabric; pink colour (pervasive hematite?); some chlorite bands with ≤1-% biotite bands; altered porphyry protolith? Pyrite fine grains – coarse grains throughout (up to 10%)	1.42	1.1	50	2	74
C156361	530143	5512954	BMS: moderate silica flooding; ~5% MSS bands; 5-7% blue quartz eyes; 1% pyrite seams and 1-2% fine grains pyrite disseminations; 2% chlorite bands containing locally up to 5% pyrite ; smoky grey quartz vein with oxidized staining hosting 1% galena + sphalerite seams; strongly sheared around quartz with increased sericite alteration	0.98 (over 0.7 m)	6.8	259	1,760	3,020
C156362	530143	5512954	BMS: moderate silica flooding; 2-3% MSS bands; 2% blue quartz eyes; mm-wide seam of galena; mm-scale massive pyrite seams; 2% very fine grains pyrite mineralization	0.76 (over 1.0 m)	0.9	64	127	267
G156110	530132	5512949	BMS; massive pyrite (~15% of sample); 15% chlorite; 60% glassy dark silica flooding;	0.754	7.5	409	560	1,060
G156058	529584	5512701	BMS: 60% biotite, 5% sericite the rest is silica flooded; moderate foliation; coarse grain pyrite seams up to 5%	0.471	4.6	5	155	175
G156054	529572	5512721	MSS: ~60% sericite + muscovite, ~30% silica; oxidized on foliation planes; trace chalcopyrite and arsenopyrite; up to 4% pyrite along foliation planes as blebs and fine grains dissemination	0.417	2.3	18	20	15
G156031	529119	5512496	BMS: moderate silica flooding; ≤1% black quartz eyes; balk quartz veinlets parallel to foliation (~30% of sample); ≤ 3% fine grains pyrite	0.41	0.6	3	18	85
G156029	529123	5512499	MSS: sericite + muscovite + silica; oxidized staining throughout; up to 3% pyrite as seams along foliation planes	0.373	0.8	8	10	26
G156114	530138	5512956	MSS: strong silica flooded with galena seam and 2% pyrite seams	0.333	2.3	188	775	768

Source: Treasury Metals (2019)

Figure 9-11: 2016 Surface Features & Diamond Drillhole Locations



Source: Treasury Metals (2019)

9.1.8.3 Eastern Alteration Corridor

Geological mapping and sampling of the eastern alteration corridor (EAC) was conducted from September 17, 2016, to October 13, 2016, by geologist Cheyenne Sica. Mapping and sampling targeted the strike extension of Goliath gold-bearing MSS and BMS geological units. A high priority of the 2016 field program was to ground truth and explore for signs of mineralization at the locations where gold chutes were projected to outcrop, utilizing the new longitudinal sections that had been constructed along the entire strike length of the EAC (red dotted line shown on Figure 9-12). Unfortunately, no outcrops were found at the exact locations of the interpreted high-grade chutes.

Grid geology mapping completed by Teck in the 1990s mapped a large package of felsic metavolcanic rocks throughout the central portion of the EAC bound to the north and south by metasedimentary rocks. The surface projection of geology from historical drillhole data reveals a more complicated stratigraphy consisting of quartz-feldspar porphyry, quartz-porphyry, strongly altered BMS and MSS rock units. The BMS and MSS units are on strike with identical rock units that host the high-grade gold mineralization at the Goliath deposit. Pervasive alteration, metamorphism and deformation make it difficult to distinguish these units throughout the EAC as definitive felsic volcaniclastic rocks, such as a silicified felsic tuff, and the MSS and BMS rocks typically have a porphyritic texture and could be interpreted as felsic intrusive porphyry rocks.

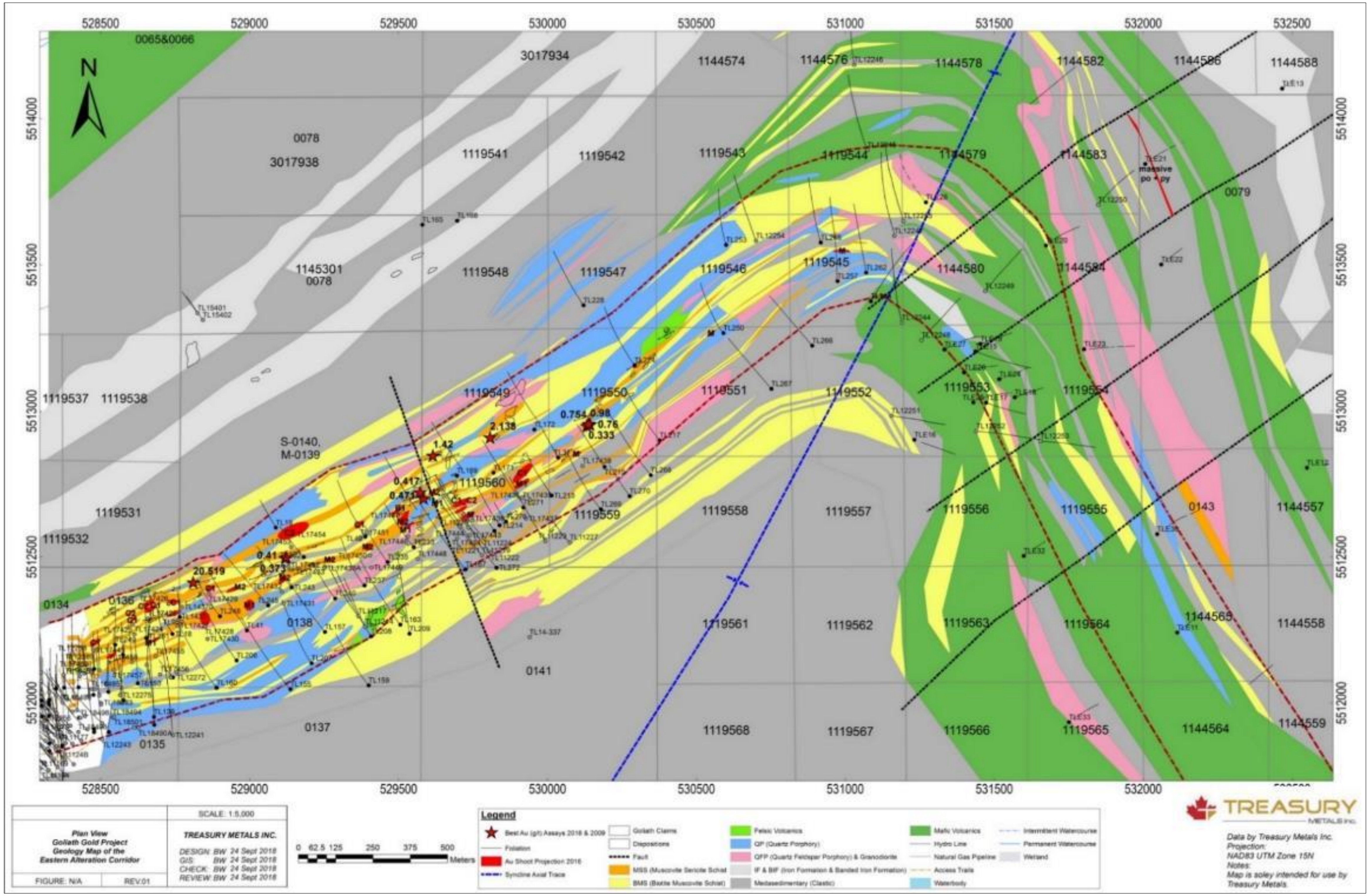
A final geology compilation map has been generated integrating (1) the geological mapping from the 2016 field program, (2) the newly interpreted drill sections utilizing all historic drillholes along the EAC, (3) the newly interpreted longitudinal sections, (4) geology from the old Teck grid mapping programs, and (5) all existing ground and airborne geophysical data.

Of the 134 grab samples collected from the EAC, 110 samples contained anomalous gold returning assays of > 0.005 g/t Au. Thirteen channel samples were taken, all of which returned > 0.10 g/t Au. The dominant sulphide phase observed was pyrite occurring as fine to coarse-grained disseminations with some massive pyrite seams. Galena and sphalerite were also observed as stringers concentrated at contacts between BMS rocks and smoky grey quartz veinlets.

- Grab samples with gold assays of > 0.3 g/t Au have been plotted on the geology map as red stars (Figure 9-12). Each of the showings have been described below starting in the western portion of the map area:
- Two grab samples returning 0.373 g/t Au (G156029) and 0.41 g/t Au (G156031) are located at the merging point of the C and B Zones. These samples contain low concentrations of Ag and were anomalous in Pb and Zn.
- Assays returns of 0.417 g/t Au (G156054) and 0.471 g/t Au (G156058) were obtained from grab samples collected from the B1 and Main Zones just west of the fault. Sample G156058 contained 4.6 g/t Ag and anomalous Pb (155 ppm) and Zn (175 ppm).
- Sample G156061 returned 1.42 g/t Au and 1.10 g/t Ag and was collected north of the best-known mineralized MSS zones.

In all three gold occurrences described above, gold is associated with silicification and pyrite and is hosted by both MSS and BMS rock units.

Figure 9-12: Geology Map of the Eastern Alteration Corridor



Source: Treasury Metals (2019).

Samples C156361, C156361, G156110 and G156114 were collected from the C Zone and are located 2 km along strike from the main Goliath deposit. C Zone grab samples returned 0.333 g/t Au (G156114) and 0.754 g/t Au (G156110). Sample G156110 also contained 7.5 g/t Ag (the highest silver assay of the 2016 program) in association with 409 ppm Cu, 560 ppm Pb and 1,060 ppm Zn. Channel sample C156361 assayed 0.98 g/t Au and 6.8 g/t Ag over a sample length of 0.7 m and also contained elevated base metals (259 ppm Cu), as well as the highest Pb and Zn assays of the program of 1,760 ppm Pb and 3,020 ppm Zn, respectively. BMS rocks host the three highest gold assays. This C Zone showing occurs further east than anticipated from historical drill best assay intersections suggesting the potential for additional chutes to occur east along strike.

Upon completion of the field sampling program, several new exploration targets were identified along strike of the main resource situated in the EAC near the nose of a regional fold structure (folded syncline) which is considered a very high priority target area. Anomalous gold assays found within the grab and channel samples warrant an additional soil sampling program (soil gas hydrocarbon) or follow up with exploratory drilling to further test the potential of the select locations

As part of this study, SRK estimated an inferred mineral resource for the western part of the EAC as outlined in Section 14 of this report.

9.1.8.4 Gossan Showing

On November 4, 2016, geologists visited the Gossan Showing exposed by the construction of a new logging road that was initially described and sampled by geologist Adam Larsen in October 2015. The access logging road has since been extended westward 700 m following the strongly gossaned (oxidized) shear zone.

The Gossan Showing coincides with a ~1 km long east-west-trending airborne EM and magnetic geophysical anomaly which also occurs in association with Pb, Zn and Cu MMI anomalies identified during the 2014 soil sampling program. During the field program, the strike extension of the Gossan Showing was traversed and sampled over a strike length of almost one kilometer.

The gossan zone itself is hosted in a moderately to strongly sheared mafic volcanic package with strong chlorite + amphibole alteration and is typically strongly oxidized along foliation planes. The strike of the zone is 260° and it dips north from 80° to 85°. Portions of the showing are weakly silicified. Pyrite was observed as semi-massive bands and fine-grained disseminations concentrated along foliation planes. Pyrrhotite (up to 2%) was also observed along foliation planes.

The intense gossan alteration zone extends for at least 1.0 km in strike length and is contacted by mafic volcanic rocks to the north and south. On certain outcrops, pink felsic dykes with irregular contacts are injected into the southern mafic volcanic rocks. South of the southern mafic volcanic unit is a large body of strongly silicified felsic intrusive rocks (a possible quartz feldspar porphyry unit).

Five samples were taken along strike length of the gossaned unit. None of the samples returned any significant gold values. Samples were found to be anomalous in Cu (up to 244 ppm), Zn (up to 184 ppm), Pb (up to 39 ppm) and Mn (up to 4,130 ppm).

9.1.9 2017 Exploration Activities

9.1.9.1 Iron Formation Mapping and Sampling Program

In 2017, Treasury Metals completed an outcrop mapping and sampling program with focus on the iron formation lithological unit. The program consisted of 36 grab samples including two coarse blanks and two standards. The sampling program occurred over 12 unpatented mining claims and one patented mining claim. The samples were mainly located along the iron formation, as well as from within the nose of the regional fold structure. The samples covered an area of approximately 5 km^2. The purpose of this program was to:

- further investigate and sample the iron formation
- map and sample newly exposed outcrops that had recently been exposed by logging activity in the area of the EAC
- identify new exploration drill targets in the nose of the regional fold to potentially increase gold ounces outside of the currently defined resource.

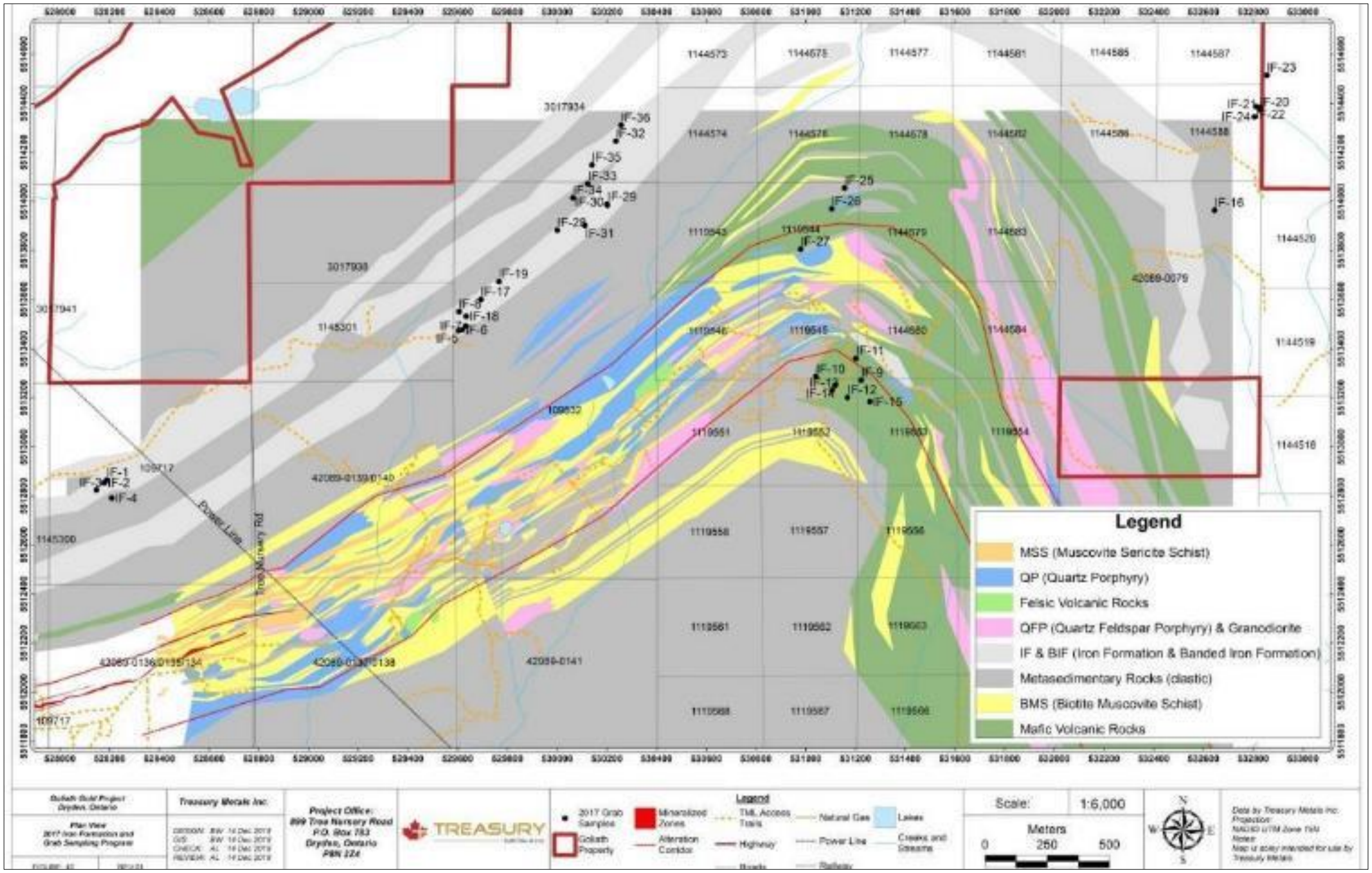
9.1.9.2 Sampling Program

Sampling of the iron formation and the EAC was conducted from August 30, 2017 to November 6, 2017 by geologists Bryan Wolfe and Eldon Phillips. A total of 36 grab samples were collected and dispatched to ActLabs in Dryden, Ontario for a fire assay gold analysis (see Figure 9-13). The areas of interest were accessed using trucks, all-terrain vehicles (ATVs), and on foot.

Mapping and sampling targeted the strike extension of the iron formation along the limbs of the regional fold structure as well as the hinge of the regional fold structure in close proximity to the EAC. Of the 26 grab samples collected from the iron formation only one sample (IF-33, sample number 303986) returned an anomalous gold value of 0.01 g/t Au. The remainder of the iron formation samples returned no significant assay results. Of the 10 grab samples collected from the nose of the regional fold structure, seven samples returned anomalous gold values ranging from 0.01 g/t Au to 0.12 g/t Au. The remaining three samples collected from the nose of the fold returned no significant assay values as summarized below.

Upon completion of the sampling program, it was determined that the program was unsuccessful at identifying any new prospective exploration targets within the iron formation. The hinge of the regional fold structure covers a large area (approximately 2 km^2) with an abundance of new outcrop exposure and was not extensively sampled at the time of the program. This area still warrants further investigation as the program was modest in size and previously identified anomalies in the 2014 MMI sampling study require follow up. Although no substantial gold assays were returned, the nose of the regional fold and the EAC still remain high priority targets with the potential to add additional gold ounces along strike of the main resource.

Figure 9-13: 2017 Iron Formation & Grab Sampling Program



Source: Treasury Metals (2019)

9.1.9.3 Mapping of the Iron Formation & Eastern Alteration Corridor

In addition to sampling, a brief geological mapping program of the iron formation and the EAC was conducted from August 30, 2017 to November 6, 2017 by geologists Bryan Wolfe and Eldon Phillips to further add to the current outcrop database.

The purpose of this program was focused on mapping the extent and continuity of the iron formation as well as mapping newly exposed outcrops due to recent logging activity in the nose of the regional fold structure situated within the EAC. Only a small amount of the exposed outcrops was mapped in detail in this program and an extensive mapping program should be undertaken to further explore the continuity of the lithologies and the structural elements that constrain them.

9.1.9.4 Eastern Alteration Corridor

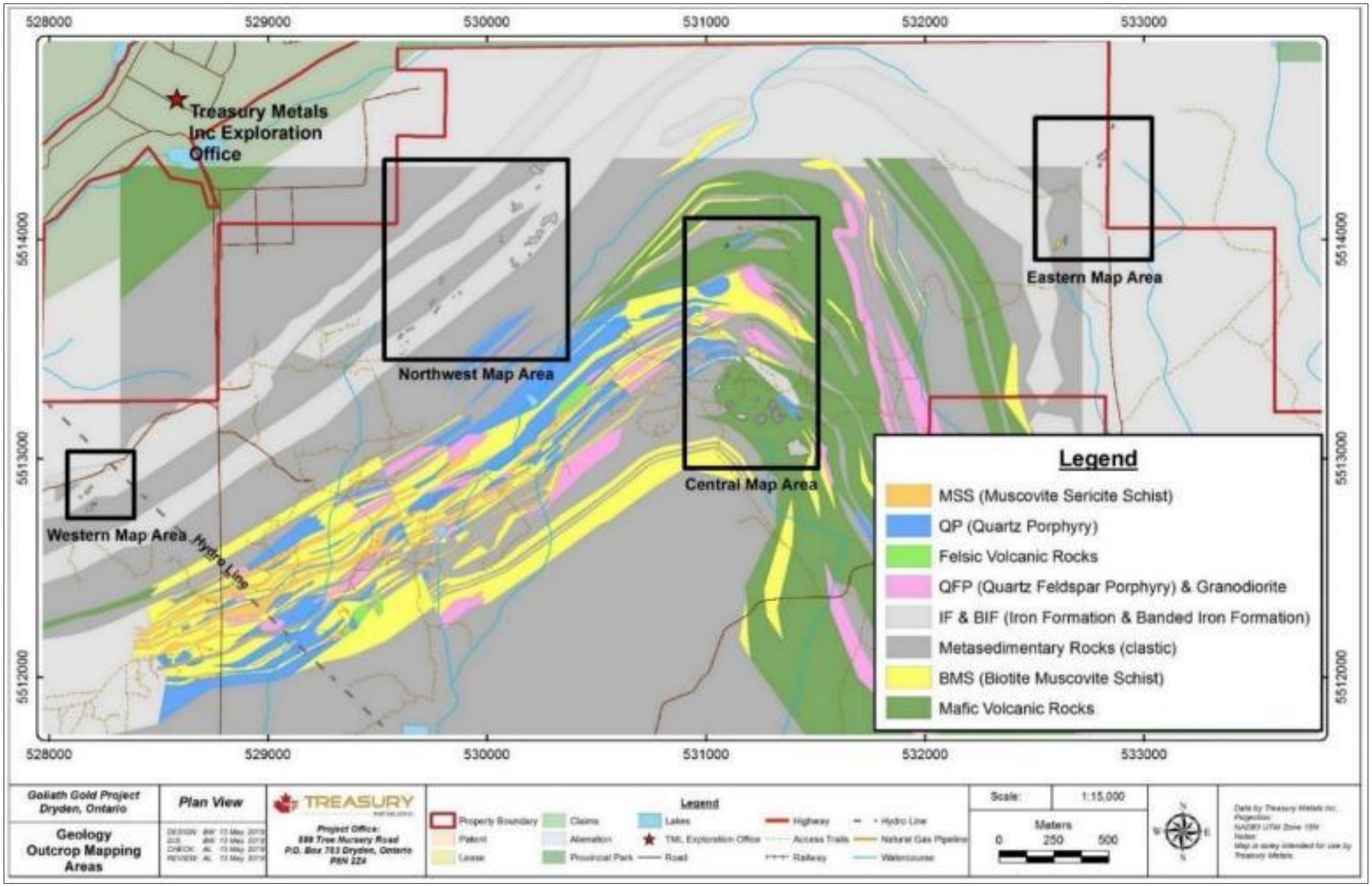
Recent timber logging of the Goliath property has exposed an extensive amount of new outcrop showings at the nose of the fold within the EAC. The area is primarily clear-cut and can easily be navigated through use of an ATV. In this area several new outcrops were observed and mapped by using a Trimble geo-explorer 600 series handheld GPS unit and traversing the circumference of the surface feature. Within the nose of the fold, in close proximity to the EAC, Treasury Metals personnel located and identified five new outcrops of BIF, six new MSED outcrops, and one outcrop of BMS. Towards the end of the mapping program the geologists were successful in locating one new outcrop of MSS. Since the MSS is the primary gold-bearing lithology within the Goliath deposit, further mapping of the area will be required to establish where the mineralized zones may be projected to the surface.

9.1.9.5 Outcrop Mapping Program

The primary purpose of the mapping and sampling program was focused on trying to establish the extents of the iron formation along the northern limb of the regional fold structure, as well as to identify any new prospective exploration targets. Treasury Metals personnel was successful in identifying several new outcrop showings in the southwest and northeast (Figure 9-14).

The iron formation is thought to extend all the way to the most northern tip of the Goliath property but was not easily accessible. To reach the northern tip of the regional fold, the geologists used an ATV on an old drill trail. Once reaching the end of the existing trail, the geologists had to follow a small ridge of outcrops and had to be traversed by foot. On the northern limb of the fold Treasury Metals was able to identify and map 25 new outcrops of BIF, five outcrops of BMS and two new MSED outcrops. Due to the short nature of the program, Treasury Metals personnel were unable to map the northernmost extent of the iron formation in the nose of the fold.

Figure 9-14: 2017 Outcrop Mapping Program



Source: Treasury Metals (2019)

9.1.9.6 Infill Core Sampling Program

From April 7 to June 14, 2017, Treasury Metals initiated a second infill sampling program intended to assay previously drilled but unsampled drill core. The program was designed to cover all mineralized zones while prioritizing intervals within and near the proposed open pit. A total of 5,256 samples were submitted including 525 blanks and standards and covered 142 separate drillholes. The three main objectives for the infill sampling program were to (1) add new gold ounces to be included in the next mineral resource estimate; (2) extend existing gold mineralization; and (3) uncover any potential new zones. Table 9-10 lists significant assay intersections greater than 1.0 g/t Au.

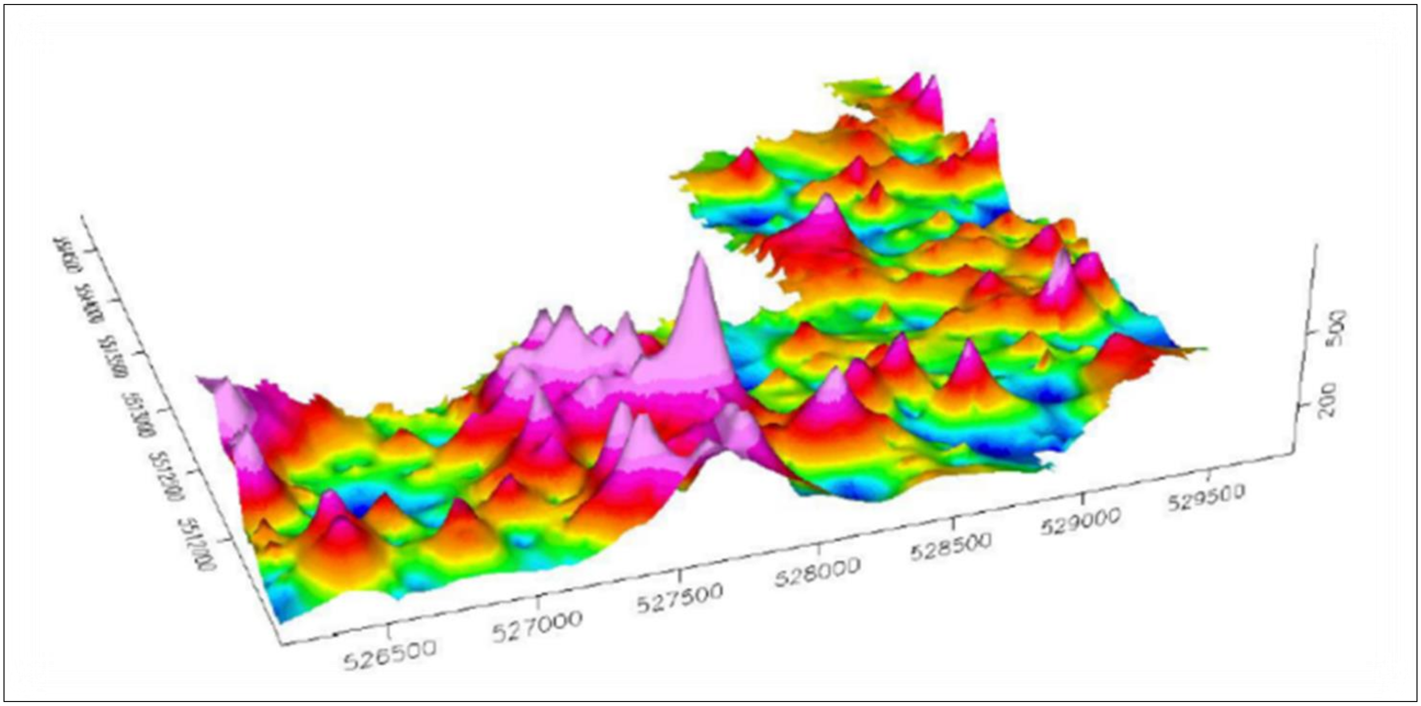
Table 9-10: Significant Assay Intersections Greater than 1.0 g/t Gold

Diamond Drillhole	Section	From (m)	To (m)	Intercept (m)	Au (g/t)	Ag (g/t)	Target
TL0849	527600E	100.00	104.00	3.00	1.61	15.97	E Zone
TL1096	527250E	206.80	211.00	4.20	11.37	P	D Zone
including		208.00	209.30	1.30	34.80	P	
TL10108	527475E	250.00	253.00	3.00	31.38	21.63	HW Zone
including		252.00	253.00	1.00	93.40	64.10	
TL11145	528500E	49.50	52.00	2.50	1.36	9.90	BMS HW
TL11167	527275E	134.30	137.00	2.70	4.52	5.18	
including		134.30	135.00	0.70	15.90	11.70	HW Zone
TL11171	527225E	279.57	284.00	4.43	4.97	1.16	B Zone
including		283.00	284.00	1.00	18.20	0.70	
TL11209A	527075E	43.00	47.00	4.00	8.61	0.99	HW Zone
including		44.00	45.00	1.00	29.80	2.20	
TL12287	527275E	292.00	294.00	2.00	4.12	2.09	HW Zone
including		292.70	294.00	1.30	6.07	2.40	
TL13306	527850E	86.00	90.00	4.00	1.12	1.65	C Zone
TL15387	527550E	143.00	145.00	2.00	3.70	5.38	HW Zone
TL164-12RE	527625E	417.00	419.30	2.25	3.01	N/A	B Zone

9.1.10 2018 Exploration Activities

A soil gas hydrocarbon (SGH) orientation survey was carried out consisting of 845 soil samples. The survey can be described as two grids (defined as "eastern" and "western" grids) with sample spacing of approximately 50 m and approximately 200 m between transects. One was conducted across the Goliath deposit and the other near the regional fold nose northeast of the deposit. The survey identified strong redox and gold pathfinder anomalies (see Figure 9-15) on and around the deposit area believed to be caused by gold mineralization with a high level of confidence (5.5 out of 6 SGH signature rating). With the capability of this surface sampling technique to detect the Goliath deposit, it is recommended to conduct additional sampling across the remaining strike length.

Figure 9-15: 3D View of Western Gold Pathfinder Class Map from 2018 Orientation Survey



Source: Treasury Metals (2019)

9.1.11 2019 Exploration Activities

9.1.11.1 Hole-to-Hole Induced Polarization (IP) Survey

A downhole spectral IP/resistivity survey was completed by Golden Mallard Corporation. Using 15 existing drillholes spanning 1.2 km along strike, this program was designed to outline the chargeability signature of Goliath, to test the high-grade down-dip extension potential below the current resource (400 m below), and to outline new drill targets and detect any previously unknown nearby mineralized concentrations.

The IP survey confirmed the project's gold-bearing zones correlate with high to moderate resistivity and chargeability high and high-low contacts. This is believed to be associated with strong silicification and an increase in disseminated sulphides, both of which are found in the Goliath gold zones. The inversion model suggests that these zones are located on a major structure and has outlined signatures of the high-grade gold shoots including the confirmation of the down-dip extension potential below the current resource to approximately a depth of 800 m below surface. The survey also identified the continuation of the resistivity and chargeability responses on the east and west sides of the resource area, indicating that the zones that host the gold extend along strike of the deposit in both directions.

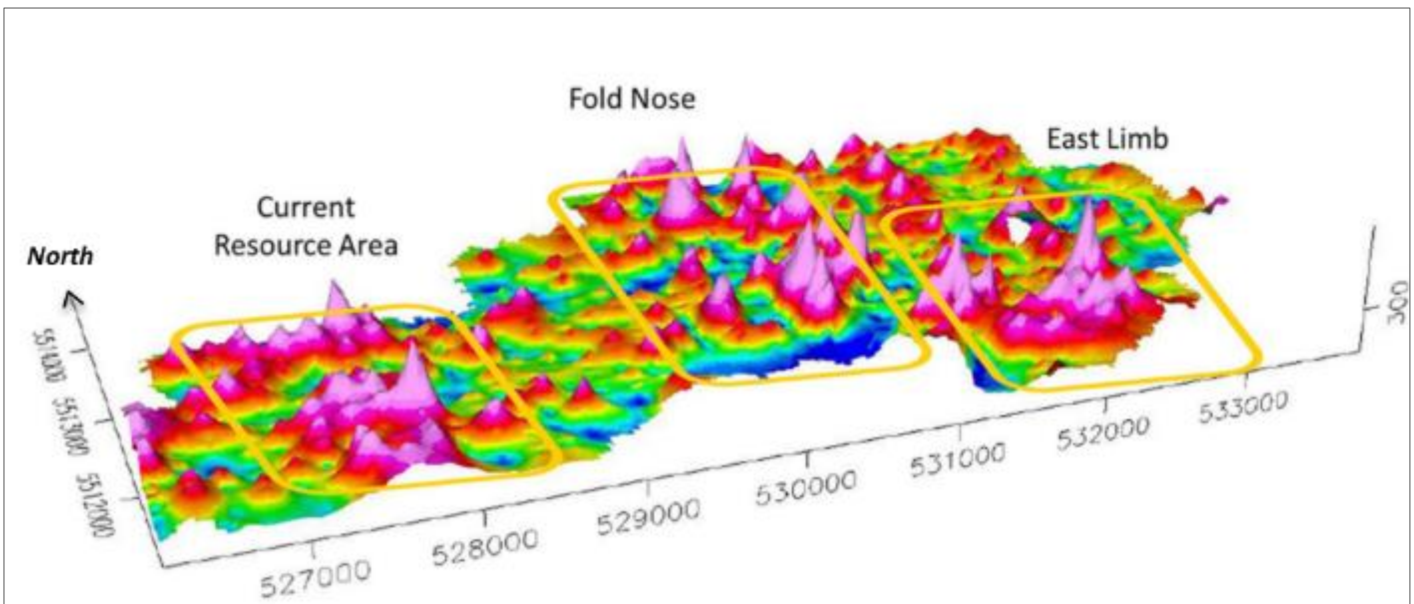
The completed holes have shown positive results and strong correlation to the currently defined resource. The IP results indicate a new valuable use of this technology and will provide Treasury Metals with the ability to define additional high-priority drill targets.

9.1.11.2 Soil Gas Hydrocarbon Sampling Follow-up Program

A follow-up program commenced in 2019 to expand sample coverage along strike length to the east of the Goliath deposit, as well as a number of other areas of interest including highly prospective areas both north of the deposit and on the eastern side of the property. Approximately 1,040 additional samples have been collected, maintaining the 50 m sample spacing and 200 m transects from the orientation survey and covering approximately 10.25 km^2.

The Activation Laboratories SGH interpretation highlighted areas of interest (see Figure 9-16), analogous to the results found across the current resource area and given a confidence rating of 4.0 out of 6.0 (the resource area survey scored 5.0 out of 6.0). Most notably are the anomalies around the nose of a large regional fold which also occur near several potential redox cells. Recommendations for future work include surface investigations of these areas of interest as well as continuing to sample the remaining strike length on the eastern half of the property.

Figure 9-16: 3D View of Main Gold Pathfinder Class Map from 2019 SGH Program



Source: Treasury Metals (2019)

9.1.12 2020 Exploration Activities

An additional SGH sampling program was executed by Axiom Exploration to complete the sample coverage along the strike length of the Goliath deposit on the eastern half of the property. Approximately 1,260 additional samples have been collected, maintaining the 50 m sample spacing and 200 m transects from the previous surveys and covering approximately 12.50 km^2. The Activation Laboratories SGH interpretation report has been received and an internal review is underway to determine if infill sampling of identified anomalies is required and to assist in the planning of future field programs.

9.1.13 2021 Exploration Activities

Other than drilling, no exploration work was carried out in 2021. Drilling is discussed in Section 10.

9.2 Goldlund Deposit

Treasury Metals has not conducted any surface exploration on the deposit since the acquisition of the property. Exploration conducted by previous owners is summarized in Section 6.

9.3 Miller Deposit

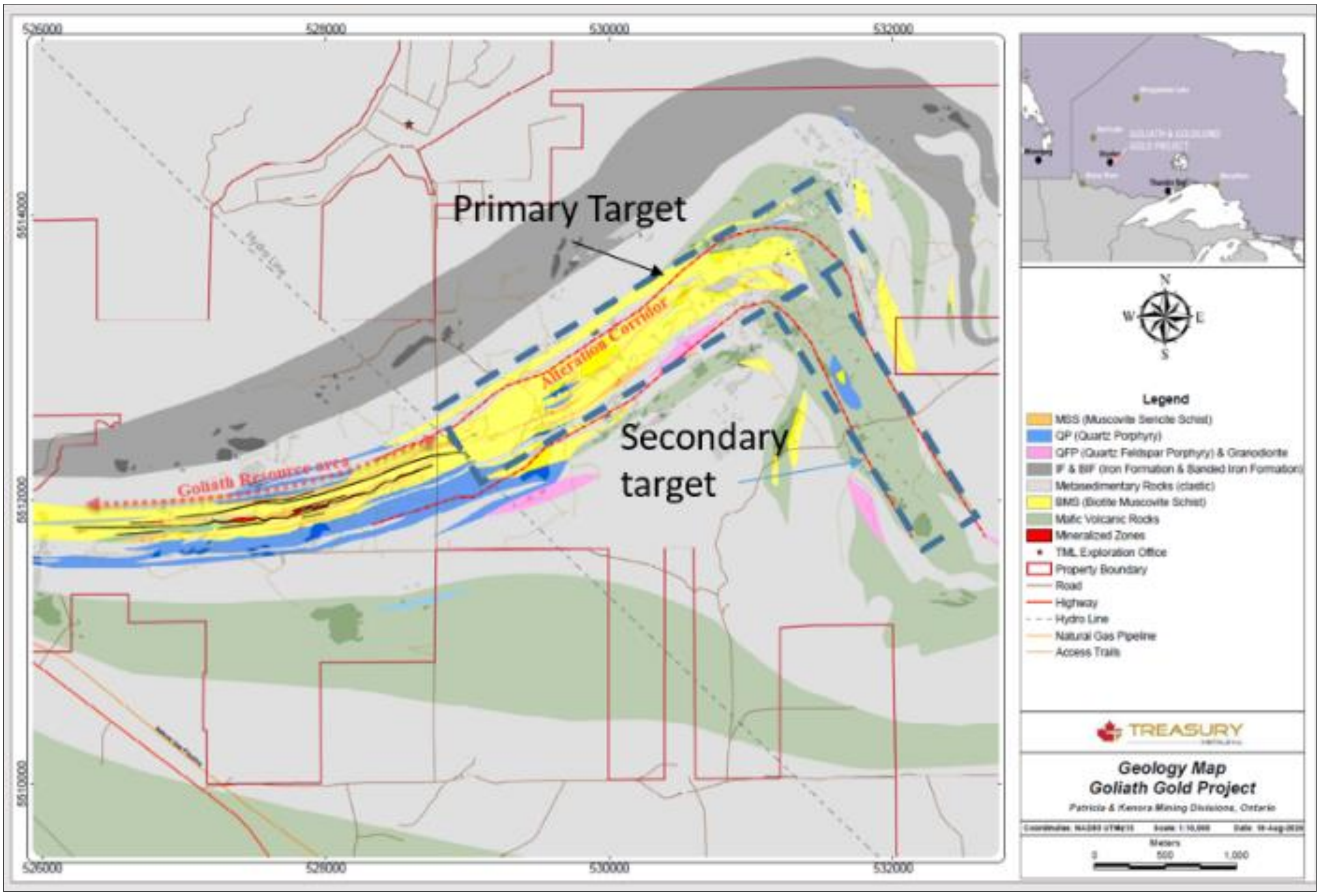
Treasury Metals has not conducted any exploration activities on the deposit since the acquisition of the property.

9.4 Exploration Potential

Mineralization at Goliath is open along strike and at depth. Work by Treasury Metals has demonstrated that the Goliath mineralization extends to the east on the EAC and continues to the south, albeit at slightly lower grade, along the folded limb of the geological units (Figure 9-17).

Mineralization at Goldlund and Miller seems to be well contained within the current drill patterns. Mineralization at Goldlund is open at depth and very little deep drilling has explored the possibility of deeper mineralization. The Miller deposit is lower-grade and of lower priority but still open down dip.

Figure 9-17: Goliath Deposit Exploration Targets



Source: Treasury Metals (2020)

10 DRILLING

10.1 Introduction

Much of the information for this section is from the NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex (Ausenco, 2021). The information about Goliath drill programs was sourced from technical reports prepared by P&E (2015, 2020), Roy et al. (2012) and Roy and Trinder (2011) as well as a number of drilling reports that have been filed for assessment credits by CCIC and Treasury Metals with the MNDM with edits from the QP.

The mineralization was sampled over the years with multiple campaigns of core drilling by Teck-Corona and Treasury Metals since the 1990s. The drill database is now a mix of historical data and more recent data collected by Treasury Metals from 2008 through to 2021[SG1] . Both data types were used in the resource estimate. The mineral resource estimate for Goliath is supported by 904 surface drillholes with an aggregated length of 290,6856 m. Drilling continued in 2022, where 53 drillholes with a length of 17,706 metres were completed on exploration targets as well as 8 drillholes with a length of 2,597 metres in support of geotechnical and metallurgical studies. The results of this additional drilling was not included in the PFS as it did not result in a significant variance from the resource published January 17, 2022.

The Goldlund-Miller property was acquired by Treasury Metals in July 2020. The information was sourced from various technical reports prepared by T. McCracken of Wardrop (2010 to 2011), T. McCracken of Tetra Tech (2012 to 2013), S. Zellerer of Tetra Tech (2014) and by T. McCracken of WSP (2015, 2017, 2019 and 2020).

10.2 Goliath Deposit Drilling

The Goliath drill program is best described in two parts between the historical Teck-Corona exploration drilling carried out between 1990 to 1998 and the more recent Treasury Metals drilling that has been carried since 2008. Historical drilling has been added to this section since it is considered highly relevant to the resource estimate discussed in Section 14. The following subsections summarize the various drill programs.

10.2.1 Teck & Teck-Corona Drilling 1990 to 1998

Thirteen drilling campaigns were undertaken by Teck Exploration and Teck-Corona over an eight-year period from 1990 to 1998. During this period, 340 diamond drillholes were completed for a total of 97,514 m of drilling (see Table 10-1 and Figure 10-1).

10.2.1.1 Teck & Teck-Corona Core Handling Procedures

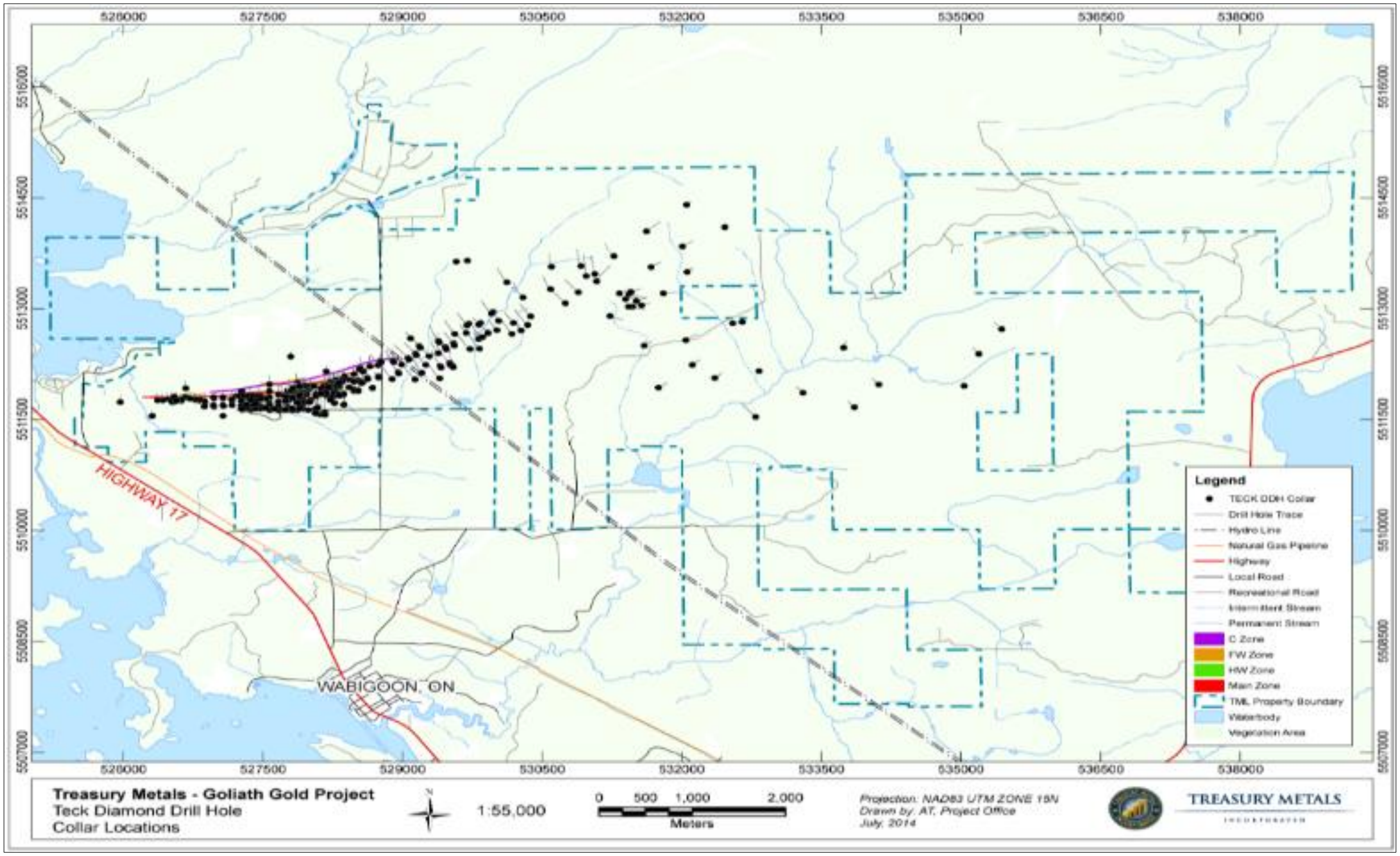
Several different drilling companies were used including Bradley Bros. Limited, Forage St. Lambert Ltd., Boart Longyear Inc. and St. Lambert Drilling Co. Ltd. Drill core size was predominantly BQ in the early years (1990 to 1996) and NQ in the later years. A majority of the drill logs record that the casing was left in the hole upon completion and the hole was capped. Downhole surveys for azimuth and dip were taken normally at 50 m intervals using initially Wel-Nav single shoot instruments and in the latter years using a Sperry-Sun Single Shot downhole instrument supplemented by acid tests when necessary. Usually, the first reading was taken immediately below the casing to ensure the hole was on course. Transit surveys of all drillhole casings within the resource area was completed by W.J. Bowman Ltd.

Upon the daily receipt of the drill core at the cores logging facility, the core was logged, marked, and tagged for assay by the geologist. The typed standard "Teck" core logs in PDF format are all available for inspection. For the major intervals, the logs record the rock type name along with a long description that typically contain the rock descriptions, mineralization, and alteration for the entire interval. The long description was often split in minor intervals describing zones of particular interest.

Table 10-1: Summary of Teck & Teck-Corona Diamond Drill Programs

Drill Program	Year	Holes	Dates Drilled	Hole Numbers	Meters Drilled
1	1990	7	October 28 to November 30, 1990	TL1 to TL7	1,096
2	1991	17	April 18 to May 15, 1991	TL8 to TL24	3,368
3	1992	13	May 13 to June 30, 1992	TL25 to TL37	4,373
4		9	October 16 to November 23, 1992	TL38 to TL43	3,800
				TL43W1 to TL43W3	
5	1993	10	August 14 to September 10, 1993	TC-1 to TC-10	1,747
6	1994	72	January 18 to November 16, 1994	TL44 to TL110	15,998
				TL44W1 to TL44W3	
				TL88W, TL96W	
7	1995	14	January 27 to February 27, 1995	TL111 to TL124	1,814
8		11	November 28 to December 25, 1995	TL125 to TL127	5,668
				TL125W1, TL125W2	
				TL126W1 to TL126W3	
				TL127W1 to TL127W3	
9	1996	18	January 7 to February 8, 1996	TL128 to TL132	6,250
				TL128W1 toTL128W3	
				TL129W1 to TL129W3	
				TLE11 to TLE17	
10		33	June 12 to October 31, 1996	TL133 to TL142	14,598
				TL133W1 to TL133W3	
				TL136W1, TL136W2	
				TL137W1, TL137W2	
				TLE18 to TLE33	
11	1997	65	January 15 to December 31, 1997	TL143 to TL206	23,232
				TL170W1	
12	1998	6	May 19 to July 1, 1998	TL207 to TL212	2,831
13		65	September 3 to December 5, 1998	TL213 to TL277	12,739
Total		340			97,514

Figure 10-1: Teck & Teck-Corona 1990 to 1998 Drill Hole Locations



Source: Treasury Metals (2014)

The legend used by Teck-Corona was different than the current legend used by Treasury Metals and the exact date of conversion between the Teck-Corona Legend to the Treasury Metals legend is not known. A significant amount of work was carried out by CCIC to recover the mineralization, veining, and alteration information from the long description of the logs and populate the appropriate database tables now used by Treasury Metals.

The samples were then sawn in half using a Target masonry saw with a 14″ diamond blade. All samples were shipped to the primary laboratory by Gardwine and Porter transport firms. The primary laboratory used was TSL Laboratories of Saskatoon, Saskatchewan with XRAL Laboratories and Interteck Testing Services used for assay verification work or whole rock analyses.

All core samples were submitted for gold and sporadically assays for silver, copper, lead, and zinc when the gold grade was expected to be high.

In 2008, all recoverable historical Teck and Teck-Corona drill core that was in long-term storage in the town of Wabigoon was moved to Treasury Metals' core farm located on the former tree nursery site. The core is in very poor condition and Treasury Metals was not able to resample the core as part of its resource evaluation work.

The highlights of the various drilling programs completed by Teck during the 1990s have been summarized below. Teck-Corona was mainly looking for high-grade zones in excess of 3.0 g/t and because of that, did not typically report assays that were considered sub-economic at the time but well above a modern open pit cut-off grade and didn't systematically sample all drill core and only carried out limited assaying for silver.

The drilling and core handling procedures used by Teck and Teck-Corona were consistent with the core handling procedures generally in place in the 1990s by most major mining companies.

10.2.1.2 1990 to 1993 Teck Drilling Programs

Teck's very first diamond drilling program on the Thunder Lake deposit commenced October 28, 1990, to November 30, 1990 with the completion of seven BQ holes (TL1 to TL7) totalling 1,096 m. The discovery hole (TL1) on the Main Zone of the deposit intersected three significant zones of polymetallic disseminated sulphide mineralization containing gold (Page, 1991):

- Zone A returned 2.23 g/t Au, 18.9 g/t Au, 0.63% Zn over 6.1 m (80.0 to 86.1 m) including 5.25 g/t Au, 16.8 g/t Ag, 0.28% Zn over 1.9 m.
- Zone B intersected 0.97 g/t Au over 10.4 m (107.4 to 117.8 m) in a pyritic alteration zone.
- Zone C assayed 7.99 g/t Au, 16.5 g/t Ag and 0.61% Zn over 6.1 m (196.7 to 202.8 m) including 17.49 g/t Au, 33.6 g/t Ag and 0.42% Zn over 2.6 m.

This hole was drilled to test a "high priority" IP chargeability anomaly determining that this exploration method was very useful in defining potential future drill targets within and on-strike with the Goliath deposit.

Following this discovery, much of the remaining historic exploration on the Thunder Lake property centred on diamond drilling programs with the most drilling having been completed in the area north of the Laramide property in the Thunder Lake West portion; there was minimal drilling on the Thunder Lake East portion in Hartman Township.

Teck completed 17 BQ diamond drillholes for a total of 3,368 m in 1991. Hole TL9 intersected an isolated high of 45.96 g/t Au over a sample length of 0.5 m (44.8 to 45.3 m) in a section of biotite-muscovite schist in the Main Zone. Holes TL21 and TL23, drilled on the same drill section, intersected three sections of high-grade gold mineralization corresponding to the Main Zone Hanging Wall, Main Zone and C Zone.

Two diamond drilling programs were completed in 1992 with Phase I initiated during the months of May and June and Phase II in the Fall in October and November. A total of 22 BQ holes were drilled (TL25 to TL43) and three wedges were turned off of hole TL43 (TL43W1, TL43W2 and TL43W3) for a total of 8,173 m of diamond drilling. Drillhole TL39 was abandoned due to excessive flattening of the hole and restarted as new hole TL39A.

In 1993, 10 BQ diamond drillholes totalling 1,747 m were drilled to test a series of ground IP geophysical anomalies located in the extreme eastern portion of the property in Hart Township (east of UTM 532400E). The holes were numbered TC1 to TC10. Hole TC6 was a failed hole ending at 135 m and no samples were taken for assay. None of the holes returned any significant gold assays (all less than 0.09 g/t Au). However, many of the IP anomalies were attributed to either the presence of graphite, elevated pyritized rocks or sulphide iron rich metasedimentary rocks.

10.2.1.3 1994 Teck-Corona Drill Program

In 1994 a total of 72 diamond drillholes totalling 15,998 m, including five wedge holes and one abandoned hole, were completed. These drillholes were numbered TL44 to TL110, TL44W1 to W3, TL88W and TL96W and were drilled using both NQ and BQ size rods.

From January to February 1994, Teck completed a 4,846 m diamond drilling program. A total of 34 holes were drilled of which 20 were NQ and 14 were BQ sized core numbered TL44 to TL77. Twelve samples were collected from hole TL44 and dispatched to X-Ray Laboratories in Don Mills, Ontario for whole rock analyses. The best gold assay intersections were obtained from the Main Zone and the most significant drillhole intersection was from TL49 that returned 21.2 g/t Au over a sample length of 8.5 m from 178.0 to 186.5 m. The better auriferous intersections in the Main Zone were characterized by (Page, 1994):

- quartz-sericite schist host rock
- rocks containing 1% to 5% disseminated pyrite with local concentrations of 5% to 20% pyrite
- trace to locally 3% to 5% disseminated and stringer sphalerite accompanied by lesser amounts of galena (trace to 2%), chalcopyrite (trace to 1%) and rare occurrences of arsenopyrite
- intense silicification containing 5% to 25% total sulphides
- rare pinpoint to mm grains of native gold and electrum.

Teck also completed a re-logging and sampling program of earlier drillholes, re-examined surface exposures, and carried out metallic screen fire assaying of most core intersections through the Main Zone (Page, 1995a).

Pulp metallic screen fire assaying determined that there were significant nugget effects present in the deposit reflected in both the assay results and the observed distribution of native gold and electrum (Page, 1995a). Roughly two-thirds (64%) of the 210 samples revealed gold assay results that compared well between the 30 g fire assay and pulp metallic methods. Just over one-tenth (12%) of the samples returned initial assays much larger than the pulp metallic and around one-quarter (24%) of the samples yielded pulp metallic gold assays much larger than the initial gold fire assay results. It was determined that, although more expensive, utilizing pulp metallic screen fire assaying proved to be most useful in defining the overall character and geometry of the deposit.

Highlights of gold assay (> 3.0 g/t Au) returns from the remaining holes drilled in 1994 include the following:

- TL80: 3.53 g/t Au over a core length of 5.6 m (174.7 to 180.3 m) including 10.50 g/t Au over a sample length of 1.5 m (178.8 to 180.3 m)
- TL81: 5.67 g/t Au over 13.2 m (215.0 to 228.2 m)
- TL82: 18.89 g/t Au over 3.7 m (266.5 to 270.2 m)
- TL84: 3.54 g/t Au over 11.0 m (48.4 to 59.4 m)
- TL96: 3.29 g/t Au over 5.4 m (375.4 to 380.8 m)
- TL44W3: 5.64 g/t over 7.9 m (535.5 to 543.4 m).

10.2.1.4 1995 Teck-Corona Drill Program

Fourteen BQ holes totalling 1,814 m, numbered TL111 to TL124, were completed in the early part of 1995. These holes were drilled to delineate a shallow gold resource in the "West Alteration Zone" (TL11 to TL117) to vertical depths of around 80 m and to partially define the west and east edges of the No. 2 shoot to depths of -50 to -85 m (TL119, TL120) and west edge of the No. 1 shoot (TL121, TL122 to a vertical depth of -140 m and -110 m, respectively). Holes TL114, TL117 and TL118 were abandoned prematurely due to drilling difficulties (Stewart, 1995).

Hole TL114 intersected the Main Zone returning 15.81 g/t Au over a core length of 3.0 m (60.2 to 63.2 m) and hole TL118 returned a Hanging Wall/Main Zone intersection of 14.73 g/t Au over a core length of 5.5 m, which includes a single 53.24 g/t Au assay over a core length of 1.5 m (87.2 to 88.7 m).

10.2.1.5 1996 Teck-Corona Drill Program

A winter drilling was program completed from November 1995 to February 1996. A total of eight deep BQ holes, numbered TL125 to TL132, were drilled for a total of 4,142 m to test the Main Zone at a vertical depth of between 400 m and 500 m to the east and west of the No. 1 and No. 2 shoot area (Stewart, 1996).

Drilling resulted in extending the Main Zone in the area of the "West Alteration Zone" in the main deposit to a vertical depth of around 450 m. Hole TL-129 intersected the Main Zone from 433.5 m to 474.0 m grading 2.31 g/t au over a 40.5 m core length which includes grades of up to 16.96 g/t Au over 2.0 m (452.5 to 454.5 m) and 15.47 g/t Au over a sample length of 1.0 m (470.0 to 471.0 m). The Main Zone in the area of the "East Alteration Zone" was extended to a vertical depth of approximately 500 m.

During the winter program, seven BQ holes were drilled (TLE11 to TLE17) for a total of 1,126 m. These were regional exploration holes in the eastern portion of the property, an area called Thunder Creek East, to test a series of both IP and VLF-EM anomalies. Most of these holes encountered amphibolite, garnet amphibolite, and meta-sedimentary rocks (argillites, conglomerates, greywacke, and chert-magnetic bearing iron formation). Geophysical target anomalies were attributed to the presence of graphite and elevated sulphides in the metasedimentary rocks. The best drillhole TLE15 intersected 11.60 g/t Au over a core length of 4.2 m (119.4 to 123.6 m) including 46.74 g/t Au over 1.0 m (122.6 to 123.6 m). Hole TLE16 returned 3.58 g/t Au over a sample length of 1.0 m (57.2 to 58.2 m).

A second phase of diamond drilling was completed from June to the end of October 1996. Ten NQ holes, numbered TL133 to TL142, 20 BQ wedges in 7 holes (2-3 wedges per hole) and nine previous drillholes were extended for a cumulative total of 1,482 m (Stewart et al., 1997). There was also a program of partial re-logging of holes TL41, TL42 and TL59.

The most significant results of the Phase II drilling program were the intersection of high-grade gold mineralization in hole TL141 and two additional intersections of lower grade mineralization at the eastern and depth extent of the resource areas (holes TL135 and TL136). In addition, the East Alteration Zone was extended eastward for another 150 m and to a vertical depth of 550 m.

Sixteen exploration holes (BQ) were drilled in the eastern portion of the property to follow-up the high-grade gold intersection by hole TLE15 earlier that year and to test additional IP and VLF-EM anomalies as well as local stratigraphy. These holes were numbered TLE18 to TLE33 totalling 3,359 m. Drilling encountered predominantly amphibolite and metasedimentary rocks (greywacke, biotite schist, mafic schist, graphitic argillites, some iron formation and garnetiferous metasedimentary rocks) some of which were intruded by quartz-feldspar and feldspar porphyry bodies. Hole TLE18 returned 2.38 g/t Au over 0.8 m (81.4 to 82.2 m) and hole TLE27 assayed 1.94 g/t Au over a core length of 1.0 m (168 to 169 m). In each case, gold mineralization was hosted in amphibolite rocks containing elevated sulphides including sphalerite.

10.2.1.6 1997 Teck-Corona Drill Program

A 64 hole diamond drilling program was completed between January 15, 1997 to December 31, 1997 (Page and Waqué, 1998). The holes, numbered TL143 to TL206, totalled 23,232 m of NQ drilling. Reconnaissance (step-out) drilling program following the eastern extension of the Thunder Lake alteration corridor, east of the deposit, included the completion of 13 drillholes covering 1,400 m of strike length. Drilling east of the resource area was disappointing with only geochemically anomalous gold values being intersected over significant to narrow widths. The best assay intersection was obtained from drillhole TL95 that returned 2.01 g/t Au over a core length of 1.2 m (77.9 to 79.1 m).

The majority of the drilling consisted of resource exploration and delineation of the No. 3 shoot (formally called the “East Alteration Zone”) in the eastern resource area and the West Alteration Zone. A total of 44 new drillholes (and one wedge cut) were completed within the resource area. Nine drillholes defined the high to moderate grade portion of the No. 3 shoot: TL144, 145, 150, 151, 174, 175, 176, 180 and TL181 (Page and Waqué, 1998). Hole TL151 returned 9.49 g/t Au over a sample length of 23.3 m (432.9 to 456.2 m) and hole TL144 intersected 11.81 g/t Au over a core length of 10.5 m (69.0 to 79.5 m).

Seven short holes drilled in the area of the No. 1 and No. 2 shoots confirmed the presence of a “dead zone” between the shoots and erratic gold distribution within the No. 2 shoot. Hole TL190 intersected the best gold intersection returning 26.04 g/t Au over a sample length of 2.3 m (52.2 to 54.5 m). Closely-spaced definition drilling at 12.5 m centres in the area confirmed some nugget effects in both the No. 1 and No. 2 shoots (Page and Waqué, 1998). For example, higher grade intersections in the No. 2 shoot did not appear to correlate well beyond two or three drillholes. The No. 1 shoot demonstrated better grade continuity both along strike and down dip.

10.2.1.7 1998 Teck-Corona Drill Program

In 1998, a total of 71 BQ diamond drillholes totalling 15,570 m numbered TL207 to TL277 were completed in a two-phased program. Previous diamond drilling programs focused on defining gold mineralization within the Main Zone alteration corridor over a strike length of about 1,800 m to vertical depths of 400 m to 500 m with only a few holes to depths of 700 m to 800 m below surface. Drilling had consisted mostly of closely spaced (25 m centres) shallow holes for resource definition, multiple wedge cuts to evaluate nugget effects, widely-spaced deeper drilling and reconnaissance drillholes located up to 1,500 m east of the main resource deposit (Page et al., 1999a).

The 1998 drilling program consisted of infill definition drilling plus reconnaissance surface diamond drilling and was completed from (1) May 19, 1998, to July 1, 1998 and (2) September 3, 1998 to December 5, 1998. Drilling was dispersed over a large area of the property and included 25 closely spaced (25 m to 50 m centres) infill holes within the gold resource area, three holes in the western portion of the property, four deep holes and seven shallow holes in the area adjacent (east) of the gold resource, and 21 reconnaissance to 100 m spaced infill holes covering an additional 2,000 m of strike length in the eastern portion of the property.

In the resource area, 23 holes tested the No. 3 shoot (Main Zone) and two holes tested for the up-dip extension of the C Zone. The C Zone holes (TL249 and TL251) returned only anomalous gold values. Four intersections of greater than 3.0 g/t Au over 3.0 m were returned from the No. 3 shoot drilling (holes TL225, TL234, TL238 and TL244).

Drillholes located west and east, and less than 1,000 m along strike of the resources did not return any significant intersections. Hole TL212 returned 1.33 g/t Au over a core length of 5.5 m (219.0 to 224.5 m) in strongly altered Main Zone rocks.

Fifteen holes totalling 3,737 m were drilled to test the alteration corridor over an additional 1,100 m strike length from grid line L14+00 E to L25+00E. These widely spaced reconnaissance and infill drillholes returned anomalous gold values with rare assays exceeding 3.0 g/t Au. Hole TL271 returned 17.36 g/t Au and 754.5 g/t Ag over a core length of 1.6 m from 59.2 to 60.8 m in a weakly sericitic zone containing abundant silver-rich electrum. However, two follow-up holes, drilled 25 m on either side of TL271, did not return any significant gold values in the target locations. These two holes returned best assays of less than 0.10 g/t Au in TL275 and 0.8 g/t Au over 1.0 m from 60.5 to 61.5 m in TL276. Hole TL208 contained an isolated stringer of visible gold yielding a high-grade single assay of 43.3 g/t Au over a core length of 1.5 m (532.5 to 534.0 m) obtained from a zone located 40 m above what is interpreted to be the Main Zone in this area. Drillhole TL272 returned a single high-grade assay of 9.47 g/t Au over a sample length of 1.1 m from 187.7 to 188.8 m.

Six holes totalling 2,013 m were also drilled in the vicinity of the regional scale synformal fold hinge (an area called the fold nose). This program was designed to test a number of anomalous sericite schist and sulphide showings, several IP anomalies, and interpreted structures. All drillholes in the fold nose returned multiple short intervals of anomalous gold hosted in virtually all rock types in this area usually associated with quartz veining and/or increased sulphide content. While two of the holes returned single high-grade assays in excess of 3.0 g/t Au, Teck could not define any localized structure or rock type that would have allowed focussing of alteration and mineralization in the fold nose area.

10.2.1.8 1998 Corona Gold Corporation (Jones Property/Lot)

Corona Gold Corporation (Corona) conducted a small diamond drilling program on its 100% owned Jones property (or "Lot"), land Parcel PA3830, from early October to early December 1998 (Page and Waqué, 1999). This parcel is located in the south part of Lot 8, Concession IV in Zealand Township. A total of 12 shallow NQ drillholes totalling 1,452 m were drilled at close spacing's (50 m centres) to intercept the western Main Zone extension targeting the zone at vertical depths of 25 m to 85 m from surface. The holes were numbered TL252, TL254 to TL256, TL258 to TL261, TL263, TL273, TL274 and

TL277. Drilling was undertaken to follow-up on favourable gold intersection obtained from the first-pass drillholes which covered the full strike length of the claim package. The initial nine drillholes (TL252 to TL263) tested 500 m of strike length along the Main Zone.

According to Page and Waqué (1999), the results of this drilling program were disappointing. In this area, the Main Zone is only weakly mineralized with sericitic alteration of variable intensity and silicification, quartz and sulphide veining as well as intense deformation fabrics was found to be generally lacking. Overall, the assay results from all drillholes completed during this program were consistent with the character of a weakened mineralized system. Hole TL274 intersected the best mineralization returning 4.30 g/t Au over a sample length of 2.6 m (29.0 to 31.6 m). The highest grade was returned from hole TL259 that intersected 5.81 g/t Au over a core length of 1.4 m (61.0 to 62.4 m).

It was concluded that the potential for gold mineralization decreases significantly further west of the main resource area along the Main Zone structure and it was recommended that no further work be completed on the Jones property.

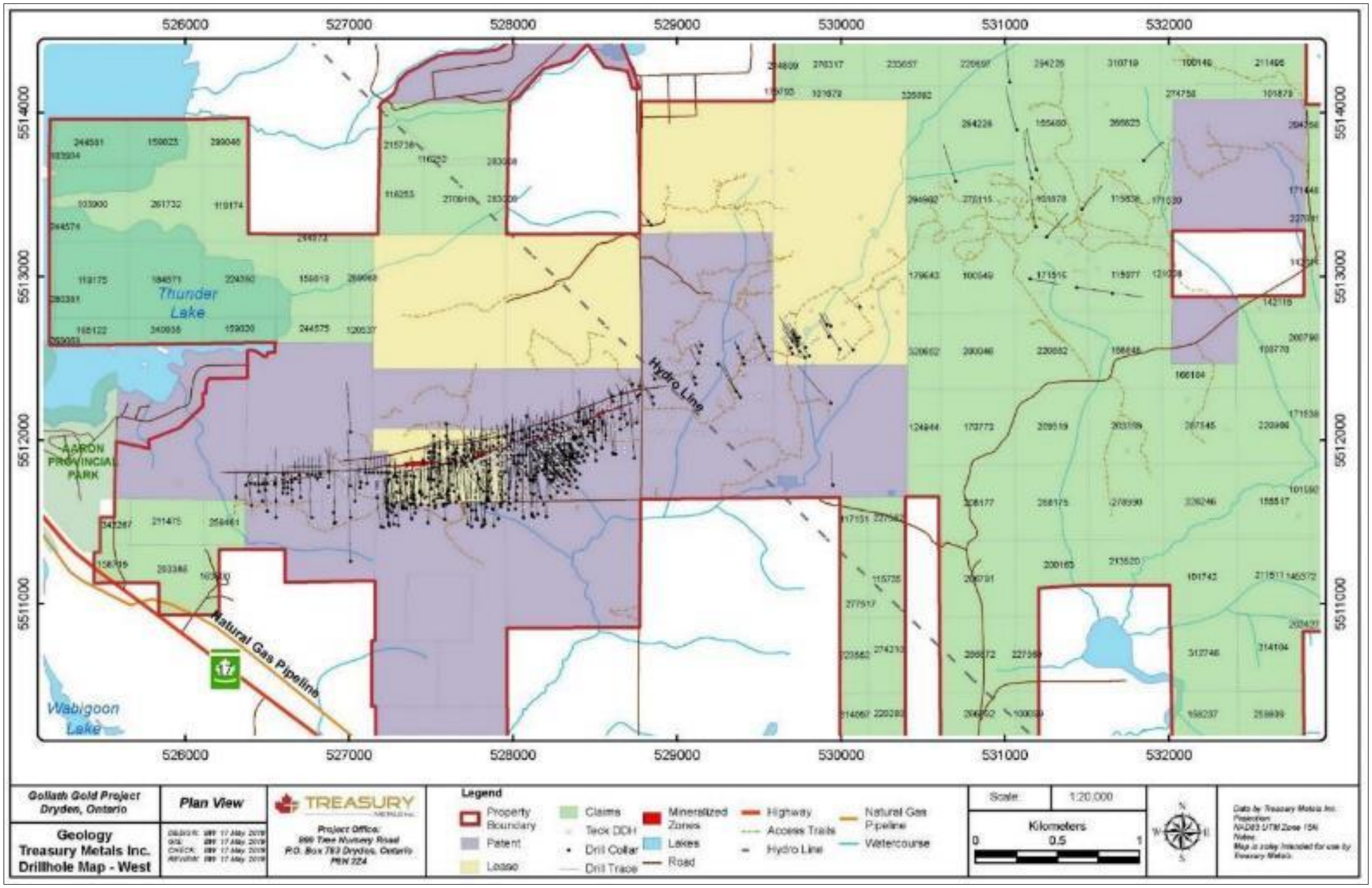
10.2.1.9 Treasury Metals, 2008 to 2020

Treasury Metals has conducted 18 diamond drilling campaigns on the Goliath property since 2008. A total of 180,269 m has been drilled by Treasury Metals on the property since 2008 including 528 newly collared holes, 30 re-entry holes and 4 wedge holes (see Table 10-2, Figure 10-2 and Figure 10-3).

Table 10-2: Treasury Metals Drill Program

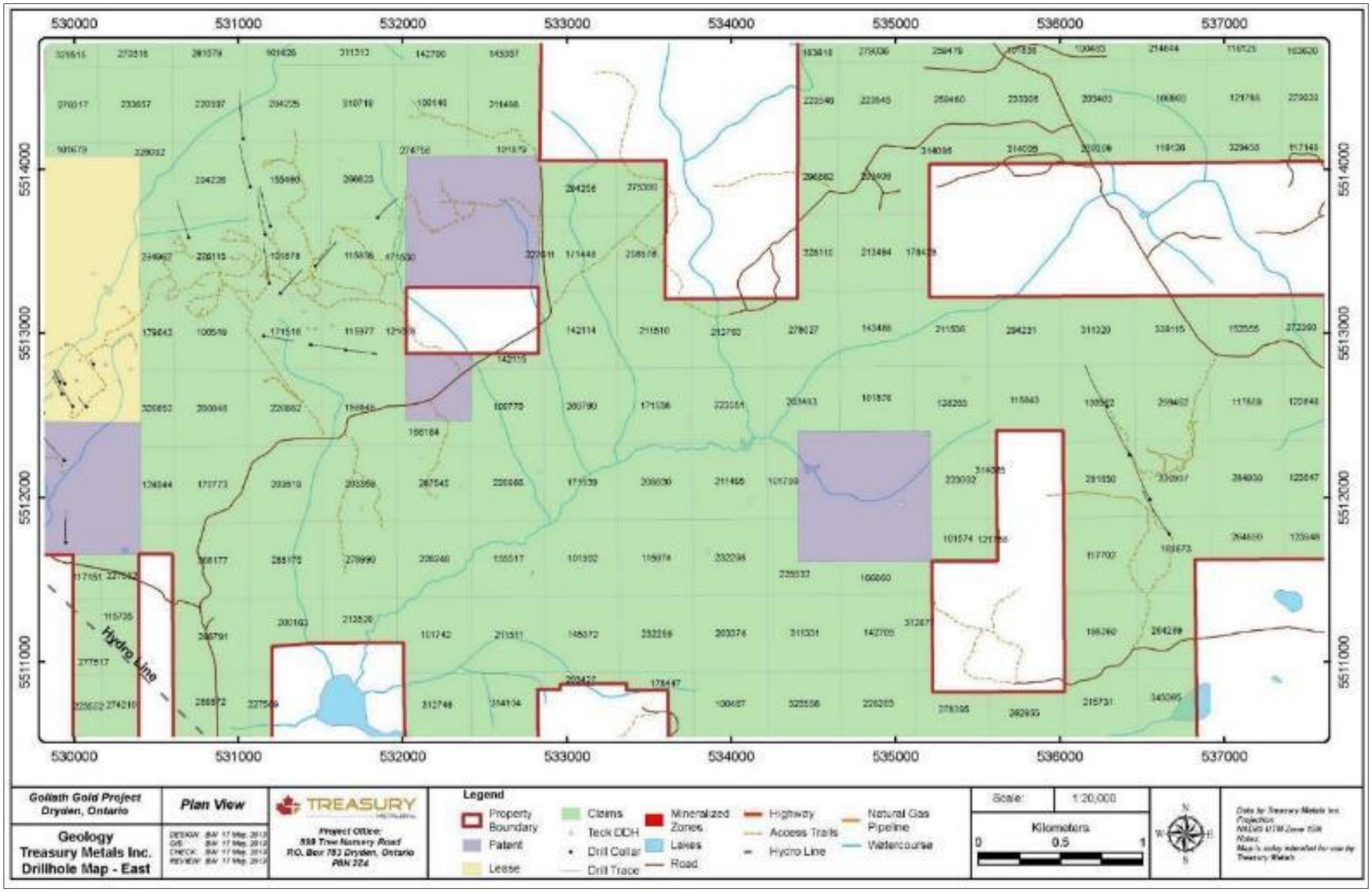
Drill Program	Year	Dates Drilled	Hole Numbers	Meters Drilled
1	2008	February 18 to September 21, 2008	TL0801 to TL0855	13,121
2	2009	October 20 to December 15, 2009	TL0956 to TL0986	4,589
3	2010	February 20 to March 29, 2010	TL1087 to TL1094	5,211
4		May 2 to June 2, 2010	TL1095 to TL10112	5,153
5		December 2 to December 19, 2010	TL10113 to TL10118	1,818
6	2011	January 17 to September 1, 2011	TL11119 to TL11229	48,538
7	2012	January 25 to June 6, 2012	TL12230 to TL12277	16,110
			TL220-12RE, TL234-12RE, TL231-12RE, TL219-12RE, TL216-12RE	
8		October 22 to December 14, 2012	TL12278 to TL12295	6,540
			TL164-12RE, TL0852-12RE, TL230-12RE	
			TL227-12RE, TL226-12RE, TL238-12RE	
			TL242-12RE, TL148-12RE, TL225-12RE	
			TL0826-12RE	
9	2013	January 7 to February 26, 2013	TL13296 to TL13336	7,772
			TL176-13RE, TL180-13RE, TL223-13RE	
			TL1095-13RE, TL10107-13RE	
			TL0827-13RE, TL10113-13RE	
10	2014	January 23 to June 23, 2014	TL14337 to TL14371	10,749
			TL0855W2b, TL166-14RE, TL161-14RE	
			TL0851-14RE, TL10109-14RE	
			TL0855W1, TL0855W2, TL0855W2b	
11		November 27 to December 19, 2014	TL14372 to TL14377	1,614
12	2015	January 8 to March 17, 2015	TL14378B to TL15402	7,263
			TL14373-15RE, TL14377-15RE	
13	2016	August 24 to January 15, 2017	TL16403 to TL16420	12,154
			TL16415W1	
14	2017	January 10 to March 16, 2017	TL17421 to TL17445	4,022
15		June 22 to October 31, 2017	TL17446 to TL17463	4,494
16	2018	January 8 to June 22, 2018	TL18464 to TL18501	20,987
17	2019	November 15 to Dec 14, 2019	TL19502 to TL19513	4,468
18	2020	January 4 to March 6, 2020	TL20514 to TL20528	5,667
19	2021	January 4 to December 31	TL21529B to TL21601	29,884
Total				210,154

Figure 10-2: Goliath Drillhole Location Map, Western Goliath



Source: Treasury Metals (2019)

Figure 10-3: Goliath Drillhole Location Map, Eastern Goliath



Source: Treasury Metals (2019)

10.2.1.10 Treasury Metals Core Handling Procedures

From 2008 to 2010, Caracle Creek International Consulting Inc. (CCIC) designed and supervised all of the drilling programs at the Goliath project. In February 2011, Treasury Metals geological staff took over the direct supervision of all Goliath exploration activities.

Over the last 11 years, Treasury Metals has used four different drilling contractors to complete the drilling programs (Table 10-3). The majority of the drill contracts were awarded to Distinctive Drilling Services Inc. of Westbank, BC, from 2009 to 2013 and George Downing Estate Drilling Ltd. (Downing Drilling) of Grenville-sur-la-Rouge, QC, from 2014 to 2020. Other contractors include G & O Diamond Drilling Contractors Ltd. (G&O) of Hay Lakes, AB, which drilled the first 37 holes of the 2008 drilling campaign and North Star Drilling Limited of Thunder Bay, ON, in 2014. All holes were drilled with NQ or NQTK (NQ2) size core which have a nominal diameter of 47.6 mm and 50.7 mm, respectively.

Table 10-3 : Treasury Metals Core Handing Procedures

Drilling Years	Drill Contractor Name
2008	G & O Diamond Drilling Contractors Ltd.
	North Star Drilling Limited (Thunder Bay)
2009 to 2013, 2018	Distinctive Drilling Services Inc. (B.C.)
2014 (January to June)	North Star Drilling Limited (Thunder Bay)
2014 to 2022	George Downing Estate Drilling Ltd.

Each drill contractor constructed drill access trails and drill pads for each setup with water supplied by pump from local beaver ponds, creeks, and streams. A Reflex single-shoot down-hole survey tool is used to survey the holes with readings taken at 50 m intervals. The drill casing is left in each hole and the hole capped to allow for future downhole geophysical testing and/or deepening of the hole.

Each hole is initially surveyed with a GPS handheld instrument in UTM coordinates (NAD83 Zone 15N) and upon completion holes are surveyed using a high precision Trimble survey instrument for higher accuracy. Oriented core drilling was implemented for holes TL0822 to TL0837 using an EzyMark tool provided by Boreinfo Ltd. The objective of this oriented core drilling was to clarify the spatial relationships between structural features and their influence on the mineralization (Roy et. al, 2012).

The drill core was logged, split, and stored at the exploration field office and core logging facility in Dryden under the supervision of the CCIC staff from 2008 to 2010. Once Treasury Metals staff took over the project management, they moved their operations to the former 136 ha Tree Nursery facility located at the end of Tree Nursery Road which they purchased in 2011 (building and surface rights). This facility includes a large office building with a core logging and core cutting room, additional large warehouses which are used for storing pulps, rejects and drill core and there is also a core farm on site. A gate has been set up on the road at the pond restricting access to the site and the main office building is monitored by a security alarm system.

As the core boxes arrive at the core logging facility from the drill, the meterage in each box is recorded and verified by a technician and hole number and meterage interval label tags are made using a dymo gun or handwritten on an aluminium tag and stapled to the end of each box. Rock-quality designation (RQD) is also determined for each hole. Overall, core recovery has been excellent.

The geologist then logs and marks out samples for assaying. Treasury Metals uses DHlogger™ and log directly into the software. Sample lengths are adjusted as necessary to reflect geological and/or mineralization contacts. Sample assay tags are placed in the box by the geologist. In general, samples range in width from 0.2 to 1.5 m with the majority of sampling being 1.0 m or 1.5 m in length. Longer sample lengths have occasionally been collected of strongly sheared core sections with poor core recoveries. All drill core boxes are photographed after they have been logged and sampled.

Samples are spilt using a core saw to retain half of the sampled sections for future verification and metallurgical testing (if required). Sample tags are placed in the bags and the sample number is written on the bag using a black permanent marker pen. Samples are then sealed in plastic sample bags using zip-straps, placed in sealed and numbered rice bags. Samples were originally shipped by courier to Accurassay in Thunder Bay. In 2016, Treasury Metals started to use the ActLabs facility in Dryden, Ontario and the samples were then delivered by company personal. Laboratory and assaying procedures are discussed in detail in Section 11 of this report. Core boxes are placed in long-term storage on site at the core farm.

Samples are analysed for gold (fire assay), silver, zinc, lead, and trace element geochemistry (ICP) as discussed in Section 11. Digital assay files provided by the laboratories are merged directly into the Datamine digital database using DHlogger and DHexplorer software to avoid errors in transferring data.

The majority (81%) of the 545 bulk density sample measurements were carried out on 10 cm core pieces submitted to the analytical laboratory. The remaining 19% were completed in-house on un-coated, air-dry samples. The core at Goliath is solid with little to no pore and the in-house density measurements compare well with the laboratory figures.

Figures 10.4 displays a representative cross-section of the Goliath deposit showing the Teck-Corona drillholes along with the Treasury Metals drilling. Each of the various drilling campaigns completed by the Company over the last ten years is summarized below.

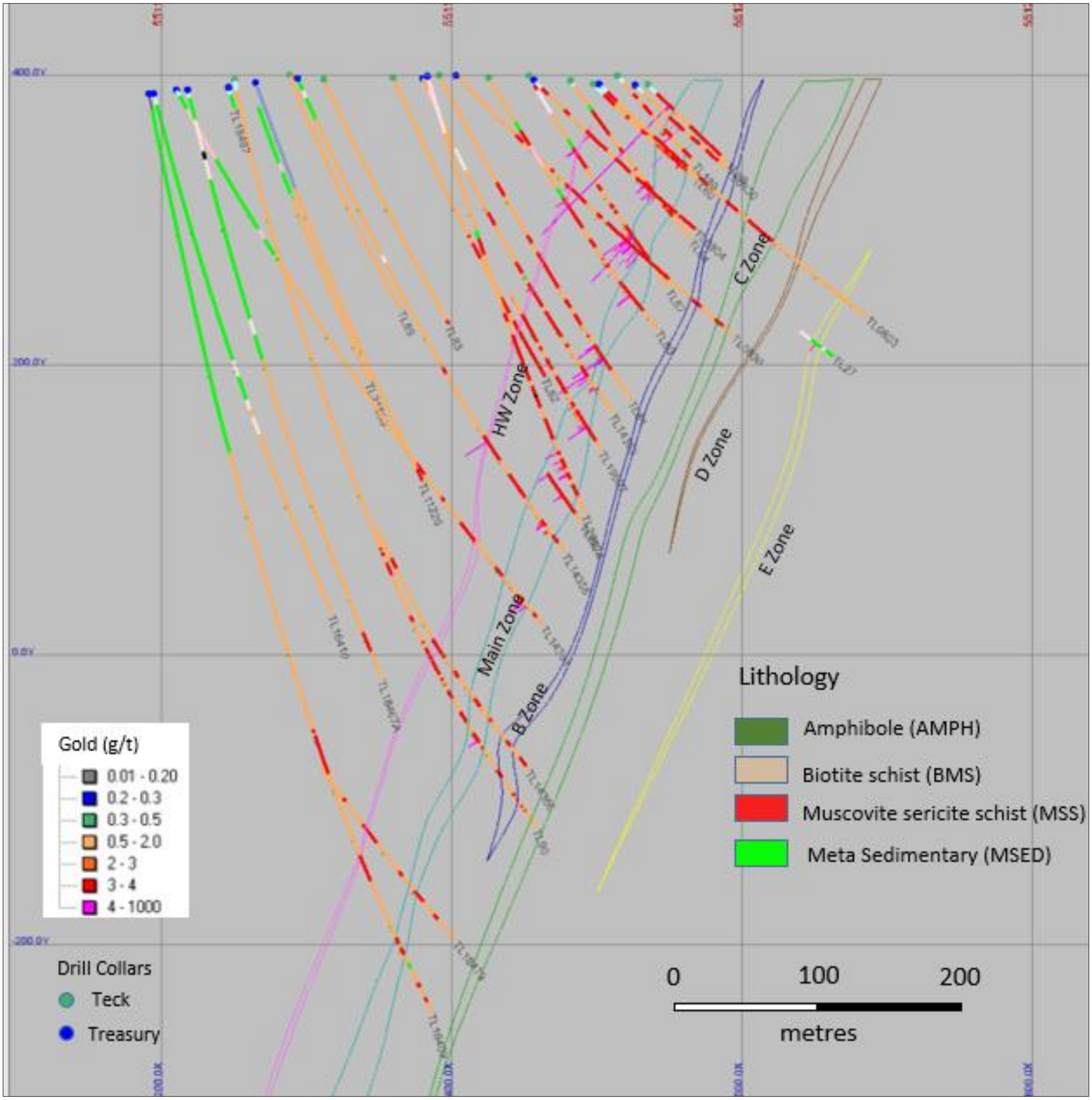
10.2.1.11 2008 Diamond Drilling Program

Fifty-five NQ2 diamond drillholes were drilled on 21 drill sections for 13,121 m from February 15, 2008 to September 22, 2008. This program targeted the Main Zone over a strike length of 1,700 m within the resource-defined area to a maximum vertical depth of around 695 m (hole TL0835). The drill contracts were awarded to G&O who drilled the first 37 holes and North Star completed the remainder. The objective of this program was three-fold (Ilieva, 2009):

- to confirm and add potential gold ounces to the historical inferred mineral resource of the Thunder Lake deposit (now referred as the "Goliath deposit")
- to include not only gold but also silver, zinc, and lead assays to eventually prepare a new resource estimate of the deposit
- to target deeper (>400 m) down dip extensions of known gold mineralized shoots.

Holes were drilled at azimuths of 360° or 180° with the inclination of each hole set at -45° or 60°. The first 10 holes (TL0801 to TL0810) were drilled in close proximity to former Teck drillholes along the deposit to confirm historical gold assays as well as testing the areas that were not previously sampled Teck. Drillholes TL0801 to TL0837 were completely sampled from top to bottom. Once it was confirmed by CCIC that the gold mineralization was associated with the MSS unit and visible occurrences of sphalerite and galena, sampling was focused mostly on these targets and the Main Zone. Magnetic susceptibility readings were collected from 7,430.1 m of drill core using a handheld KT-9 Kappameter instrument.

Figure 10-4: Cross Section 527925E Looking West of Goliath Mineralization



Source: SRK, 20223

All of the diamond drillholes intersected and tested the Main Zone which consisted of the Hanging Wall (M1) and Footwall (M2) sub Zones. Intersection core lengths of this zone ranged from 5.0 to 30.4 m (hole TL0836). Mineralized intervals were often narrow (up to 0.5 m) zones enriched with 3% to 5% visible sulphide, locally up to 15% (Ilieva, 2009). The main sulphide mineral phases identified were pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite, arsenopyrite and dark grey needles of stibnite in decreasing order of abundance. These sulphides occur as disseminations, blebs, and stringer as well as cubic in the case of galena.

Visible occurrences of gold and electrum (gold-silver) are rare and are observed mostly in the MSS units and in leucocratic sericite-rich bands. For example, very rare specks of visible gold were found in holes TL0815 and TL0817 and downhole depths of 50.8 m and 129.2 m, respectively.

All of the holes intersected gold-bearing sulphide mineralization many returning significant assay results for gold silver, zinc, and lead.

Gold concentrations were found to be independent of pyrite content. However, an increase in the pyrite (especially coarser grained pyrite) and sphalerite content corresponded to increases in both gold and silver grades. Grade-wise, it was determined that an increase in chalcopyrite and galena did not seem to affect the overall gold content or grade.

CCIC concluded that "low-grade gold-silver mineralization is pervasive throughout the Main Zone, but the high-grade gold (>3.0 g/t Au) is concentrated in steeply west-plunging "shoots" with relatively short strike-lengths up to 50 m, good down-plunge continuity and remained open at depth". Very rare flakes aquamarine green mica (fuchsite- Cr muscovite) were found to occur in the strongly altered sericite alteration in association with high-grade gold.

10.2.1.12 2009-2010 Diamond Drilling Program

Four phases of drilling were completed from October 2009 to the end of 2010. The purpose of this drilling program was to (1) follow-up on the results of the 2008 drilling program with "infill" drilling to better define the resource in and around the Main Zone and expand it at depth and along strike, and (2) to conduct exploration drilling to expand the known resource along strike to the west and to the east and at depth (Magyarosi and Peshkepia, 2011).

Sixty-three NQ holes were drilled on 28 drill sections for a total of 16,672 m testing the gold potential of the main deposit over a strike length of around 2.0 km. The drill contract was awarded to Distinctive Drilling Services Inc. (Distinctive Drilling). All holes were drilled approximately perpendicular to the mineralization with azimuths of 360°and 320° and dips ranging between -45° and -87°.

Drilling was conducted in four phases over the 14-month period. Phase I was carried out in the fall of 2009 with 31 holes drilled for a total of 4,590 m (TL0956 to TL0986) with most of this work being concentrated in the western portion of the deposit. Phase II was completed in the spring of 2010 and includes eight holes numbered TL1087 to TL1094 for a total of 5,111 m. Phase III was initiated in the summer of 2010 where 18 holes were drilled for a total of 5,153 m (TL1095 to TL10112). The final phase of drilling was carried out in December 2010 with the completion of six holes totalling 1,818 m numbered TL10113 to TL10118. The majority of the 2010 drill program tested primarily the eastern flank of the Main Zone as well as its down-dip gold potential.

The drilling program was successful by extending the known mineralization and alteration corridor an additional 650 m to the west, 200 m to the east and tested the gold potential of the Main Zone to a vertical depth of 720 m.

10.2.1.13 2011-2012 Diamond Drilling Program

10.2.1.13.1 Overview

Treasury Metals completed three diamond drilling programs from January 17, 2011 to December 13, 2012 with the completion of 192 NQ2 drillholes totalling 70,775 m (Table 10-2). This drilling included 15 re-entry holes to extend historical Teck drillholes (Krocker and Wolfe, 2013). The objective of this drilling was three-fold:

- to confirm and increase the confidence level of indicated gold resources at Goliath
- to locate additional gold mineral resources at depths no more than 400 m from surface in and around the Main Zone focusing on the western shoot and on the eastern flank of the Main Zone; several former Teck holes were re-entered in 2012 to test the gold potential of the C Zone
- to test new exploration targets that reside on strike with the Goliath deposit to the northeast following the known alteration corridor and other potential targets elsewhere on the property (reconnaissance exploration drilling)

Drilling was contracted to Distinctive Drilling. The 2011 holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 312° to 005° and dips ranging from -50° to -87°. Most of the 2011 drilling was concentrated in the eastern portion of the main resource deposit. The new drilling data collected from the main deposit was integrated into the new mineral resource estimate prepared by A.C.A. Howe International Limited PEA in 2010 and updated resource estimate in 2011 (Roy and Trinder, 2011; Roy, 2010).

According to Krocker and Wolfe (2013), compilation work indicated that there was approximately 11.5 km of potential strike length of the alteration corridor that hosts the Goliath deposit heading east throughout the remainder of the property to the far northeast corner of the property claim block. The folded stratigraphy (nose area) is clearly illustrated by the Fugro airborne magnetic survey data.

The 2012 drilling program including further drilling of the main resource deposit and exploration the gold potential of this 11.5 km proposed alteration corridor. A reconnaissance exploration drilling program was initiated to:

- drill test the northeast strike extension of the main deposit in areas where Teck had previously intersected some high gold assay values (Parcel 0138 and legacy claims 1119559 and 1119560)
- drill test the large fold nose centred around claim 1144580 where F2 folds were thought to possibly concentrate gold mineralization (holes TL12244 to TL12254)
- explore for similar Goliath deposit geology utilizing a north-northwest-trending fence of four holes (covering legacy claims 3017880 and 1144553) to test 1,200 m of prospective stratigraphy to a vertical depth of 300 m where alteration and gold mineralization was anticipated to occur (holes TL12266, TL12262, TL12271 and TL12277)

Hole azimuths for the 2012 drilling ranged from 320°to 360° with hole dips ranging between -45° and -70°.

10.2.1.13.2 2012 Drilling Results

- Highlights of the 2012 drilling include the following:
- Hole TL12245 intercepted 2.27 g/t Au and 2.5 g/t Ag over a sample length of 3.0 m (51.0 to 54.0 m)

- Hole TL12235 drilled to test the westernmost strike extension of the main resource area mineralization returned 1.05 g/t Au and 1.25 g/t Ag over a sample length of 3.32 m (199.18 to 202.5 m) within the C Zone
- Re-entry hole TL148-12RE assayed 17.13 g/t Au and 9.0 g/t Ag over 1.5 m from 201.0 to 202.5 m within a lower grade C Zone of 1.05 g/t Au and 1.2 g/t Ag from 172.5 to 202.5 m
- Hole TL164-12RE intersected 5.87 g/t Au and 9.26 g/t Ag over a sample length of 17.13 m (485.31 to 502.44 m) including 18.64 g/t Au and 26.94 g/t Ag over 5.2 m (485.31 to 490.50 m) with visible gold
- Hole TL12293 returned 2.47 g/t Au and 2.70 g/t Ag over a core length of 10.65 m (33.25 to 43.90 m) including 6.65 g/t Au and 7.0 g/t Ag over 2.25 m (33.25 to 35.50 m) near surface in the C Zone

The most northwest exploration fence hole TL12266 on legacy claim 1144553 returned 2.62 g/t Au and 2.48 g/t Ag over a core length of 2.1 m (336.16 to 338.25 m), including 3.67 g/t Au over 1.0 m (337.25 to 338.25 m), hosted in an MSS unit surrounded by BMS rocks in association with elevated pyrite and trace chalcopyrite. The other three holes to the south did not return any significant assays. These results clearly demonstrate that the alteration corridor hosting gold mineralization is still present in the eastern portion of the Goliath property.

Two exploration drillholes (TL12247 and TL12255) intersected several massive to semi-massive sulphides, mostly consisting of pyrrhotite and pyrite bands up to 30 cm wide hosted in mafic volcanoclastic amphibolite rocks with minor meta-sedimentary rocks. These holes were collared on claim 1119545 in the nose of the regional fold structure. Hole TL12247 intersected several 20 to 30 cm wide semi-massive sulphide intervals containing predominantly pyrrhotite with lesser amounts of pyrite from 291.0 to 343.0 m. The second hole intersected seams and stringers of massive sulphides hosted in biotite schist and amphibolite rocks within seams 1 to 10 cm thick. The sulphide enriched units did not contain any significant base metal mineralization. However, hole TL12247 returned 17.52 g/t Au and 2.0 g/t Ag over a sample length of 1.5 m (22.5 to 24.0 m) in a metasedimentary rock and 4.86 g/t Au and 2.0 g/t Ag over 1.0 m (103.0 to 104.0 m) in a biotite mica schist.

10.2.1.14 2013 Drilling

From January 7, 2013, to February 26, 2013, Treasury Metals completed 48 NQ2 diamond drillholes totalling 7,773 m. This program consisted of 41 holes numbered TL13296 to TL13336 and seven re-entry holes on former Teck drillholes.

The primary objective of the drilling program was to further delineate the C Zone within the proposed open pit to convert inferred gold resources to the indicated resource category and to add ounces to the open pit. Drilling was focused along the main deposit over a strike length of 1.5 km. Additional exploration work focused on the C Zone high-grade gold shoot discovered in the central part of the Goliath deposit intersected approximately 50 m after the Main Zone mineralization. A re-interpretation of the geology concluded that the re-entry holes were required in order to extend the Teck holes past the Main Zone to test the gold potential of the C Zone that was largely unknown during the Teck drilling programs in the 1990s. The C Zone mineralization within MSS rocks usually starts downhole around 30 to 60 m past the Main Zone.

This drill contract was awarded to Distinctive Drilling. Holes were drilled north with azimuths ranging from 355° to 045° with the exception of hole TL13315 that was drilled south at 190°. Collar dips ranged from -45° to -80°.

A number of significant C Zone intersections were reported on company press releases. It was concluded that drilling of the proposed open pit mine shell was successful in providing significant gold intersections of the central shoot of the C Zone and in adding ounces to the resource inventory and reducing overall waste to potential ore stripping ratios, especially in the eastern portion of the deposit. The hole extensions also lead to the discovery of several new mineralized zones, including the B Zone intercepts hosted in the BMS unit located between the Main Zone and C Zone.

At the completion of the program, Treasury Metals performed a gap analysis to determine what further diamond drilling would be required for future resource conversion from inferred to the indicated classification within the proposed open pit design focusing on the Main and C Zones to propose an expanded 2014 infill diamond drilling program.

10.2.1.15 2014 Drilling, Phase I

In 2014, Treasury Metals completed Phase I diamond drilling program from January 23, 2014 to June 23, 2014. A total of 42 NQ2 holes were drilled for a total of 10,294 m. This drilling consisted of 35 holes numbered TL14337 to TL14371, five re-entry holes of both Teck and Treasury Metals historical holes and three wedge holes turned off of Treasury Metals hole TL0855 previously drilled in 2008 (Table 10-1). Drillhole TL14363 was abandoned at a depth of 50 m. None of the core in that hole was mineralized.

This program consisted of infill and expansion drilling of the Main and C Zones, further delineation of the new high-grade zone discovered in the central portion of the C Zone and exploration drill testing of targets on its Norman property acquisition, located east of the deposit, which Treasury Metals purchased the surface rights to in 2014 (holes TL14337 and TL14338). The Norman property is contiguous to and located along strike and down dip of the eastern end of the mineral resource at Goliath. Prior to that purchase, Treasury Metals was not allowed surface easement on that property. The new acquisition allowed for the first-time access for drilling on an additional 1.6 km of potential deposit strike length given that the resources defined at that time were interpreted to project towards the northeast portion of this new ground.

This program focused considerably on both exploring and developing the C Zone target both near surface and at depth to add to potential open pit and underground resources. The purpose of the re-entry holes was to extend drillholes to evaluate the C Zone where these original holes were initially terminated after the Main Zone. Further delineation efforts of the Main Zone were also implemented to tighten grades and extend limits of known mineralization within the westward plunging shoots, which included additional infill drilling.

The drill contract was awarded to North Star Drilling. The majority of the holes along the main deposit were drilled north with azimuths ranging from 320° to 005° with the exception of hole TL14356 that was drilled southeast at 145° in the central portion of the deposit. Collar dips ranged from -49° to -77°.

Highlights of the drilling program include the following notable intersections of the C Zone:

- TL14343: 4.32 g/t Au and 32.50 g/t Ag over 3.0 m (16.3 to 19.3 m) in the western portion of the C Zone
- TL14346A: 4.69 g/t Au and 6.67 g/t Ag over 6.4 m (317.0 to 323.4 m) including 27.23 g/t Au and 29.0 g/t Ag over 1.0 m (319.4 to 320.4 m) in the western area of the C Zone
- TL14349: 2.2 g/t Au and 3.48 g/t Ag over 9.3 m (112.7 to 122.0 m) approximately 30 m below hole TL14350
- TL14350: 5.39 g/t Au and 14.59 g/t Ag over 6.7 m (79.33 to 86.00 m) including 28.41 g/t Au and 93.0 g/t Ag over 1.0 m (81.33 to 82.33 m) was intersected in the C Zone at a vertical depth of 60 m from surface
- TL14356: 2.69 g/t Au and 8.87 g/t Ag over 13.5 m (111.5 to 125.0 m) in the C Zone that was drilled down dip on the mineralization
- Wedge hole TL0855W2b: a step-out exploration hole that that intersected 3.64 g/t Au and 2.5 g/t Ag over 5.75 m (561.50 to 567.25 m) in the C2 sub Zone with visible gold located 36 m west of previous C Zone hole TL164-12RE (18.64 g/t Au over 5.2 m reported above)
- TL161-14RE: 4.94 g/t Au and 44.0 g/t Ag over a sample length of 4.0 m (485.0 to 489.0 m).

At the conclusion of drilling program, Treasury Metals determined that the C Zone remained a "high priority" exploration target that remained open to the west and down plunge at depth.

Two exploration holes were drilled on the Norman ground collared on land Parcel 0141 with only one gold assay intersection. Hole TL14337 was targeting an EM anomaly identified from the Fugro airborne geophysical survey as well as testing the potential to intercept down dip MSS mineralization intersected by nearby Teck hole TL272 that returned 9.47 g/t Au over a sample length of 1.1 m. However, this hole did return an isolated assay of 2.79 g/t Au over a sample length of 1.0 m (444.5 to 445.5 m) hosted in a BMS unit with patches of moderate to strong sericite alteration in associated with elevated concentrations of copper (98 ppm Cu) and zinc (761 ppm Zn). It is possible that this hole just intersected the fringe of alteration located just south of the main alteration corridor that hosts the Goliath mineralization. A strong magnetic iron formation containing both magnetite and pyrrhotite were intersected from 118.0 to 120.0 m and the core was determined to be very conductive using a multi-meter resistivity instrument which was most likely the source of the EM target.

A second drillhole TL14338 drilled further to the south was found to be meta-sedimentary rocks with a small iron formation unit intersected from 74.0 to 89.0 m containing patches of blebby pyrrhotite and pyrite. This hole did not return any significant gold assays.

10.2.1.16 2014-2015 Drilling

The Phase II drilling program on the Goliath property was completed between November 27, 2014 and March 17, 2015. A total of 31 NQ2 holes were drilled for a total of 8,769 m. Twenty-nine new holes were drilled numbered TL14372 to TL15402 and two re-entry holes (TL14373-15RE and TL14377-15RE) were extended to evaluate the gold potential of the C Zone (Table 10-1).

This drilling program was initiated for the purpose of resource category conversion and expanding known gold mineralization by drill testing high-grade gold intercepts down plunge and along the perimeter of the gold-bearing shoots outside of the main shoots to complete the current mineral resource update. The program focused on further developing and expanding the resource potential of the C Zone and Main Zone mineralization and Western shoots at depth in areas that had not been previously drill tested. A short two-hole exploration drilling program was also completed to test one of the best gold MMI anomalies defined by the 2014 soil sampling program.

The drill contract was awarded to Downing Drilling. In February 2015, a second drill was added accelerate the drilling program. This program focused predominantly along a 1.6 km strike length of the main resource deposit with holes drilled north at azimuths ranging from 325° to 002°. Collar dips ranged from -45° to -79°.

Significant Main Zone Intersections consisted of the following:

- TL14372 returned an interval of 3.86 g/t Au and 1.67 g/t Ag over 4.5 m (267.0 to 271.5 m) through the western Main Zone shoot.
- TL14374 intersected the western Main Zone shoot containing an interval with visible gold that assayed 199.75 g/t Au and 13.25 g/t Ag over 2.0 m (234.5 to 236.5 m). This hole was drilled around 41.0 m down plunge of the same zone tested by hole TL11204A that returned 17.83 g/t Au over a sample length of 6.0 m (223.5 to 229.5 m).
- TL14375 returned 4.87 g/t Au over 3.5 m in a Hanging Wall Zone from 133.0 to 136.5 m and then intersected 3.81 g/t Au and 8.38 g/t Ag over 8.0 m (185.0 to 193.0 m) through the Main Zone (western shoot).

- TL15396 intersected a well mineralized and quartz veined unit that returned 7.93 g/t Au and 43.57 g/t Ag over a sample length of 2.74 m (45.00 to 47.74 m) at a depth of just 36.0 m vertically from surface in the Main Central Zone. This result is within the proposed reserve pit and came from an area considered to contain low gold concentration.

In an area located 400 meters west of the main proposed pit, Treasury Metals completed seven infill holes to discover and potentially delineate additional shallow open pit-able resources. The program was following up on TL 14367, which intersected 12.8 m at 2.71 g/t (68.0 to 75.0 m) in the Main Zone at a vertical depth of 52 m identified by the 2014 Phase I program. Hole TL15400 returned 6.68 g/t Au and 1.97 g/t Ag over a sample length of 3.6 m (23.4 to 27.0 m) in a Hanging Wall (HW) Zone at a depth of 21.0 m from surface. Main Zone intersections included holes TL15395 that returned 1.43 g/t Au and 1.44 g/t Ag over 8.0 m (107.0 to 115.0 m), and hole TL15397 that assayed 2.44 g/t Au and 0.5 g/t Ag over 4.6 m (M1: 109.4 to 114.0 m) followed by 6.20 g/t Au and 0.5 g/t Ag over 2.0 m (M2: 120.0 to 122.0 m). The latter hole also returned the best C Zone (C2) intersection of 2.07 g/t Au and 0.5 g/t Ag over a sample length of 2.0 m (189.0 to 191.0 m).

The B Zone has been previously intersected by other historical holes throughout the deposit that have also returned significant gold assays. This program, including the 2015 infill core sampling program, further emphasized the importance of the B Zone located in the BMS rocks situated between the Main Zone and C Zone and their potential to add additional gold ounces to the Goliath deposit. In the 2015 drilling, hole TL15-390B intersected the B Zone in BMS rocks with no significant base metal mineralization but containing coarse visible gold on the selvage edge of a well mineralized grey glassy quartz vein.

Two exploration holes numbered TL15401 and TL15402 were drilled just northeast of Tree Nursery Road on claim 1145301 to test the gold potential of a “high priority” mobile metal ion (MMI) Anomaly P in iron formation. This was a moderately strong gold (RR=60) and copper anomaly that occurred in association with weak silver and arsenic RR’s. Treasury Metals interpreted that F2 structures at the main resource deposit could be possibly extrapolated northeast to potentially intersect this target anomaly.

Both holes were drilled as a fence across the target anomaly and they intersected a series of iron formational units separated by strong to moderately garnetiferous metasedimentary rocks (MSED) that were locally weakly magnetic. Small sections of chert-magnetite banded iron formation (BIF) were also recorded. The iron formation was periodically intercalated with chloritized amphibolite rocks, which could represent mafic volcanoclastic rocks or inter-pyroclastic flows.

A bleached silicified and possibly weakly sericite altered zone was intersected by both drillholes at the point where the gold MMI high was centred. All cores were split for assay. None of the samples returned any significant gold or base metal assays.

10.2.1.17 2016 Drilling

A single-phase diamond drill program on the Goliath property was completed from August 24, 2016 to January 15, 2017. A total of 28 NQ2 holes were drilled for a total of 12,154 m. Eighteen new holes were drilled numbered TL16403 to TL16420, including one wedged hole (TL16415W1) in order to recover 2 m of lost core in the main zone of mineralization. In this program, ten drillholes were abandoned due to bad ground conditions causing the drill to deviate from the planned pierce points.

The objective of this drilling program was to:

- convert and increase indicated gold resources at Goliath through means of infill drilling
- locate and identify additional gold resources at depth with focus on the down plunge potential of the eastern, western, and central high-grade chutes of the Main Zone as well as the C Zone chute

- further delineation of the new high-grade zone discovered in the central portion of the C Zone
- to continue drill testing high-grade gold intercepts down plunge to depth's up to 723.0 m (TL16404D) to potentially add to underground resources.

Drilling was contracted to George Downing Estate Drilling Ltd. The 2016 holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 345° to 357° and dips ranging from -67° to -83°. Most of the drilling was concentrated along the peripherals of known high-grade chutes of the Main Zone and the C Zone to further delineate the chutes and convert mineral resources from inferred to indicated. The remainder of the drilling focused on testing the down dip potential of the high-grade chutes to add additional ounces of gold to the current resource. The average core recoveries were excellent and the RQD was good.

Drilling of the proposed underground mineral resource was successful in providing significant gold intersections of the central chute of the Main Zone and the C Zone. In addition to the Main Zone there was also significant gold intercepts occurring in the Hanging Wall and B Zones. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from the inferred to the indicated category and assist in further delineating the high-grade chutes of the Main Zone and C Zone.

Out of a total of 5,078 individual samples the highest gold assay obtained from the 2016 drilling program was from drillhole TL16405 that returned 63.1 g/t Au over a sample length of 1.0 m. Additional significant intervals from the 2016 drill program include:

- TL16403B intersected 5.44 g/t Au and 5.90 g/t Ag over 1.0 m (529.0 to 530.0 m) followed by 3.94 g/t Au and 4.28 g/t Ag over 4.0 m (541.0 to 545.0 m) as well as 14.3 g/t Au and 6.60 g/t Ag over a 1.0 m sample in the Main Zone that contained visible gold. This hole is located in the main zone central chute approximately 475 m from surface.
- TL16405 encountered several specks of visible gold in the B Zone returning 13.3 g/t Au and 6.68 g/t Ag over a sample length of 5.15 m (582.85 to 588.0 m) including 19.27 g/t Au and 9.51 g/t Ag over 3.45 m (582.85 to 586.3 m).
- TL16410 returned 10.95 g/t Au and 12.44 g/t Ag over a sample length of 7.0 m (544.0 to 551.0 m) including 24.47 g/t Au and 22.7 g/t Ag over 3.0 m (547.0 to 550.0 m). Visible gold was observed within this interval which was centrally located in the M2 portion of the Main Zone.
- TL16413 returned 6.54 g/t Au and 7.04 g/t Ag over a sample length of 11.50 m (657.0 to 668.5 m) including 11.32 g/t Au and 9.38 g/t Ag over 5.5 m (663.0 to 668.5 m) in the M2 footwall of the Main Zone. This hole was drilled to a depth of 717.0 m to test the down plunge potential of the eastern chute.

At the conclusion of the drilling program, and given the excellent gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the C Zone remained a high priority exploration target that remained open to the west and down plunge at depth.

10.2.1.18 2017 Drilling

Treasury Metals conducted a diamond drill program from January 10, 2017 through to October 31, 2017. A total of 43 NQ2 drillholes totalling 8,516 m was completed, not including two holes that were abandoned due to poor sub-surface conditions. A total of 6,176 samples were taken over the span of the year, not including a total of 686 blanks and standards. The objectives of the drilling program were:

- to conduct condemnation and exploration drilling of areas where proposed mining infrastructure will be situated, including milling, tailings storage facility and mining operations
- to convert and increase indicated gold resources at Goliath property through infill and expansion drilling
- to locate and identify additional gold resources at depth with focus on the down plunge potential of the eastern, western, and central high-grade chutes of the Main Zone as well as the C Zone chute.

This drilling program consisted of condemnation/exploration drilling along strike of the main resource as well as infill and expansion drilling of the Main and C Zone, further delineating the extents of the high-grade chutes. The program also included drill testing high-grade gold intercepts down plunge to depths up to 774.0 m (TL17412A) to potentially add to underground resources.

Drilling was contracted to George Downing Estate Drilling Ltd. The holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 319.2° to 355° and dips ranging from -48.6° to -82.9°. The condemnation/exploration drilling was concentrated along strike, northeast of the known resource and outside of the current proposed open-pit. The purpose of the condemnation/exploration program was to drill test areas along strike of the main resource where proposed mining infrastructure is to be located, including milling, tailings storage facility and mining operations. In addition, to test locations of potential gold chutes interpreted by Exploration Manager Paul Dunbar from historical drillhole compilation and newly prepared longitudinal sections of the EAC. The condemnation/exploration drilling was comprised of a series of shallow drillholes ranging in depth from 57.0 m to 204.0 m.

Treasury Metals spent the remainder of the drilling in 2017 focused on infill and resource conversion around the perimeter of known high-grade chutes of the Main Zone and the C Zone. The purpose of this program was to further delineate the chutes and convert resources from the inferred to indicated classification, while also testing the down dip extents of the mineralized chutes. The average core recoveries were excellent and the RQD was good.

Infill drilling of the proposed underground resource was successful in providing significant gold intersections of the Main Zone and C Zone. In addition to the Main Zone and C Zone there was also significant gold intercepts occurring in the Hanging Wall, D Zone and E Zone. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from the inferred to the indicated classification and assist in further delineating the high-grade chutes of the Main Zone and C Zone.

Two infill/resource conversion drillholes numbered TL17422 and TL17460 were drilled on mining patent 47122 and mining lease 109717. The purpose of these infill holes was to test the gold potential of the eastern most edge of the western high-grade chute of the Main Zone and to test the down dip potential of the central chute of the Main Zone. TL17422 intersected good gold assays within the Main, C, and B Zones, therefore successfully expanding the Main Zone’s western chute to the east and warranting further drilling to test the continuity. TL17460 also intersected good gold grades within the Main Zone and was able to expand the continuity of the high-grade central chute further down dip. It was determined from TL17460 that the central chute of the Main Zone remains open down dip and remains to be a high priority target for future drill programs. Infill holes TL17445 and TL17459 were drilled just southwest of where the main hydro line intersects with Tree Nursery Road on mining patent 46017. These are both near surface holes with the goal of identifying and expanding the C Zone resource along strike to the east of the known deposit. TL17445 returned a number of anomalous gold assays within the C zone and the highest gold sample of the program within the D Zone. TL17459 intersected a single high-grade gold assay in the C Zone. Both of these holes display that there is some continuity down dip between the high-grade lenses within the eastern side of the C Zone, but more investigation is required to determine their trend and extent.

The condemnation and exploration drilling took place along strike of the main resource area, stepping out to the Northeast over a distance of approximately 1.4 km from the current known resource. Low-grade gold intersections were encountered to the northeast of the proposed tailings pond in what was previously a sparsely drilled portion of the property. Hole TL17442 and TL17443 intersected discontinuous low-grade mineralization confirming the grade observed in the 2011 drill program. The near surface mineralization appears to be in two poorly define zones extending 200 m below surface with a short strike length on 50 to 60 m..

Out of a total of 6,176 individual samples the highest gold assay obtained from the 2017 drilling program was from drillhole TL17445 that returned a single assay of 33.3 g/t Au over a sample length of 1.0 m corresponding to the D Zone. Additional significant intervals from the 2017 program include:

- TL17422 intercepted 3.67 g/t Au and 3.58 g/t Ag over a sample length of 4.0 m (348.0-352.0 m) in the Main Zone. This hole also intersected 7.13 g/t Au and 6.20 g/t Ag over a sample length of 0.9 m (392.0-392.9 m) in the B Zone. In the C Zone this hole intersected 4.10 g/t Au and 26.46 g/t Ag over a sample length of 5.0 m (457.0-462.0 m), including 18.2 g/t Au and 119.0 g/t Ag over a sample length of 1.0 m (459.0-460.0 m) which contained several specks of visible gold and electrum.

- TL17445 was targeting the C and D Zones and returned several high-grade assay values. In the C Zone this hole found 9.92 g/t Au and 3.60 g/t Ag over a sample length of 1.0 m (47.0-48.0 m) with several specks of visible gold within a wider zone grading 2.67 g/t Au and 4.49 g/t Ag (43.17-48 m). In the D Zone this hole intersected 16.79 g/t Au and 1.90 g/t Ag over a sample length of 2.0 m (68.0-70.0 m), including 33.30 g/t Au and 2.10 g/t Ag over a 1.0 m sample length (69.0-70.0 m). No visible gold was noted in this interval.

- TL17459 intercepted 13.8 g/t Au and 19.90 g/t Ag over a sample length of 1.0 m (122.0-123.0 m) in the C Zone within a wider zone grading 3.88 g/t Au and 6.58 g/t Ag over 4.0 m (122-126 m).

- TL17460 intersected 4.53 g/t Au and 29.90 g/t Ag over 1.0 m (576.0-577.0 m) in the Hanging Wall. Followed by 3.34 g/t Au and 5.94 g/t Ag over 5.0 m (641.0-646.0 m), including 4.80 g/t Au and 8.83 g/t Ag over 3.0 m (643.0-646.0 m) and 3.41 g/t Au and 56.50 g/t Ag over 2.0 m (663.0-665.0 m), including 6.47 g/t Au and 80.10 g/t Ag over 1.0 m (664.0-665.0 m).

Upon completion of the drilling program and given the excellent gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the C Zone remained a high priority exploration target that remained open to the east and west as well as down plunge at depth.

10.2.1.19 2018 Drilling

Treasury Metals conducted a diamond drill program on the Goliath property from January 8, 2018 through to June 22, 2018, totalling 20,987 m. This consisted of 38 new holes drilled (TL18464 to TL18501), not including 14 holes that were abandoned due to bad ground conditions causing deviation from the intended target. A total of 10,251 samples and 1,139 blanks and standards were tested over the span of the year. The objective of the drilling program was to:

- convert and increase indicated gold resources in the Main and C Zones of the Goliath property, through means of infill and expansion drilling

- investigate the extent of high-grade mineralization found in historic Teck drillholes in the East C Zone

This drilling program consisted of infill and resource conversion drilling within the Main and C Zones and further delineation of the high-grade chutes of each. The program included drill testing of high-grade gold intercepts down plunge of the Main Zone to depths up to 762.0 m (TL18471A) to potentially add to underground resources.

Drilling was contracted to George Downing Estate Drilling and Distinctive Drilling. The 2018 holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 350° to 0° and dips ranging from -58° to -78°. This program consisted of drilling at depths ranging from 195.0 m (TL18480) to 831.0 m (TL18473A) and targeted areas along the outer edges and down plunge of the high-grade chutes in the central, western, and eastern chutes of the Main Zone as well as the C Zone. Additionally, 5,000 m of drilling was conducted on the East C zone area where historic Teck drillholes intercepted moderate to high-grade mineralization. The average core recoveries were excellent and the RQD rock mass quality was good.

Drilling of the underground resource was successful in providing significant gold intersections in both the Main and C Zone. Upon completion of the program, Treasury Metals performed a gap analysis to further determine what diamond drilling would be required for future resource conversion from inferred to the indicated classification and assist in further delineating the high-grade chutes of the Main Zone and C Zones.

Out of a total of 10,251 individual samples the highest gold assay obtained from the 2018 drilling program was from drillhole TL18494 that returned 111 g/t Au over a sample length of 1.0 m within a large zone of lower grade mineralization grading 6.28 g/t Au and 1.71 g/t Ag over 19.0 m (425-444 m). This was drilled as a follow up to Teck drillhole TL205 which intersected 1.0 g/t Au over 23.5 m and is located near the eastern most extent of drilling in the C Zone. Additional significant intervals from the 2018 program include:

- TL18469 intersected 14.88 g/t Au and 5.33 g/t Ag over 6.0 m (558.0-564.0 m), including 79.6 g/t Au and 3.8 g/t Ag over 1.0 m (559.0-560.0 m) in the Main Zone. This hole is situated along the eastern edge of the east chute. Three small specks of visible gold (< 1 mm grain size) was observed between 559.75 m to 559.58 m.

- TL18474 intersected 10.35 g/t Au and 5.89 g/t Ag over a sample length of 7.0 m (445.0-452.0 m), including 64.5 g/t Au and 1.8 g/t Ag over 1.0 m (451.0-452.0 m). This hole was drilled along the eastern edge of the west chute in the Main Zone.

- TL18489 intersected 48.71 g/t Au and 310.67 g/t Ag over a sample length of 3.0 m (542.0-545.0 m), including 145.00 g/t Au and 921.00 g/t Ag over 1.0 m (543.0-544.0 m) in the C Zone in addition to 5.28 g/t Au and 143.00 g/t Ag over 1.0 m (528.4-529.4 m). This hole was drilled at the deepest extent of the C Zone and was successful in confirming the continuity of gold mineralization down plunge. Minor visible gold was observed between 528.4-529.4 m depth and approximately 20 specks of visible gold ranging in size from 1-5 mm was observed between 543.2-543.3 m depth.

- TL18494 intersected 25.20 g/t Au and 3.98 g/t Ag over a 4.50 m sample length (426.0-430.5 m), including 1.0 m (426.0-427.0 m) at 111.00 g/t Au and 11.10 g/t Ag. This drillhole was drilled to investigate nearby Teck drillhole TL205 which intersected 1.0 g/t Au over 23.5 m and is located near the eastern most extent of drilling in the C Zone.

- TL18499A intersected 3.81 g/t Au and 34.65 g/t Ag over 13.0 m (516.0-529.0 m), including 10.17 g/t Au and 120.47 g/t Ag over 3.0 m (516.0-519.0 m) in the Main Zone. This hole was drilled as a follow up to TL18469 and is located on the eastern edge of the east chute within the Main Zone. Visible gold was observed in four small specks (< 1 mm grain size) between 518.4-518.5 m depth.

Upon completion of the drilling program and given the gold grade intersections, Treasury Metals determined that the eastern and western chutes of the Main Zone and the C Zone remained a high priority exploration target that remained open to the east as well as down plunge.

10.2.1.20 2019-2020 Drilling

Treasury Metals conducted a diamond drill program on the Goliath property from November 15, 2019 through to March 7, 2020, totalling 10,135 m. This program consisted of 27 new holes drilled (TL19502 to TL20528), not including six holes that were abandoned due to bad ground conditions causing deviation from the intended target. A total of 6,468 core samples and 680 blanks and standards were tested over the span of the program. The three objectives of the drilling program were to:

- convert and increase measured gold resources in the Main Zone east and central shoots of the Goliath deposit for inclusion as potential estimate ounces for the initial mine life years and for grade control purposes through infill drilling
- convert and increase indicated and inferred gold resources in the C East area through infill and expansion drilling
- investigate and expand the Main Zone east shoot at depth on the eastern side through exploration and expansion drilling

Drilling was contracted to George Downing Estate Drilling Ltd. The 2019 and 2020 holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 350° to 0° and dips ranging from -60° to -78°. This program consisted of drilling intersections at depths from surface ranging from 120 m (TL20528) to 625 m (TL20517). Out of the total program, 3,816 m were for Main Zone measured infill, 4,176 m targeted the C East area, and 2,143 m at depth adjacent to the Main Zone eastern shoot. The average core recoveries were excellent, and the RQD rock mass quality was good.

Out of a total of 6,468 individual samples the highest gold assay obtained from the 2019-2020 drilling program was from drillhole TL20520 that returned 152.0 g/t Au over a sample length of 1.0 m within an intersection grading 51.5 g/t Au over 3.0 m (523.5-526.5 m). This was drilled in the C East area 100 m down dip from hole TL18494 which returned 25.2 g/t Au over 4.5 m including 111.0 g/t Au over 1.0 m. Additional significant intervals from the program include:

- TL19503, also in the C East area, intersected 17.1 g/t Au over 7.0 m including 117.0 g/t Au over 1.0 m
- TL19505, located in the Main Zone central shoot, intersected 9.2 g/t Au over 6.3 m including 13.0 g/t Au over 4.0 m
- TL20517, drilled at depth adjacent to the Main Zone eastern shoot, intersected 4.6 g/t Au over 4.4 m including 13.2 g/t Au over 1.0 m in the Main Zone and 2.4 g/t Au over 6.0 m including 10.6 g/t Au over 1.0 m in a Hanging wall Zone

Highlights of the program are summarized in Table 10-4. In the table, duplicate samples were averaged together to calculate intersection grade; all grades are reported uncut and interval lengths were reported at core length. In general, true width at the Goliath deposit typically range between 70% to 90% of the sample length but can occasionally reach as low of 44% and a high of 96%.

Table 10-4: Highlights of the 2019 -2020 Goliath Drill Program

Drillhole	Zone	From (m)	To (m)	Sample Length (m)	Au g/t
TL19503	Main	356.0	361.0	5.0	2.0*
	C	449.0	456.0	7.0	17.08*
	including	449.0	450.0	1.0	117.00*
TL19505	Main	214.7	221.0	6.3	9.23*
	including	217.0	221.0	4.0	13.02
TL20515	Main	348.0	352.0	4.0	5.38*
	including	348.0	349.0	1.0	20.90*
	C	446.0	462.4	16.4	0.52
	C	477.0	483.1	6.1	0.52
TL20517	HW	454.0	460.0	6.0	2.42*
	including	454.0	455.0	1.0	10.60*
	Main	658.6	663.0	4.4	4.64*
	including	658.6	659.6	1.0	13.20*
TL20518	HW	129.0	140.0	11.0	0.45
	C	403.1	417.1	14.0	0.67*
	including	413.1	417.1	4.0	1.21*
	C	432.7	438.2	5.5	0.70
TL20519	HW	55.7	58.3	2.6	1.32
	Main	308.8	310.5	1.7	0.43
	C	419.4	428.0	8.6	1.30*
	including	427.0	428.0	1.0	7.10*
	C	448.7	452.3	3.6	0.68
TL20520	C	495.0	509.7	14.7	1.19*
	including	507.0	508.0	1.0	8.27*
	C	523.5	526.5	3.0	51.60*
	including	524.5	525.5	1.0	152.00*
TL20521	Main	205.0	231.0	26.0	1.00*
	including	211.0	215.0	4.0	1.44*
	and including	222.0	223.0	1.0	9.89*
TL20522	Main	265.0	280.0	15.0	1.63*
	including	267.0	271.0	4.0	3.95*
	including	269.0	270.0	1.0	9.72*
	Main	285.1	290.0	4.9	2.70*
TL20523	HW	140.0	142.5	2.5	1.47
	Main	221.0	240.5	19.5	4.04*
	including	222.0	234.0	12.0	6.02*
	including	222.0	224.0	2.0	27.30*
TL20525	Main	157.5	166.5	9.0	6.04*
	including	162.5	166.5	4.0	12.92*
	Main	174.0	176.0	2.0	1.69
TL20527	Main	197.0	207.0	10.0	3.59*
	including	204.0	205.0	1.0	18.00*
	Main	219.0	226.0	7.0	7.03
	including	224.0	225.0	1.0	40.6
TL20528	Main	118.4	122.0	3.6	0.84

Note: * Includes metallic screen assays. Source: Treasury Metals (2020).

10.2.1.21 2021 Drilling

Treasury Metals continued the diamond drill program on the Goliath property and between April 2021 and December 2021 drilled an additional 18,155 m. This program consisted of 73 new holes drilled (TL21529A to TL20601). The primarily focus of the drill program was to evaluate areas within the resource which could benefit future studies and mine design. Holes TL21561 and TL21534 targeted gaps in the PEA mine stopes along the Eastern Shoot of the Main Zone which the geological interpretation suggested high-grade mineralization should be present, but historical drilling past the outer edge of the shoots restricted the extension of mineable material. Both returned significant gold mineralization where TL21561 intersected 52.60 g/t Au over 15.0 meters including 870 g/t Au over 0.84 meters, 25.80 g/t Au over 0.74 meters, and 47.50 g/t Au over 0.50 meters; and TL21534 intersected 2.10 g/t Au over 8.00 meters including 3.81 g/t Au over 1.00 meter and 7.77 g/t Au over 1.0 meter.

In addition, drilling was conducted to convert any remaining underground resources from inferred to the indicated classification while expanding the areas of known mineralization. TL21536 intersected 2.31 g/t Au over 14.05 meters including 31.40 g/t Au over 0.80 meters and TL21535 intersected 0.42 g/t over 13.15 meters including 3.59 g/t Au over 0.55 meters. Both intersections are located along the western side of the Eastern Shoot of the Main Zone and will provide greater drill density to help upgrade nearby blocks to the indicated classification.

At the time of this report, assay data were available for 40 of the 73 holes drilled since the PEA providing an additional 11,383 core samples were tested over the span of the program. The three objectives of the drilling program were to:

- 23 holes have been drilled within the resource area focusing on resource conversion with the goal to connect stopes across identified gaps in the mine plan;
- 18 holes have been drilled to the west of the proposed PEA pit to explore near surface potential for pit expansion; and
- 5 holes have been drilled on the Far East exploration target approximately 11 km along strike to the Goliath Deposit where 2012 drilling had encountered felsic volcanic rock similar to Goliath which contained gold mineralization.

Drilling was contracted to George Downing Estate Drilling Ltd. All drill collar locations are initially located with hand-held GPS and final collar locations are verified with DGPS delivering real-time accuracy in the centimeter range. Down hole deviation is controlled with the use of Reflex Gyro readings collected at 15 to 20 m intervals. Core is logged by Treasury Metals employees and data is entered into both Excel and MX Deposits for capture and safe keeping.

The holes were drilled approximately perpendicular to the mineralized zone with azimuths ranging from 350° to 0° and dips ranging from -60° to -78°. Out of the total program, 22,358 m targeted the Goliath Main Zone, 5,527 m targeted the Goliath East area and 1,999 m were drilled to test an exploration target known as the Far East Goliath target. at depth adjacent to the Main Zone eastern shoot. The average core recoveries were excellent, and the RQD rock mass quality was good.

Out of a total of 11,383 samples collected at 1.0 or 1.5 m intervals. Sample intervals were generally broken on geological contacts resulting is some samples being shorter than 1.0 m but not less than 0.1m in length. The highest gold assay obtained from the program was from drillhole TL21561 that returned 79.8 g/t over 9.8 m including 870.0 g/t Au over a sample length of 0.8 m within an intersection of the Main Zone at 383 m down hole depth.

Highlights of the program are summarized in Table 10-5. In the table, all grades are reported uncut and interval lengths were reported at core length. The QP notes that true width at the Goliath deposit typically range between 74% to 90% of the sample length but can occasionally reach as low of 44% and a high of 96%.

Table 10-5: Significant intersections from the 2021 Goliath Drill Program

Drillhole		Zone	From (m)	To (m)	Sample Length (m)	Au g/t
TL21529B		HW	460	465	5	1.06
TL21529B		Main	640.55	646	5.45	0.89
TL21530		HW	99	100.07	1.07	6.78
TL21530		HW	105	111	6	0.64
TL21530		C	462	470	8	0.39
TL21530		C	506.95	511	4.05	0.87
TL21531B		HW	184	188.4	4.4	2.25
	including		186	187	1	8.6
TL21531B		HW	200	211	11	0.47
TL21531B		HW	243	245	2	1.42
TL21532		HW	543	551.6	8.6	0.29
TL21532		HW	568	570.85	2.85	2.57
	including		569	570	1	5.82
TL21532		C	828	834	6	1.27
	including		828	829	1	4.53
TL21533B		HW	528.75	538.15	9.4	0.96
	including		530.65	532.5	1.85	2.57
TL21533B		HW	563	567	4	2.47
	including		563	564	1	8.49
TL21534		Main	244	252	8	2.1
	including		245	246	1	3.81
	and including		251	252	1	7.77
TL21534		C	347.5	351.5	4	0.65
TL21534		C	380.9	384.2	3.3	0.65
TL21535		Main	508.85	522	13.15	0.42
	including		516.9	517.45	0.55	3.59
TL21535		C	588.02	591.15	3.13	0.87
TL21536		Main	639.9	653.95	14.05	2.31
	including		651.5	652.3	0.8	31.4
TL21539		West - Main	122.05	146.5	24.45	0.75
	Including	West - Main	135	146.5	11.5	1.28
	Including		135	136	1	6.01
TL21549		Main	612.2	621	8.8	1.56
	Including		612.2	613	0.8	12.1
TL21553		Main	711.3	723.1	11.8	1.51
	including		711.3	712.3	1	6.87
	Including		722.1	723.1	1	6.13
TL21559		West - Main	162	167.5	5.5	9.55
	Including		164.5	166	1.5	24.4
	Including		166	167.5	1.5	8.6
TL21559		West - C	212	237.7	25.7	0.57
	Including		237	237.7	0.7	7.8
TL21566		West - C	194.9	207.5	12.6	1.08
	Including		206	207.5	1.5	4.94
TL21561		HW	169.5	177	7.5	0.34
TL21561		HW	320	324.6	4.6	0.63
TL21561		Main	375	390	15	52.6
	including		375	383.6	8.6	1.37
	and including		383.6	384.44	0.84	870
	and including		384.44	385.18	0.74	25.8
	and including		385.18	385.68	0.5	47.5

10.3 Goldlund Deposit Drilling

Diamond drilling on the Goldlund project has been carried out since the 1940s. There is a total of 1,933 drillholes totalling 250,320 m of drilling in the Goldlund database, including 480 drillholes totalling 18,626 m of underground drilling. Also included are 188 trenches totalling 1,442 m and 247 underground channel samples totalling 4,130 m. The underground channels and trench samples were excluded from the mineral resource estimate.

10.3.1 Historical Drilling

Very little information is available on the drill programs at Goldlund prior to the work done in 2007. Between 1941 and 1947 a total of 183 holes were drilled, totaling 16,482 m of E-X core. Between 1976 and 1989 a total of 323 holes were drill for a total of 29,645 m.

Drill logs, assay summaries, and assay certificates for the majority of these historical drillholes are available and were compiled into a digital format to support the mineral resources estimate. A summary of the historical work is described in Section 6.

The procedures of the various historical drilling programs are not documented. Sampling details for the historical programs prior to 2006 have not been verified by the Qualified Person for this section of the report. No QA/QC programs are believed to have been conducted at that time. The legible quality of the diamond drill logs, and assay certificates has allowed for the construction and validation of the historical drilling, sampling, and assay results in the drillhole database.

10.3.2 2007 – 2008 Drilling

In 2007 and 2008, Tamaka carried out a drilling program of 109 holes totalling 29,259 m of surface drilling on the project. The drilling was completed by Bradley Brothers of Timmins. All holes were drilled NQ (47.6 mm) and NQ2 (50.6 mm) and all drilling runs were in 10 ft intervals (3 m). The collars were initially spotted with a hand-held GPS and the final completed collars were surveyed by a land surveyor from Dryden. Downhole surveys were completed using the Reflex Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore system is not affected by the magnetic influence in the surrounding environment.

The NQ or NQ2 core was received at the logging facility and the run lengths were measured to confirm the block markers. The core recovery and RQD were measured and then entered into a Microsoft Excel template. Magnetic susceptibility measurements were taken at 0.5 m intervals using a hand-held unit. The core was photographed both wet and dry. Logging of the lithology, structure, alteration, and sulphide content were recorded in an Excel spreadsheet template. Sample lengths were marked and range from 0.20 to 1.5 m, but do not cross lithological boundaries.

The samples were taken continuously from collar to the end of the hole. The drill core was sawn in half, with one half placed in a plastic sample bag, and the other half returned to the core box. One of the sample tags was placed in the sample bag, while the other tag was stapled into the core box. The sample bags were then sealed with fibre tape. QA/QC samples were inserted into the sample stream and the samples were placed in rice bags, then sealed and stored in the secure logging facility until shipment. The samples were delivered by a Tamaka employee to Manitoulin Transport in Dryden, Ontario for delivery to the Accurassay Laboratory in Thunder Bay, Ontario. The laboratory returned all coarse rejects and pulps to Tamaka for safe and secure storage at the project.

10.3.3 2011 Drilling

In 2011, Tamaka carried out a drilling program of 31 holes totalling 12,782 m of surface drilling. The drilling was completed by C3 Drilling of Ithaca, New York. All holes were drilled NQ (47.6 mm) and all drilling runs were in 10 ft intervals (3 m). The drilling program was managed independently by geologists employed by Fladgate Exploration based in Thunder Bay and monitored by the Vice President of Exploration for Tamaka.

The collars were initially spotted using a hand-held GPS and the final completed collars were surveyed with a handheld GPS. Downhole surveys were completed using the Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore® system is not affected by magnetic influence in the surrounding environment.

Drill core was delivered by C3 Drilling to the Tamaka core logging facility located on site and the run block measurements were checked. The core recovery and RQD were recorded, and magnetic susceptibility measurements were made using a hand-held instrument for each 3 m length of core. Drillholes K11-110 to K11-120 were logged into Microsoft Excel spreadsheets, while from K11-121 onwards, holes were logged into a Gemcom© Gemslogger (Gemslogger) Microsoft Access database. A geologist logged the core, recording lithology, alteration, structure, and mineralization in Gemslogger on the spreadsheet, marking the intervals with a grease pen. Sample lengths range between 0.2 and 2.6 m in length, with an average sampling length of around 0.7 m. The samples did not cross lithological boundaries and at least two shoulder samples are taken on either side of the mineralization. Core was photographed after logging and sampling was completed, both wet and dry.

The core was sawn using a top-mounted diamond saw blade. Half of the core was placed in a sample bag while the other half was replaced in the core box. The QA/QC samples consisting of standard reference material (SRM), blanks and duplicates were inserted into the sample stream. For field duplicates, the remaining half of the core was quarter split and placed in a sample bag. For coarse duplicates, a sample tag was placed in an empty sample bag. The sample tag was stapled to the inside of the sample bag and the sample bag is stapled sealed. The samples were placed into rice bags and stored in crates awaiting shipment. Crates were shipped every week to Accurassay in Thunder Bay by Manitoulin Transport. The laboratory returned all course rejects and pulps to Tamaka for storage at the project.

10.3.4 2013-2014 Drilling

In 2013 to 2014, Tamaka carried out a drilling program of 24 holes totalling 9,000 m of surface drilling. The drilling was completed by C3 Drilling of Ithaca NY and North Star Drilling of Thunder Bay. All holes were drilled NQ (47.6 mm) and all drilling runs were in 10 ft (3 m) intervals. The drilling program was managed independently by geologists employed by Fladgate Exploration based in Thunder Bay and monitored by the Tamaka employees (Tamaka, 2016).

The collars were initially spotted with a hand-held GPS and the final completed collars were surveyed with a differential GPS. The downhole surveys were completed using the Reflex Maxibore® tool. Survey readings were collected at 3 m intervals from the top of the hole. The Maxibore® system is not affected by magnetic influence in the surrounding environment.

The NQ core was received at the logging facility and the run lengths were measured to confirm the block markers. The core recovery and RQD were measured and then entered into a Microsoft Excel template. Magnetic susceptibility measurements were taken at 0.5 m intervals using a hand-held unit. The core was photographed both wet and dry. Logging of the lithology, structure, alteration, and sulphide content were recorded directly into a Microsoft Excel spreadsheet template. Sample lengths are variable and range from 0.20 to 1.5 m; however, the samples do not cross lithological boundaries.

The drill core selected to be sampled was sawn in half with one half placed in a plastic sample bag, with the other half returned to the core box. One of the sample tags was placed in the sample bag while the other tag was stapled into the core box. The sample bags were then sealed with fibre tape. QA/QC samples were inserted into the sample stream and the samples were placed in rice bags, then sealed and stored in the secure logging facility until shipment.

The samples were delivered by a Tamaka employee to Manitoulin Transport in Dryden, Ontario for delivery to the Accurassay Laboratory in Thunder Bay, Ontario. The laboratory returned all coarse rejects and pulps to Tamaka for safe and secure storage at the project.

10.3.5 First Mining 2017 – 2020

The most recent drilling was carried out by First Mining Gold Corp. in three phases of drilling between 2017 and 2020. Phase I was completed between January 2017 and July 2017 and targeted Zone 7 of the deposit. Phase II was completed between June 2017 and March 2018 and primarily targeted Zone 1. Phase III was completed between November 2019 and July 2020 and targeted Zones 2 and 3. The programs were designed to better understand and define the potential resource in the Goldlund deposit by infill drilling.

A total of 100 infill holes were drilled during the Phase I drill program, for a total of 24,299 m targeted mainly at Zone 7. The primary goal of this drilling campaign was to upgrade Inferred resources at Zone 7 into a higher resource category and to better define the geology and gold mineralization. Of the 100 holes drilled, 86 holes intersected intervals of significant gold mineralization.

The second phase of drilling by First Mining consisted of 38 infill holes for 14,961 m. The program was designed to provide greater confidence in the gold mineralization within Zone 1. While 33 out of the 38 drillholes intersected gold mineralization, this phase of drilling was limited in extent in order to avoid intersecting historic underground workings. Areas of Zone 1 have previously been mined and therefore contain several levels of existing underground workings. Accordingly, new holes had to be positioned to avoid drilling through existing levels or stopes, and as a result some of the holes may not have reached the key mineralized zones which occur closer to the footwall of the zone.

The third and last phase of drilling by First Mining consisted of 48 holes totalling 8,958 m. this program was mainly targeted at providing better confidence in the gold mineralization within Zones 2 and 3 of the deposit.

10.3.5.1 Drilling

All three drill programs were conducted by Rodren Drilling of Manitoba with HQ sized core. The drillhole collar locations were initially surveyed using a handheld Garmin GPS, then after drilling was completed, collars were surveyed by differential GPS. After Treasury Metals acquired the Goldlund project they independently surveyed the collar locations using a Trimble GeoExplorer 6000 Series, Model: 88950 GPS and found that all but one drillhole (GL-19-034) had similar locations to the planned drillhole collars. The location of drillhole GL-19-034 has been corrected in the drillhole database used for the mineral resources estimate.

Down-the-hole surveying was done using an EZ Gyro survey tool to determine the deviation of inclined drill paths. The path of the drillhole was surveyed upon completion of the hole, with readings taken approximately every 30 m. There were optimized readings (consisting of three consecutive readings taken at the same interval and averaged together), taken at the top and bottom of the drillhole.

The core logging methodology and QA/QC procedures were overseen by Mr. Miro Mytny, P.Geo., Senior Exploration Manager for First Mining. The logging procedures applied during the 2019 and 2020 drilling programs at Goldlund are summarized below. Figure 10-5 displays photographs of the core logging area on the left-hand side and the core sampling area with a diamond saw on the right-hand side of the figure.

Figure 10-5: Core Logging and Sampling facilities at the Goldlund Exploration Camp

Source: SRK, 2023.

10.3.5.2 Logging

The HQ diameter (63.5 mm) drill core was cleaned, and the run blocks checked. After this, the runs were measured for recovery. The recovery percentage was then used to mark off the adjusted meters within the run. The core was logged for lithology, alteration, minerology, veining, and structure, and entered into the DH Logger® software, which synchronizes with First Mining's central Fusion® SQL drilling database. The RQD was measured and recorded in an Excel sheet, for importing into the Datamine DH Logger® software. The core was photographed twice, both dry and wet.

10.3.5.3 Drill Core Sampling

Most samples collected were in one-meter intervals except at lithological contacts and in zones of poor recovery, where sample size was adjusted accordingly. The core was sawn in half on site, with one half bagged and labelled to be sent for assay. The remaining half core was placed in core boxes which were stored in a secure on-site facility to serve as a permanent record.

For field duplicates, the core was quartered and one quarter was sent for regular assay, while the other quarter was sent as a duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the pulverized sample to run a duplicate assay.

Standards and blanks were inserted in the sample stream at the required intervals. Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates.

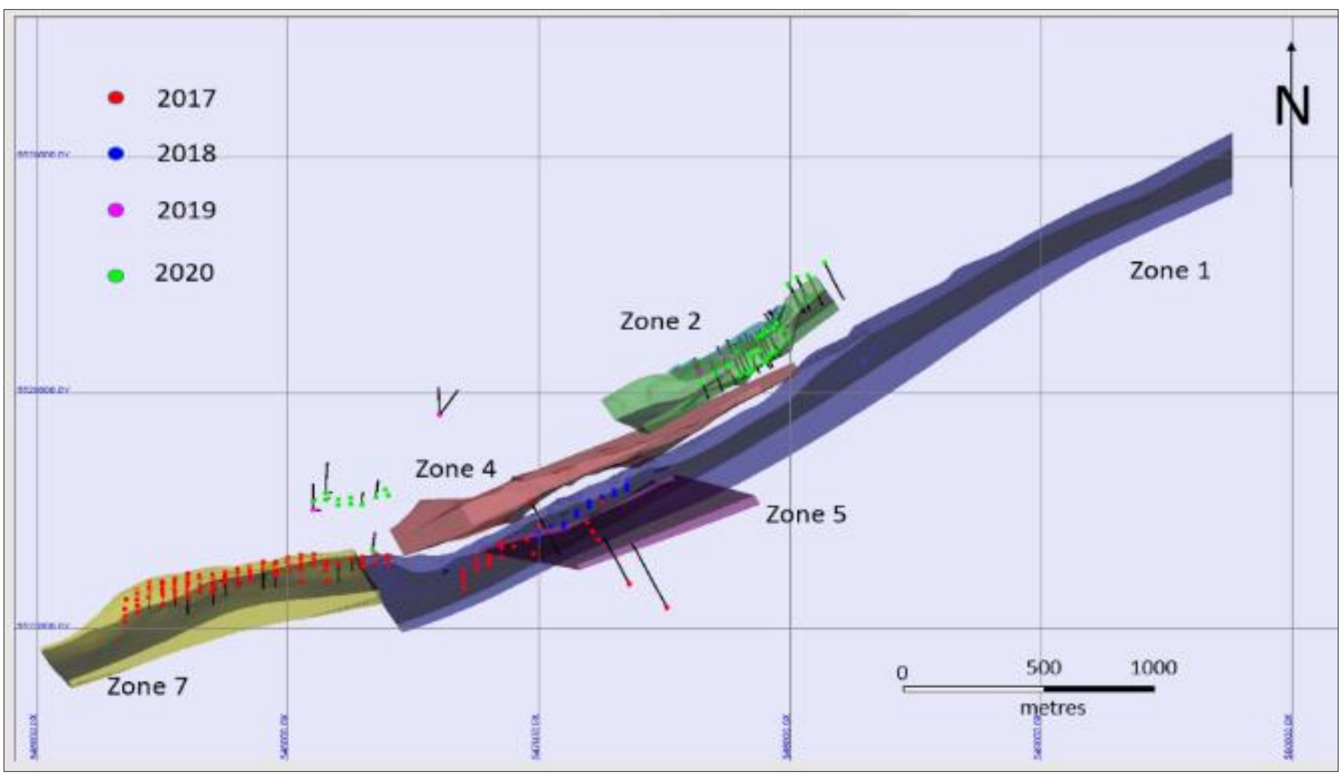
The sample bags were placed in zip-tied rice bags and shipped to SGS Laboratory facilities either in Red Lake, Ontario or Burnaby, British Columbia for fire assay. Intact pieces of drill core were selected and measured for specific gravity using the buoyancy methodology.

The SGS laboratories returned all coarse rejects and pulps to First Mining for permanent and secure storage on site at the Goldlund project. The remaining drill core is securely stored in open core racks or in core racks inside temporary structures. The drill core recovery was good, with an average core recovery of approximately 100%, and only 0.6% of the core intervals had less than 90% core recovery.

Hole GL-19-008 intersected 21 m of 5.36 g/t Au within highly mineralized granodiorite and porphyry units, as well as within andesite, and was successful in confirming the high grades within Zone 2 that were encountered in historical drilling.

Hole GL-19-010 was drilled to intersect the area between the known mineralized areas at Zones 2 and 3 and encountered significant gold mineralization hosted within andesite (15.0 m at 1.68 g/t Au), before intersecting the mineralized granodiorite and porphyries of Zone 2 towards the end of the hole. Table 10-5 on the following page summarizes the significant drill intersections from the First Mining drilling and Figure 10-6 below shows the relative locations of the First Mining drillholes with the various mineralized zones.

Figure 10-6: Plan View showing the First Mining 2017 to 2020 Drillholes in Relation to the Mineralized Zones



Source: SRK, 2023.

Table 10-6: Significant Intersections from First Mining Drill Programs

Hole ID	From (m)	To (m)	Length (m)	Au g/t	Target
GL-19-003	23.57	25.00	1.43	10.91	Main Zone (Zone 2)
and	44.00	47.00	3.00	0.62	
including	45.00	46.00	1.00	1.51	
and	72.10	74.10	2.00	0.25	
and	102.40	107.46	5.06	0.95	
including	106.62	107.46	0.84	4.57	
GL-19-004	32.86	36.21	3.35	1.28	Main Zone (Zone 2)
and	51.90	55.05	3.15	0.60	
and	149.91	155.00	5.09	1.72	
including	149.91	151.00	1.09	4.73	
and	166.00	172.00	6.00	1.57	
including	166.00	167.00	1.00	3.03	
and incl.	170.00	172.00	2.00	2.47	
GL-19-005	58.90	64.00	5.10	0.33	Main Zone (Zone 2)
and	83.00	84.00	1.00	0.30	
and	133.00	135.00	2.00	1.98	
including	133.00	134.00	1.00	3.58	
and	169.30	174.00	4.70	1.05	
including	172.21	174.00	1.79	2.40	
and	189.00	195.00	6.00	0.52	
GL-19-006	82.00	86.00	4.00	3.08	Main Zone (Zone 2)
including	83.00	85.00	2.00	5.72	
and incl.	83.00	83.67	0.67	9.53	
and	107.00	114.00	7.00	0.96	
including	107.00	108.00	1.00	3.14	
and incl.	112.00	114.00	2.00	1.63	
and	134.00	134.50	0.50	1.80	
and	137.54	137.85	0.31	5.13	
and	147.76	148.09	0.33	48.03	
GL-19-008	1.40	25.00	23.60	0.33	Main Zone (Zone 2)
including	10.00	16.00	6.00	1.06	
and incl.	13.00	15.00	2.00	1.90	
and	57.00	66.00	9.00	0.82	
and	83.00	104.00	21.00	5.36	
including	88.00	89.00	1.00	5.49	
and incl.	96.00	97.00	1.00	89.60	
GL-19-010	69.00	84.00	15.00	1.68	Main Zone (Zone 2, 3)
including	69.00	70.00	1.00	8.02	
and incl.	71.00	72.00	1.00	4.86	
and incl.	80.00	81.00	1.00	4.89	
and	143.00	148.00	5.00	1.26	
including	147.00	148.00	1.00	5.24	
and	167.00	175.00	8.00	0.97	
GL-19-012	9.40	9.71	0.31	0.69	Main Zone (Zone 2)
and	48.00	65.00	17.00	1.11	
including	48.00	53.00	5.00	2.27	
and incl.	48.00	49.00	1.00	4.14	
and	86.00	87.00	1.00	3.59	
and	96.00	97.00	1.00	0.98	
and	103.00	104.00	1.00	0.74	
GL-19-013	32.00	34.00	2.00	0.66	Main Zone (Zone 2)
and	63.00	77.00	14.00	1.15	
including	70.00	77.00	7.00	2.20	
and incl.	70.00	71.00	1.00	5.32	
and incl.	75.00	76.00	1.00	9.42	
GL-19-014	25.00	27.00	2.00	0.75	Main Zone (Zone 2)
and	36.00	37.00	1.00	4.07	
and	56.00	58.00	2.00	0.71	
GL-19-021	139.00	140.00	1.00	9.19	Main Zone (Zone 3)
and	188.00	191.00	3.00	3.20	
including	188.00	189.00	1.00	6.54	
and	286.00	288.61	2.61	1.97	
including	286.00	286.70	0.70	6.64	
GL-19-034	25.94	27.17	1.23	8.63	Main Zone (Zone 2)
and	30.72	31.20	0.48	1.81	
and	53.00	55.00	2.00	1.46	
and	60.00	62.00	2.00	3.40	
GL-20-005	52.13	57.07	4.94	0.38	Main Zone (Zone 3)
and	60.00	94.57	34.57	0.28	
GL-20-006	153.00	211.00	58.00	0.83	Main Zone (Zone 3)
including	153.00	166.00	13.00	2.10	
and incl.	161.00	162.00	1.00	12.07	
and incl.	165.00	166.00	1.00	5.10	
and incl.	202.00	211.00	9.00	1.67	
and incl.	208.00	209.00	1.00	9.00	
GL-20-008	94.00	95.00	1.00	2.74	Main Zone (Zone 3)
and	123.00	167.00	44.00	0.27	
including	147.00	166.00	19.00	0.47	

Hole ID	From (m)	To (m)	Length (m)	Au g/t	Target
and incl.	147.00	148.00	1.00	1.64	
and incl.	165.00	166.00	1.00	2.19	
and	175.00	176.00	1.00	1.33	
GL-20-009	37.00	100.00	63.00	0.27	Main Zone (Zone 3)
including	80.00	100.00	20.00	0.52	
and incl.	99.00	100.00	1.00	7.90	
GL-20-010	119.00	122.00	3.00	3.06	Main Zone (Zone 3)
including	120.00	121.00	1.00	7.86	
and	148.00	192.00	44.00	1.20	
including	152.00	153.00	1.00	6.70	
and incl.	166.00	183.00	17.00	1.94	
and incl.	182.00	183.00	1.00	15.90	
and	199.00	210.00	11.00	0.26	
including	209.00	210.00	1.00	1.72	
GL-20-011	88.00	130.00	42.00	0.26	Main Zone (Zone 3)
including	88.00	107.00	19.00	0.54	
and incl.	93.00	99.00	6.00	1.03	
GL-20-012	12.00	102.00	90.00	0.31	Main Zone (Zone 3)
including	19.00	23.00	4.00	1.10	
and	175.00	225.00	50.00	0.14	
GL-20-013	17.00	61.00	44.00	0.27	Main Zone (Zone 3)
including	20.00	21.00	1.00	1.21	
and incl.	54.00	58.00	4.00	0.67	
GL-20-014	1.15	29.00	27.85	0.42	Main Zone (Zone 3)
including	2.00	3.00	1.00	2.92	
and incl.	16.00	17.00	1.00	1.75	
and incl.	27.00	28.00	1.00	1.52	
and	41.00	123.00	82.00	0.10	
and	131.00	140.00	9.00	0.25	
and	158.00	166.00	8.00	0.32	
GL-20-015	10.00	171.00	161.00	0.12	Main Zone (Zone 3)
including	87.00	88.00	1.00	1.29	
and incl.	102.00	115.00	13.00	0.20	
and incl.	140.00	171.00	31.00	0.22	
and incl.	164.00	171.00	7.00	0.52	
and incl.	164.00	165.00	1.00	2.57	
GL-20-017	87.00	93.00	6.00	1.67	Main Zone (Zone 3)
including	88.00	89.00	1.00	8.49	
and	130.00	141.00	11.00	0.15	
including	130.00	134.00	4.00	0.33	
GL-20-018	45.00	76.00	31.00	0.14	Main Zone (Zone 3)
including	71.00	72.00	1.00	2.14	
and	87.00	88.00	1.00	1.11	
and	126.00	136.00	10.00	5.42	
including	129.00	131.00	2.00	22.03	
and incl.	135.00	136.00	1.00	5.10	
GL-20-019	102.00	104.00	2.00	0.95	Main Zone (Zone 2)
and	128.43	130.17	1.74	0.78	
and	145.27	146.33	1.06	0.19	
GL-20-020	34.89	35.76	0.87	0.60	Main Zone (Zone 2)
and	87.00	109.00	22.00	1.25	
including	103.00	109.00	6.00	2.71	
and incl.	103.00	104.00	1.00	5.46	
and incl.	107.00	108.00	1.00	6.37	
GL-20-021	82.50	83.50	1.00	0.39	Main Zone (Zone 3)
and	116.00	117.00	1.00	0.88	
and	121.00	122.00	1.00	0.43	
and	141.00	142.00	1.00	1.75	
GL-20-022	15.00	18.40	3.40	0.59	Main Zone (Zone 3)
and	30.00	32.00	2.00	0.18	
and	108.00	116.00	8.00	0.35	
including	108.00	109.00	1.00	1.21	
and incl.	112.00	113.00	1.00	1.14	
GL-20-023	14.50	15.50	1.00	0.75	Main Zone (Zone 2, 3)
and	52.00	62.00	10.00	1.42	
including	52.00	54.54	2.54	5.24	
and	131.86	139.00	7.14	1.05	
including	131.86	132.86	1.00	2.90	
and incl.	138.00	139.00	1.00	2.72	
and	146.00	147.00	1.00	0.81	
and	157.00	158.00	1.00	0.41	
and	173.00	193.00	20.00	0.50	
including	173.00	185.00	12.00	0.77	
and incl.	184.00	185.00	1.00	6.95	
GL-20-024	26.00	27.00	1.00	0.32	Main Zone (Zone 3)
and	107.00	129.00	22.00	0.48	
including	107.00	114.00	7.00	1.22	
and incl.	107.00	109.00	2.00	3.36	
and	156.00	157.00	1.00	0.24	
and	159.00	160.00	1.00	0.26	
and	182.00	183.00	1.00	3.03	

Hole ID	From (m)	To (m)	Length (m)	Au g/t	Target
GL-20-025	23.00	54.18	31.18	1.82	Main Zone (Zone 2, 3)
including	23.00	39.00	16.00	3.08	
and incl.	24.00	25.00	1.00	20.12	
and incl.	33.05	33.65	0.60	7.58	
and incl.	35.00	36.00	1.00	6.03	
and	118.00	134.00	16.00	1.54	
including	126.00	134.00	8.00	2.95	
and	145.78	167.27	21.49	0.56	
including	150.00	160.00	10.00	0.84	
and incl.	159.00	160.00	1.00	3.77	
and incl.	166.00	167.27	1.27	2.18	
GL-20-026	5.00	6.00	1.00	0.18	Main Zone (Zone 3)
and	31.00	32.00	1.00	6.22	
and	38.00	39.00	1.00	0.12	
and	43.00	44.00	1.00	0.10	
and	55.00	56.00	1.00	0.28	
and	78.00	79.00	1.00	0.14	
and	97.00	119.00	22.00	0.17	
including	97.00	98.00	1.00	2.16	
and	137.00	137.76	0.76	0.25	
GL-20-027	28.00	66.71	38.71	1.39	Main Zone (Zone 2)
including	31.00	32.61	1.61	5.22	
and incl.	35.67	37.01	1.34	19.54	
and incl.	37.01	38.00	0.99	3.01	
and incl.	55.45	57.00	1.55	4.42	
and	83.00	98.00	15.00	0.33	
GL-20-028	16.03	38.00	21.97	2.51	Main Zone (Zone 2)
including	20.00	35.00	15.00	3.58	
and incl.	20.00	29.55	9.55	5.46	
and incl.	28.00	29.55	1.55	24.08	
and	46.00	59.00	13.00	0.55	
including	54.00	59.00	5.00	1.15	
and	64.14	65.00	0.86	1.17	
and	72.00	77.00	5.00	0.97	
including	73.00	74.00	1.00	3.87	
GL-20-029	73.00	80.00	7.00	0.21	Main Zone (Zone 2)
and	92.00	93.00	1.00	3.54	
and	123.00	124.00	1.00	1.81	
and	133.00	151.00	18.00	1.69	
including	141.00	151.00	10.00	2.98	
and incl.	150.00	151.00	1.00	19.93	
and	175.00	176.00	1.00	0.96	
GL-20-030	97.00	101.00	4.00	0.15	Main Zone (Zone 2)
and	152.00	154.00	2.00	0.29	
and	169.00	180.00	11.00	0.42	
including	175.00	179.00	4.00	0.72	
GL-20-031	30.00	38.00	8.00	0.49	Main Zone (Zone 2)
including	37.00	38.00	1.00	1.73	
and	71.00	94.00	23.00	0.28	
including	73.00	89.00	16.00	0.36	
and incl.	85.00	86.00	1.00	1.28	
GL-20-032	57.00	58.00	1.00	0.42	Main Zone (Zone 2)
and	125.00	126.00	1.00	0.22	
and	138.00	139.00	1.00	0.47	
and	172.00	173.00	1.00	1.89	
and	200.71	202.21	1.50	0.26	
GL-20-033	61.00	66.00	5.00	0.63	Main Zone (Zone 2)
and	73.00	74.00	1.00	173.80	
and	99.00	100.00	1.00	0.34	
and	119.00	122.00	3.00	0.62	
and	197.00	198.00	1.00	0.57	
GL-20-034	36.50	37.50	1.00	0.21	Main Zone (Zone 2)
and	101.00	136.00	35.00	0.32	
including	104.00	111.00	7.00	1.14	
and incl.	104.00	105.00	1.00	5.10	
and incl.	110.00	111.00	1.00	1.65	

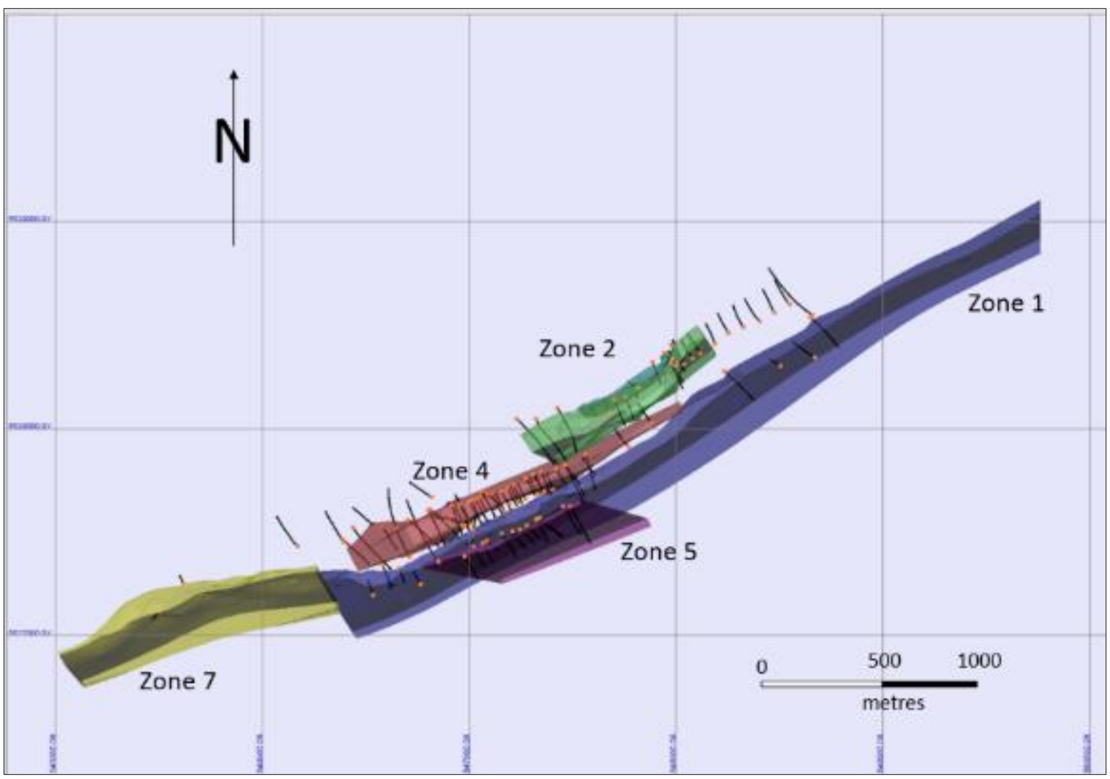
10.3.6 Treasury Metals

After acquiring the Goldlund Property in 2020, Treasury Metals drilled 104 holes totalling 19,124 m. The drilling was focused within and around the defined resource area at Goldlund (Zones 1 and 4), with an initial target of defining and extending mineralization in the eastern and western portions of the deposit. The procedures and results of the 2021 drill program are summarized below.

Drilling procedures and sampling methodology for the 2021 Treasury Metals drilling were the same as described for the Goliath deposit in Section 10.2.1.21 above. Hole GL-21-073 intersected 5.50 m grading 66.56 g/t Au including 0.70 m grading 210.00 g/t Au, 0.80 m grading 78.40 g/t Au and 1.00 m grading 138.00 g/t Au on the inter-zone mineralization related to Zone 6 at the hanging wall contact of a felsic intrusive unit.

Holes GL-21-080 and GL-21-082 were drilled to reach Zone 6 located in the footwall approximately 30 meters from Zone 4. Hole GL-21-082 intersected 4.60 m grading 12.70 g/t Au, including: 0.6 m grading 6.51 g/t Au and 1.40 m grading 37.50 g/t Au from 140.4 m to 145 m downhole. Hole GL-21-082 also intersected 8.00 m grading 3.53 g/t Au, including 1.00 m grading 26.10 g/t Au in Zone 6 from 155 m to 163 m downhole. Also, from Zone 6, hole GL-21-080 intersected 6.90 m grading 7.07 g/t Au, including: 0.50 m grading 61.20 g/t Au and 0.50 m grading 26.40 g/t Au. Results in Zone 6 from GL-21-080 and GL-21-082 were associated with mafic flows adjacent to felsic intrusive dykes. Table10.6 on the following page summarizes the significant intersections from the Treasury Metals 2021 drilling. Figure 10-7 below shows the location of the Treasury Metals 2021 drillholes.

Figure 10-7: Plan View showing the Location of the 2021 Drillholes in relation to the Mineralized Zones



Source: SRK, 2023.

Table 10-7: Significant Drillhole Intervals for 2021 Goldlund Drilling

Drill Hole		Zone	From (m)	To (m)	Length (m)	Au (g/t)
GL-21-052		Zone 5	137	150	13	1.05
GL-21-055		Zone 3	25.25	25.75	0.5	22.8
GL-21-055		Zone 3	70	71	1	18.3
GL-21-068		Zone 5	50	60.6	10.6	0.63
GL-21-068		Zone 5	101	102.1	1.1	11.8
GL-21-073		Zone 6	22.5	28	5.5	66.56
	including		22.5	23.2	0.7	210
	and including		23.2	24	0.8	78.4
	and including		24	25	1	138
GL-21-073		Zone 6	36.8	63.8	27	0.37
GL-21-073		Zone 3	157	163	6	2.08
	including		161	162	1	4.05
GL-21-073		Zone 3	171	173	2	6.05
	including		171	172	1	9.48
GL-21-074		Zone 3	142	153	10.9	2.57
	including		148	149	1	4.72
	and including		150	151	1	16.2
GL-21-078		Zone 4	242.2	249	6.8	11.58
	Including		247.5	249	1.5	48.8
GL-21-080		Zone 4	165.1	181	15.9	0.37
GL-21-080		Zone 6	282	288.9	6.9	7.07
	including		282	282.5	0.5	61.2
	and including		282.5	283	0.5	26.4
		Zone 6	294	294.5	0.5	5.92
GL-21-081		Zone 4	25.5	49	23.5	1.91
	including		25.5	27	1.5	19.2
	and including		46	47	1	10.6
GL-21-082		Zone 4	66	68.1	2.1	1.74
GL-21-082		Zone 4	79.4	88	8.6	1.07
GL-21-082		Zone 6	134	136	2	1.12
		Zone 6	140.4	145	4.6	12.7
	including		140.4	141	0.6	6.51
	and including		143.6	145	1.4	37.5
GL-21-082		Zone 6	155	163	8	3.53
	including		162	163	1	26.1
GL-21-086		Zone 1	23.1	41.2	18.1	0.56
	Including		23.1	24.2	1.1	3.74
	Including		33.5	35	1.5	1.43
GL-21-086		Zone 4	138.7	140	1.3	8.79
GL-21-094		Zone 5	12.7	16.1	3.4	2.96
	Including		14.9	16.1	1.2	7.81

10.4 Miller Deposit Drilling

Drilling on the Miller deposit began in 2018 when First Mining completed nine diamond drillholes on three targets on the Miller property intended to test the potential to host gold mineralization similar to that at the Goldlund project. These targets included Miller, Eaglelund and Miles. A follow-up drill program in 2019 was conducted along strike of the 2018 drillholes based on significant gold intercept results from the initial drill program.

10.4.1 2018-2019 First Mining

Drilling was completed by Rodren Drilling Ltd., based in Winnipeg, Manitoba. Drill core size was HQ (63.5 mm) from the 2018 drilling program and NQ (47.6 mm) from the 2019 drilling program (Figure 10-8).

Drillholes were surveyed downhole using a Reflex or EZ Shot device. The downhole survey was carried out at approximately 30 m to 60 m intervals. Drillholes were initially located in the field using either a differential or handheld GPS.

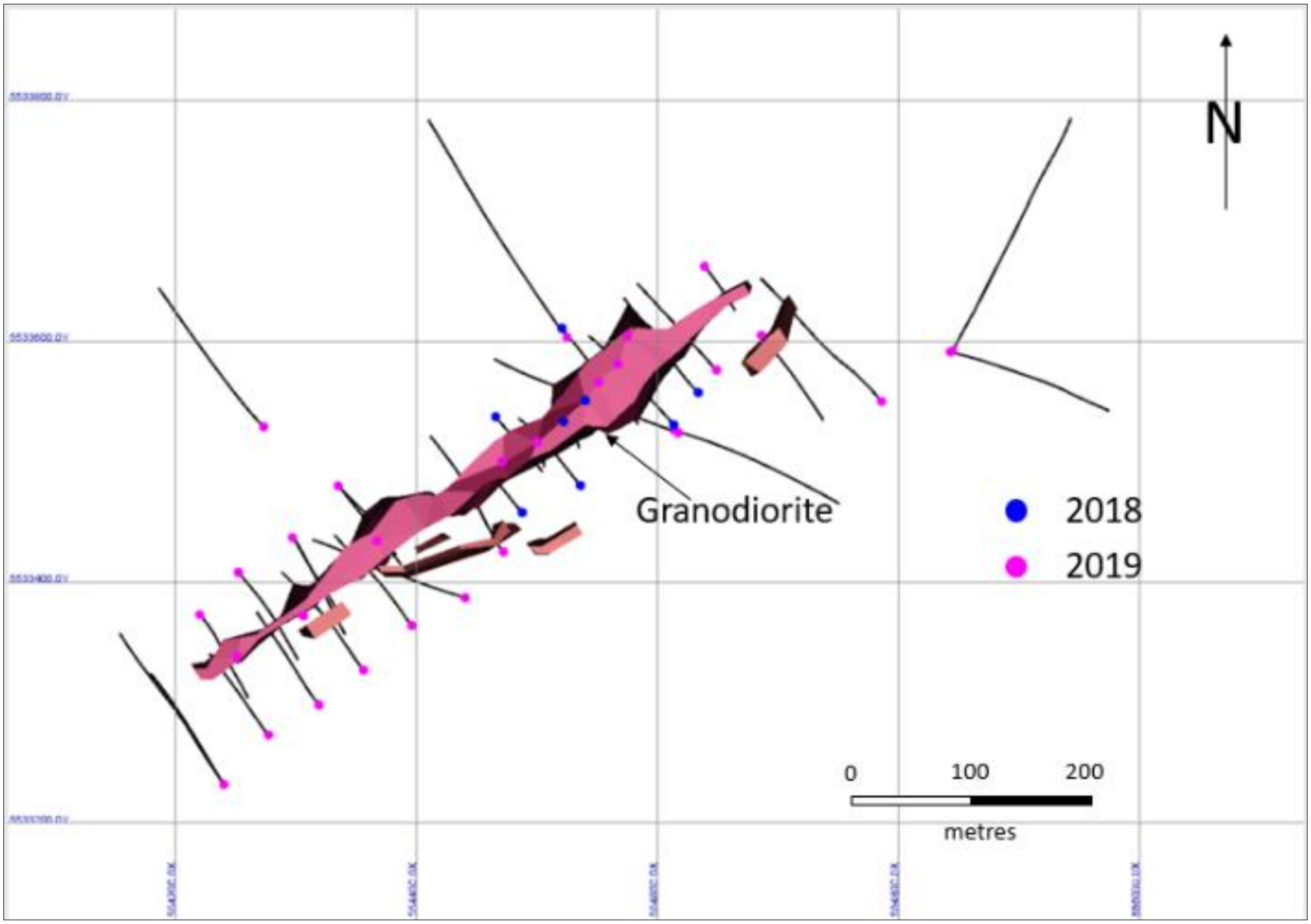
Drill core was transported to the Goldlund exploration camp for logging and sampling.

10.4.2 Core Logging and Sampling

The core logging methodology and QA/QC procedures were overseen by Mr. Miro Mytny, P.Geo, Senior Exploration Manager for First Mining. The logging procedures applied during the Miller drill programs were as follows:

- Drill core was cleaned, and the run (meterage) blocks checked. After this, the core measured for recovery. The recovery percentage was then used to mark-off the adjusted meters within the run. The RQD was measured and recorded in an Excel® spreadsheet, for importing into Datamine DH Logger software.
- The core was logged for lithology, alteration, minerology, veining, and structure directly into DH Logger, which synchronizes with First Mining's central Fusion SQL drilling database.
- One-meter sample intervals were marked-off, except at lithological contacts, and in zones of poor recovery, where sample size could be adjusted accordingly.
- Standards and blanks were inserted in the sample stream at the required intervals.
- Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates.
- Core pieces were selected and measured for specific gravity.
- The core was photographed twice, both dry and wet.
- The core was sawn in half on site, with one half bagged and labelled to be sent for assay. For field duplicates, the core was quartered, and one quarter was sent for the regular assay and the other quarter was sent for the duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the coarse reject or the pulverized sample to run a duplicate assay.
- The remaining half core was placed in core boxes which are stored in a secure on-site facility to serve as a permanent record.
- Sample bags were placed in zip-tied rice bags and shipped to SGS Laboratory facilities in Red Lake, Ontario and Lakefield, Ontario for fire assay analysis.

Figure 10-8: Planview Showing 2018 – 2019 Drilling and Mineralized Zones



Source: SRK, 2023.

10.4.2.1 Drill Results

The 2018 and 2019 drill programs at the Miller consists of 40 drillholes where 28 drillholes intersected the core of the deposit. Drilling was completed over the Miller deposit at approximately 50 m to 100 m spacing and covers an area approximately 500 m x 100 m. Where the 2018 drill program discovered a core of gold mineralization at Miller, the 2019 drill program defined the extension of the mineralization along strike, mainly to the southwest.

Table 10-8 lists selected drillhole intercepts in the Miller deposit with significant gold values.

Table 10-8: Significant Drillhole Intercepts 2018-2019 Drilling

Drillhole		From (m)	To (m)	Width (m)	Au (g/t)
MI-18-001		7	114.6	107.6	0.33
	including	15	88.6	73.6	0.41
	including	16	18.3	2.3	1.93
	including	18	18.3	0.3	8.59
	including	23.3	29.6	6.3	0.91
	including	27.3	27.6	0.3	8.67
	including	77.6	88.6	11	1.17
	including	87.6	88.6	1	6.27
MI-18-002		0.42	142.5	142.08	1.9
	including	1.5	109.5	108	2.44
	including	57.5	88.5	31	4.44
	including	75.5	82.5	7	14.67
	including	81.5	82.5	1	88.8
	including	102.5	109.5	7	9.6
	including	108.5	109.5	1	54.47
MI-18-003		69	72	3	1.12
	and	90	138	48	1.07
	including	90	90.5	0.5	17.23
	including	94	97.5	3.5	2.28
	including	105	106	1	3.9
	including	115	130	15	1.41
	including	125	125.5	0.5	10.55
	including	137.7	138	0.3	9.87
MI-18-004		34	57.8	23.8	0.54
	including	34	35	1	2.56
	including		52	57.8	5.8
	including		55	56	1
MI-18-005		46	47	1	4.18
	including	68	78	10	0.43
	including		72	74	2
	including	109	110	1	1
MI-18-006		76	77	1	1.38
	and	102	124	22	0.69
	including	103	109.4	6.4	2.09
	including	103.62	104	0.38	21.66
	including	109	109.4	0.4	4.69
	and	145	147	2	1.48
	and	169	170	1	3.01
MI-18-007		66	69	3	4.24
	including	66	67	1	9.16
	and	89	138	49	2.53
	including	94.5	116	21.5	5.43
	including	107.5	109	1.5	8.83
	including	114	115	1	91.41
MI-18-008		135	149	14	0.58
	including	135.5	138	2.5	1.59
	including	146	147	1	2.14
MI-19-013		46	228	182	1.09
	including	46	50	4	9.15
	including	47	48	1	35.19
	including	88	109	21	2.73
	including	107	113	6	3.95
	including	134	147	13	2.67
MI-19-014		3	210	207	1.57
	including	42	91	49	2.34
	including	56	70	14	4.53
	including	60	61	1	26.43
	including	142	183	41	4.07
	including	168	182	14	7.38
	including	168	169	1	55.28
MI-19-015		1	168	167	1.01
	including	1	26	25	1.62
	including	5	8	3	5.4

Drillhole		From (m)	To (m)	Width (m)	Au (g/t)
	including	108	141	33	1.84
	including	120	122	2	5.82
MI-19-017		6	7	1	1.48
	and	32	201	169	0.88
	including	56	93	37	3.42
	including	79	93	14	7.27
	including	83	84	1	65.97
	including	85	86	1	11
MI-19-018		18	141	123	0.86
	including	67	141	74	1.18
	including	100	134	34	2.08
	including	105	106	1	6.49
	including	113	114	1	12.91
	including	129	130	1	23.96
	and	168	169	1	4.24
MI-19-019		65	101	36	0.41
	including	68	69	1	2.78
	including	83	85	2	2.09
	including	100	101	1	1.62
MI-19-020		133	139	6	1.77
	including	134	135	1	8.15
MI-19-021		111	118	7	0.99
	including	112	113	1	4.78
MI-19-022		115	122	7	0.82
	including	119	120	1	1.56
	including	121	122	1	2.58
MI-19-032		39	143	104	0.25
	including	60	80	20	0.40
	including	79	80	1	3.56
	and	107	143	36	0.38
	including	126	127	1	5.50
MI-19-040		60	119	59	1.35
	including	60	62	2	5.91
	including	78	83	15	3.88
	including	80.88	81.88	1	6.83
	including	86.88	87.88	1	44.07

10.4.3 Treasury Metals 2021

The 2021 drill program was designed to upgrade the mineral resource classification within areas in the planned open pit and to expand the resources along strike of the current resource and pit areas. A total of 21 holes were drilled for 2,985 m.

Drilling was contracted to George Downing Estate Drilling Ltd. All drill collar locations are initially located with hand-held GPS and final collar locations are verified with DGPS delivering real-time accuracy in the centimeter range. Down hole deviation is controlled with the use of Reflex Gyro readings collected at 15 to 20 m intervals. Core is logged by Treasury Metals employees and data is entered into both Excel and MX Deposits for capture and safe keeping.

The holes were drilled approximately perpendicular to the mineralized zone with azimuths being either 135° or 300° and dips ranging from -45° to -70°. The average core recoveries were excellent, and the RQD rock mass quality was good.

Most of the 2,764 samples collected were at 1.0 m intervals. Sample intervals were generally broken on geological contacts resulting is some samples being shorter than 1.0 m but not less than 0.3m in length. The highest gold assay obtained from the program was from drillhole MI-21-046 that returned 27.3 g/t over 1.0 m.

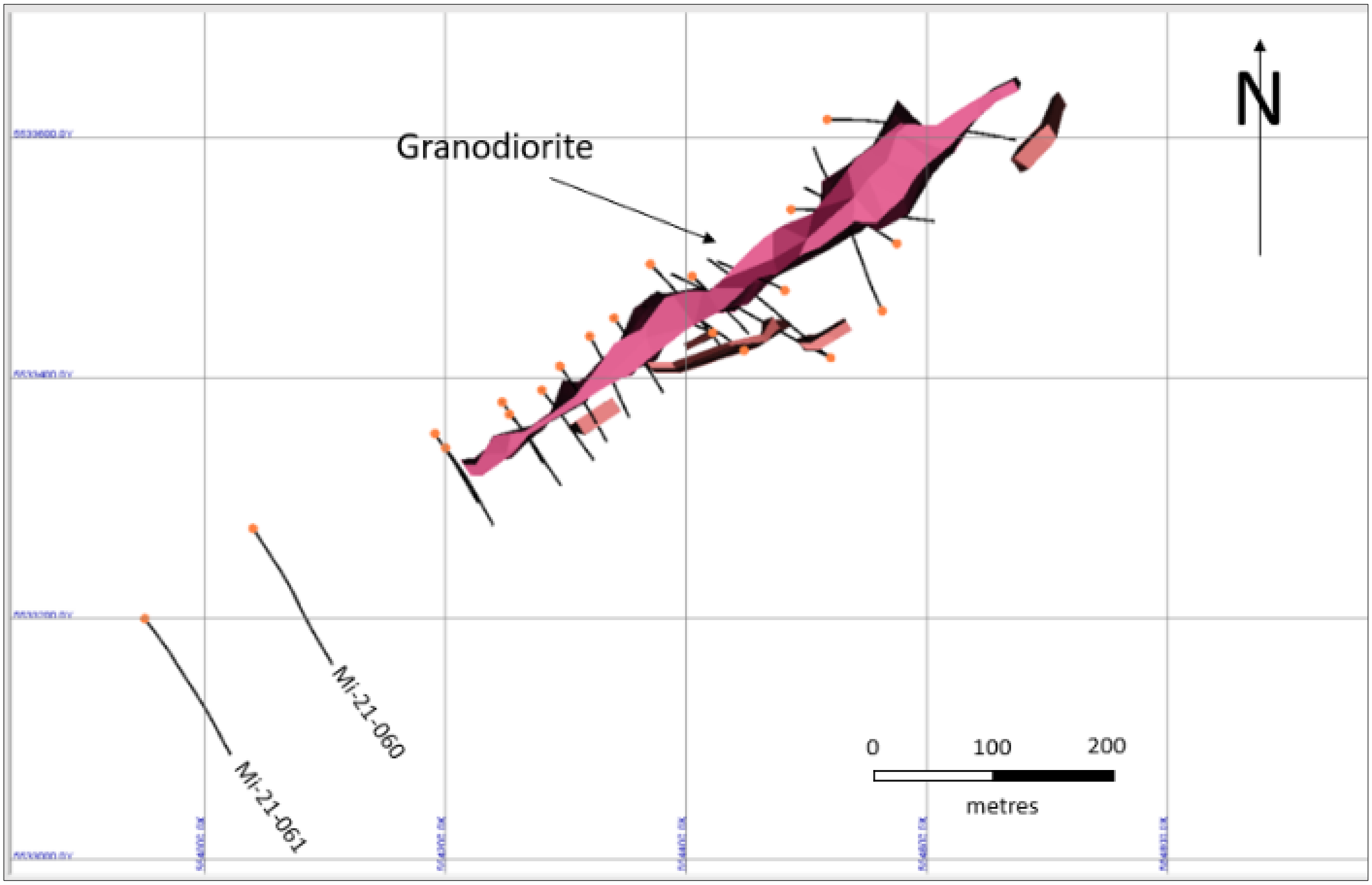
Drillhole MI-21-041 intersected 44.3 m grading 0.68 g/t Au (including 11.3 m grading 2.01 g/t Au), MI-21-042 intersected 26.7 m grading 1.59 g/t Au (including 14.7 m grading 2.58 g/t Au), MI-21-047 intersected 52.6 m @ 0.72 g/t Au (including 10.7 m grading 2.39 g/t Au) and MI-21-048 intersected 18.1 m grading 0.65 g/t Au (including 1.0 m grading 7.58 g/t Au). Table 10-9 summarizes the most significant intersections from the 2021 Miller drill program.

Table 10-9: Significant Intersections from 2021 Miller Drilling

Hole ID	Zone	From (m)	To (m)	Length (m)	Au (g/t)
MI-21-041	Miller	83	127.3	44.3	0.68
including		108	119.3	11.3	2.01
including		113	114	1	5.75
And including		117	118	1	6.23
And including		118.6	119.3	0.7	7.22
MI-21-042	Miller	52.3	79	26.7	1.59
including		62.45	77.15	14.7	2.58
including		68	69	1	14.4
MI-21-043	Miller	129.9	140.1	10.2	0.95
MI-21-044	Miller	162	171.1	9.1	0.92
MI-21-046	Miller	39.7	63	23.3	2.73
including		42.7	43.9	1.2	14
MI-21-047	Miller	59	111.6	52.6	0.72
including		72.35	83	10.65	2.39
MI-21-048	Miller	47	65.06	18.06	0.65
including		62	63	1	7.58
MI-21-049	Miller	66	83.15	17.15	1.01
including		79	80.05	1.05	7.75
MI-21-056	Miller	33	44	11	0.72
MI-21-060	Miller	90	91	1	6.3

Two step-out drillholes were also completed approximately 100 and 200 meters respectively along strike from previous drilling and 500 meters from the planned open pit (Figure 10-9). Both drillholes encountered the gold hosting diorite lithology and MI-21-060 identified visible gold with an intersection of 1.0 m grading 6.30 g/t Au. These holes give confidence in the potential growth of the Miller resource for future mining studies.

Figure 10-9: Planview of 2021 Miller Drill Program



Source: SRK, 2023.

10.5 Sample Length/True Thickness

Most holes drilled at the Goliath Complex, Goliath, Goldlund and Miller were generally drilled to intersect the known mineralization at right angles as much as possible. Most Goliath holes were drilled to intersect the mineralized zones between 60 to 90°. Some historical drillholes were drilled sub-parallel to the mineralization but for the most part, drill intersections are 1.3 to 1.5 times the true width of the mineralized zones.

The mineralization at Goldlund and Miller is hosted in stockwork of variable orientation. At Goldlund, at least two preferred orientations are known to occur (239°/58°N (70 set) and 189°/53°W (20 set). Drill intercepts are not always reflective of true zone thickness. Instead, drillholes were planned to intersect the boarder zones of stockwork as perpendicular as feasible. Down hole intervals are not truly representative of individual higher grade veins structures but wider intervals are reasonably representative of the thicknesses of the wider stockwork zones.

10.6 Comments on Drilling

10.6.1 Goliath Drill Programs

The drillhole orientation was found to be appropriate for the deposit style and the orientation of the mineralization. Drill spacing in the most densely drilled areas is less than 25 m and is deemed sufficient to adequately define the grade of the mineralization and the spatial grade distribution.

Drill core logging is appropriate for the mineralization style and carried out to industry standards.

Very high-grade intersections above 10 g/t Au are rare and are sometime isolated (> 99th percentile in the MSS lithology). Intercepts above 3 g/t Au occur more frequently (> 98th percentile in the MSS lithology) and have documented continuity of at least 20 m strike length in the Teck bulk sample area.

Drill core handling, surveying, and chain of custody from the rig to the core logging facility was found to meet or exceed industry standards. The Qualified Person believes that the drill data are sufficiently accurate to be reliable and is therefore suitable for use in the estimation of mineral resources.

10.6.2 Goldlund Drill Programs

The Qualified Person believes, that based on the review of selected drill core and the description of the logging and sampling methodology provided in various technical reports, that the drilling and sampling was undertaken in accordance with industry standards and best practices at the time for which each campaign was carried out. The Qualified Person also believes that the drill data are sufficiently accurate to be reliable and is therefore suitable for use in the estimation of mineral resources.

10.6.3 Miller Drill Programs

The Qualified Person believes, that based on the review of selected drill core and the description of the logging and sampling methodology provided in various technical reports, that the drilling and sampling was undertaken in accordance with industry standards and best practices at the time for which each campaign was carried out. The Qualified Person compared the selected drill core to the drill logs to verify that the descriptions, lithological and sampling intervals were correctly described. The Qualified Person believes the data are sufficiently accurate to be reliable and is suitable for use in the estimation of mineral resources.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sampling Methods

11.1.1 Goliath

11.1.1.1 Teck-Corona Sample Preparation & Analysis, 1990-1998

Teck-Corona samples were typically 0.5, 1.0 and 1.5 meters, but could range between a low of 0.3 to 2.5 meters with very few exceptions. All samples were shipped to the primary laboratory by Gardwine and Porter transport firms. The primary laboratory used was TSL Laboratories (TSL) of Saskatoon, Saskatchewan. XRAL Laboratories and Interteck Testing Services were used for assay verification work or whole rock analyses.

11.1.1.2 Teck-Corona Quality Control & Quality Assurance (QA/QC), 1990-1998

Not much detail is available on sample preparation and analysis procedures during that period. The following was extracted from the Teck bulk sample program, and it is assumed that the analytical procedure at the TSL Laboratory in Saskatoon for the face and muck samples was similar to what was use for the drill core submitted to that laboratory.

The samples were prepared by crushing the whole samples 90% passing -10 mesh and then splitting into 250 g sub-sample. The pulverized sub-sample was then analysed by fire assays with either atomic absorption (FA-AA) or gravimetric (FA-GRAV) finish. Silver was analysed by dissolution (aqua regia digestion?) and atomic absorption spectrometry (AAS). High-grade samples were known to have be analysed by 1000 g pulp metallics. Teck-Corona Quality Control & Quality Assurance (QA/QC), 1990-1998

No details were available with regard to the quality control and quality assurance (QA/QC) program during that period.

11.1.1.3 Treasury Metals Sample Preparation & Analysis

As described in Section 10 of this report, the drill core for the Goliath project was logged and split with a core saw lengthwise, with the majority of samples ranging from 1.0 to 1.5 m in length. All samples were kept on site in the secured sampling area and shipped to the assay laboratory by Company transport on a weekly basis. Half of the core was retained for future verification and the other half was sent to the analytical laboratories.

11.1.2 Goldlund

11.1.2.1 Historical Drilling

No information is available on the historical drill programs prior to the involvement by Tamaka (pre-2007).

11.1.2.2 Tamaka Drill Programs

11.1.2.2.1 2007, 2008 and 2011

Drill core was prepared by a technician who recoded Rock Quality Designation (RQD) of the core on a computer form. Magnetic susceptibility was recorded over the entire hole length at 0.5 meter intervals. All core was photographed (both wet and dry) and logging is completed by the geologist directly into a Microsoft® Excel® spreadsheet template form.

Sample lengths were variable, 20 centimeters minimum sample length, 1.5 meters maximum sample length and all samples were collected by sawing the core with a diamond saw lengthwise. Sample bags were sealed with fiber tape and placed in rice bags for shipping to the assay laboratory. A Tamaka employee delivered the samples to Manitoulin Transport in Dryden for delivery to Accurassay Laboratories in Thunder Bay. The laboratory returned all coarse rejects and pulps to Tamaka for storage at the Goldlund project.

11.1.2.2.2 2013 and 2014

A geologist or geotechnician measured run lengths to confirm block markers, checked the core, and completed a quick log. A geotechnician recorded RQD of the core and the core was logged by a geologist for lithology, structure, alteration veining, and sample intervals, directly into a Gemcom© Gemslogger (Gemslogger) Microsoft® Access database.

Samples were collected at 1.5-meter intervals downhole and marked with a red lumber crayon. The sample intervals were adjusted (to a range of 0.3 to 2 meters) to ensure that they did not cross lithological boundaries. All drill core was photographed on the log benches in sets of four.

The core was split in half, and one half was placed in a plastic sample bag and the other half was returned to the core box. One portion of the sample tag was placed in the sample bag, one was stapled into the core box at the beginning of the sample interval, and one tag remained in the sample book which was filed in the site office.

11.1.2.3 First Mining Drill Programs

The core was logged for lithology, alteration, minerology, veining and structure, and entered into DH Logger, which synchronizes with First Mining's central Fusion SQL drilling database. Samples were collected at two-meter intervals, except at lithological contacts, and in zones of poor recovery, where sample size was adjusted accordingly. Standards and blanks were inserted in the sample stream at the required intervals. Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates. Core pieces were selected and measured for bulk density.

The core was photographed twice, both dry and wet. The core was sawn in half on site, with one half bagged and labelled to be sent for assay. For field duplicates, the core was quartered, and one quarter was sent for the regular assay and the other quarter was sent for the duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the pulverized sample to run a duplicate assay. The remaining half core was placed in core boxes which were stored in a secure onsite facility to serve as a permanent record.

11.1.2.4 Treasury Metals 2021

Drill core for the Goldlund project was logged and information was entered into MX Deposit while logging. The core was then split lengthwise with a diamond saw by Treasury Metals employee. The majority of samples ranged from 0.30 to 1.5 meters in length. All samples were kept on site in the secured sampling area and shipped to the assay laboratory by

Company transport on a weekly basis. Half of the core was retained for future verification and the other half was sent to the analytical laboratories.

11.1.3 Miller

11.1.3.1 First Mining 2018-2019

The RQD was measured and recorded in an Excel® spreadsheet, for importing into Datamine DH Logger software. The core was logged for lithology, alteration, minerology, veining, and structure directly into DH Logger, which synchronizes with First Mining's central Fusion SQL drilling database.

Samples were collected at one-meter sample intervals, except at lithological contacts, and in zones of poor recovery, where sample size could be adjusted accordingly.

Standards and blanks were inserted in the sample stream at the required intervals. Duplicates were inserted between the blanks and standards, alternating between field and laboratory duplicates. Core pieces were selected and measured for bulk density determination.

The core was photographed twice, both dry and wet and then sawn in half on site, with one half bagged and labelled to be sent for assay. For field duplicates, the core was quartered, and one quarter was sent for the regular assay and the other quarter was sent for the duplicate assay. For the laboratory duplicates, an empty sample bag with a sample ID was sent to the laboratory where a split was taken from the coarse reject or the pulverized sample to run a duplicate assay.

The remaining half core was placed in core boxes which are stored in a secure onsite facility to serve as a permanent record.

Sample bags were placed in zip-tied rice bags and shipped to SGS Laboratory facilities in Red Lake, Ontario and Lakefield, Ontario for fire assay analysis.

11.1.3.2 Treasury Metals 2021

Drill core for the Miller project was logged and information was entered into MX Deposit while logging. Holes drilled prior to June 2021 were entered into DH Logger. The core was then split lengthwise with a diamond saw by Treasury Metals employee. The majority of samples ranged from 0.30to 1.5 meters in length. All samples were kept on site in the secured sampling area and shipped to the assay laboratory by Company transport on a weekly basis. Half of the core was retained for future verification and the other half was sent to the analytical laboratories.

11.2 Sample Preparation and Analysis

11.2.1 Goliath

Accurassay Laboratory (Accurassay), an independent analytical laboratory, was used by Treasury Metals from 2008 to 2015. Once the rock samples were received at the Accurassay's facilities in Thunder Bay, Ontario, they were entered into the Laboratories Local Information System (LIMS).

The samples were prepare using procedure code ALP1. Samples were dried then jaw crushed to 8 mesh size. A 500 g split was then pulverized to approximately 90% passing -150 mesh and then matted to ensure homogeneity. Silica abrasive sand was used to clean out the pulverizing dishes between each sample to prevent cross contamination. Some certificates listed ALP2 procedure code which is similar to the ALP1 but crushing at 90% passing -8 mesh and collecting a 1000 g split instead of the 500 g. Once prepared, the samples were then sent to the fire assay laboratory or the wet chemistry laboratory depending on the required analysis. For gold, all samples were assayed using code “ALFA1”, denoting a 30 g fire assay with an AAS finish.

Starting during the 2009 drill program, samples grading above 5 g/t Au were re-assayed using the code “ALFA7”, which indicated a gold fire assay with a gravimetric finish. This was altered to all samples grading above 3 g/t Au for the 2010-2012 drill programs. It reverted to samples above 5 g/t Au for drill programs occurring between 2013-2015.

From 2008 to 2015, samples returning values in excess of 5.0 g/t Au were analysed with the pulp metallic method code “ALPM1”. The 2015 drilling program used 6.0 g/t Au as the threshold limit. Accurassay described the pulp metallic method as a procedure that is able to overcome the “nugget effect” of gold by increasing the sub-sample size to 1,000 g and physically collecting the free gold within the system using a 150 mesh (106 µm) sieve. This procedure is most effective when the whole sample is used for the analysis. The sub-sample is pulverized to ~90% - 150 mesh (106 µm) and subsequently sieved through a 150-mesh (106 µm) screen. The entire +150 metallics portion is assayed along with two duplicate sub-samples of the -150 pulp portion. Results are reported as a weighted average of gold in the entire sample.

Geochemistry for silver and a suite of six or nine additional elements from 2008 to the beginning of the 2010 drill campaign. Late in 2010 through to 2015 Treasury Metals ran geochemistry for silver and 29 other elements using procedure code “ALMA1”, which is described as a multi-acid digestion with an inductively coupled plasma with optical emission spectrometry (ICP-OES) finish.

A certificate was produced from the LIMS laboratory database system. The laboratory manager checks the data, validates the certificates, and issues the results as a PDF file and a Microsoft® Excel® file.

Accurassay was accredited by ISO/IEC 17025 was accountable to the Standards Council of Canada for its quality management at the time the samples were processed. Accurassay filed for bankruptcy on May 16, 2017.

Starting in 2016 Treasury Metals submitted samples to the Activation Laboratory Ltd. (ActLabs) in Dryden. At the ActLabs facility, the samples were processed using procedure code RX1, which is described as crushing up to 80% passing 2 mm, riffle splitting a sub-sample of 250 g, and pulverizing to 95% passing 105 µm.

Sample pulps were then assayed using procedure code 1A2-50, which is a 50 g fire assay with AA finish. Samples grading above 3 g/t Au were re-assayed with code 1A3-50, which is a 50 g fire assay with gravimetric finish.

High-grade samples in excess of 5 g/t Au were assayed using procedure code 1A4-1000, which is a metallic screen assay. For this type of assay, a representative 500 g split (1,000 g for 1A4-1000) is sieved at 100 mesh (149 µm) with fire assays performed on the entire +100 mesh and 2 splits on the -100-mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fraction is weighed for assay reconciliation.

Starting in 2016, Treasury Metals assayed the sample for silver and an additional 37 elements on selected samples within the mineralized zones only. The samples were analysed using ActLabs code 1E3, which is described as a partial digestion by aqua regia with an ICP-OES for the analysis. The method quantitatively dissolves base metals for the majority of geological materials, but major rock-forming elements and more resistive metals are only partially dissolved. As such, the leach should be considered partial for most elements.

ActLabs in Dryden, an independent analytical laboratory, was assessed by TRC Inc. and found to be in conformance to the ISO 9001:2015 standard (Certificate number TRC 01028).

11.2.2 Goldlund

11.2.2.1 Historical Drilling

There is no information regarding the sampling preparation and analysis for the historical drilling at Goldlund.

11.2.2.2 Tamaka Drill Programs

11.2.2.2.1 2007 and 2008

Samples for the Tamaka 2007 and 2008 drilling program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverizers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverized to 90% passing -150 mesh (0.104 mm) and sent for fire assay.

Gold and silver were analysed using a 50 g aliquot from a 500 g pulp by lead fusion fire assay with an inductively coupled plasma mass spectrometry (ICP-MS) finish.

11.2.2.2.2 2011 – 2014

Samples for the Tamaka 2011 drilling program, including the standard, duplicate, and blank samples, were shipped to the Accurassay in Thunder Bay where they were prepared for fire assay analysis using jaw crushers and ring and puck mill pulverizers. Samples were dried, crushed to 90% passing -8 mesh (2 mm) and a 1,000 g split was taken and pulverized to 90% passing -150 mesh (0.104 mm) and sent for fire assay.

For samples from drillholes K11-110 to K11-118, a 30 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish. For the samples from drillholes K11-119 to K11-2-140, a 50 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish for gold and silver. For samples more than 10 g/t Au, a second lead fusion fire assay was carried out for gold using either a 30 or 50 g aliquot from a second 500 g pulp with a gravimetric finish.

All other samples were analysed using a 50 g aliquot taken from a 500 g pulp and gold and silver were assayed by conventional lead fusion fire assay with an AAS finish. For samples assaying more than 10 g/t Au a second lead fusion fire assay was carried out for gold using a second 50 g aliquot from the 500 g pulp with a gravimetric finish.

11.2.2.3 First Mining Drill Programs

11.2.2.3.1 2013 - 2014

The samples from the 2013 and 2014 drilling program were analysed by Accurassay in Thunder Bay. A 50 g aliquot was taken from a 500 g pulp and analysed for gold and silver by conventional lead fusion fire assay with an AAS finish. For samples assaying more than 10 g/t Au a second lead fusion fire assay was carried out for gold using a second 50 g aliquot from the 500 g pulp with a gravimetric finish.

11.2.2.3.2 2017 – 2018

The samples from the 2017 and 2018 drilling program were analysed for gold at either the SGS laboratory in Vancouver, an independent analytical laboratory, using a BLEG methodology, or the SGS laboratory in Red Lake, using a lead fusion fire assay methodology. SGS in Vancouver holds ISO/IEC 17025 accreditation.

The BLEG methodology uses a large sample (1,000 g) that is digested, or leached, with a cold cyanide solution (LeachWellTM CN) for two hours. The gold in the sample is dissolved as cyanide complexes. The leachate is then concentrated in a solvent exchange type procedure and analysed by AAS or ICP. The large sample sizes and solvent extraction technology used in bulk leach extractable gold analysis provides detection limits as low as 0.1 ppb. The precision of BLEG test results is high due to the large sample size. However, this methodology is not a total assay, so a fire assay of the residual material is also required. This methodology was considered to improve the reproducibility of the gold assays for the "nuggety" Goldlund mineralization.

The pulverized sample material was weighed and placed into labelled bottles and the cyanide reagent was added. The bottles were agitated using a bottle roll with a leach time of two hours to homogenize the sample with the cyanide solution. Once settled, a layer of clear solution is available for analysis by AAS. The residue sample is then filtered and washed to remove the cyanide solution. The residue is dried, homogenized and a 200 g split is collected, with a 50 g aliquot taken and analysed for gold by a lead fusion fire assay. The final assay is then a combination of the cyanide leachable gold and the residual fire assay gold.

In addition to the gold assay, a 50 g split from each sample was sent for ICP multi-element analysis by two-acid aqua regia digestion with an ICP-MS and atomic emission spectroscopy (AES) finish.

The samples that were sent to the SGS laboratory in Red Lake were assayed for gold using either a 30 g or a 50 g aliquot for lead fusion gold fire assay with an AAS finish.

11.2.2.3.3 2019 – 2020

The samples from the 2019 and 2020 drilling program were analysed by SGS laboratories in Red Lake or Vancouver. A 50 g aliquot was taken from a 250 g pulp and analysed for gold by conventional lead fusion fire assay with an AAS finish. For drillholes GL-19-003, GL-19-008, GL-20-006, GL-20-009, and GL-20-010 selected assay repeats were done for gold by screen "metallics" lead fusion fire assay on 1 kg size samples at the SGS laboratories in Lakefield and Vancouver.

11.2.2.4 Treasury Metals 2021

Treasury Metals submitted samples to the Activation Laboratory Ltd. (ActLabs) in Dryden. At the ActLabs facility, the samples were processed using procedure code RX1, which is described as crushing up to 80% passing 2 mm, riffle splitting a sub-sample of 250 g, and pulverizing to 95% passing 105 µm.

Sample pulps were then assayed using procedure code 1A2-50, which is a 50 g fire assay with AA finish. Samples grading above 3 g/t Au were re-assayed with code 1A3-50, which is a 50 g fire assay with gravimetric finish.

High-grade samples in excess of 5 g/t Au were assayed using procedure code 1A4-1000, which is a metallic screen assay. For this type of assay, a representative 500 g split (1,000 g for 1A4-1000) is sieved at 100 mesh (149 µm) with fire assays performed on the entire +100 mesh and 2 splits on the -100-mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fraction is weighed for assay reconciliation.

Sample within the mineralized zones were assayed for silver and an additional 37 elements. The samples were analysed using ActLabs code 1E3, which is described as a partial digestion by aqua regia with an ICP-OES for the analysis. The method quantitatively dissolves base metals for the majority of geological materials, but major rock-forming elements and more resistive metals are only partially dissolved. As such, the leach should be considered partial for most elements.

ActLabs in Dryden was assessed by TRC Inc. and found to be in conformance to the ISO 9001:2015 standard (Certificate number TRC 01028).

11.2.3 Miller

11.2.3.1 First Mining Drilling

Samples from the 2018 and 2019 drill programs at the Miller deposit were analysed at the SGS laboratories in Red Lake or Lakefield, Ontario or Burnaby, BC by 50 g fire assay and atomic absorption (AA) finish (SGS Code: GE_FAA515). Additionally, a 51 multi-element analysis (SGS Code: ZMS_ICM14B) was completed on the first eight drillholes but was discontinued for the remained for the drill program.

Due to the frequent occurrence of visible gold in the drillholes and the course, nuggety nature of the gold mineralization, analyses were followed up on selected samples with a more definitive assay protocol of metallic screen fire assay using a 1,000 g sample size to minimize the high nugget effect (SGS Code: GO_FAS30M).

11.2.3.2 Treasury Metals Drilling

The 2021 Miller samples were submitted to the Activation Laboratory Ltd. (ActLabs) in Dryden. At the ActLabs facility, the samples were processed using procedure code RX1, which is described as crushing up to 80% passing 2 mm, riffle splitting a sub-sample of 250 g, and pulverizing to 95% passing 105 µm.

Sample pulps were then assayed using procedure code 1A2-50, which is a 50 g fire assay with AA finish. Samples grading above 3 g/t Au were re-assayed with code 1A3-50, which is a 50 g fire assay with gravimetric finish.

High-grade samples in excess of 5 g/t Au were assayed using procedure code 1A4-1000, which is a metallic screen assay. For this type of assay, a representative 500 g split (1,000 g for 1A4-1000) is sieved at 100 mesh (149 µm) with fire assays performed on the entire +100 mesh and 2 splits on the -100-mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fraction is weighed for assay reconciliation.

Sample within the mineralized zones were assayed for silver and an additional 37 elements. The samples were analysed using ActLabs code 1E3, which is described as a partial digestion by aqua regia with an ICP-OES for the analysis. The method quantitatively dissolves base metals for the majority of geological materials, but major rock-forming elements and more resistive metals are only partially dissolved. As such, the leach should be considered partial for most elements.

ActLabs in Dryden was assessed by TRC Inc. and found to be in conformance to the ISO 9001:2015 standard (Certificate number TRC 01028).

11.3 Quality Assurance and Quality Control

11.3.1 Goliath

Treasury Metals implemented and monitored a thorough QA/QC program for the diamond drilling and sampling undertaken at the Goliath property from 2008 through 2021. QC protocol included inserting control samples into every batch sent for analysis. The QA/QC protocols were altered somewhat over the program, as described in the following sections by year.

A number of certified reference materials (CRMs) were used throughout the years. During the 2008 drill program, CRMs were supplied by Accurassay and CDN Resource Laboratories Ltd of Delta, BC, and ORE Pty Ltd (now OREAS). The CDN Laboratory CRMs were found to be more reliable, and Treasury Metals exclusively used the CRMs supplied by CDN Laboratory for the subsequent years. Table 11-1 summarized the various CRMs used throughout the years.

The Treasury Metals QA/QC programs were reviewed by AGP as part of the 2021 PEA work. The discussion that follows on the historical QA/QC programs is taken from the NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex prepared by Ausenco in 2021.

11.3.1.1 2008 QA/QC Program

To monitor accuracy, CRMs (or standards) and blanks were inserted into the sample stream by Treasury Metals at a rate of at least 1 in every 20 samples submitted. A total of nine CRMs were utilized to monitor gold results over the course of the 2008 drill program including the AuQ1, Au43, CDN-FCM4, AuG1, AuH2, OREAS_61D, Au48, CDN-GS-5D, CDN-SE2. Treasury Metals selected a mixture of low-, medium-, and high-grade CRMs to monitor lab accuracy.

Treasury Metals uses a mean ±3x standard deviation as control limit and mean ± 2x standard deviation as warning limit. Any single standard analysis beyond the upper and lower control limit is considered a “failure”. Treasury Metals also considers a failure when three successive standard analyses are outside the upper and lower warning limits on the same side of the mean.

11.3.1.1.1 Performance of Certified Reference Materials

It was reported that most standard failures occur at the beginning of the drill program. Oreas61D and Au48 retuned erratic results and were replaced by CCIC with more reliable standards. Failure of a standard within the mineralized horizon prompted the resubmission of the pulps for the entire batch.

Most of the CRMs monitoring accuracy within the mineralized zone returned values within three standard deviations from the mean. The CDN_GS-5D mean value is low when compared to the certify mean and likely a matrix match issue. There were 20 failures within the mineralized zone and the pulp samples from all 20 batches were re-analysed at Accurassay to confirm results.

11.3.1.1.2 Performance of Blank Material

The blank material used for the QC monitoring was a prepared blank supplied by Accurassay that was pulverized to 200 mesh, blended and packaged in 60-gram packets. The blank was inserted at a rate of at least one in 20 samples and has a gold concentration of less than 15 ppb. A tolerance limit of 45 ppb was set by the Company to evaluate for contamination. AGP note that the blank material used in 2008 is unsuitable to monitor cross contamination at the crushing stage of the sample preparation.

Table 11-1: Summary of CRM Used from 2008 to 2021

Standard (CRM)	Recommended Value Au(ppm)	Standard Deviation Au (ppm)	Supplier	Drill Program Year													
				2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	2018	2019	2020	2021
AuQ1	1.33	0.114	Accurassay	X													
Au43	12.686	0.859	Accurassay	X													
CDN-FCM4 *	0.97	0.04	CDN	X													
AuG1	1.019	0.04	Accurassay	X													
AuG2	1.013	0.02	Accurassay	X													
OREAS_61D *	4.76	0.14	OREAS	X	X	X	X										
Au48	16.15	0.964	Accurassay	X													
CDN-GS-5D	5.06	0.125	CDN	X	X	X	X										
CDN-SE-2 *	0.242	0.009	CDN	X	X	X	X										
CDN-GS-1D	1.05	0.05	CDN		X												
CDN-GS-1F	0.242	0.009	CDN			X	X										
CDN-CGS-13	1.01	0.055	CDN			X	X										
CDN-CM6 **	1.43	0.045	CDN			X	X										
CDN-ME-6 *	0.27	0.014	CDN			X	X										
CDN-GS-P2A	0.229	0.015	CDN					X	X								
CDN-CM-26 **	0.372	0.024	CDN					X	X	X	X	X	X	X			
CDN-GS-2K	1.97	0.09	CDN					X	X	X	X						
CDN-GS-5J *	4.96	0.21	CDN					X	X								
CDN-GS-1P5K	1.44	0.065	CDN							X	X	X	X	X			
CDN-GS-5P *	4.78	0.155	CDN							X	X	X		X			
CDN-GS-1P5P	1.59	0.075	CDN									X	X	X			
CDN-GS-5T *	4.76	0.105	CDN									X	X	X	X		
CDN-CM-26 **	0.372	0.024	CDN										X		X		
CDN-GS-1P5Q	1.329	0.05	CDN											X	X		
CDN-CM-43	0.309	0.02	CDN												X	X	X
CDN-GS-1P5R	1.81	0.07	CDN												X	X	X
CDN-GS-4H	5.01	0.15	CDN												X	X	X
CDN-GS-4L	4.01	0.15	CDN														X
CDN-GS-5X	5.04	0.165	CDN														X
CDN-GS-P4J	0.479	0.024	CDN														X
CDN-GS-P5H	0.497	0.028	CDN														X
CDN-GS-1P5T	1.75	0.085	CDN														X

Notes: *Denotes CRM is also certified for silver. ** Denotes CRM with a provisional or indicated silver value.

Sixteen samples of 636 returned results greater than the 45 ppb tolerance limit, and of those 16, three lay within the mineralized zone. Two were sample misallocations, where a standard was used instead of a blank, and the remaining sample is not considered by the author to be of significant impact to the resource.

11.3.1.1.3 Performance of Duplicate Samples

During the 2008 drill campaign, Treasury Metals did not insert any duplicate samples into the sample stream.

11.3.1.1.4 Check at Umpire Laboratory

In many QA/QC programs, pulp duplicates are also submitted for external check analyses at an umpire laboratory to provide an independent check of relative bias and accuracy. The submission rate is usually 5% of the pulps. Treasury Metals did not submit check samples to a second laboratory during the 2008 exploration program.

11.3.1.2 2009 QA/QC Program

Treasury Metals undertook a similar QA/QC program throughout the 2009 drill program, with every tenth sample being either a low- or medium-grade CRM or blank. The insertion of quarter-core (field) duplicates was implemented for this program. Insertion rates are summarized in Table 11-2.

Table 11-2: QA Sample Insertion Rate for 2009 Drill Program

Insertion Rate	QA/QC Sample Type
10 samples	
Insert	Low-grade CRM
10 samples	
Insert	Blank
5 samples	
Collect	Quarter-core duplicate
5 samples	
Insert	Medium-grade CRM
10 samples	
Insert	Blank
5 samples	
Collect	Quarter-core duplicate
5 samples	
Insert	High-grade CRM
10 samples	
Insert	Blank

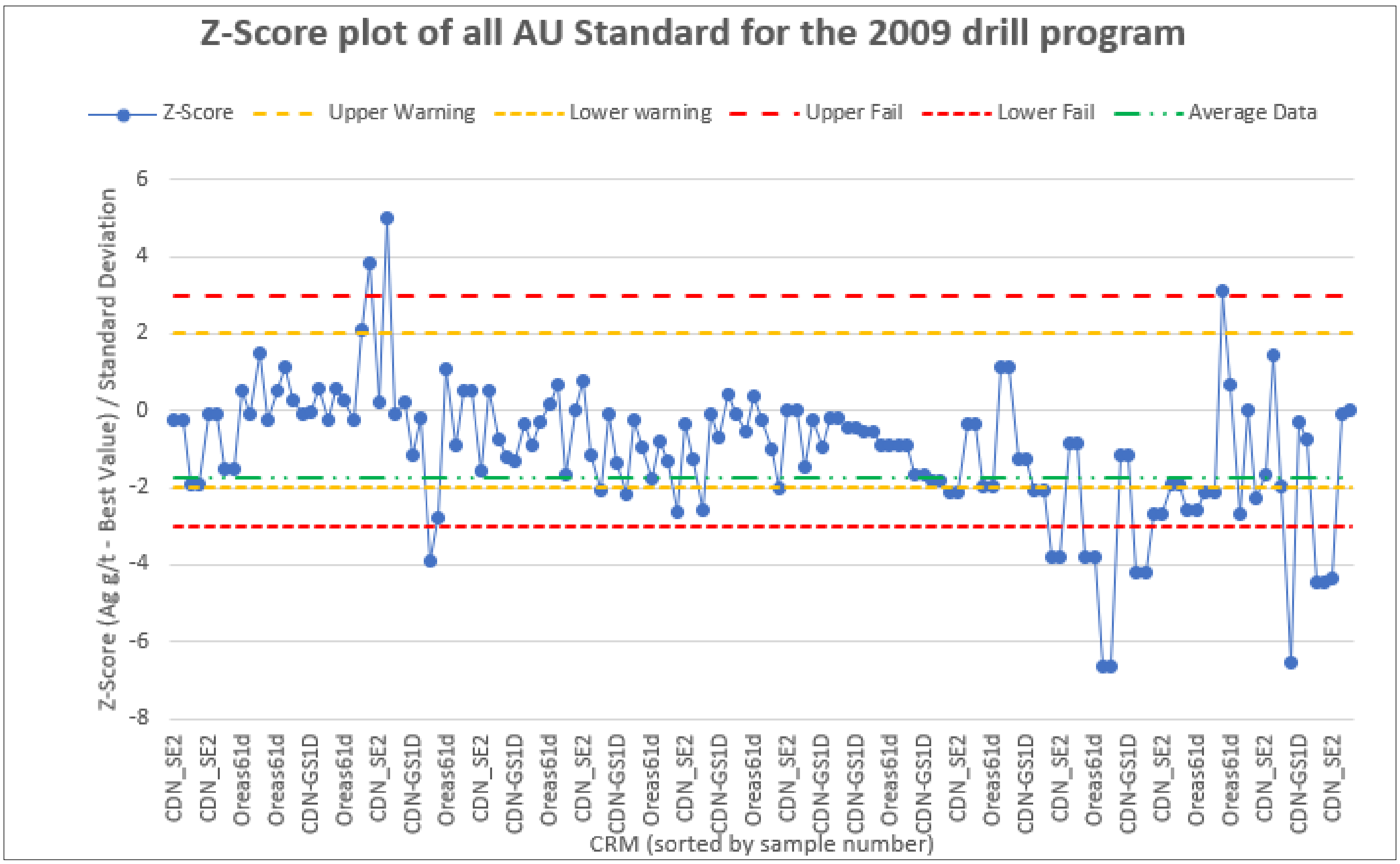
11.3.1.2.1 Performance of Certified Reference Materials

The Company utilized five CRMs to monitor gold results for the 2009 drill program, including the OREAS 61D, CDN-GS-1D, CDN-GS-5D and the CDN-SE2 (Table 11-1).

CRM results were monitored the same as in the 2008 drill program. The majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. There were eight failures within the mineralized zone, so Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent six samples in the batch to confirm results.

The Z-Score chart of the CRM results are shown in Figure 11-1. The chart reveals a low failure rate, but also a degradation in precision with higher sample number near the end of the program.

Figure 11-1: Z-Score Chart for 2009 CRM



Source: AGP, 2020

11.3.1.2.2 Performance of Blank Material

The same blank material that was used for the 2008 QC monitoring was used again in 2009. A tolerance limit of 45 ppb was set by the Company to evaluate for contamination.

There were 184 data points for the blank material and all results were below three times the detection limit of the analysis type (15 ppb).

11.3.1.2.3 Performance of Quarter-Core Duplicate

Details on the performance of the quarter-core duplicate could not be located.

11.3.1.3 2010-2011 QA/QC Program

Treasury Metals continued their QA/QC program in similar fashion throughout the 2010 and 2011 drill program.

11.3.1.3.1 Performance of Certified Reference Materials

The Company utilized seven CRMs to monitor gold results for the 2010 and 2011 drill programs, including the CDN-SE-2, CDN-GS-1F, CDN-GS-5D, OREAS 61D, CDN-CGS-13, CDN-CM-6, and CDN-ME-6 (Table 11-1).

CRM monitoring continued in the same fashion as the previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent five samples for any batches where failures occurred within the mineralized zone (if any samples were greater than 5.0 g/t Au) to confirm results. Some failures were considered to be misallocated blanks or standards and the records were changed accordingly. The QP reviewed the chart provided in a report authored by Julie Selway and the chart reported by A.C.E. Howe Ltd (2012) and concurred that standard Oreas-61D was showing a low bias first quarter of the program then a then a high bias in the last three quarters of the program.

11.3.1.3.2 Performance of Blank Material

The same blank material that was used for the 2008 and 2009 QC monitoring was used again in 2010. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There was a total of 291 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (15 ppb). One result was considered a misallocated CDN-CM-6 standard and the other sample (at 0.045 g/t Au) was considered the only failure.

11.3.1.3.3 Performance of Duplicate Samples

Treasury Metals submitted 970 quarter-core duplicate samples into the 2010 and 2011 drill programs. A.C.A. Howe (2012) reported the results of the field duplicate data and a plot of the original versus duplicate material. The data shows acceptable correlation between the original samples and quarter-core duplicates. Most deviation can be attributed to the nugget effect. A.C.A. Howe reported that very few high-grade samples were submitted and recommended that Treasury

Metals collect additional quarter-core duplicates from the mineralized zones. AGP noted that about one-third of the samples collected graded in excess of 0.2 g/t with approximately nine samples above 3 g/t.

11.3.1.4 2012-2013 QA/QC Program

The 2012-2013 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low- or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample.

11.1.4.4.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2012 and 2013 drill programs, including the CDN-GS-P2A, CDN-CM-26, CDN-GS-2K and CDN-GS-5J (Table 11-1).

A slightly higher rate of failures was noted by the Company at the commencement of the 2012-2013 drill program for the CDN-GS-2K standard, with four out of 18 failures in total. Overall, 28 standards failed, where results were greater than three standard deviations away from the CRM mean value. Out of these 28 failures, 22 samples were selectively chosen to retest due to their proximity to mineralized zones and magnitude of failure.

11.3.1.4.1 Performance of Blank Material

The same blank material was continued to be used for the 2012-2013 QC monitoring. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 197 data points for the blank material, and all results except three were below three times the detection limit of the analysis type (15 ppb). None of these failures were of significant impact to the resource estimate.

11.3.1.4.2 Performance of Duplicate Samples

The Company submitted 750 quarter-core duplicate samples for assaying during the 2012-2013 drill program. The results of the original and duplicate data display a poor precision as is often the situation with gold deposits. AGP re-plotted the data with seven outliers removed and found the regression showed a R2 value of 0.86 with a slope of regression of 0.81.

11.3.1.4.3 Re-assay Comparison

Treasury Metals re-assayed 742 samples due to failure of control samples. The re-assay results were charted in a QA/QC report. The re-assayed sampled for 2012 and 2013 compared well with the original assay as evidenced by a R2 value of 0.92 and a slope of regression of 1.08 which indicates virtually no bias.

11.3.1.4.4 Pulp Re-submitted to ALS Chemex

Treasury Metals re-submitted pulps analysed at Accurassay for five drillholes (TL13316, TL13318, TL13319, TL13322, TL13323) to be assayed at ALS labs in response to the high failure rate at the beginning of the 2013 drill program. ALS's service was prompt and had zero failed standards throughout the five holes.

11.3.1.5 2014-2015 QA/QC Program

The 2014-2015 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low- or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample.

11.3.1.5.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2014-2015 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-2K and CDN-GS-5P (refer to Table 11-1).

CRMs were monitored in the same fashion as the previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. Starting in 2015, Treasury Metals elected to re-analyse the pulps from the preceding five and subsequent five samples for any batches where failures occurred within the mineralized zone to confirm results as opposed to re-submit the entire batch.

Overall, 21 standards failed out of a total of 274, where results were greater than three standard deviations away from the CRM mean value. Out of these 21 failures, 10 samples were selectively chosen to retest due to their proximity to mineralized zones and magnitude of failure. Additional failures were considered to be misallocated blanks or standards and the records were altered accordingly.

11.3.1.5.2 Performance of Blank Material

The same blank material continued to be used for the 2014-2015 QC monitoring. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There was a total of 277 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (15 ppb). None of these failures were considered to be of significant impact to the resource.

11.3.1.5.3 Performance of Duplicate Samples

The Company submitted quarter-core duplicate samples only for assaying during the 2014-2015 drilling program. The results of the original and duplicate data were plotted on a scatter plot and poor (but acceptable) correlation was displayed for these coarse level duplicates.

Treasury Metals did not insert any other duplicate samples into the sample stream; however, Accurassay's pulp duplicates and crusher replicate samples were available for analysis. All data was analysed for gold and the pulp duplicates displayed excellent precision.

11.3.1.5.4 Laboratory change (Accurassay – ActLabs) & Assay Verification

For the 2016 drill program, Treasury Metals started using the Activation Laboratories (ActLabs) in Dryden due to the closure of the Accurassay facility in Thunder Bay.

In order to validate the analytical results from both laboratory, Treasury Metals submitted 328 pulp samples from Accurassay Laboratory for check assaying to ActLabs Laboratory in Thunder Bay. Pulp samples were taken from 29 drillholes, drilled over the 2014 to 2015 period. Samples were sent in two batches of 134 and 194 pulp samples.

Scatter plots and line graphs of the ActLabs results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples. The results indicated a good correlation with R2 of 0.99. The slope of regression was 0.95 which indicate a slight negative bias.

11.3.1.6 2016 QA/QC Program

The 2016 QA/QC program carried out by Treasury Metals followed the same protocol as earlier years, with every tenth sample being either a low or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample. The laboratory derived blank material was replaced by a crushable blank material in 2016 which is suitable to monitor contamination at the sample preparation stage.

11.3.1.6.1 Performance of Certified Reference Materials

Five CRMs were used to monitor gold results for the 2016 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P, CDN-GS-5T and CDN-GS-5P (refer to Table 11-1).

CRMs were monitored in a similar fashion as the previous years and the majority of the CRMs within the mineralized zone returned values within the acceptable limits of three standard deviations from the mean. Overall, 11 standards failed out of a total of 276, where results were greater than three standard deviations away from the CRM mean value. A slightly higher rate of failures was noted by the Company at the commencement of the 2016 drill program for the CDN-CM-26 standard, which accounted for five out of the 11 failures in total. There was also a slightly elevated failure rate for the CDN-GS-5T standard, which accounted for three out of the 11 failures. None of these failures were considered to be of significant impact to the resource.

Standard CDN-GS-1P5K showed a slight positive bias and CDN-GS-5T displayed a slight negative bias and resulted in the higher failure rate.

11.3.1.6.2 Performance of Blank Material

In 2016, a coarse blank made from bags of crushed granite replaced the packaged blank (CDN-BL-10) used in previous years. A total of 10 test samples were sent to the lab to ensure that the material was suitable for use. All test samples returned values below the detection limit. A tolerance limit of 15 ppb was maintained by Treasury Metals to evaluate for contamination.

There were 281 samples of blank material and all results, except one, were below three times the detection limit of the analysis type (5 ppb).

11.3.1.6.3 Performance of Duplicate Samples

The Company submitted 278 quarter-core duplicate samples for assaying during the 2016 drill program. The results of the original and duplicate data were plotted on a scatter plot and show acceptable correlation for these coarse level duplicates.

11.3.1.7 QA/QC Program

The 2017 QA/QC program carried out by Treasury Metals followed the same protocol as 2016 with the addition of checks samples submitted at Agat Laboratory.

11.3.1.7.1 Performance of Certified Reference Materials

Four CRMs were used to monitor gold results for the 2017 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P and CDN-GS-5T (Table 11-1).

CRMs were monitored in the same fashion as the previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean value. Overall, 12 standards failed out of a total of 343, where results were greater than three standard deviations away from the CRM mean value. A slightly higher rate of failures was noted by the Company at the commencement of the 2017 drill program for the CDN-CM-26 standard, with five out of twelve failures in total. There was also an elevated failure rate for the CDN-GS-5T standard with six out of twelve failures. Out of these 12 failures, 11 were actual failures not selected for retesting as the failures were considered to have minimal impact to the resource. The remaining sample flagged for failure, was not an actual failure but a misallocated standard that fell within acceptable limits.

Again, Standard CDN-GS-5T showed a slight negative bias during the first half of the drill program then a slight positive bias in the second half of the program. This suggests that a change occurred at the laboratory or slight degradation of the standard material.

11.3.1.7.2 Performance of Blank Material

The same blank material was again used for the 2017 QC monitoring (coarse crushed granite). A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 343 samples of blank material and all results, except five, were below three times the detection limit of the analysis type (5 ppb). None of these failures were considered to be of significant impact to the resource.

11.3.1.7.3 Performance of Duplicate Samples

The company submitted 341 quarter-core duplicate samples for assaying during the 2017 drill program. The results of the original and duplicate data were plotted on a scatter plot and show acceptable correlation for these coarse level duplicates.

11.3.1.7.4 Agat Laboratory Check Samples

Treasury Metals submitted 172 pulp samples to AGAT Laboratory located in Mississauga, Ontario for check assaying from ActLabs Laboratory in Thunder Bay. Pulp samples were taken from 10 drillholes drilled during 2017.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

The results of this program produced an R2 of 0.95 with a slope of regression of 1.06 with 3 outliers removed from the data set. The average differences between the assays were 0.001 g/t Au indicating no apparent biases.

11.3.1.8 2018 QA/QC Program

The 2018 QA/QC program carried out by Treasury Metals followed the same protocol implemented in 2016 including the use of an umpire laboratory.

11.3.1.8.1 Performance of Certified Reference Materials

Six CRMs were used to monitor gold results for the 2018 drill program, including the CDN-CM-26, CDN-GS-1P5K, CDN-GS-1P5P, CDN-GS-1P5Q, CDN-GS-5P and CDN-GS-5T (Table 11-1).

CRMs were monitored in the same fashion as previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. Overall, 23 standards failed out of a total of 569, where results were greater than three standard deviations away from the CRM mean value. A higher rate of failures was noted by the Company for the CDN-GS-5T standard, accounting for 11 out of the 23 failures. Of these failures, 8 standards were selected for retesting due to their proximity to significant mineralization. All standards selected for retesting have fallen within acceptable limits and no further action is deemed necessary. The remaining failed standards were not considered to be of significant impact to the resource.

Standard CDN-CM-26 showed a slight positive bias.

11.3.1.8.2 Performance of Blank Material

The Company continued to use the coarse crushed granite blank material for the 2018 QC monitoring program. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 569 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (5 ppb). Neither of the two failures was considered to be of significant impact to the resource

11.3.1.8.3 Performance of Duplicate Samples

The Company submitted 569 quarter-core duplicate samples for assaying during the 2018 drill program. The results of the original and duplicate data were plotted on a scatter plot and acceptable correlation is displayed for these coarse level duplicates.

11.3.1.8.4 Agat Laboratory Check Samples

Treasury Metals submitted 560 pulp samples to AGAT Laboratory for check assaying to ActLabs Laboratory in Thunder Bay in 2018. Pulp samples were taken from 25 drillholes, drilled over the 2018 period.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in several samples.

11.3.1.9 2019-2020 QA/QC Program

The 2019-2020 QA/QC program carried out by Treasury Metals followed the same protocol implemented in 2016 including the use of an umpire laboratory.

11.3.1.9.1 Performance of Certified Reference Materials

Six CRMs were used to monitor gold results for the 2019 - 2020 drill program, including the CDN-CM-26, CDM-CM-43, CDN-GS-1P5Q, CDN-GS-1P5R, CDN-GS-4H, and CDN-GS-5T (Table 11-1).

CRMs were monitored in the same fashion as previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. Overall, 23 standards failed out of a total of 357, where results were greater than three standard deviations away from the CRM mean value. A higher rate of failures was noted by the Company for the CDN-GS-4H standard, accounting for 14 out of the 23 failures. Of these failures, three standards were selected for retest due to their proximity to significant mineralization. All standards selected for retesting have fallen within acceptable limits and no further action is deemed necessary. The remaining failed standards were not considered to be of significant impact to the resource.

11.3.1.9.2 Performance of Blank Material

The company continued to use the coarse crushed granite blank material for the 2019-2020 QC monitoring program. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 354 data points for the blank material and all results, except one, were below three times the detection limit of the analysis type (5 ppb). The failure was not considered to be of significant impact to the resource.

11.3.1.9.3 Performance of Duplicate Samples

The Company submitted 357 quarter-core duplicate samples for assaying during the 2019-2020 drill program. The results of the original and duplicate data were plotted on a scatter plot and acceptable correlation is displayed for these coarse level duplicates.

11.3.1.9.4 Agat Laboratory Check Samples

Treasury Metals submitted 323 pulp samples to AGAT Laboratory for check assaying to ActLabs Laboratory in Thunder Bay in 2018. Pulp samples were taken from 14 drillholes, drilled over the 2019-2020 period.

Scatter plots and line graphs of the AGAT results were compared to the original Accurassay results and the comparison was very good, considering test results were from two separate laboratories. Nugget effect was also evident in a number of samples.

11.3.1.10 2021 QA/QC Program

For the 2021 drill program, Treasury Metal merged the QA/QC programs for all three deposits, Goliath, Goldlund and Miller and continued to follow the same protocols initiated in 2016 at Goliath. Every tenth sample inserted was either a low or medium-grade CRM or blank and a quarter-core (field) duplicate was inserted every 20th sample. Field duplicate samples were inserted at a rate of one in twenty on sample numbers ending in 05, 25, 45, 65 and 85.

Blank material made up of crushed rock or gravel and pre-packaged for insertion in the assay stream.

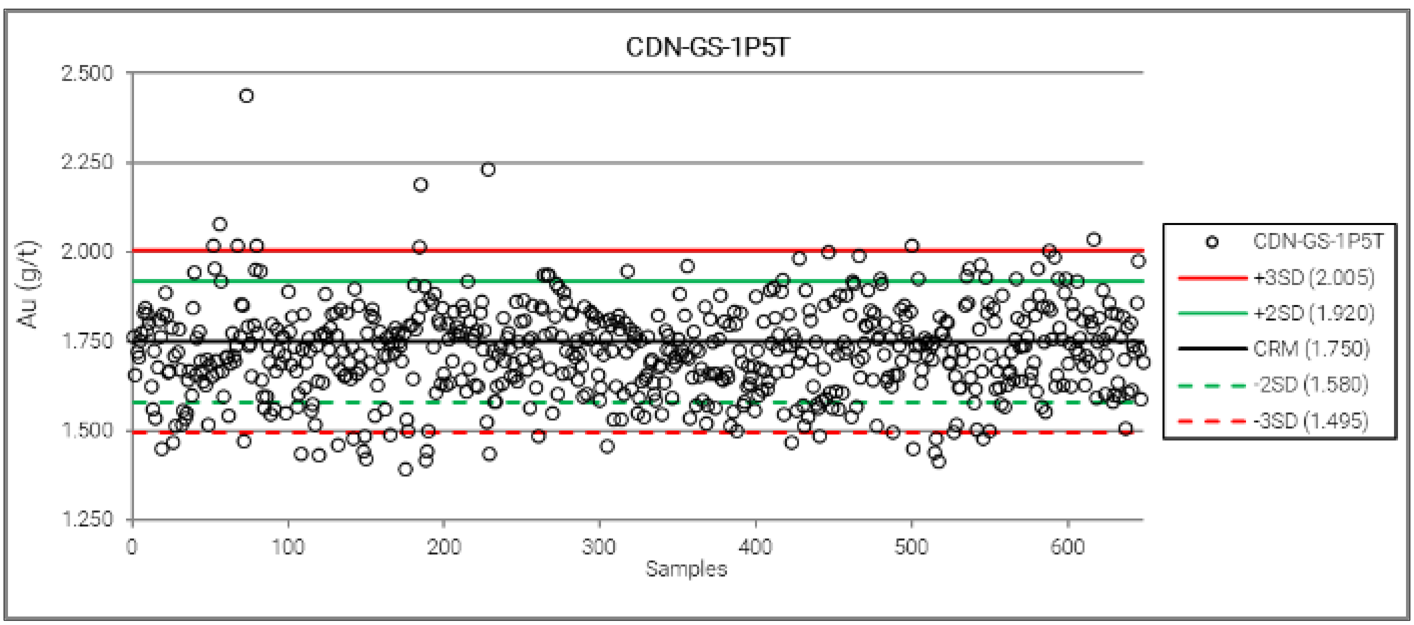
When samples are returned from the laboratory, the results are checked by the data administrator before importing onto the database. Standard results that fall outside of two standard deviations of the expected results are flagged as warning and assay results that fall outside of three standard deviations are labelled as failed. For all failed standards, Treasury Metals requests that nine samples above and below in sequence with the failed standard be re-assayed. Once satisfied that the assay results are acceptable, all assays are imported into MX Deposits by the data administrator

11.3.1.10.1 Performance of Certified Reference Materials

Five new CRMs were used to monitor gold results for the 2021 drill program, CDN-GS-1P5T, CDM-GS-4L, CDN-GS-5X, CDN-GS-P4J and CDN-GS-P5H (Table 11-1).

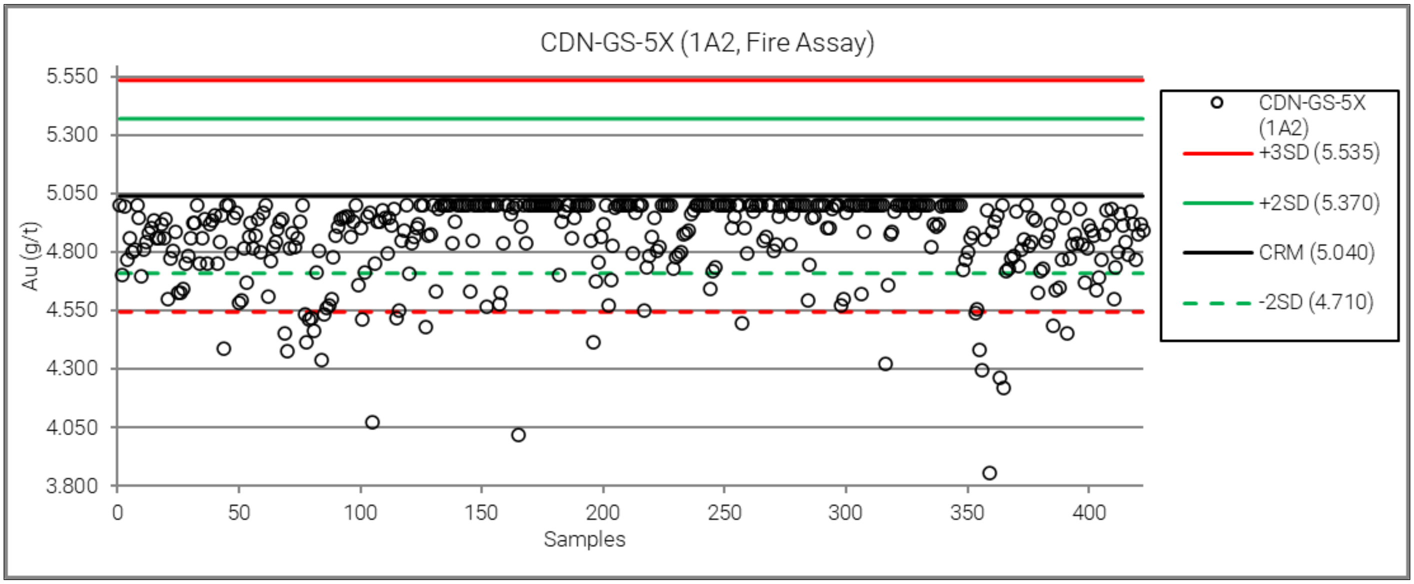
CRMs were monitored in the same fashion as previous years and the majority of the CRMs within the mineralized zone returned values within three standard deviations from the mean. Overall, from 2,056 CRM analysed, 100 samples failed with results greater than three standard deviations away from the CRM mean value (53 samples from Goldlund, 32 from Goliath and 15 from Miller). Most standard performed well with two standards accounting for 66 of the 100 failures. Standard CDN-GS-1P5T with 34 failures shows a wide spread of values spanning the ±3 standard deviation (SD) but showing no evidence of bias (Figure 11-2). Standard CND-GS-5X with 32 failures shows that the lab is consistently reporting a lower value by fire assay (1A2 method) for this standard (figure 11.3). The failure rate improves when the gravimetric method is applied (1A3 method) (Figure 11-4). The failed standards are not considered to be of significant impact to the resource.

Figure 11-2: Performance of Standard CDN-GS-1P5T



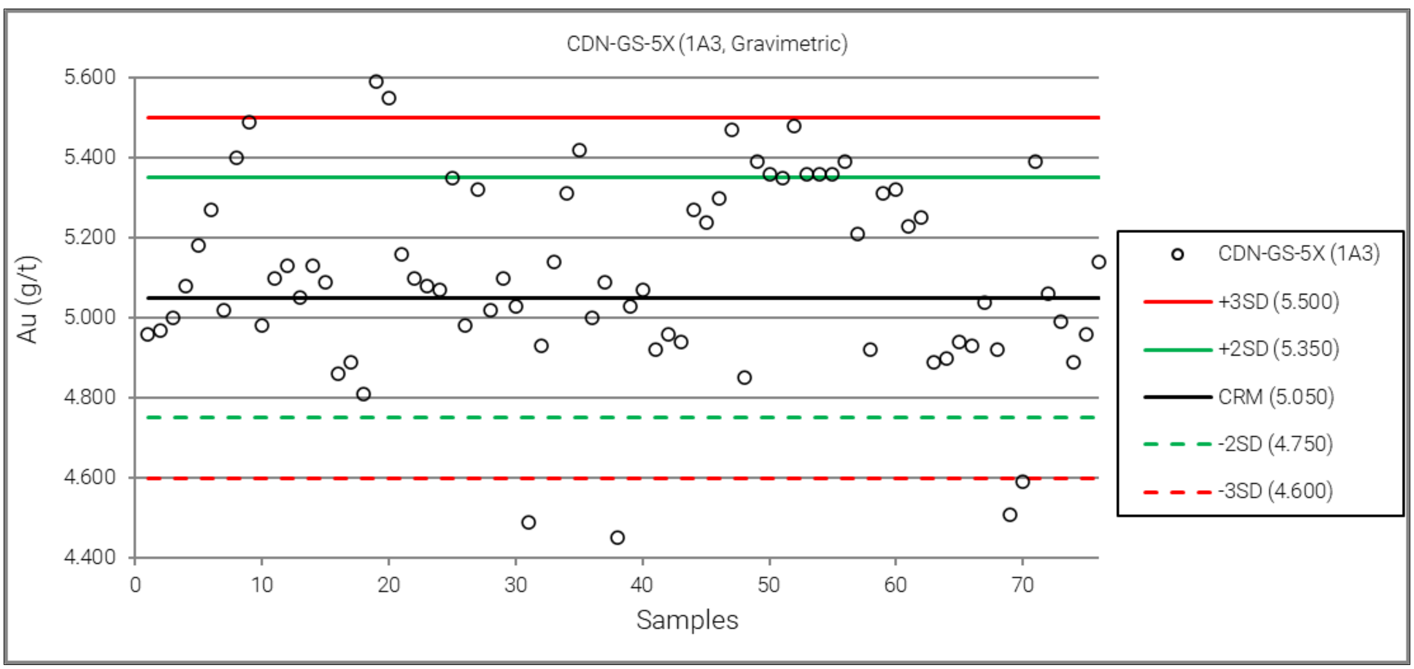
Source: SRK, 2023

Figure 11-3: Performance of CRM CDN-GS-5X (1A2 Fire Assay Method)



Source: SRK, 2023

Figure 11-4: Performance of CDN-GS-5X (1A3 Gravimetric Assay Method)



Source: SRK, 2023

11.3.1.10.2 Performance of Blank Material

The company continued to use the coarse crushed granite blank material for the 2021 QC monitoring program. A tolerance limit of 15 ppb was set by Treasury Metals to evaluate for contamination.

There were 2028 data points for the blank material and all results, except two, were below three times the detection limit of the analysis type (5 ppb). The failure was not considered to be of significant impact to the resource.

11.3.1.10.3 Performance of Duplicate Samples

The Company submitted 2,027 quarter-core duplicate samples for assaying during the 2021 drill program (672 from Goliath, 1,192 from Goldlund and 163 from Miller). The results of the original and duplicate data were plotted on a scatter plot and demonstrate acceptable correlation with scatter to be expected for gold deposits with coarse gold. All Goliath duplicate samples returned an R^2 value of 0.53 while the Goldlund samples returned an R^2 of 0.57 for samples with less than 1.0 g/t gold. Goldlund samples with values greater than 1.0 g/t gold only had a R^2 value of 0.39 which could be indicative of a higher nugget effect for the Goldlund and Miller deposits, however, the dataset is relatively small with only 32 pairs.

11.3.2 Goldlund

For samples prior to 2006, it is not known if any QA/QC programs were carried out, other than those inserted by the respective assay laboratories at the time.

11.3.2.1 2007-2021 QA/QC Program

Both Tamaka (2007 to 2014) and First Mining (2017 to 2020) carried out QA/QC programs that consisted of the insertion and analysis of blanks, CRMs, and duplicate samples to monitor the precision and accuracy or the reliability of the assay results from their drilling and sampling programs. This is in addition to the quality control samples that are inserted by the respective assay laboratories that would consist of blanks, standards, and duplicates.

11.3.2.1.1 Tamaka, 2007-2008

Tamaka's 2007 and 2008 QA/QC program consisted of the insertion of blanks and CRM samples into the sample stream at specified intervals. The standards were inserted every 20th sample, or 5% of the samples, while blanks were inserted every 30th sample, or 3% of the samples. Tamaka did not include any field duplicates in the QA/QC program. In addition to the Tamaka field-inserted QA/QC program, Accurassay operates its own QA/QC protocols. The laboratory inserts quality control materials, blanks, and duplicates with each analytical batch.

The blanks were obtained from ALS Chemex as pre-packaged samples. There were 741 results for the blanks in the QA/QC data files with 40 failures, a failure rate of 5.4%. These blanks have assayed more than 0.022 g/t, the upper control limit. This was a concern for Tamaka, and they replaced this standard with the Nelson granite in future QA/QC programs.

There were 10 different CRM samples incorporated into the samples for assay for the 2007-2008 drilling program. All 10 standards were purchased from Rocklabs (part of Scott Automation or SCOTT® since 2008), and range in expected value from 2.645 g/t Au to 30.104 g/t Au. Table 11-3 lists the standards with their expected values and standard deviation, along with the number of assay results and the average grade of the assays. Those assays that were outside the limit of ±3 standard deviations were considered failures.

There were 1,355 assays of the various standards with 27 being outside the acceptance criteria, or an overall failure rate of approximately 2%. Failure rates for the individual standards range from 0.0% up to 6.7%, with only one being more than 3%, as shown in Table 11-3. The average assayed grade of the standards is typically below the expected value for all the standards. These results confirm that Accurassay was producing sufficiently accurate and precise results such that these assays can be considered reliable.

Table 11-3: Summary of SRM used for the 2007-2008 Drill Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2007-2008	Accurassay	FAAU	Rocklabs	OXP39	14.890	0.090	184	13.468	4	2.2
	Accurassay	FAAU	Rocklabs	OXP61	14.920	0.130	174	13.916	3	1.7
	Accurassay	FAAU	Rocklabs	SJ32	2.645	0.027	45	2.439	3	6.7
	Accurassay	FAAU	Rocklabs	SL34	5.893	0.057	136	5.555	0	0.0
	Accurassay	FAAU	Rocklabs	SL46	5.867	0.066	209	5.549	6	2.9
	Accurassay	FAAU	Rocklabs	SN26	8.543	0.072	92	8.168	2	2.2
	Accurassay	FAAU	Rocklabs	SP27	18.100	0.270	87	17.637	2	2.3
	Accurassay	FAAU	Rocklabs	SP37	18.140	0.150	124	16.555	3	2.4
	Accurassay	FAAU	Rocklabs	SQ36	30.040	0.240	133	28.766	1	0.8
	Accurassay	FAAU	Rocklabs	SQ28	30.104	0.300	171	28.573	3	1.8

11.3.2.1.2 Tamaka, 2011 & 2012

Tamaka's 2011 and 2012 QA/QC programs consisted of the insertion of blanks, CRM samples, field duplicates of one-quarter core and coarse duplicates from coarse reject material into the sample stream at specified intervals. The standards were inserted every 20th sample, while blanks were inserted every 30th sample. Field and coarse duplicates were inserted into the sample stream only for the latter portion of the 2011 drilling campaign with a frequency of one field duplicate every 30th sample, and one coarse duplicate every 30th sample. In addition to the Tamaka field-inserted QA/QC program, Accurassay conducts their own QA/QC protocols consisting of quality control materials, blanks, and duplicates with each analytical batch.

The blank sample material was obtained from the Nelson granite quarry near Vermillion Bay, in Northwestern Ontario. There were 400 assays of blank material in the QA/QC data files with only 10 failures, which are blanks that assayed more than the upper control limit of 0.013 g/t Au. This failure rate is considered as acceptable.

The CRMs were obtained from RockLabs (part of Scott Automation or SCOTT® since 2008), and from Geostats Pty Ltd. A total of 11 different standards were used during the 2011 and 2012 sampling campaigns with three in use at any one time. Table 11-4 lists the different standards and a summary of the results, including the number of failures and the percentage of failures. There is a total of 568 assays for the standard material with only 11 failures, which are samples that are outside the ± 3 standard deviations. This is a failure rate of approximately 2%, which is acceptable. The failure rates for the individual standards are shown in Table 11-4 and they range from 0.0 to 11.4%. The failure rate for standard G907-2 is high, but there are only 35 assay results for that standard. The performance of the other standards is acceptable.

Table 11-4: Summary of Standards for the 2011-2012 Drill Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2011-2012	Accurassay	FAAU	Geostats	G907-2	0.890	0.060	35	0.944	4	11.4
	Accurassay	FAAU	Geostats	G302-6	0.990	0.050	50	1.023	0	0.0
	Accurassay	FAAU	RockLabs	SH55	1.375	0.014	42	1.314	2	4.8
	Accurassay	FAAU	RockLabs	SJ53	2.637	0.016	42	2.548	0	0.0
	Accurassay	FAAU	Geostats	G301-10	5.570	0.210	85	5.591	3	3.5
	Accurassay	FAAU	RockLabs	SL46	5.867	0.066	60	5.584	2	3.3
	Accurassay	FAAU	Geostats	G308-5	13.300	0.56	30	13.417	0	0.0
	Accurassay	FAAU	Geostats	G904-3	13.660	0.620	52	13.491	0	0.0
	Accurassay	FAAU	RockLabs	OxP76	14.980	0.080	56	14.554	0	0.0
	Accurassay	FAAU	RockLabs	SP37	18.140	0.15	58	16.799	0	0.0
	Accurassay	FAAU	RockLabs	SP49	18.340	0.12	58	16.799	0	0.0

The field duplicate and coarse duplicate results are summarized in Table 11-5. As this program was carried out in the latter part of the 2011 drilling program, there are a limited number of results. The failure rates of 13.5% for the field duplicates and 15.8% for the coarse duplicates, as shown in Table 11-5, are typical for this style of gold mineralization. The presence of coarse gold in samples often results in poor coarse duplicate results.

Table 11-5: Summary of Duplicates Samples for the 2011-2012 Drill Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2011	Accurassay	FAAU	Field Dup.	37	0.930	3.155	0.992	30%	5	13.5
2011	Accurassay	FAAU	Field Dup.	38	0.497	0.519	0.773	20%	6	15.8

Considering the good results observed for the blanks and standards, it appears that Accurassay, which assayed the samples for the 2011 and 2012 sampling campaigns, has produced sufficiently accurate and precise results such that these results can be considered reliable.

11.3.2.1.3 Tamaka, 2013-2014

The 2013-2014 QA/QC program consisted of the insertion of CRMs, blanks, field duplicates, and coarse duplicates into the sample stream at specified intervals. QA/QC samples were inserted every 30th sample such that for each group of 30 samples there was one of three standards: one blank, one field duplicate, and one coarse duplicate. This gives an overall insertion rate for the QA/QC samples of approximately 12%, which is believed to be sufficient to determine the reliability of the assay results.

The blank sample material was obtained from the Nelson granite quarry near Vermillion Bay, in Northwestern Ontario. There were 238 assays of blank material in the QA/QC data files with no failures, which are blanks that assayed more than the upper control limit of 0.010 g/t Au. Accurassay's results for the blank samples are considered as good.

The CRMs or "standards" were obtained from Geostats Pty Ltd. Three different standards were used during the 2013-2014 QA/QC program. Table 11-6 lists the different standards and provides a summary of the results, including the number of failures and the percentage of failures. There is a total of 274 assays for the standard material with only 11 failures, which are samples that are outside the ±3 standard deviation acceptance criteria. This is a failure rate of approximately 4%, which is acceptable. The failure rates for the individual standards, shown in Table 11-6, range from 3.4 to 5.2%. The performance of the standards for the Accurassay laboratory results is considered acceptable.

The field duplicate and coarse duplicate results are summarized in Table 11-7. The results for the coarse duplicates are good, with a failure rate of 8 out of 268 (3%). The average grades are also similar, and the linear correlation is strong at 0.96. The results for the field duplicates are acceptable, with a failure rate of 17 out of 268, or 5.6%. This higher failure rate for the field duplicates is likely due to the nature of the "nuggety" gold mineralization at Goldlund.

Table 11-6: Summary of Standards for the 2013-2014 Drill Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Average Assay Au (g/t)	No. of Failures	% Failures
2013-2014	Accurassay	FAAU	Geostats	G907-2	0.890	0.060	89	0.905	3	3.4
	Accurassay	FAAU	Geostats	G301-10	5.570	0.210	89	5.418	3	3.4
	Accurassay	FAAU	Geostats	G308-5	13.300	0.560	96	13.170	5	5.2

Table 11-7: Summary of Duplicate samples for the 2013-2014 Drill Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2013-2014	Accurassay	FAAU	Field Dups	303	0.037	0.072	0.503	30%	17	5.6
	Accurassay	FAAU	Coarse Dups	268	0.060	0.059	0.962	20%	8	3.0

11.3.2.1.4 First Mining, 2017-2018

The First Mining 2017-2018 QA/QC program consisted of the insertion of CRMs or "standards", blanks, field duplicate samples and coarse duplicate samples at specified intervals. Blanks and standards were inserted at a rate of one standard for every 20 samples (5% of the total), and one blank for every 30 samples (3% of the total). Field duplicates from quartered core, as well as coarse duplicates taken from 1 kg crushed rejects, were also inserted at regular intervals with an insertion rate of 4% for field duplicates, 4% for coarse duplicates and 4% for pulp duplicates. As well, selected samples were sent to Activation Laboratories (ActLabs) in Thunder Bay and Ancaster, Ontario, for independent umpire check assay.

In addition to the QA/QC program implemented by First Mining, the SGS laboratories each operate their own internal QA/QC protocols, inserting quality control materials, blanks, laboratory replicates and laboratory duplicates for each analytical batch. Blank samples of barren "garden rock" purchased from a local hardware store were used. An upper control limit of 0.020 g/t Au was used to determine if there was a blank failure, indicating potential contamination between samples. Any assays above this threshold were reviewed on a case-by-case basis to determine if any corrective action was required at that laboratory.

As a general rule, for samples of granodiorite being assayed at the SGS laboratory in Vancouver, BC, if a single blank or standard was deemed to have failed, that QA/QC sample plus five samples either side in the same batch were sent for re-analysis. If a blank/standard plus one or more consecutive standards were deemed to have failed, then the failed samples plus ten samples to either side and all the samples in between were sent for re-analysis. For samples of non-granodiorite material, which were sent for fire assay at the SGS Red Lake, Ontario laboratory, if only a single standard failed within a batch where the other standards or blanks passed, the entire batch was deemed to have passed and no corrective action was taken.

A total of 600 blanks were submitted for assay for the 2017-2018 program. Two blanks from the SGS Vancouver, BC, laboratory and three from the SGS Red Lake, Ontario, laboratory exceeded the upper control limit, and a portion of those batches were re-run in accordance with the corrective action protocols detailed above. Table 11-8 shows a summary of results for the blanks from the SGS laboratories. Overall, the SGS laboratories performed well.

Table 11-8: Summary of Assay Results for Blanks for the 2017-2018 Drill Program

Year	Assay Laboratory	Method	Source	Type	No of Assays	Average Assay Au (g/t)	No. of Failures	% Failures
2017-2018	SGS Red Lake	FAAU	"Garden Rock"	blank	100	0.005	3	3.0
	SGS Vancouver	BLEG	"Garden Rock"	blank	500	0.006	2	0.4

There were essentially eight different standards used in the 2017-2019 drilling program and all were supplied by CDN Resource Laboratories Ltd. (CDN) of Langley, BC. While there were four other standards considered, they were used only 1 to 3 times so there are insufficient results for statistical analysis and their results will not be presented here. The range in expected value of the eight standards is 0.968 g/t Au to 9.0 g/t Au. A standard was deemed as a failure if the result fell outside 3 standard deviations from its expected value as defined by the standard's certificate. Any assay results outside this acceptance criteria were reviewed on a case-by-case basis to determine if any corrective action was required.

Table 11-9 presents a summary of the standards that includes the expected value and associated standard deviation, along with the number of assays, the average assay grade, the number of failures and the percentage of failures. For the SGS Red Lake, Ontario laboratory there are 101 assays of standard material and there are no failures. For the SGS Vancouver, BC laboratory there are 698 assays of standard material and there are 18 failures, or a failure rate of 2.6%, which is considered acceptable. The individual standard percentage failure rates for the SGS Vancouver, BC laboratory results ranges from 0% up to 4.4%, which is also considered acceptable.

Table 11-9: Summary of Standards used for the 2017-2018 Drill Program

Year	Assay Lab	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2017-2018	SGS Red Lake	FAAU	CDN	GS-1U	0.968	0.086	46	0.986	0	0.00
		FAAU	CDN	GS-1M	1.070	0.090	40	1.079	0	0.00
		FAAU	CDN	GS-2S	2.380	0.160	15	2.344	0	0.00
	SGS Vancouver	BLEG	CDN	GS-1U	0.968	0.086	54	0.961	1	1.85
		BLEG	CDN	GS-1M	1.070	0.090	159	1.042	7	4.40
		BLEG	CDN	GS-2P	1.990	0.150	68	1.980	2	2.94
		BLEG	CDN	GS-2R	2.030	0.140	39	1.975	1	2.56
		BLEG	CDN	GS-2S	2.380	0.160	24	2.316	0	0.00
		BLEG	CDN	GS-3P	3.060	0.180	152	2.956	1	0.66
		BLEG	CDN	GS-5M	3.880	0.380	145	3.893	5	3.45
		BLEG	CDN	GS-9B	9.020	0.750	57	8.722	1	1.75

Table 11-10 presents a summary of the duplicate assay results for the 2017-2018 drilling program. Duplicate samples, regardless of whether they were BLEG duplicates, metallic screens, or check duplicates for the umpire laboratory, utilized 1 kg splits from the original 3 kg pulverized sample. The only exception to this in the BLEG QA/QC program were the field duplicates which were done on separately prepared, quarter-core samples.

There are 420 duplicate samples assayed by the SGS Red Lake, Ontario laboratory with 20 failures, or a failure rate of approximately 5%. The field duplicate and the coarse duplicate results have low failure rates, while the pulp re-runs failure rate is higher than would be expected. However, overall, the duplicate results for the SGS Red Lake, Ontario laboratory are considered acceptable. The duplicate results for the SGS Vancouver, BC laboratory consist of five different types of samples: field duplicates, coarse duplicates, pulp duplicates, re-run of pulp duplicates and check assays, as shown in Table 11.10.

The SGS Vancouver results for the field duplicates (647) shows a high failure rate, which is an indication of the high "nugget effect" in this style of gold mineralization. The failure rate for the coarse duplicate samples (74) and pulp duplicate samples (514) are somewhat higher than preferred at 8.1 and 9.7%, respectively. However, these results are still considered acceptable, as the failure rate is less than 10%. The duplicate results for the re-run on the pulps (234 assays) shows a good failure rate of only 2.6%.

The comparison between the BLEG methodology and the screen fire assays or "metallics" assay methodology shows that 28 assays of the 294 were failures, for a failure rate of 9.5%. While this failure rate is higher than preferred, it is still considered acceptable for the comparison of two different methodologies.

The last comparison of duplicate sample results is for the SGS Vancouver versus Activation Laboratories BLEG assays. There are 326 results with 18 failures for a failure rate of 5.5%, which is considered acceptable. There is a bias in the mean of approximately 10%, with the SGS Vancouver, BC assays having a higher average grade of 2.13 g/t Au, compared to the ActLabs average grade of 1.91 g/t Au. This difference is expected given the high nugget effect observed for the Goldlund mineralization.

Table 11-10: Summary of Duplicate Results for the 2017-2018 Drill Program

Year	Assay Laboratory	Method	Type	No. of Assays	Avg. 1	Avg. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2017-2018	SGS Red Lake	FAAU	Field Dups	125	0.099	0.190	0.782	30%	3	2.4
		FAAU	Coarse Dups	116	0.080	0.061	0.811	20%	3	2.6
		FAAU	Re-Run Pulp Dups	179	0.164	0.093	0.241	10%	14	7.8
	SGS Vancouver	BLEG	Field Dups	647	0.438	0.346	0.835	30%	105	16.2
		BLEG	Coarse Dups	74	0.215	0.225	0.954	20%	6	8.1
		BLEG vs. Metallics	Check Assays	294	6.433	6.758	0.992	20%	28	9.5
		BLEG	Pulp Dups	514	0.335	0.336	0.951	10%	50	9.7
		BLEG	Re-Run Pulp Dups	234	0.516	0.498	0.997	10%	6	2.6
	SGS vs. ActLabs	BLEG	Check Assays	326	2.131	1.908	0.987	20%	18	5.5

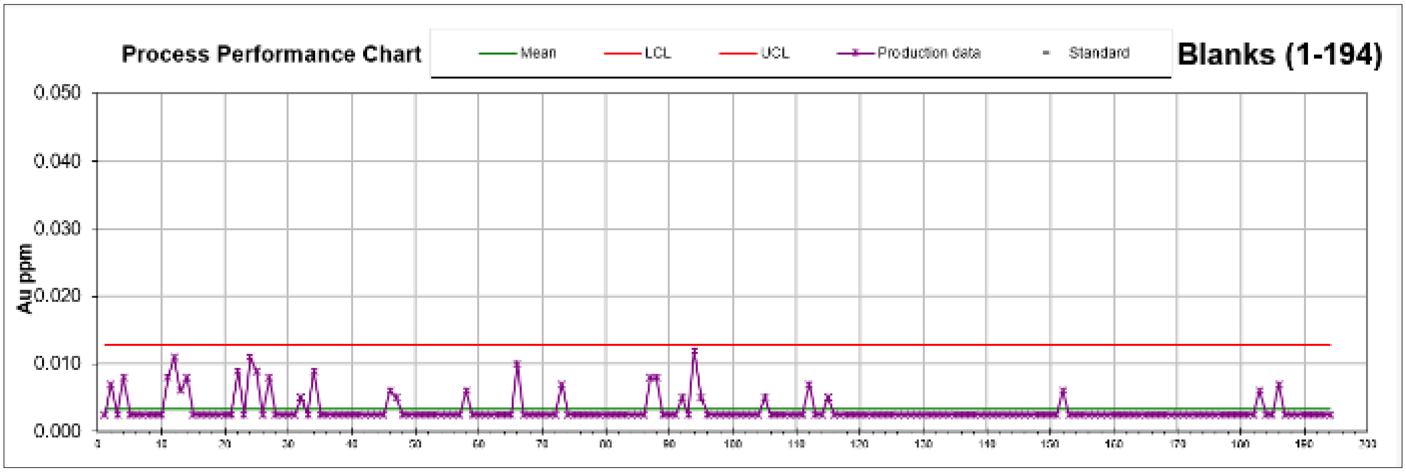
The statistical analysis of the 2017-2018 QA/QC sample assays indicates that both the SGS Vancouver and the SGS Red Lake laboratories are producing results that are sufficiently accurate and precise, such that these results can be considered as reliable.

11.3.2.1.5 First Mining, 2019-2020

The QA/QC employed by First Mining for the 2019-2020 drilling program to assess the quality of the drilling results consisted of the submission of CRMs at an insertion rate of 5%, a sample of blank material at an insertion rate of 3%, a field duplicate from quartered drill core at an insertion rate of 4%, a coarse duplicate taken from a second split of the crushed material at an insertion rate of 4% and pulp duplicates taken from pulverized material with an insertion rate of 4%. In addition to the QA/QC program carried out by First Mining, SGS also uses an internal laboratory QA/QC program consisting of CRMs, blanks, laboratory repeats and laboratory duplicates for each analytical batch.

Blanks are made from barren decorative stone purchased from a local hardware store, "garden rock". Figure 11-5 displays a control chart of the 194 assay results for the blanks inserted into the 2019-2020 sample stream, with an upper control limit of 0.013 g/t Au that is determined as 4 times the average grade of the blanks. There are no failures for the blank samples.

Figure 11-5: Control Chart of Blank Sample Results for the 2019-2020 Drill Program



Source: SRK, 2023

There were five different commercial CRMs incorporated into the 2019-2020 drillhole sample program. All five standards were prepared by CDN Resource Laboratories Ltd. of Langley, BC, and range in grade from 0.562 g/t Au up to 9.02 g/t Au. Table 11-11 presents a listing of the five standards, including the expected value and standard deviation at a 95% confidence limit, the number of assays of each of the standards, the average assay by SGS and the number and percentage of failures.

Table 11-11: Summary of Standards used for the 2019-2020 Drill Program

Year	Assay Lab.	Method	SRM Source	SRM	Expected Value Au (g/t)	95% Confidence Limits Standard Deviation	No. of Assays	Avg. Assay Au (g/t)	No. of Failures	% Failures
2019-2020	SGS	FAAU	CRL	GS-1W	1.063	0.076	63	1.055	0	0.00
	SGS	FAAU	CRL	GS-2U	2.120	0.130	75	2.114	0	0.00
	SGS	FAAU	CRL	GS-4F	3.830	0.240	62	3.856	2	3.23
	SGS	FAAU	CRL	GS-9B	9.020	0.750	39	9.033	1	2.56
	SGS	FAAU	CRL	GS-P5G	0.562	0.054	54	0.557	0	0.00

A review of the control charts for the five different standards used show that all of the standards performed very well and that the SGS laboratory assay results were similar to the expected values for each of the standards. There were two failures observed for GS-4F and one failure observed for GS-9B.

The summary results shown in Table 11-11 show that the SGS laboratories are reproducing the grade of the expected values for each of the standards and are therefore producing reliable assay results.

The 2019 and 2020 QA/QC program also included duplicate analysis using field duplicates, coarse duplicates, and pulp duplicates. Table 11-12 presents a summary of the results for the three different duplicate samples for the 2019-2020 Drilling Program. This summary includes the average grade of the original assay (Avg. 1) and duplicate assay (Avg. 2) results, along with the linear correlation coefficient. The pass/fail criteria are ±30% for field duplicates, ±20% for coarse duplicates and ±10% for the pulp duplicates. The number of failures and the percentage of failures is also provided in Table 11.12.

Table 11-12: Summary of Duplicate Sample Results for the 2019-2020 Drill Program

Year	Assay Laboratory	Method	Type	No. of Assays	Ave. 1	Ave. 2	Correlation	Pass/Fail	No. of Failures	% of Failures
2019-2020	SGS	FAAU	Field Dups	238	0.285	0.288	0.818	30%	30	12.6
2019-2020	SGS	FAAU	Coarse Dups	149	0.568	0.547	0.984	20%	5	3.4
2019-2020	SGS	FAAU	Pulp Dups	119	0.573	0.602	0.995	10%	8	6.7

There is a total of 30 failures out of 238 assay results. This is higher than the desired maximum of 10% and is an indication of the amount of variability due, largely, to the "nuggety" gold mineralization found at Goldlund. The samples with the red circles are those that are considered failures.

For the coarse duplicates, there are only 5 failures out of 149 assay results. This is a failure rate of only 3.4%, which is considered acceptable for this type of mineralization.

There are 8 failures out of 119 assay results for pulp duplicates, this is a failure rate of only 6.7% which is also considered acceptable, because it is less than the 10% failure limit.

The statistical analysis of the QA/QC sample results shows that the SGS laboratories that assayed the 2019 and 2020 drillhole samples are producing sufficiently accurate and precise results such that the 2019 and 2020 assays can be considered as being reliable.

11.3.3 Miller

11.3.3.1 First Mining 2018-2019

The QA/QC program consisted of submitting duplicate samples and inserting CRMs at regular intervals. Blanks and CRMs were inserted at a rate of one CRM for every 20 samples, and one blank for every 30 samples. Field duplicates from quartered core, as well as alternating pulp and coarse duplicates (taken from coarse reject materials or pulverized splits) were also inserted at regular intervals, with an insertion rate of 4% for field duplicates, and 4% for pulp and coarse duplicates. Check assays were submitted to a second independent laboratory. Table 11-13 summarizes the control samples.

Table 11-13: Summary of Quality Control Samples used in 2018-2019 Drill Program

Description	2018	2019
Total Number of Samples	951	2955
Number of Control Samples	180 (19%)	571 (19%)
Distribution		
Blanks	34 (4%)	116 (4%)
Standards	54 (6%)	158 (5%)
CDN-GS-5M	10	
CDN-GS-9B	3	18
CDN-GS-1U	3	52
CDN-GS-2S	17	
CDN-GS-P4E	11	
CDN-GS-P4G	10	
CDN-GS-1W		1
CDN-GS-2U		38
CDN-GS-4F		26
CDN-GS-P5G		23
Duplicates	92 (10%)	297 (10%)
Field Duplicates	44	141
Coarse Duplicates	22	81
Pulp Duplicates	26	75

11.3.3.1.1 Standards

Ten different standards were used for the QA/QC program. The standards were supplied by CDN Resource Laboratories Ltd. of Vancouver, BC. A standard was deemed suspect as a failure if the result fell outside three standard deviations (± 3STDEV) from its expected value as defined by the standard's certificate. Any assays outside of this threshold were reviewed on a case-by-case basis to determine if any corrective action was required. Table 11-14 presents a summary of failures and those resolved by reanalysis or where no further action was taken due to the occurrence within the unmineralized host rock. Ten failures occurred out of the 212 standard samples assayed. The failure rate is elevated but acceptable.

Table 11-14: Summary of Standards used in 2018-2019 Drill Program

Standards	2018	Failures	Action	2019	Failures	Action
CDN-GS-5M	10	0				
CDN-GS-9B	3	0		18	2	Reanalysis
CDN-GS-1U	3	0		52	2	No Action
CDN-GS-2S	17	1	No action			
CDN-GS-P4E	11	1	Reanalysis			
CDN-GS-P4G	10	0				
CDN-GS-1W				1	0	
CDN-GS-2U				38	0	
CDN-GS-4F				26	4	Reanalysis
CDN-GS-P5G				23	2	Reanalysis

11.3.3.1.2 Duplicate Samples

Field duplicate samples were produced by quarter-splitting the core and placing the quartered core into separate sample bags with sequential sample numbers. A field duplicate assay was taken approximately every 30 samples. A total of 185 field duplicates were assayed as part of the Miller QA/QC program. Alternating coarse and pulp duplicates were carried at every 25 samples in the sample stream. An empty sample bag containing the duplicate's sample tag was provided in the rice bag of samples shipped to the laboratory. A total of 103 coarse duplicates and 101 pulp duplicates were assayed as part of the Miller QA/QC program. Only one major departure was found in the duplicates.

The duplicate data shows expected similarities in grades; however, due to the nuggety nature of the gold mineralization, some samples are difficult to reproduce and often show differences greater than 20% difference between samples.

11.3.3.1.3 Blank Samples

Coarse blanks for the Miller drill program were taken from barren garden rocks purchased from a local hardware store. A threshold of ten times the lower detection limit (LDL) was used as a guide to determine potential contamination. Any assays above this threshold were reviewed on a case-by-case basis to determine if any corrective action was required at that laboratory.

As a general rule, if a single blank was deemed to have failed, that QA/QC sample plus five samples on either side in the same batch were sent for reanalysis. If a blank/standard plus one or more consecutive standards were deemed to have failed, then the failed samples plus ten samples on either side and all the samples in between were sent for re-analysis.

In 2018, only one sample failed the threshold limit, but no action was taken as it occurred within unmineralized host rock. There were no blanks failures reported from the 2019 drill program.

11.4 Density Determinations

11.4.1 Goliath

There are 545 bulk density measurements in the Goliath drillhole database. Bulk density was measured using the water immersion method. The core samples were weighted in air and then in water, using an Acculab VIC-612 electronic balance.

11.4.2 Goldlund

There are a total of 2,155 specific gravity measurements that were made by Tamaka, First Mining and Treasury Metals on representative pieces of drill core. The core samples were weighted in air and then in water, the buoyancy method, using an Acculab VIC-612 electronic balance, with a maximum weight of 610 g and an accuracy of 0.01 g (refer to Figure 11-3).

Figure 11-6: Bulk Density Measurement Equipment at Goldlund

Source: SRK, 2023

11.4.3 Miller

Density measurements were collected by First Mining on selected drill core samples from all 40 drillholes and all lithologies using the water immersion (wet/dry) method. A total of 101 measurements were collected during the 2018 drill program and an additional 285 measurements were collected during the 2019 drill program. The density measurements were collected by a hanging a wire cage below the scale (Acculab VIC-612) on the lower hook and the scale was zeroed. Core samples were placed within the cage and the dry weight taken. A bucket of water was raised below the hanging samples until the rock was fully submerged and not touching the bucket, the wet weight was then taken (WSP, 2020).

The wet and dry values were entered into the following formula.

$$Density = \frac{(Weight\ dry)}{(Weight\ dry - Weight\ wet)}$$

11.5 Sample Security and Storage

11.5.1 Goliath

The drill core for Goliath project was logged and split at the Goliath project site by employees or contractors at the time of drilling. Half of the core was retained for future verification and stored in racks at Goliath. Historical drill core from the Tech and Teck-Corona drilling was stored outside and has now deteriorated to a point that is it no longer useful.

All samples sent for analysis were bagged and sealed once collected and then placed in rice sacks and sealed. Samples from the Treasury Metals drill programs were stored in Treasury's field office in Wabigoon, Ontario under the supervision of Treasury staff until they were securely shipped to the laboratory for analysis.

11.5.2 Goldlund

11.5.2.1 Historical Drilling

The chain of custody for the drilling and sampling programs prior to 2006 is not documented.

11.5.2.2 Tamaka Drilling

Chain of custody and sample security are documented for the Tamaka (2007-2008, 2011, 2013-2014) drilling programs. For these drilling and sampling programs, the sample bags were sealed and kept secure by Tamaka in the Goldlund logging and sampling facility until they were transported to Accurassay in Thunder Bay, Ontario. A Tamaka employee delivered the samples to Manitoulin Transport in Dryden for delivery to Accurassay Laboratories in Thunder Bay. The laboratory returned all coarse rejects and pulps to Tamaka for storage at the Goldlund project.

11.5.2.3 First Mining Drilling

Chain of custody and sample security are also documented for the First Mining (2017-2018, 2019-2020) drill programs. For these drilling and sampling programs, the sample bags were sealed and kept secure by First Mining in the Goldlund logging and sampling facility until they were transported to the SGS Laboratories in either Red Lake, Ontario or Vancouver, BC.

11.5.2.4 Treasury Metals Drilling

The drill core for Goldlund project was logged and split at the Goldlund exploration camp by employees of Treasury Metals. Half of the core was retained for future verification and stored in racks at Goliath.

All samples sent for analysis were bagged and sealed once collected and then placed in rice sacks and sealed. Samples from the Treasury Metals drill programs were stored in Treasury's field office in Wabigoon, Ontario under the supervision of Treasury staff until they were securely shipped to the laboratory for analysis,

11.5.3 Miller

11.5.3.1 First Mining Drilling

Chain of custody and sample security are also documented for the First Mining drill program. Drill core was transported to the Goldlund Exploration camp by the drill contractor daily and stored on racks outside of the core logging area. After sampling, the sample bags were sealed and kept secure by First Mining in the Goldlund logging and sampling facility until they were transported to the SGS Laboratories in either Red Lake, Ontario or Vancouver, BC.

11.5.3.2 Treasury Metals Drilling

Drill core was transported to the Goldlund Exploration camp by the drill contractor daily and stored on racks outside of the core logging area. The core was logged and split at the Goldlund exploration camp by employees of Treasury Metals. Half of the core was retained for future verification and stored in racks at Goliath.

All samples sent for analysis were bagged and sealed once collected and then placed in rice sacks and sealed. Samples from the Treasury Metals drill programs were stored in Treasury's field office in Wabigoon, Ontario under the supervision of Treasury staff until they were securely shipped to the ActLab laboratory for analysis in Dryden.

11.6 Comments on Sample Preparation, Analyses and Security

Treasury Metals routinely charts all QA/QC samples. If a trend exists or samples deviate from the norm, either the entire batche or a number of samples surrounding the "failure" are re-submitted to the laboratory for check assays.

The data collected by Treasury Metals for the Goliath, Goldlund and Miller deposits show no evidence of systemic contamination during the assaying process and since a crushable blank material was utilized, the data shows no evidence of systemic cross-contamination between samples at the sample preparation facility.

The Treasury Metals quarter-core sample duplicate shows evidence of a rather strong nugget effect, but the variations are to be expected with gold deposits with a high nugget effect and the presence of visible gold.

11.6.1 Goliath

The Qualified Person reviewed the sample preparation, analytical and security procedures, as well as the insertion rates and performance of blanks, CRM and duplicates from the data provided and concluded that the observed failure rates are within the expected ranges and that no significant assay biases are present.

All charts and figures presented by Treasury Metals focussed on gold. Blanks and CRM that have a certified silver value should also be charted.

Based upon the review of the QA/QC program and of the analytical procedures undertaken by Treasury Metals and the previous operators at Goliath, the QP is of the opinion that the sample preparation, security , and analytical procedures are acceptable for use in the current mineral resource estimate.

11.6.2 Goldlund

The Qualified Person believes that the preparation and analyses of the samples are satisfactory for this type of deposit and style of gold mineralization and that the sample handling and chain of custody, as documented, meet standard industry practice and are acceptable for inclusion in the estimation of mineral resources and mineral reserves.

The Qualified Person has reviewed the QA/QC program and deems it to be in accordance with standard industry practice and CIM's "Exploration Best Practice Guidelines" (2018). Both Tamaka and First Mining personnel have taken reasonable measures to ensure the sample analysis completed is sufficiently accurate and precise such that the assays can be considered as reliable.

11.6.3 Miller

Based upon the review of the QA/QC program and of the analytical procedures undertaken by Treasury Metals and First Mining, the QP is of the opinion that the sample preparation, security, and analytical procedures are acceptable for use in the current mineral resource estimate.

12 DATA VERIFICATION

12.1 Introduction

The Goliath Gold Complex has been reviewed by several independent qualified persons in the past. The Goliath deposit has been reviewed by Pierre Desautels, P. Geo. (Ausenco, 2021), by Yungang Wu, P. Geo., in 2018 and 2015 (P&E.2015 and 2018). The Goldlund deposit was validated by Chris Keech, P. Geo. (Ausenco, 2021) and by Todd McCracken, P. Geo., (WSP, 2017 and 2010) and the Miller deposit was verified by Paul Daigle, P. Geo. (Ausenco, 2021).

Dr. Arseneau, QP for this section of the report, reviewed all the work completed by previous qualified persons and carried out independent reviews and checks on the historical data to verified and validate the previous independent reviews. Based on these reviews, Dr. Arseneau is satisfied that the data for the Goliath, Goldlund and Miller deposits is acceptable for inclusion in the preparation of mineral resources and mineral reserves. Dr. Arseneau accepts the work completed by the previous independent QPs as a valid verification of the historical work. The QP recognizes that the historical drilling at Goldlund could not be fully validated because of the limited information available for the drillholes drilled prior to 1970. However, because of the limited influence of the historical data on the global resources at Goldlund, the QP feels that the historical data was acceptable for inclusion in the estimate of mineral resources. In addition to the reviews by the previous independent reviews, Dr. Arseneau also carried out a site visit and a review of all recent assay data against original assay certificates.

12.2 Verification Performed by the QP

12.2.1 Site visit Validation

The qualified person carried out a site visit on July 7 and July 8 of 2021. The site visit provided an opportunity to review the property access and site facilities at Goliath and Goldlund. The surface geology at Goliath, Goldlund and Miller was examined. As part of the site visit the local geology and exploration history of the project was reviewed with Treasury metals staff. Drillhole collar locations were examined in the field and compared with locations provided in the digital database for the project. A total of 36 drill collars were verified with hand-held GPS, all collars were found to be within the margins of error allowed by the instrument.

In addition to drill collars, the core logging and sampling procedures along with the quality control and quality assurance measures were reviewed for all three deposits.

Typical drill core was reviewed for the three deposits, a total of 18 holes were reviewed and geological logging was validated along with sampling methodology. Drill core was compared with core logging sheets and procedures which included commentary on typical lithologies, alteration and mineralization styles, and contact relationships at the various lithological boundaries.

12.2.2 Assay Data Validation

Assays in the database were validated using information derived from assay values written in historical drillhole logs and original laboratory assay certificates in Excel™ and pdf formats.

The QP checked some random drill logs against the data provided from the historical assay laboratories and validated the digital database by verifying for overlapping intervals, missing intervals, and missing data. No significant errors were noted. All of the Treasury Metals assays were validated against original assay sheets provided directly from the independent assay laboratory. Most of the discrepancies noted between the GEMS database and the certificates originated from re-assays and these were mostly all resolved once the correct certificates were located.

12.2.3 Independent Sampling Validation

Independent characterization samples were collected during the site visit. The QP supervised the quartering of the core samples and personally delivered the samples to ActLabs in Dryden, Ontario for analysis. The sample analysis was completed to confirm the presence of gold in the deposit and assess differences in terms of grade ranges. Samples were analysed for gold with fire assay with gravimetric finish.

Table 12-1: Independent Sampling Validation Results

Sample No	Drillhole	From	To	QP Check Assay Au (g/t)	Treasury Metals Original Assay Au (g/t)
21592	MI-21-046	60.00	61.00	0.33	1.35
21593	MI-21-046	61.00	62.00	2.57	3.98
21594	MI-21-046	62.00	63.00	0.50	1.69
21595	TL-16-413	658.85	659.85	0.92	1.59
21596	TL-16-413	659.85	660.90	0.71	0.71
21597	TL-16-413	660.90	662.00	3.33	6.37
215798	GL-21-058	115.50	116.50	2.08	2.21
21599	GL-21-058	116.50	117.50	4.23	5.20
21600	GL-21-058	117.50	118.50	0.99	1.00

The independent check samples collected by the QP show the presence of gold in similar values to that which has been reported for the Goliath Gold Complex deposits. The QP does note that the higher-grade samples show more variability due to nugget effect which is often the situation with gold deposits with coarse gold.

12.3 Comments on Data Verification

The Qualified Person is of the opinion that the assay data for the Goliath, Goldlund and Miller deposits are of sufficient quality to be included in a mineral resource estimate supporting a prefeasibility study.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Summary

The objective of the metallurgical study was to quantify the metallurgical response of ores from deposits in the Goliath Gold Complex. The program was designed with the intent to confirm the parameters for process design criteria for leaching. The metallurgical program was conducted at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as project BL0840 in September 2021, and was performed on composites from MZ, CZ, MS, MP, and WP ore zones.

All testwork conducted as a part of this prefeasibility level project phase was intended to validate or update design choices made in previous studies.

13.2 Historical Testwork

Testwork was previously completed for other studies on the Goliath Gold Complex. Previous testwork reports consulted are shown in Table 13-1.

Report Name	Laboratory	Date	Description
KM3406	G&T Metallurgical Services Ltd.	2011	Mineralogy, comminution testing, flotation testing, gravity concentration, leach testing on a single Goliath composite sample.
KM3406	ALS Metallurgy	2012	Comminution testing, gravity concentration, leach testing on two master composite samples and 10 variability samples from the Goliath deposit.
Project No. 13665-001	SGS Mineral Services Inc.	2013	Comminution, gravity concentration, flotation testing and leach testing on 16 Goldlund samples
KM5262	ALS Metallurgy	2017	Comminution testing on ten Goliath composite samples
BL0172	Base Metallurgical Laboratories Ltd.	2017	Leaching testing and cyanide destruction testing on a Goliath master composite sample.
BL0697	Base Metallurgical Laboratories Ltd.	2020	Leaching testing on Goliath and Goldlund individual samples and cyanide destruction testing on a Goliath/Goldlunc composite sample

Historical metallurgical testing on Goliath samples showed recoveries typically in excess of 93% Au with a combination of gravity concentration and leaching at relatively coarse grinds. Comminution testing showed samples to be soft to medium hardness as measured by Bond Rod and ball mill work indices. Low weak acid dissociable cyanide concentrations (<1 mg/L) were achieved using SO_2/air cyanide detoxification testing.

Results from the only historical Goldlund program showed samples to have recoveries with gravity and leaching ranging from 85% to 96% Au with fine to moderate grinds. Comminution testing with Bond ball mill work index testing showed samples had medium to hard hardness.

Summaries of the historic testwork listed above can be found in previous technical reports on the Goliath Gold Complex, as follows:

- NI 43-101 Technical Report & Preliminary Economic Assessment of the Goliath Gold Complex for Treasury Metals Inc. by Ausenco Engineering Canada Inc. 8 March 2021.
- Technical Report and Resource Estimation Update Goldlund Gold Project, Sioux Lookout, Ontario by WSP 1 April 2019.
- Preliminary Economic Assessment Update on the Goliath Gold Project, Kenora Mining Division, Ontario by CSA Global 17 April 2017.

13.3 Prefeasibility Study Testwork

13.3.1 Sample Descriptions

Ausenco designed the latest testwork program based on preliminary review of historical sample locations and results. Objectives included the following:

- comminution testing
- bulk mineralogical analysis
- evaluate the inclusion of gravity concentration in the process flowsheet
- conduct a series of grind-recovery tests to confirm grind selection and optimize leach conditions
- based on optimal conditions from previous programs, evaluate testing conditions for typical telluride treatment (high lime (CaO) conditions, remove free gold prior to leaching with gravity concentration)
- determine optimal cyanide destruction operating parameters
- conduct tailings thickening tests.

13.3.1.1 Goliath Samples

Composite samples from MZ and CZ were selected for the testwork program. Two primary composites were selected to represent the two zones (Central and Main) within the 2020 Goliath PEA open pit. These samples were used for flowsheet development, including extended gravity recoverable gold (E-GRG) tests and cyanide destruction tests. Ten variability samples were selected to cover all pits and major lithologies for a full suite of tests. Six additional samples were selected for variability Bond ball mill tests. The main lithological units include:

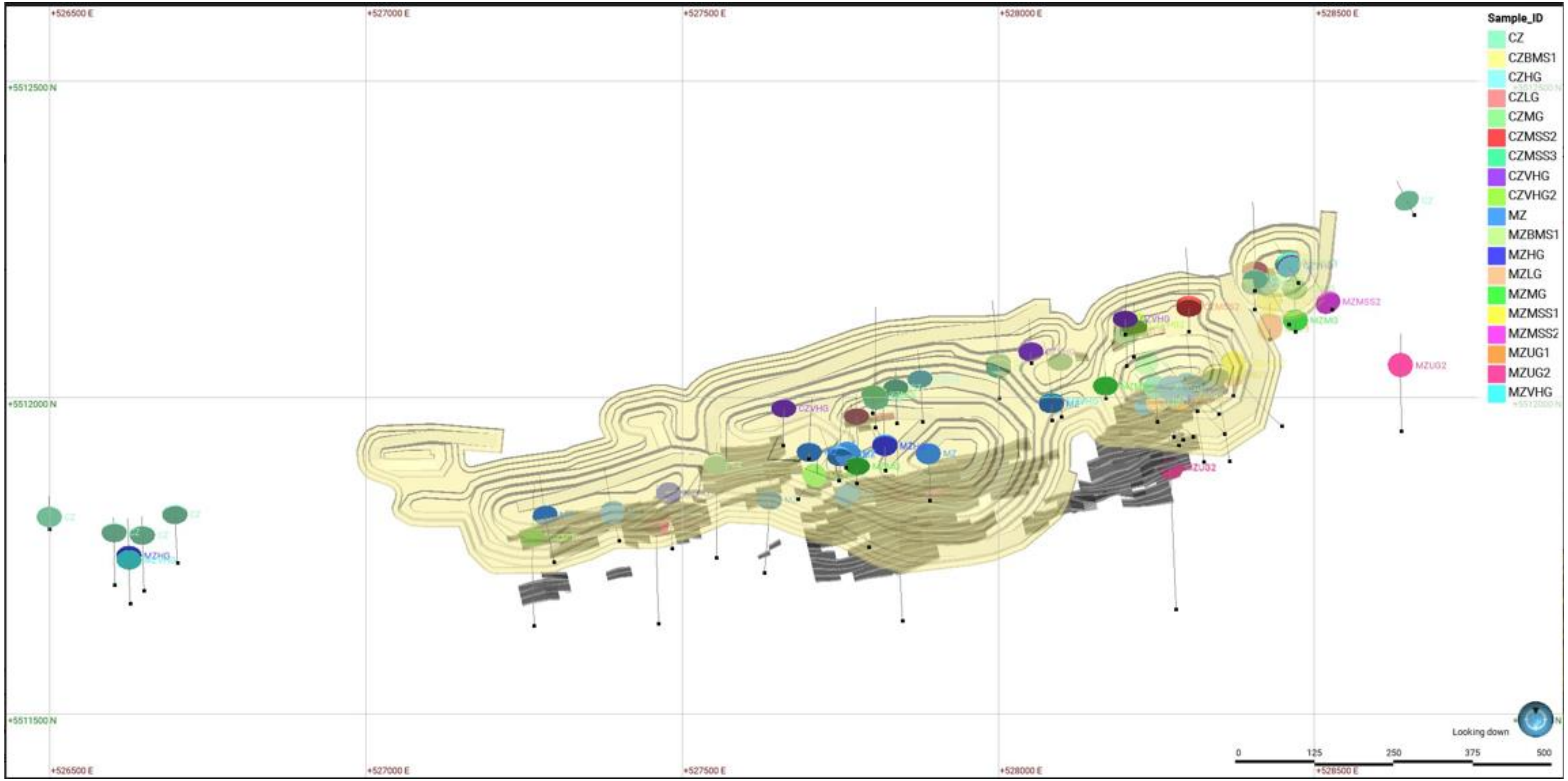
- biotite-muscovite schist (BMS)
- muscovite sericite schists (MSS)
- metasediments (MSED).

Table 13-1 shows the sample composition breakdown from the 2021 inventory. Figure 13-1 provides the physical location of the selected surface samples, whereas Figure 13-2 provides the physical location of the underground Goliath samples.

Table 13-1: Goliath sample compositions

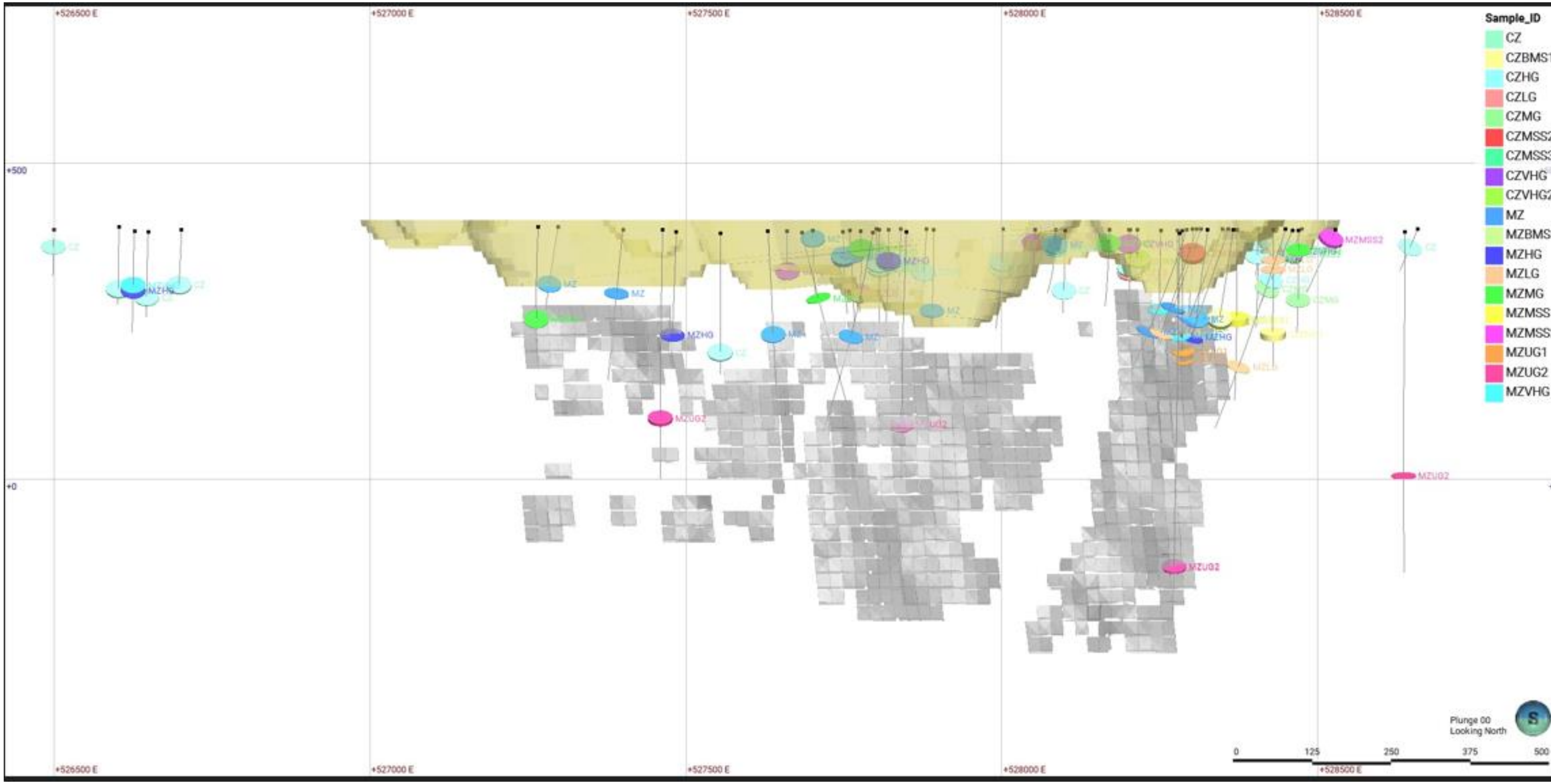
Zone	Composite Sample ID	Grade Au (g/t)	Lithology	Testing
CZ	CZ BMS1	0.42	BMS	BWi
CZ	CZ MSS2	0.45	MSS	BWi
CZ	CZ MSS3	0.02	MSS	BWi
CZ	CZ Comp.	1.27	BMS 26%, MSS 55%, MSED 19%	All except BWi
CZ	CZVHG	1.12	BMS 23%, MSS 77%	Variability
CZ	CZHG	1.77	MSS	Variability
CZ	CZMG	0.87	BMS 20%, MSS 80%	Variability
CZ	CZLG	0.34	BMS 25%, MSS 75%	Variability
MZ	MZ BMS1	0.07	BMS	BWi
MZ	MZ MSS1	0.59	MSS	BWi
MZ	MZ MSS2	0.06	MSS	BWi
MZ	MZ Comp.	1.07	BMS 51%, MSS 49%	All except BWi
MZ	MZVHG	5.63	MSS	Variability
MZ	MZHG	1.60	BMS 30%, MSS 70%	Variability
MZ	MZMG	0.79	BMS 53%, MSS 47%	Variability
MZ	MZLG	0.55	BMS 15%, MSS 85%	Variability
MZ	MZUG1	6.72	MSS	Variability
MZ	MZUG2	1.16	BMS 23%, MSS 77%	Variability

Figure 13-1: Goliath Surface Sample Locations



Source: TMI, 2021

Figure 13-2: Goliath Underground Sample Locations



Source: TMI, 2021

13.3.1.2 Goldlund Sample

Composite samples from the 2020 PEA Main Pit (MPE, MPW), and West Pit (WP) were selected for the testwork program. Six samples from unique drill holes were selected for the comminution program that included Bond rod mill, ball mill and abrasion index tests and SMC tests. Ten additional samples from unique drill holes were selected for variability Bond ball mill tests. Three primary composites were selected for flowsheet development (MPEC, MPWC and WPC) included E-GRG, grind size and leach optimization and cyanide destruction testing. Two additional composites were required (MPEC2 and MPWC2) as the initial composites were exhausted during development work. Twelve variability samples were selected from both pits.

The primary lithologies include the following:

- granodiorite (GRD)
- andesite (AND)
- gabbro (GAB)
- porphyry (POR)
- dacite (DAC)
- other lithologies include mafic volcanic (VM) and felsic intrusive (INTF).

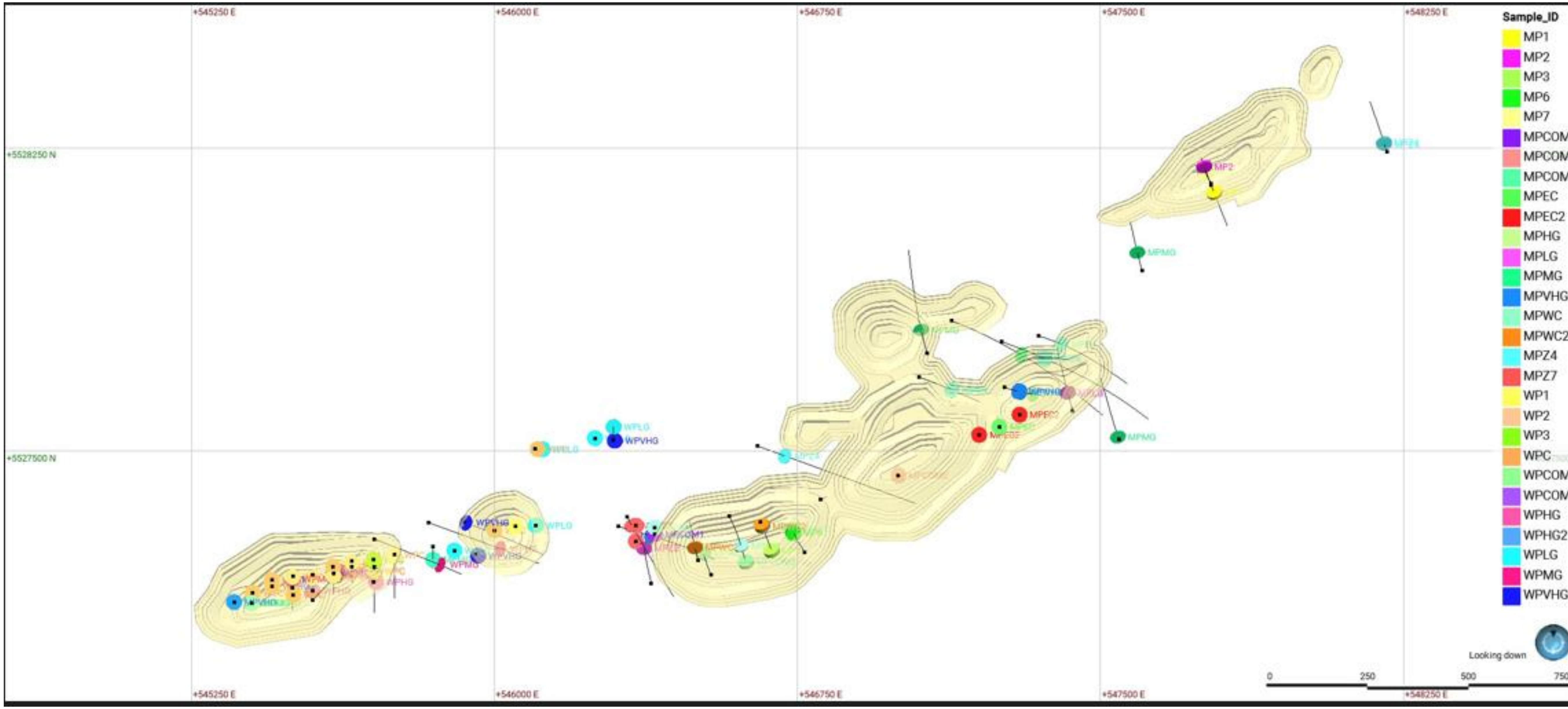
The Goldlund deposit is defined by a series of sub-parallel mineralized zones. The Main Pit is predominantly Zone 1, with lesser amounts of Zone 4 and Zone 7. The West Pit is almost entirely Zone 7. Other zones include 3, and 10.

Table 13-2 shows the sample composition breakdown from the 2021 inventory, while Figures 13-3 and 13-4 show the physical location of the selected samples from above and in cross-section.

Table 13-2: Goldlund Sample Compositions

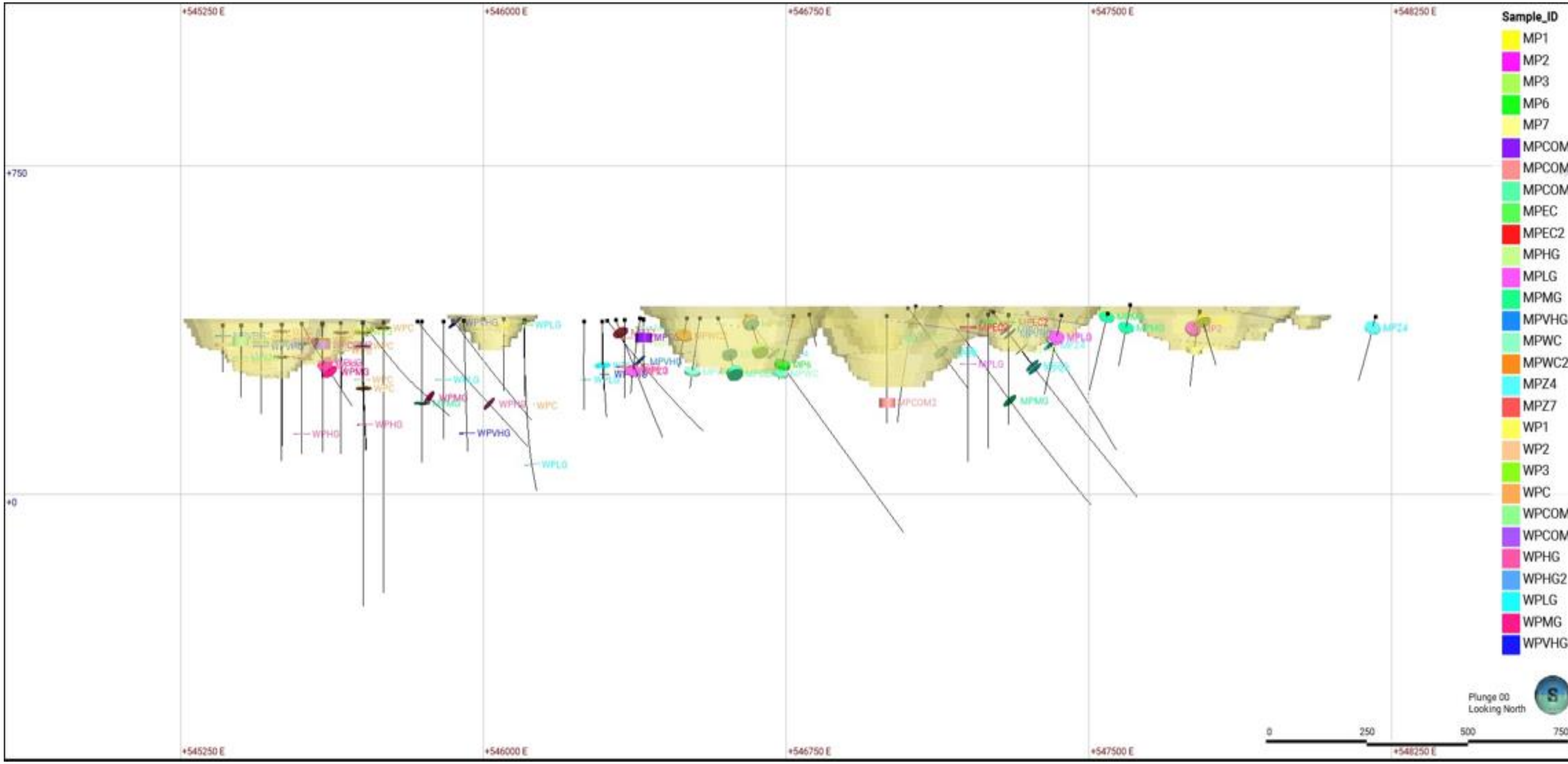
Pit	Composite Sample ID	Grade Au (g/t)	Lithologies	Zones	Testing
MP	MPCOM1	0.13	GRD	7	Comminution
MP	MPCOM2	2.08	GRD	1	Comminution
MP	MPCOM3	0.02	GRD 50%, GAB 31%, POR 8%, AND 6%, DAC 5%	1	Comminution
MP	MPCOM4	0.01	GRD91%; DAC 9%	1	Comminution
WP	WPCOM1	0.1	GRD100%	7	Comminution
WP	WPCOM2	0.4	GRD 100%	7	Comminution
MP	MP1	0.05	AND 100%	3	BWi only
MP	MP2	0.12	POR 75%; AND 25%	2	BWi only
MP	MP3	0.01	GRD 100%	1	BWi only
MP	MP4	0.00	GAB92%; GRD 8%	1	BWi only
MP	MP5	0.01	GRD	1	BWi only
MP	MP6	0.01	GRD	1	BWi only
MP	MP7	0.09	GRD	2	BWi only
WP	WP1	0.27	GRD	7	BWi only
WP	WP2	0.01	GRD 67%; AND 33%	7	BWi only
WP	WP3	0.25	GRD	7	BWi only
MP	MPEC	0.65	GRD 98%, POR 2%	1	All
MP	MPWC	1.21	GRD 97%, VM 3%	1	All
MP	MPVHG	3.72	GRD 93%, POR 7%	1, 7	Variability
MP	MPHG	1.13	GRD	1, 7	Variability
MP	MPMG	3.51	GRD 82%, AND 18%	7, 6, 10, 4, 3	Variability
MP	MPLG	0.65	GRD	1, 7	Variability
MP	MPZ4	0.67	AND 54%, DAC 19%, TRO 15%, GRD 6%, INTF 6%	4	Variability
MP	MPZ7	0.79	GRD	7	Variability
WP	WPC	0.35	GRD 99%, POR 1%	7, 1	All
WP	WPVHG	3.72	GRD 81%, POR 19%	7, 3, 10	Variability
WP	WPHG	1.30	GRD	7	Variability
WP	WPMG	2.25	GRD 97%, POR 3%	7, 10	Variability
WP	WPLG	0.84	GRD 70%, POR 20%, AND 10%	7, 3, 10	Variability

Figure 13-3: Goldlund Surface Sample Locations from Above



Source: TMI 2021

Figure 13-4: Goldlund Surface Sample Locations

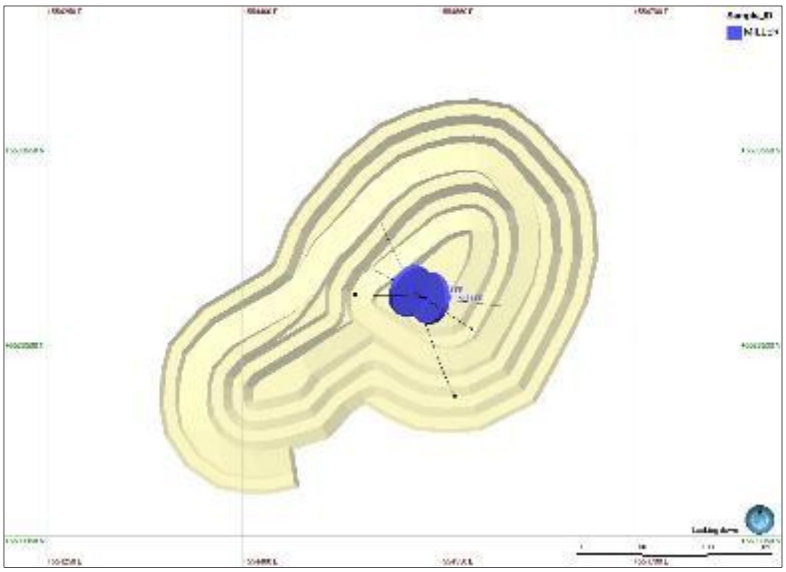


Source: TMI 2021

13.3.1.3 Miller Sample

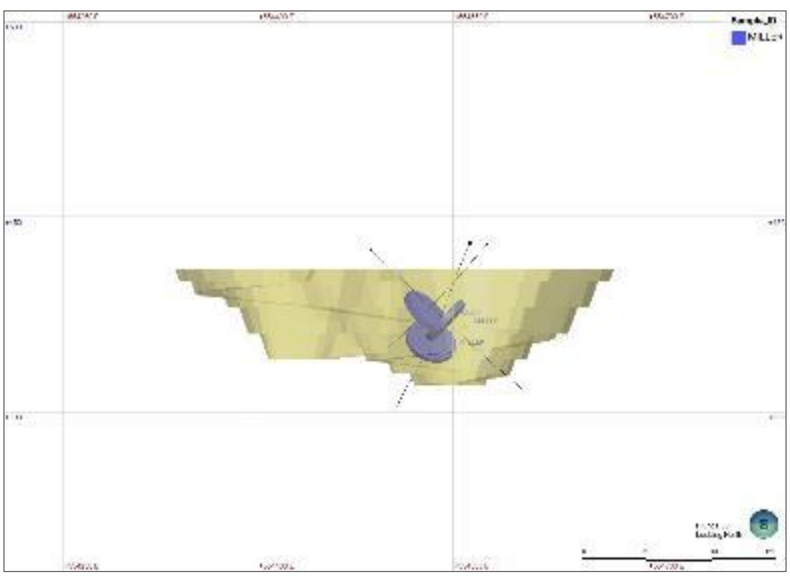
One variability sample was collected from the Miller deposit. The sample was used as variability sample and for Bond ball mill work index testing (see Figures 13-5 and 13-6).

Figure 13-5: Miller Surface Sample Locations from Above



Source: TMI, 2023

Figure 13-6: Miller Surface Sample Locations



Source: TMI, 2023

13.3.2 Head Grade Analysis

13.3.2.1 Goliath

Screened metallics gold assays were conducted on 12 composites. Aliquots of 0.5 kg from each composite were pulverized and then screened at 106 µm with the oversize and undersize fractions assayed separately. The head grade was calculated from the weighted assays from the two fractions. The results are shown in Table 13-3. Several samples contain coarse gold within the coarse fraction that is significantly above the overall mass. The lower grade samples also display similar effects with coarse gold. The results indicate coarse gold occurs in samples that is potentially amenable to gravity concentration.

Table 13-3: Goliath Screen Metallics Sample Assays

Sample	+106 µm Fraction		-106 µm Fraction	Calculated Grade (g/t Au)
	Au (g/t)	Au Distribution (%)	Au (g/t)	
MZ	1.94	9.80	0.98	1.02
CZ	10.4	29.5	1.61	2.15
MZVHG	66.8	53.7	4.24	8.52
MZHG	44.9	56.5	2.27	4.89
MZMG	12.0	42.4	1.08	1.75
MZLG	0.58	9.71	0.39	0.40
MZUG1	21.3	19.4	5.86	6.81
MZUG2	16.7	33.4	2.06	2.91
CZVHG	5.05	21.7	1.14	1.37
CZHG	8.30	16.8	2.62	2.95
CZMG	3.91	17.2	1.16	1.31
CZLG	0.50	7.78	0.33	0.34

Composites were submitted to characterize the sample with a full suite of assays which included:

- gold of all samples by direct assay
- sulphur (total S_T, sulphide sulphur S^{2-})
- ICP scan for 39 elements.

The head analysis of the samples is shown in Table 13-4 and ICP assays are shown in Table 13-5.

Table 13-4: Goliath Sample Gold, Silver and Sulphur Species Analysis

Sample	Au (g/t)	Ag (g.t)	S_T (%)	$S^=$ (%)
MZ BMS 1	0.07	2.20	0.74	0.72
MZ MSS 1	0.59	4.00	0.93	0.91
MZ MSS 2	0.06	0.70	0.74	0.71
CZ BMS 1	0.42	1.80	1.62	1.60
CZ MSS 2	0.45	2.10	1.83	1.84
CZ MSS 3	0.02	0.40	0.72	0.70
MZ	1.07	21.7	0.96	1.08
CZ	1.27	14.7	1.61	1.75
MZVHG	5.63	1.90	1.01	1.00
MZHG	1.60	5.40	1.45	1.44
MZMG	0.79	41.5	0.94	0.92
MZLG	0.55	1.90	0.81	0.78
MZUG1	6.72	42.0	1.96	1.94
MZUG2	1.16	9.40	1.20	1.18
CZVHG	1.12	8.10	2.31	2.30
CZHG	1.77	6.10	2.16	2.15
CZMG	0.87	4.90	2.01	1.99
CZLG	0.34	1.90	1.79	1.78

Table 13-5: Goliath Samples ICP Analyses

Sample	Ag (g/t)	Cd (g/t)	Cu (g/t)	Pb (g/t)	Mn (g/t)	Ni (g/t)	Zn (g/t)	Fe (%)	As (g/t)	Sb (g/t)	Hg (mg/t)	S (%)
MZ	21.7	1.8	45	473	526	8	852	1.45	38	13	226	1.09
CZ	14.7	2.5	102	192	385	37	877	2.77	74	17	472	1.71
MZVHG	21.4	2.4	82	1010	342	8	988	1.21	35	10	309	1.17
MZHG	5.4	1.6	38	361	255	7	719	1.42	56	9	195	1.41
MZMG	41.5	1.7	48	458	468	6	826	1.25	27	13	282	0.94
MZLG	1.9	1	39	156	203	6	482	0.92	17	3	130	0.84
MZVG1	42	10.7	80	2480	569	7	3930	2.02	55	34	1550	1.98
MZVG2	9.4	1.9	33	526	489	9	942	1.46	25	7	347	1.22
CZVHG	8.1	11.9	186	840	212	22	4050	2.5	61	13	3370	2.32
CZHG	6.1	2.5	90	414	131	28	969	2.19	148	43	916	2.1
CZMG	4.9	2.3	86	470	136	21	848	1.87	99	12	838	1.94
CZLG	1.9	1	32	92	298	27	393	2.04	51	4	140	1.53
MZBMS	2.2	0.8	35	148	538	6	402	1.16	17	< 2	45	0.67
MZMSS1	4	4.3	56	1010	172	8	1660	0.86	32	9	663	1.01
MZMSS2	0.7	< 0.5	23	47	501	4	171	0.97	18	< 2	17	0.66
CZBMS1	1.8	1.9	56	171	565	36	497	2.8	58	9	206	1.27
CZMSS2	2.1	4.6	55	139	58	9	1890	1.5	70	11	1740	1.91
CZMSS3	0.4	< 0.5	23	30	501	22	76	2.2	13	< 2	9	0.61

The samples tested had gold assays ranging from 0.02 to 6.72 g/t. Sulphur occurs primarily as sulphide sulphur and is associated predominantly with pyrite. Silver assays by ICP are not as accurate as fire assays but do show high concentrations in some samples. Copper concentrations are below the threshold that may cause elevated cyanide consumption. Some of the mercury (Hg) assays show elevated concentrations which should be further investigated to determine if retorting is required prior to the refining stage.

13.3.2.2 Goldlund

Screened metallics gold assays were conducted on 13 composites with the same procedure used for the Goliath samples. The results are shown in Table 13-6. The results are similar to the Goliath samples with samples showing coarse gold that may be amenable to gravity concentration.

Composites were submitted to characterize the sample with a full suite of assays which included:

- gold content of all samples by direct assay, silver of MP and WP composites by direct assay
- sulphur (total S_T, sulphide sulphur S^{2-})
- ICP scan for 39 elements.

Table 13-6: Goldlund Screen Metallics Sample Assays

Sample	+106 µm Fraction		-106 µm Fraction	Calculated Grade (g/t Au)
	Au (g/t)	Au Distribution (%)	Au (g/t)	
MPEC	0.59	2.71	0.92	0.91
MPWC	0.80	3.64	1.27	1.24
MPVHG	8.08	8.99	5.34	5.51
MPHG	2.27	8.40	1.28	1.32
MPMG	33.7	33.8	4.23	6.00
MPLG	0.60	4.11	0.79	0.78
MPZ4	3.49	19.6	0.77	0.91
MPZ7	1.05	6.89	0.90	0.90
WPC	0.50	4.28	0.70	0.69
WPVHG	30.9	31.7	3.89	5.38
WPHG	0.51	1.59	1.47	1.42
WPMG	3.08	8.97	2.06	2.12
WPLG	20.2	38.6	1.25	1.96

Observations from the pit composite head assay results:

- The samples tested had gold assays ranging from 0.01 to 8.12 g/t.
- All tested samples had silver grades above 1 g/t.
- Almost all sulphur occurs as sulphide sulphur.

Observations from the pit composite ICP analysis are as follows:

- All samples assayed low levels of Cu, Zn, and Ni.
- All samples showed low levels of mercury, less than 1 g/t.
- All samples showed tellurium concentrations comparable to the gold assays, up to 7 g/t. Gold telluride minerals were identified as the likely source of Goldlund Zone 1 samples low recoveries.
- No other potentially deleterious elements were identified.

Gold, silver and sulphur species assays for Goldlund samples are shown in Table 13-7 and head ICP assays presented in Table 13-8.

Table 13-7: Goldlund Samples Gold, Silver and Suphur Species Assays

Sample	Au (g/t)	Ag (g/t)	S_T (%)	$S^=$ (%)
MPCOM 1	0.08	1.4	0.49	0.48
MPCOM 2	0.36	1.0	0.13	0.12
MPCOM 3	0.02	1.6	0.25	0.25
MPCOM 4	<0.01	1.0	0.11	0.11
WPCOM 1	0.03	1.2	0.10	0.09
WPCOM 2	0.06	1.0	0.11	0.09
MP1	0.06	<0.2	0.41	0.19
MP2	0.26	<0.2	0.30	0.29
MP3	0.03	<0.2	0.09	0.08
MP4	0.02	<0.2	0.20	0.15
MP5	0.01	<0.2	0.10	<0.01
MP6	0.01	<0.2	0.08	-
MP7	0.18	<0.2	0.07	-
WP1	0.31	<0.2	0.23	-
WP2	0.02	<0.2	0.17	-
WP3	0.21	<0.2	0.26	-
MPEC	0.65	<0.1	0.55	0.53
MPWC	1.21	-	0.97	0.96
MPWC Recut 2	1.00	-	-	-
MPWC Recut 3	1.70	-	-	-
MPVHG	8.12	21.4	1.34	-
MPHG	1.13	<0.2	1.74	-
MPMG	3.51	3.60	1.24	-
MPLG	0.65	<0.2	0.84	-
MPZ4	0.67	0.60	0.62	-
MPZ7	0.79	<0.2	1.68	-
WPC	0.35	-	0.52	0.51
WPVHG	3.72	0.90	1.21	-
WPHG	1.30	-	0.28	-
WPMG	2.25	-	0.94	-
WPLG	0.84	1.20	0.31	-

Table 13-8: Goldlund Samples ICP Analysis

Sample	Ag (g/t)	Cd (g/t)	Cu (g/t)	Pb (g/t)	Mn (g/t)	Ni (g/t)	Zn (g/t)	Fe (%)	As (g/t)	Sb (g/t)	Hg (mg/t)	S (%)
MPCOM 1	0.4	< 0.5	6.0	< 2.0	290	3.0	31.0	3.87	13.0	5.0	65.0	0.45
MPCOM 2	< 0.2	< 0.5	2.0	< 2.0	325	2.0	24.0	3.18	< 2.0	3.0	8.0	0.09
MPCOM 3	0.5	< 0.5	16.0	< 2.0	377	5.0	39.0	3.57	< 2.0	4.0	11.0	0.2
MPCOM 4	< 0.2	< 0.5	6.0	< 2.0	308	4.0	36.0	3.48	< 2.0	3.0	8.0	0.1
WPCOM 1	< 0.2	< 0.5	27.0	6.0	321	3.0	36.0	3.79	< 2.0	2.0	6.0	0.08
WPCOM 2	< 0.2	< 0.5	7.0	12.0	369	3.0	171	4.16	< 2.0	3.0	6.0	0.09
MPEC	0.2	< 0.5	8.0	2.0	524	3.0	43.0	4.21	< 2.0	3.0	< 5	0.49
MPWC	0.3	< 0.5	28.0	2.0	457	5.0	32.0	4.49	< 2.0	4.0	10.0	0.83
WPC	< 0.2	< 0.5	7.0	3.0	350	2.0	33.0	3.81	< 2.0	3.0	< 5	0.44

13.3.2.3 Miller

Screened metallics gold assays were conducted on one Miller pit sample. The results are shown in Table 13-9. The results show the gold is predominantly fine and not likely amenable to gravity concentration.

Table 13-9: Miller Screen Metallics Sample Assay

Sample	+106 µm Fraction		-106 µm Fraction	Calculated Grade (g/t Au)
	Au (g/t)	Au Distribution (%)	Au (g/t)	
MS	0.32	1.75	0.93	0.90

The Miller pit sample was submitted to a full suite of assays which included the following:

- Gold and silver by direct assay
- sulphur (total S_T, sulphide sulphur S^{2-}).

Results of the Miller pit sample head assay are shown in Table 13-10.

Table 13-10: Miller Sample Assay

Sample	Au (g/t)	Ag (g/t)	S_T (%)	$S^=$ (%)
MS	0.94	<0.2	0.93	0.93

13.3.3 Mineralogy

13.3.3.1 Goliath

The primary Goliath development composites from the Main and Central Zones underwent QEMSCAN rapid mineral scan to identify the bulk mineralogy, as presented in Table 13-11. The deportment of sulphide minerals is presented in Table 13-12.

Key observations are as follows:

- Quartz, plagioclase, muscovite/sericite, and biotite make up the majority of non-sulphide gangue.
  - Muscovite/sericite content ranged between 20% to 25%.
- Main sulphide mineral is pyrite, which this represents in excess of 88% of the sulphide sulphur.
- Elevated levels of pyrrhotite observed in CZ Comp and represents ~8.5% of the global sulphides in this composite. Pyrrhotite can sometimes result in high oxygen and cyanide consumption in leaching but had no such effects in the samples tested in this program.
- No arsenopyrite is present; therefore, arsenic removal in the effluent treatment plant is likely not required.

Table 13-11: Goliath Samples Bulker Mineralogy

Mineral	Mineral Proportions (wt %)	
	MZ Comp	CZ Comp
Pyrite	1.60	3.58
Pyrrhotite	0.04	0.49
Chalcopyrite	0.01	0.04
Sphalerite	0.16	0.15
Other Sulphides	0.04	0.02
Quartz	48.5	48.8
Plagioclase	19.6	8.92
K-Feldspar	1.15	1.12
Sericite/Muscovite	20.0	25.0
Biotite	6.49	8.10
Chlorite	1.10	1.29
Clays	0.41	0.81
Other Silicates	0.61	1.06
Oxides	0.10	0.17
Carbonates	0.06	0.12
Apatite	0.14	0.22
Other	0.04	0.06

Table 13-12: Goliath Sulphide Minerals Deportment (% of Total S)

Mineral	% of Total S	
	MZ Comp	CZ Comp
Pyrite	91.2	88.5
Pyrrhotite	1.80	8.43
Chalcopyrite	0.33	0.63
Sphalerite	5.66	2.27
Galena	0.27	0.04
Other Sulphides	0.64	0.09
Other	0.09	0.07

13.3.3.2 Goldlund

Primary development composites from the Main Pit East, Main Pit West, and West Pit Zones underwent QEMSCAN rapid mineral scan to identify the bulk mineralogy, as presented in Table 13-13. The distribution of sulphides is presented in Table 13-14. Key observations are as follows:

- Quartz and plagioclase make up the majority of non-sulphide gangue.
- Main sulphide mineral is pyrite, which this represents typically > 92% of the sulphide sulphur.
- No arsenopyrite is present; therefore, arsenic removal in effluent treatment is likely not required.

Table 13-13: Goldlund Mineral Proportions (wt%)

Mineral	Mineral Proportions (%)		
	MPEC	MPWC	WPC
Pyrite	0.68	1.88	1.21
Pyrrhotite	0.06	0.00	0.02
Chalcopyrite	0.00	0.00	0.01
Sphalerite	0.01	0.00	0.01
Other Sulphides	0.01	0.00	0.00
Quartz	39.1	30.5	35.7
Plagioclase	36.6	46.4	45.0
K-Feldspar	0.26	0.25	0.11
Sericite/Muscovite	3.73	0.44	1.62
Biotite	1.79	2.24	1.45
Chlorite	7.40	4.33	4.04
Clays	0.87	0.81	0.68
Other Silicates	1.02	0.92	0.75
Oxides	3.27	6.17	4.34
Carbonates	4.65	5.40	4.61
Apatite	0.37	0.40	0.36
Other	0.17	0.26	0.16

Table 13-14: Goldlund Sulphur-Bearing Minerals (% of Total S)

Mineral	% of Total S		
	MPEC	MPWC	WPC
Pyrite	91.8	99.6	98.3
Pyrrhotite	5.72	0.00	0.90
Chalcopyrite	0.17	0.14	0.19
Sphalerite	0.95	0.04	0.36
Galena	0.01	0.00	0.00
Other Sulphides	0.85	0.00	0.10
Other	0.46	0.17	0.19

13.3.4 Goldlund Gold Deportment

Goldlund master composites MPEC and MPWC were included in an abbreviated visible gold deportment study by gravity concentration and by tailing size fraction, as follows:

- grind to P_{80} 100 µm
- Knelson gravity concentration followed by Mozley mineral separation
- three products produced: concentrate, middling and tail
- screen tail fraction at 53 µm.

Goldlund gold associations are provided in Table 13-15.

Table 13-15: Goldlund Gold Mineral Liberation and Association

Gold Associations	Composite	
	MPEC	MPWC
Liberated (>95% liberated)	14.9	53.6
Au – Sphalerite	0.0	0.0
Au – Pyrite	44.5	4.4
Au – Arsenopyrite	0.0	0.0
Au – Fe Oxides	0.0	0.0
Au – Carbonates	0.0	0.0
Au – Gangue	27.4	36.9
Complex Association	13.2	0.8

Gold mineral species are summarized in Table 13-16. Significant portions of gold is present as gold telluride minerals petzite, calaverite with lesser amounts as sylvanite. Gold telluride minerals can be slow leaching as a tellurium oxide layer can form on the mineral surface, preventing leaching action from cyanide. Poor leach recoveries from these samples are likely from gold telluride minerals.

Table 13-16: Goldlund Gold Mineral Species

Gold Mineral Species (by mass %)	Composite	
	MPEC	MPWC
Native Gold	51.2	39.2
Electrum	0.1	0.0
Petzite	19.0	7.9
Calaverite	28.6	52.9
Sylvanite	1.2	0.1
Hessite	0.0	0.0
Fischesserite	0.0	0.0

13.3.5 Comminution

The objective of the comminution testing was to characterize the variability of the ore competency and hardness/grindability of the deposit.

13.3.5.1 Historical Testing

13.3.5.1.1 Goliath

Bond ball mill work index testing was completed on seven Goliath samples in the 2012 ALS Metallurgy report referenced at the beginning of this section. Comprehensive comminution testing was completed on another eleven Goliath samples during the 2017 ALS Metallurgy Testwork Program. The 2017 comminution testwork included the following:

- JK drop weight and Steve Morrell comminution (SMC) tests
- Bond rod mill work index tests
- Bond ball mill work index (BWi) tests.

The results of the historical comminution testwork for the Goliath pit are summarized in Table 13-17.

Table 13-17: Historical Goliath Comminution Testwork Results

Sample	Report	Year	Ai (g)	RWi (kWh/t)	BWi, 106 µm (kWh/t)	BWi, 150 µm (kWh/t)	Axb
Master Composite 2	ALS Metallurgy, Feasibility Metallurgical Testing	2012	N/A	N/A	10.8	N/A	N/A
Variability Composite 3			N/A	N/A	13.9	N/A	N/A
Variability Composite 4			N/A	N/A	10.8	N/A	N/A
Variability Composite 7			N/A	N/A	10.2	N/A	N/A
Variability Composite 8			N/A	N/A	10.4	N/A	N/A
Variability Composite 9			N/A	N/A	8.9	N/A	N/A
Variability Composite 10			N/A	N/A	9.2	N/A	54.1
VS11_MSS_MZ_C_UG_FR	ALS Metallurgy, Metallurgical Testwork on Goliath Gold Samples	2017	0.086	11.9	N/A	8.9	41.0
VS12_MSS_MZ_C_UG_HR			0.093	11.5	N/A	8.9	43.0
VS13_MSS_MZ_W_UG_HR			0.072	13.0	N/A	11.0	38.0
VS14_MSS_MCZ_W_UG_FR			0.086	12.7	N/A	10.1	37.0
VS15_MSS_MZ_W_UG_MWR			0.066	13.2	N/A	10.5	38.0
VS16_MSS_CZ_UG_HR			0.048	12.0	N/A	10.7	39.0
VS17_MSS_CZ_UG_MWR			0.072	12.8	N/A	11.9	39.0
VS18_MSS_MCZ_W_OP_MWR			0.068	13.5	N/A	11.1	39.0
VS19_MSS_MZ_WC_OP_HR			0.069	12.2	N/A	8.5	35.0
VS20_BMS_MZ_OP_HR			0.085	12.9	N/A	9.4	33.0
VS21_BMS_MZ_UG_FR			0.104	11.6	N/A	8.5	37.0

13.3.5.1.2 Goldlund

During the re-issue of the 2020 Goldlund Gold Project Technical Report that was referenced at the beginning of this section, Bond ball mill work index tests were performed on four Goldlund samples. The results are summarized in Table 13-18.

Table 13-18: Summary of Historical Goldlund Bond Ball Mill Work Index Test Results

Sample	Report	Year	BWi, 105 µm (kWh/t)	BWi, 75 µm (kWh/t)
Sample 1	Technical Report Re-Issue, Goldlund Gold Project	2020	13.4	14.0
Sample 4			N/A	14.0
Sample 3			N/A	20.7
Sample 6			13.7	14.0

13.3.5.2 PFS Goliath Comminution Testing

Bond ball mil work index testing was performed on three samples from each the CZ and MZ composites. The Bond ball mill work index tests were conducted using a 150 µm closing screen size. The results are summarized in Table 13-19.

Table 13-19: Summary of Goliath BWi Test Results

ID	P_{80} (µm)	BWi (kWh/t)
MZ BMS1	100	7.50
MZ MSS1	101	9.10
MZ MSS2	102	6.40
CZ BMS1	100	10.5
CZ MSS2	104	10.1
CZ MSS3	100	10.0

13.3.5.3 PFS Goldlund Comminution Testing

Testing of the crushed Goldlund composite material comprised Steve Morrell mill comminution (SMC) testing, Bond crushing work index (CWi) Bond rod mill work index (RWi), Bond ball mill (Bwi) work index tests, and Bond abrasion index (Ai) testing. Bond rod mill tests were conducted using a 1,180 µm closing screen size. Bond ball mill tests were conducted using a 150 µm closing screen size, aiming to achieve a grind size of P_{80} of 100 µm.

Ten additional samples were submitted for Bond ball mill work index tests conducted using a 120 µm closing screen size to achieve a grind size of P_{80} of 85 µm.

The results of all these tests are presented in Table 13-20. The 75th percentile of the Bond ball mill work index results is 16.0, classified as hard.

Table 13-20: Summary of Goldlund Comminution PFS Test Results

ID	Ai (g)	Cwi (kWh/t)	Rwi (kWh/t)	Bwi (kWh/t)	Axb (SMC)
MPCOM1	0.62	10.6	16.8	14.0	27.0
MPCOM2	0.51	10.2	16.2	13.1	27.4
MPCOM3	0.50	7.51	18.4	16.0	25.0
MPCOM4	0.58	8.33	16.8	14.2	27.0
WPCOM1	0.65	11.0	18.0	16.3	26.0
WPCOM2	0.52	10.6	18.7	17.4	27.5
MP1	N/A	N/A	N/A	13.2	N/A
MP2	N/A	N/A	N/A	15.1	N/A
MP3	N/A	N/A	N/A	17.5	N/A
MP4	N/A	N/A	N/A	13.3	N/A
MP5	N/A	N/A	N/A	15.5	N/A
MP6	N/A	N/A	N/A	18.6	N/A
MP7	N/A	N/A	N/A	15.8	N/A
WP1	N/A	N/A	N/A	11.3	N/A
WP2	N/A	N/A	N/A	11.7	N/A
WP3	N/A	N/A	N/A	13.7	N/A

13.3.5.4 Miller

Bond ball mill testing was performed on a singular sample from the Miller deposit. The Bond ball mill test was conducted using a 150 µm closing screen size. The result is summarized in Table 13-21.

Table 13-21: Summary of Miller BWi Test Result

ID	P_{80} (µm)	Bwi (kWh/t)
WS	115	14.5

13.3.6 Extended Gravity Recovery Gold (E-GRG) Testing

E-GRG tests were conducted on both Goliath composites, CZ and MZ, and all three Goldlund composites, WPC, MPEC, and MPWC. 20 kg of each sample was crushed to produce a K_{80} of approximately 1.2 mm. The crushed material was passed through a Knelson concentrator, from where the concentrate is retained and sized for assay and the tailings are sized, reground to a grind target, K_{80}, of 250 µm, and passed through a second concentrator. Again, the concentrate is retained and sized, whereas the tailings are reground to a K_{80} of 75 µm and passed through a third concentrator. Final tailings are sampled, sized, and assayed.

13.3.6.1 Goliath

Goliath samples were processed as described above. CZ and MZ E-GRG test results are summarized in Table 13-22. The Goliath E-GRG test results demonstrate that samples are amenable to gravity concentration in the grinding circuit to remove coarse free gold prior to leaching.

Table 13-22: Goliath E-GRG Test Results

Composite	Product	Feed Size (K_{80}) per Stage (µm)	Mass (%)	Assay (g/t Au)	Au Distribution (%)
MZ	Stage 1 Concentrate	1042	0.41	70.3	15.0
	Stage 2 Concentrate	113	0.56	112	32.7
	Stage 3 Concentrate	85	0.50	29.2	7.60
	Tailing	N/A	98.5	0.87	44.6
	Combined Concentrate	N/A	1.47	72.1	55.4
	Feed (Calculated)	N/A	N/A	1.91	N/A
CZ	Stage 1 Concentrate	936	0.43	33.0	11.5
	Stage 2 Concentrate	117	0.57	45.5	21.1
	Stage 3 Concentrate	78	0.49	49.6	19.5
	Tailing	N/A	98.5	0.60	47.9
	Combined Concentrate	N/A	1.49	43	52.1
	Feed (Calculated)	N/A	1.24	N/A	N/A

The E-GRG test results of the two Goliath samples indicate a fair amount of coarse gold and high amenability to gravity gold recovery with high recoveries ranging from 52 to 55%.

13.3.6.2 Goldlund

Goldlund samples were processed as described above. WPC, MPEC, and MPWC E-GRG test results are summarized in Table 13-23.

The Goldlund WPC shows some amenability to gravity concentration with a gravity recovery of 54%. The two Main Pit composite samples show less amenability to gravity with recoveries of 28.5% and 43.5% for the MPEC and MPWC samples, respectively. Gravity concentration was not carried forward in the testing program due to the lower recoveries with the two main pit samples.

Table 13-23: Goldlund E-GRG Results

Composite	Product	Feed Size (K_{80}) per Stage (µm)	Mass (%)	Assay (g/t Au)	Au Distribution (%)
WPC	Stage 1 Concentrate	1208	0.53	25.1	18.8
	Stage 2 Concentrate	163	0.52	31.3	22.7
	Stage 3 Concentrate	67	0.41	21.6	12.5
	Tailing	N/A	98.5	0.33	46.0
	Combined Concentrate	N/A	1.46	26.0	54.0
	Feed (Calculated)	N/A	N/A	0.71	N/A
MPEC	Stage 1 Concentrate	1255	0.58	10.9	8.70
	Stage 2 Concentrate	171	0.52	16.6	11.8
	Stage 3 Concentrate	72	0.42	13.9	8.00
	Tailing	N/A	98.5	0.53	71.5
	Combined Concentrate	N/A	1.53	14.0	28.5
	Feed (Calculated)	N/A	N/A	0.74	N/A
MPWC	Stage 1 Conc.	1120	0.46	36.4	9.80
	Stage 2 Conc.	143	0.58	66.8	22.8
	Stage 3 Conc.	75	0.56	33.1	11.0
	Tailing	N/A	98.4	0.98	56.5
	Combined Concentrate	N/A	1.60	46.0	43.5
	Feed (calc.)	N/A	N/A	1.7	N/A

13.3.7 Leaching Testing

13.3.7.1 Goliath

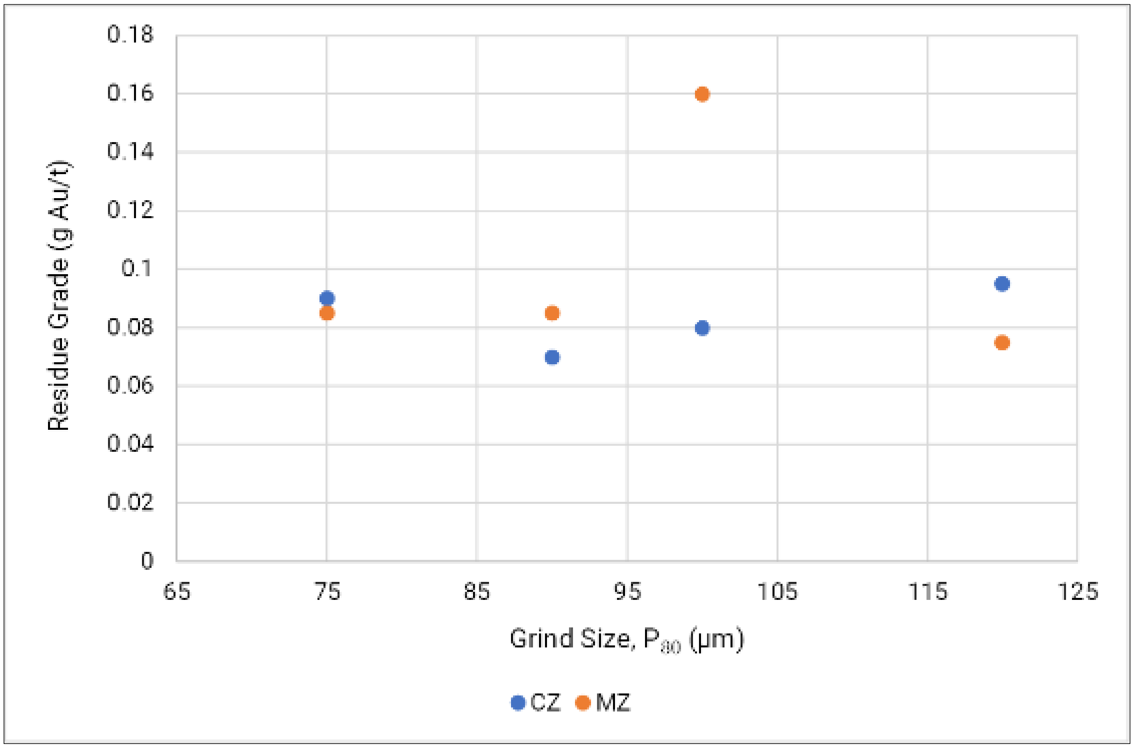
13.3.7.1.1 Leach Grind Series

Gravity tails leach tests were conducted at varying target grind k_{80} sizes ranging from 75 to 120 µm, summarized in Table 13-24 and Figure 13-7. MZ shows a decrease in residue from 120 to 90 µm and flat thereafter to 75 µm, excluding an outlier residue grade at 100 µm. CZ residue gradually increases between 90 and 120 µm. A grind size of 100 µm was nominated for the Goliath leaching testwork based on a trade-off study of grind versus recovery.

Table 13-24: Goliath Grind Size Leach Tests Results

Test ID	Sample ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)		
									Head (calc)	Residue	Grav.	Leach	Total
				g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au	36/48	Au
CN48	CZ	75	36	1.00	1.83	0.77	0.39	0.77	1.74	0.09	44.5	50.4	94.9
CN49	CZ	90	36	1.00	1.77	0.46	0.34	0.46	0.92	0.07	15.9	76.5	92.4
CN50	CZ	120	36	1.00	1.80	0.51	0.36	0.51	1.37	0.10	17.6	75.4	93.1
CN84	CZ	100	48	1.00	1.87	1.37	1.87	1.37	2.41	0.08	50.3	46.3	96.7
CN51	MZ	75	36	1.00	1.73	0.52	0.29	0.52	0.77	0.09	14.5	74.4	88.9
CN52	MZ	90	36	1.00	1.78	0.42	0.28	0.42	0.83	0.09	7.6	82.2	89.8
CN53	MZ	120	36	1.00	1.73	0.27	0.27	0.27	0.97	0.08	5.3	87.0	92.3

Figure 13-7: Effect of Grind on Goliath Deposit Leach Residue



Source: Ausenco, 2023

13.3.7.1.2 Leach Variability Testwork

Optimized leaching conditions were developed for the two Goliath deposit composites (MZ and CZ). All samples underwent gravity recoverable gold (GRG) testing with leaching completed on gravity concentrate and tails. Cyanide leach testwork was evaluated to optimize grind size and investigate CaO dosage.

Baseline leach testwork was completed using 1 kg of material on bottle rolls measuring leach kinetics after 24 hours, at which point the leach was terminated. The CaO dosage was investigated at various lime addition rates for the baseline tests. The following leach conditions were maintained throughout all baseline tests:

- pulp density = 40 wt% solids
- NaCN concentration = 0.5 g/L (maintained)
- retention time = 24 hours
- grind size K_{80} = 100 µm.

The results of all leach testing are shown in Table 13-25.

Average recoveries from the variability tests include:

- Main Zone samples = 93.9% Au
- Central Zone samples = 94.8% Au.

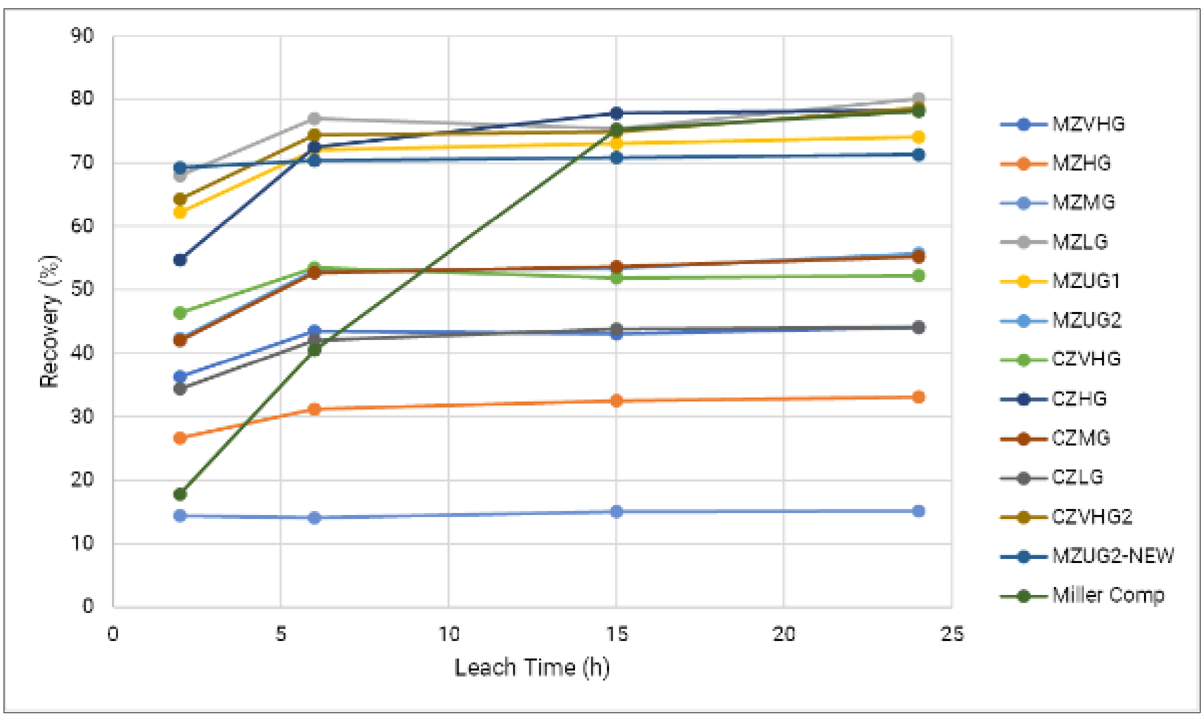
Previous testwork results were analyzed along with the 2021 results summarized here to determine the optimum Goliath leach conditions:

- 20 hours overall residence time
- grind size K_{80} of 100 µm

The leach circuit pulp density selected was 55 wt% solids to keep the required volume of the tanks as low as is reasonable with the addition of a pre-leach thickener.

The condition of 20 hours lech residence time was elected as results from well preformed tests shown in Figure 13-8 indicate a plateau in gold recovery between 8 and 24 hours, but no data points were available to confirm this assumption. Additional leach kinetic testwork at shorter intervals should be taken in future project phases.

Figure 13-8: Goliath Leach Kinetics



Source: Ausenco, 2023

Table 13-25: Goliath Variability Leach Tests Results

Test ID	Sample ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)		
									Head (calc)	Residue	Grav.	Leach	Total
				g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au	24	Au
CN88	MZVHG	100	24	0.50	0.88	1.03	0.16	1.03	4.06	0.16	51.2	44.1	95.2
CN89	MZHG	100	24	0.50	0.82	1.12	0.11	1.12	4.16	0.14	63.7	33.1	96.8
CN90	MZMG	100	24	0.50	0.83	1.09	0.12	1.09	3.50	0.12	81.6	15.1	96.7
CN91	MZLG	100	24	0.50	0.84	1.00	0.13	1.00	0.71	0.06	11.5	80.1	91.5
CN92	MZUG1	100	24	0.50	0.86	1.07	0.14	1.07	5.71	0.48	17.6	74.1	91.7
CN93	MZUG2	100	24	0.50	0.83	1.20	0.09	1.20	2.39	0.21	35.8	55.7	91.4
CN94	CZVHG	100	24	0.50	0.91	1.22	0.28	1.22	1.58	0.13	39.6	52.2	91.8
CN95	CZHG	100	24	0.50	0.89	1.08	0.24	1.08	3.11	0.15	16.9	78.3	95.2
CN96	CZMG	100	24	0.50	0.83	1.03	0.12	1.03	2.48	0.12	40.2	55.2	95.4
CN97	CZLG	100	24	0.50	0.85	0.99	0.13	0.99	1.04	0.05	51.6	44.1	95.7
CN98	CZVHG2	100	24	0.50	0.81	1.14	0.10	1.14	0.47	0.04	14.0	78.6	92.6
CN99	MZUG2	100	24	0.50	0.87	1.12	0.19	1.12	2.40	0.11	24.1	71.3	95.4

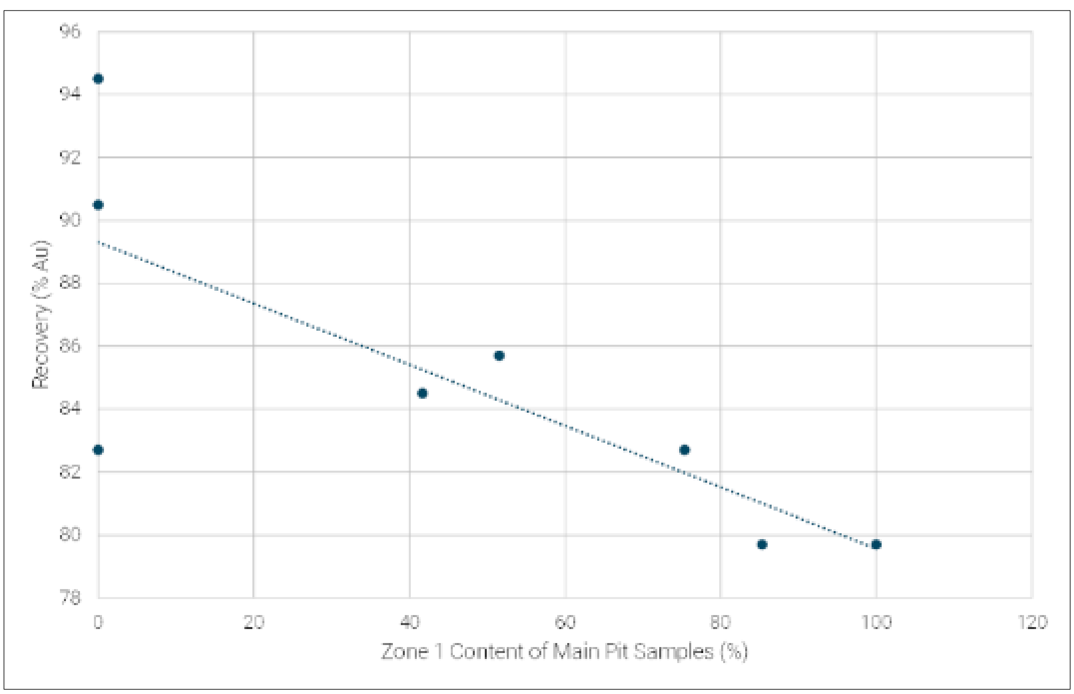
13.3.7.2 Goldlund

13.3.7.2.1 Leach Grind Series

Gravity tails leach tests were conducted at varying target grind k_{80} sizes ranging from 60 to 120 µm under standard conditions used for the Goliath tests. Gravity concentrate was not included in the Goldlund tests as a result of the E-GRG test results. Initial leach test results at typical operating conditions showed low recoveries with the two Main Pit composites, MPEC and MPWC. Figure 13-10 demonstrates the impact of Zone 1 mineralization on the on recovery in the MPEC and MPWC samples. Zone 1 mineralization is the major part of the Goldlund Main Pit. As discussed in Section 13.3.4, gold telluride minerals were predominant in these samples, which led to the inclusion of telluride leach conditions to the program. Telluride leach conditions include extensive pre-aeration at pH 12.0 to 12.5, leaching at similar pH range typically for a minimum of 48 hours. The telluride leach conditions improved recoveries from 72.8% Au to 88.4% Au for the MPEC sample and from 71.1% Au to 91.3% Au for the MPWC sample. Figure 13-11 and Figure 13-12 demonstrate the effect telluride leach conditions have on the gold recoveries of the MPEC and MPWC samples at 60 µm, respectively.

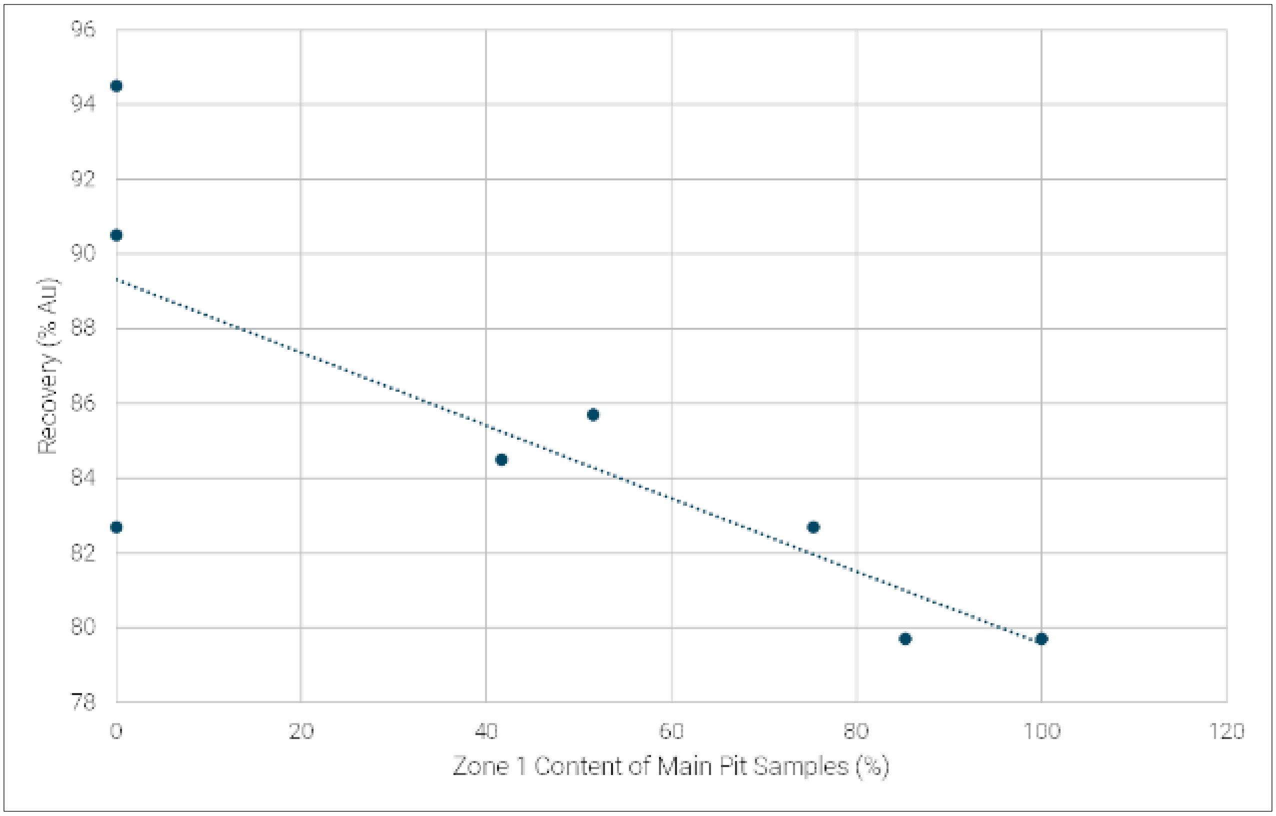
Table 13-26 provides a summary of the results of all tests performed at telluride and non-telluride conditions. MPW shows a gradual increase in residue grade throughout all tested grind sizes. However, multiple tests performed at k_{80} = 75 µm demonstrate varying residue grades. MPE residue grade remains relatively constant throughout all grind sizes, with lowest residue grades occurring at k_{80} = 75 µm. WP demonstrates a linear increase in residue grade between grind k_{80} sizes of 60 and 90 µm. WP was not tested at grind sizes coarser than 90 µm. Based on an analysis of the results of this grind test series, a target grind k_{80} of 90 µm was selected for the Goldlund variability program.

Figure 13-9: Effects of Grind on Goldlund at Telluride Conditions



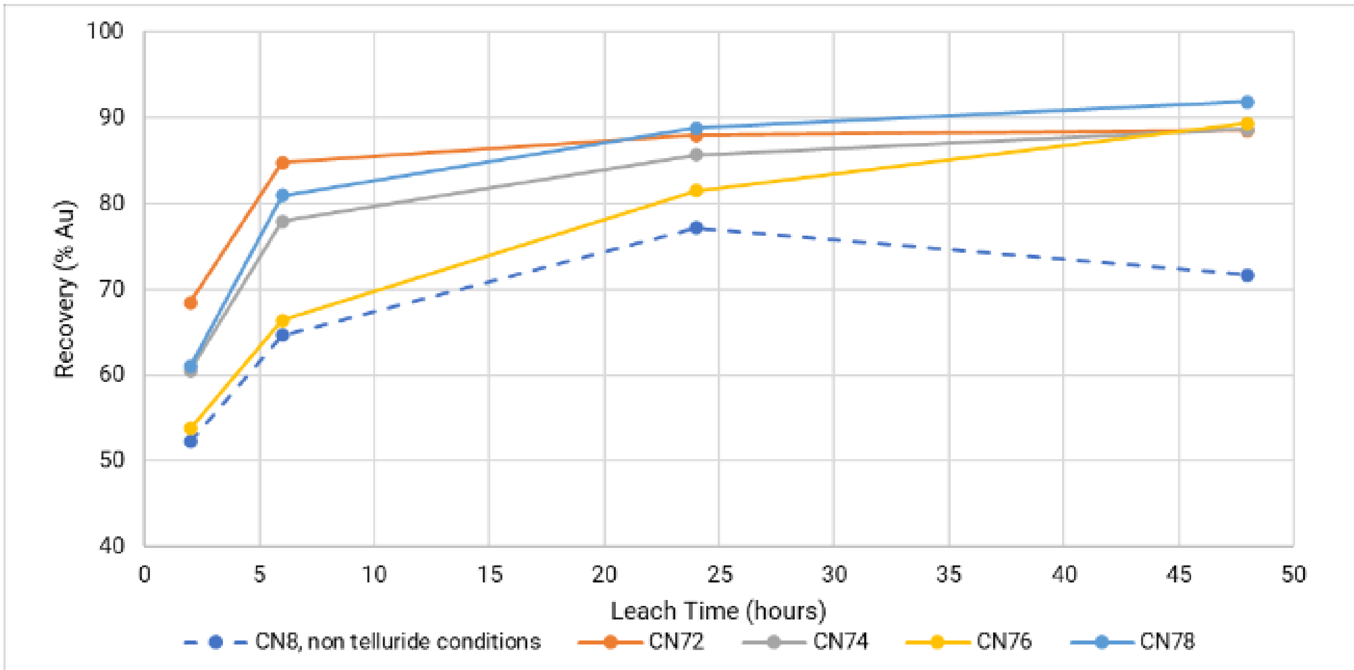
Source: Ausenco, 2023

Figure 13-10: Effects of Zone 1 Content of MPEC and MPWC Samples on Gold Recovery



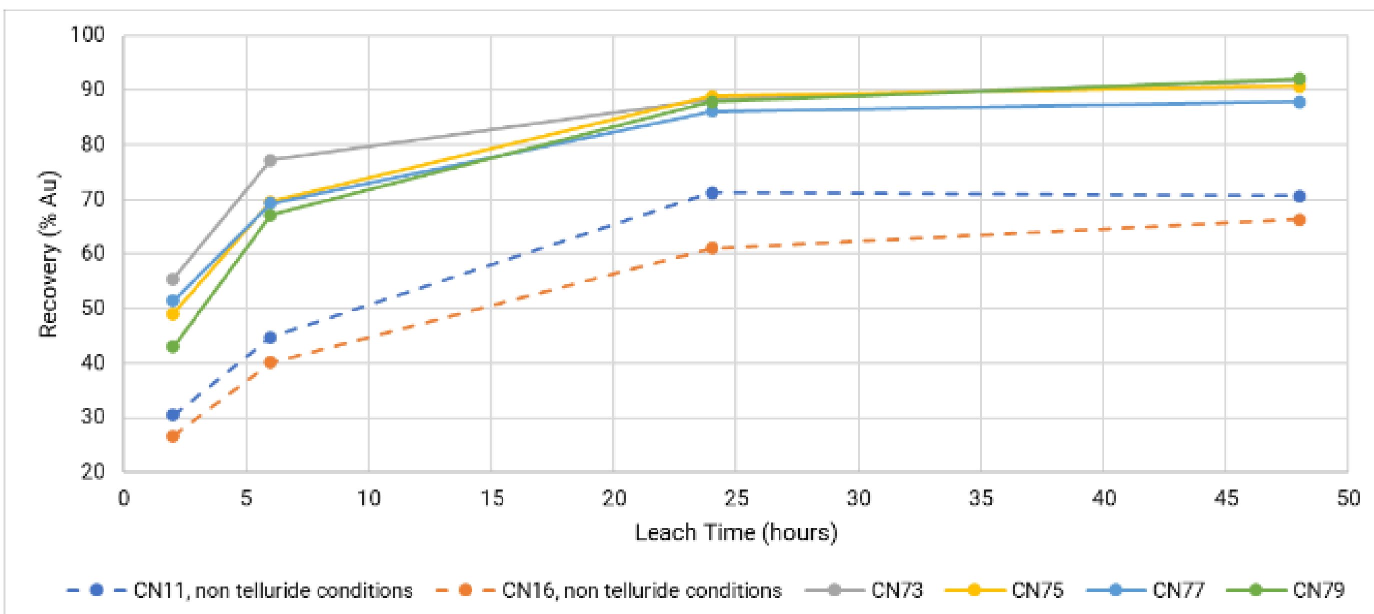
Source: Ausenco, 2023

Figure 13-11: Effect of Telluride Leaching Conditions on MPEC Samples at 60 µm



Source: Ausenco, 2023

Figure 13-12: Effect of Telluride Leaching Conditions on MPWC Samples at 60 µm



Source: Ausenco, 2023

Table 13-26: Goldlund Grind Series Leach Tests Results

Test ID	Sample ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)		
									Head (calc)	Residue	Grav.	Leach	Total
				g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au	36/48	Au
Telluride Conditions													
CN60	MPEC	60	36	1.00	2.07	5.86	0.66	5.86	0.60	0.11	N/A	82.5	82.5
CN72	MPEC	75	48	2.00	3.33	8.92	0.34	8.92	0.57	0.07	N/A	88.5	88.5
CN74	MPEC	75	48	2.00	3.21	5.37	0.42	5.37	0.62	0.07	N/A	88.7	88.7
CN76	MPEC	75	48	2.00	3.27	2.10	0.45	2.10	0.61	0.07	N/A	89.4	89.4
CN86	MPEC	60	48	1.00	1.77	14.16	0.27	14.2	0.88	0.09	N/A	89.8	89.8
CN78	MPEC	75	48	1.00	1.63	4.02	0.14	N/A	0.61	0.05	N/A	91.8	91.8
CN80	MPEC	90	48	1.00	1.62	5.59	0.18	N/A	0.84	0.10	N/A	88.7	88.7
CN82	MPEC	120	48	1.00	1.65	5.64	0.21	N/A	0.67	0.08	N/A	88.1	88.1
CN73	MPWC	75	48	2.00	3.33	8.67	0.33	8.67	1.54	0.13	N/A	91.9	91.9
CN75	MPWC	75	48	2.00	3.24	5.16	0.45	5.16	1.19	0.11	N/A	90.8	90.8
CN77	MPWC	75	48	2.00	3.36	2.15	0.48	2.15	1.28	0.16	N/A	87.9	87.9
CN87	MPWC	60	48	1.00	1.76	16.52	0.27	16.5	1.48	0.12	N/A	92.2	92.2
CN79	MPWC	75	48	1.00	1.56	4.47	0.13	N/A	1.26	0.10	N/A	92.0	92.0
CN81	MPWC	90	48	1.00	1.66	5.58	0.25	N/A	1.58	0.12	N/A	92.7	92.7
CN83	MPWC	120	48	1.00	1.72	5.60	0.31	N/A	1.57	0.14	N/A	91.4	91.4
Non-Telluride Conditions													
CN4	WPC	60	48	1.00	1.60	0.97	0.10	0.97	0.80	0.08	N/A	90.6	90.6
CN5	WPC	75	48	1.00	1.59	1.09	0.10	1.09	0.88	0.10	N/A	89.2	89.2
CN6	WPC	90	48	1.00	1.59	0.96	0.12	0.96	0.64	0.12	N/A	81.3	81.3
CN7	MPEC	60	48	1.00	1.62	1.11	0.19	1.11	0.65	0.14	N/A	78.3	78.3
CN8	MPEC	75	48	1.00	2.86	0.95	1.36	0.95	0.76	0.22	N/A	71.6	71.6
CN9	MPEC	90	48	1.00	1.65	1.12	0.19	1.12	0.69	0.23	N/A	67.2	67.2
CN54	MPEC	60	36	1.00	1.79	0.69	0.33	0.69	1.01	0.21	13.7	66.1	79.7
CN56	MPEC	60	36	1.00	1.66	1.00	0.23	1.00	0.75	0.23	N/A	69.2	69.2
CN57	MPEC	60	36	1.00	1.67	0.79	0.26	0.79	0.62	0.18	N/A	71.0	71.0
CN10	MPWC	60	48	1.00	1.61	0.64	0.11	0.64	2.29	0.92	N/A	59.7	59.7
CN11	MPWC	75	48	1.00	1.66	0.75	0.19	0.75	1.39	0.41	N/A	70.6	70.6
CN12	MPWC	90	48	1.00	1.65	0.78	0.18	0.78	2.62	0.46	N/A	82.7	82.7
CN15	MPWC	60	48	1.00	1.98	0.96	0.54	0.96	1.44	0.42	N/A	71.2	71.2
CN16	MPWC	75	48	1.00	1.88	0.91	0.45	0.91	1.25	0.42	N/A	66.3	66.3
CN17	MPWC	90	48	1.00	1.72	0.95	0.32	0.95	1.49	0.40	N/A	73.1	73.1
CN55	MPWC	60	36	1.00	1.82	0.62	0.38	0.62	1.24	0.35	13.8	58.0	71.8
CN58	MPWC	60	36	1.00	1.67	1.00	0.21	1.00	1.42	0.43	N/A	70.1	70.1
CN59	MPWC	60	36	1.00	1.74	1.03	0.24	1.03	1.35	0.34	N/A	74.8	74.8
CN18	MPVHG	60	48	1.00	2.41	0.84	2.41	0.84	5.17	0.80	N/A	84.5	84.5
CN19	MPVHG	75	48	1.00	2.17	0.82	0.77	0.82	4.82	0.93	N/A	80.7	80.7
CN20	MPVHG	90	48	1.00	2.12	0.93	0.68	0.93	5.21	0.95	N/A	81.8	81.8
CN21	MPHG	60	48	1.00	2.28	0.88	0.87	0.88	1.01	0.15	N/A	85.7	85.7
CN22	MPHG	75	48	1.00	2.23	1.04	0.83	1.04	0.98	0.15	N/A	85.3	85.3
CN23	MPHG	90	48	1.00	2.26	0.78	0.83	0.78	1.04	0.18	N/A	82.7	82.7
CN24	MPMG	60	48	1.00	2.54	0.87	1.13	0.87	3.88	0.22	N/A	94.5	94.5
CN25	MPMG	75	48	1.00	2.32	0.77	0.92	0.77	3.48	0.26	N/A	92.5	92.5
CN26	MPMG	90	48	1.00	2.32	0.80	1.01	0.80	3.69	0.28	N/A	92.5	92.5
CN27	MPLG	60	48	1.00	2.42	0.96	0.93	0.96	0.74	0.15	N/A	79.7	79.7
CN28	MPLG	75	48	1.00	2.18	0.89	0.77	0.89	0.72	0.18	N/A	75.8	75.8
CN29	MPLG	90	48	1.00	2.23	0.80	0.83	0.80	0.71	0.16	N/A	77.4	77.4
CN30	MPZ4	60	48	1.00	2.51	0.86	1.16	0.86	0.79	0.08	N/A	90.5	90.5
CN31	MPZ4	75	48	1.00	2.29	0.97	0.92	0.97	0.63	0.08	N/A	88.0	88.0
CN32	MPZ4	90	48	1.00	2.35	0.89	0.97	0.89	0.75	0.11	N/A	85.9	85.9
CN33	MPZ7	60	48	1.00	2.33	0.63	0.98	0.63	0.81	0.17	N/A	79.1	79.1
CN34	MPZ7	75	48	1.00	2.11	0.73	0.71	0.73	0.81	0.14	N/A	82.7	82.7
CN35	MPZ7	90	48	1.00	2.07	0.62	0.69	0.62	0.79	0.16	N/A	79.9	79.9
CN36	WPVHG	60	48	1.00	2.53	0.56	1.30	0.56	4.71	0.18	N/A	96.2	96.2
CN37	WPVHG	75	48	1.00	2.58	0.52	1.26	0.52	5.15	0.18	N/A	96.6	96.6
CN38	WPVHG	90	48	1.00	2.46	0.49	1.23	0.49	5.09	0.21	N/A	95.9	95.9
CN39	WPHG	60	48	1.00	2.07	0.45	0.78	0.45	1.28	0.03	N/A	97.6	97.6
CN40	WPHG	75	48	1.00	1.93	0.49	0.64	0.49	1.62	0.07	N/A	95.7	95.7
CN41	WPHG	90	48	1.00	2.66	0.34	1.37	0.34	1.56	0.08	N/A	94.9	94.9
CN42	WPMG	60	48	1.00	2.49	0.44	1.21	0.44	2.14	0.38	N/A	82.2	82.2
CN43	WPMG	75	48	1.00	3.25	0.35	2.02	0.35	2.11	0.38	N/A	82.2	82.2
CN44	WPMG	90	48	1.00	2.80	0.34	1.42	0.34	2.19	0.33	N/A	85.0	85.0
CN45	WPLG	90	48	1.00	2.56	0.41	1.21	0.41	1.10	0.04	N/A	96.4	96.4
CN46	WPLG	75	48	1.00	2.44	0.46	1.09	0.46	1.18	0.04	N/A	97.0	97.0
CN47	WPLG	90	48	1.00	2.44	0.46	1.01	0.46	1.71	0.05	N/A	97.1	97.1

13.3.7.2.2 Variability Leach Testwork

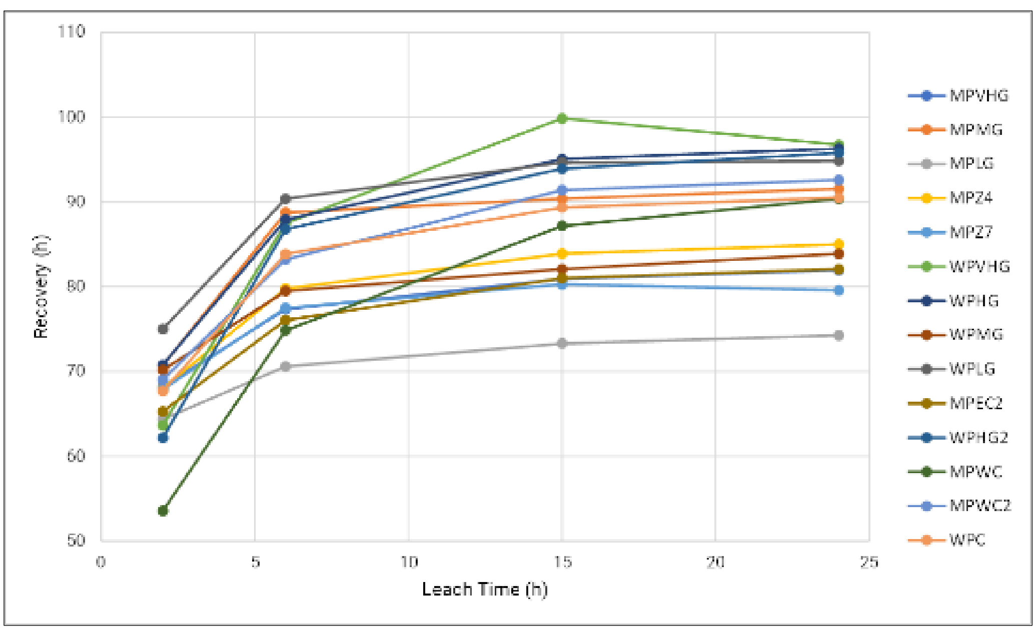
Optimized leaching conditions were developed for the two Goldlund deposit composites (MPEC, MPWC and WP). No samples underwent gravity recoverable gold (GRG) testing prior to leaching. Cyanide leach testwork was evaluated to optimize grind size and investigate CaO dosage.

Baseline leach testwork was completed using 1 kg of material on bottle rolls measuring leach kinetics after 36 or 48 hours, at which point the leach was terminated. The CaO dosage was investigated at various lime addition rates for the baseline tests. The following leach conditions were maintained throughout all baseline tests and are generally referred to as "telluride leach conditions" in this report due to the use of higher dissolved oxygen and pH at 20 ppm and 12-12.5, respectively.

- pulp density = 40 wt% solids
- NaCN concentration = 1.0 g/L (maintained)
- retention time = 48 hours (samples taken at 2, 6, 24, and 48 hours)
- grind sizes K_{80} = 90 µm
- dissolved oxygen = 20 ppm
- pH = 12-12.5 (maintained using hydrated lime).

The results of all leach testing are shown in Table 13-27. Average recoveries from the tests include 87.8% Au from the Main Pit variability samples and 93.0% Au from the West Pit variability samples.

Figure 13-13: Goldlund Variability Samples Leach Kinetics



Source: Ausenco, 2023

Table 13-27: Goldlund Variability Leach Tests Results

Test ID	Sample ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)
									Head (calc)	Residue	Leach
				g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au
CN101	MPVHG	90	48	1.00	1.71	7.50	0.25	7.50	5.24	0.95	82.0
CN103	MPMG	90	48	1.00	1.72	6.83	0.25	6.83	3.31	0.28	91.5
CN104	MPLG	90	48	1.00	1.67	6.40	0.21	6.40	0.84	0.22	74.3
CN105	MPZ4	90	48	1.00	1.73	7.10	0.27	7.10	0.97	0.15	85.0
CN106	MPZ7	90	48	1.00	1.71	6.49	0.24	6.49	0.86	0.18	79.6
CN107	WPVHG	90	48	1.00	1.74	6.11	0.30	6.11	5.06	0.17	96.7
CN108	WPHG	90	48	1.00	1.70	6.36	0.23	6.36	1.75	0.07	96.3
CN109	WPMG	90	48	1.00	1.67	6.41	0.20	6.41	1.96	0.32	83.9
CN110	WPLG	90	48	1.00	1.68	6.53	0.22	6.53	1.46	0.08	94.8
CN111	MPEC2	90	48	1.00	1.64	6.75	0.20	6.75	0.75	0.14	82.1
CN112	WPHG2	90	48	1.00	1.65	6.33	0.16	6.33	2.26	0.10	95.8
CN113	MPWC	90	48	1.00	1.69	6.04	0.22	6.04	1.45	0.14	90.4
CN120	MPWC2	90	48	1.00	1.66	11.39	0.22	11.39	1.69	0.13	92.6
CN114	WPC	90	48	1.00	1.62	6.23	0.15	6.23	0.68	0.07	90.5

13.3.7.2.3 Goldlund Zone 4 Variability Leach Testwork

Six composite samples from Zone 4 of the Goldlund Main Pit underwent variability testwork. None of the samples were subject to GRG testing prior to leaching. Testing was completed using 1 kg of sample on bottle rolls measuring leach kinetics after 24 hours, at which point the leach was terminated. The CaO dosage was investigated at various lime addition rates for the baseline tests. The following leach conditions were maintained throughout all baseline tests:

- pulp density = 40 wt% solids
- NaCN concentration = 1.0 g/L (maintained)
- retention time = 24 hours (samples taken at 2, 6, and 24 hours)
- grind sizes K_{80} = 85 µm
- dissolved oxygen = 20 mg/L (from oxygen addition)
- pH = 12-12.5 (maintained using hydrated lime).

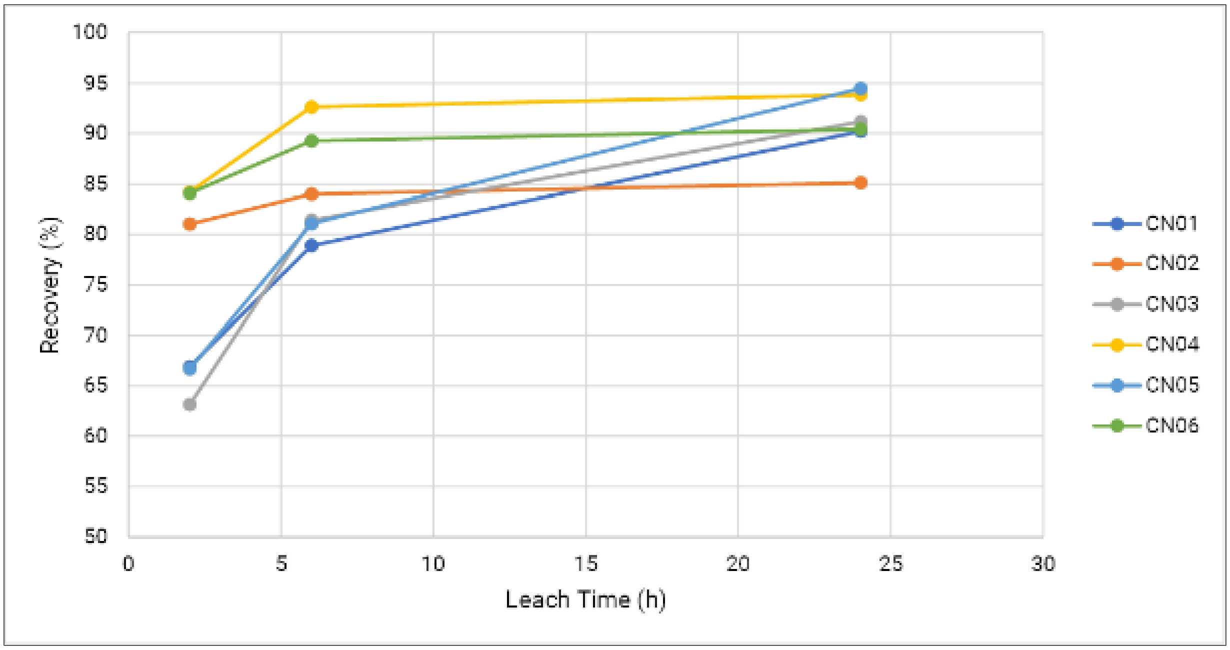
The results of all leach testing are shown in Table 13-28 with an average recovery from the variability tests is 90.9%.

Table 13-28: Goldlund Zone 4 Variability Leach Test Results

Sample ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)
								Head (calc)	Residue	Leach
			g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au
Z4 Comp 1	85	24	1.0	1.70	1.63	0.29	1.63	1.74	0.17	90.2
Z4 Comp 2	85	24	1.0	1.65	1.37	0.21	1.37	0.50	0.08	85.1
Z4 Comp 3	85	24	1.0	1.63	1.44	0.22	1.44	4.81	0.43	91.2
Z4 Comp 4	85	24	1.0	1.65	1.38	0.21	1.38	1.06	0.07	93.9
Z4 Comp 5	85	24	1.0	1.65	0.84	0.18	0.84	0.73	0.04	94.5
Z4 Comp 6	85	24	1.0	1.63	1.21	0.17	1.21	0.58	0.06	90.4

Kinetic curves of the six Zone 4 composite samples are shown in Figure 13-14.

Figure 13-14: Goldlund Main Pit Zone 4 Variability Leach Kinetics



Source: Ausenco, 2023

13.3.7.2.4 Goliath and Goldlund Bulk Leach Tests

Bulk leach tests were completed on a series of composites from Goliath and Goldlund to provide slurry for cyanide detox testing and tailings samples for geochemical testing. The samples included:

- Goldlund master composite: 41.7% MPEC2, 41.7% MPWC2, 16.6% WPC. MPEC2 and MPWC2 were additional Main Pit east and west composites prepared to provide sample for bulk leach tests.
- Goliath master composite: 50% of MZ and 50% of CZ samples.
- Blend 1: 69% of Goldlund master composite and 31% of Goliath master composite.
- Blend 2: 65% of Goldlund master composite and 35% of Goliath master composite.

The blend compositions were largely driven by requirements for geochemical testing.

Bulk leach tests were completed with the following conditions, which are telluride leach conditions:

- K_{80} grind = 85 µm
- Gravity concentration for the Goliath sample.
- pulp density = 40 wt% solids
- NaCN concentration = 1.0 g/L (maintained)

- 8 hours pre-aeration with oxygen (for Goldlund master composite and blend samples)
- pH = 12-12.5 (maintained with lime)
- 24 hours leach retention time at 1 g/L NaCN maintained.
- dissolved oxygen = 20 mg/L (from oxygen addition).

The results of the bulk leach tests are shown in Table 13-29. The results included the following information:

- The Goliath bulk leach test included gravity concentration that provided 50.5% Au recovery, for a total recovery of 94.7% Au. A 1 kg batch test competed on this sample had a recovery of 91.4% Au with no gravity concentration stage.
- The calculated head grade from the Goliath bulk leach test was 2.28 g/t Au compared to the assayed head grade of 1.18 g/t Au. The 1 kg batch test had a calculated head grade of 1.57 g/t Au.
- Telluride leach conditions do not appear to impair Goliath recoveries.
- The Goldlund bulk leach test produced a recovery of 87.6% Au with a calculated head grade of 0.97 g/t Au.
- The Blend 1 sample produced a recovery of 90.5% Au and the Blend 2 sample produced a recovery of 90.3% Au. These correspond to the ratio of the respective ratios of the Goldlund and Goliath recoveries.

Table 13-29: Goliath and Goldlund Bulk Leach Test Results

Sample ID	Grind (µm)	Leach Time (h)	NaCN	Consumption (kg/t)		Au Grade		Recovery (%)
						Head (calc)	Residue	Leach
			g/L	NaCN	CaO	g/t	g/t	Au
Goldlund Master Comp.	85	24	1.0	0.19	5.15	0.97	0.12	87.6
Goliath Master MC	85	24	1.0	0.56	2.64	2.28	0.12	94.7
Blend 1	85	24	1.0	0.23	5.76	1.42	0.14	90.5
Blend 2	85	24	1.0	0.21	4.60	1.14	0.11	90.3

13.3.7.2.5 Leach Circuit Design Considerations

Previous testwork results was analyzed along with the 2021 results presented above to determine the optimum leach conditions for plant operations for processing of blends of Goliath and Goldlund ore based on the available mine plan:

- 5 hours of pre-aeration time
- 20 hours overall residence time between leach and adsorption with a CIL configuration.
- grind size K_{80} of 85 µm

The leach circuit pulp density selected was 55 wt% solids, to minimize tank volume with the addition of a pre-leach thickener.

The leach retention time of 20 hours lech residence time was selected as results from leach tests indicate a plateau in gold recovery between 12 and 24 hours, for most tests but no data points were available to confirm this assumption. Additional leach kinetic testwork at shorter intervals should be taken in future project phases.

13.3.7.3 Miller

One Miller composite sample underwent leach testwork. The sample underwent gravity recoverable gold (GRG) testing with leaching completed on gravity tails. The following leach conditions were maintained throughout the test:

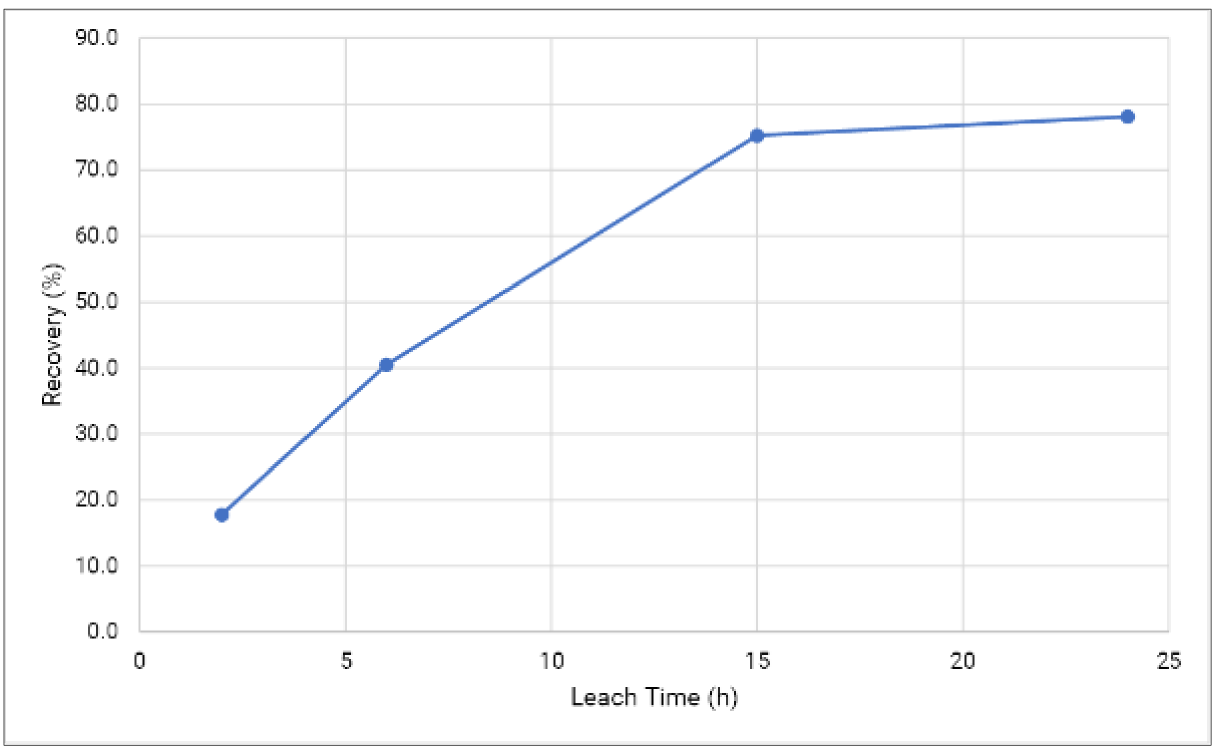
- pulp density = 40 wt% solids
- NaCN concentration = 0.5 g/L (maintained)
- retention time = 24 hours
- grind sizes K_{80} = 100 µm.

The result is summarized in Table 13-30 and leach kinetics are demonstrated in Figure 13-15.

Table 13-30: Miller Composite Leach Testwork Results

Test ID	Grind (µm)	Leach Time (h)	NaCN	Addition (kg/t)		Consumption (kg/t)		Au Grade		Recovery (%)		
								Head (calc)	Residue	Grav.	Leach	Total
			g/L	NaCN	CaO	NaCN	CaO	g/t	g/t	Au	Au	Au
CN100	100	24	0.5	0.88	1.24	0.17	1.24	1.27	0.03	19.5	78.1	97.6

Figure 13-15: Miller Composite Leach Kinetics



Source: Ausenco, 2023

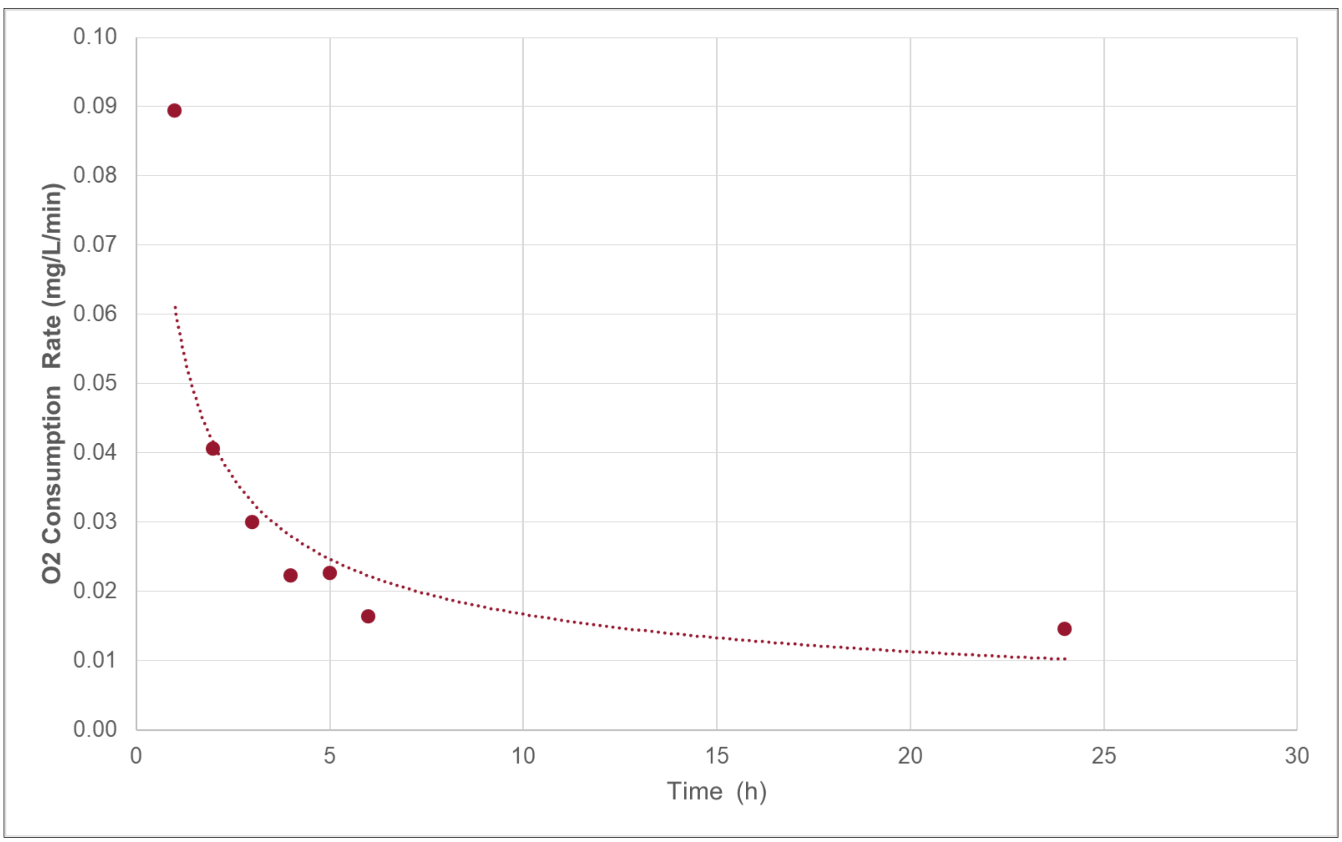
13.3.8 Oxygen Uptake Testing

Oxygen uptake testing was completed on a 1 kg sample at the K_{80} grind size of 85 µm from the Goldlund Master sample. The following conditio.ns were maintained during testwork:

- pulp density = 40 wt% solids
- NaCN concentration = 1.0 g/L (maintained)
- pH = 12 (maintained with lime)
- dissolved oxygen target = 10 mg/L with oxygen addition.

The intention of the testwork was to determine oxygen demand in leaching. The testwork indicated moderate to high oxygen demand. The test results also provide oxygen consumption data for selecting the required oxygen plant capacity or liquid oxygen supply. The resulting oxygen consumption rates are show based on 15-minute intervals are shown in Figure 13-16.

Figure 13-16: Oxygen Consumption (15 Minute Average Rates)



Source: Ausenco, 2023

13.3.9 Cyanide Detoxification Testing

13.3.9.1 The SO_2/Air Process

The chemical reaction for the oxidation of weak-acid dissociable cyanide (CN_{WAD}) using sodium metabisulphite ($Na_2S_2O_5$ as a source of SO_2) is widely used throughout the industry. The technology is proven and capable of achieving low CN_{WAD} concentrations.

Process development testing for the SO_2/air process is completed in two stages. The first stage is batch testing, followed by second stage continuous testing. The batch reactor is first filled with feed slurry and the required copper sulphate is added. The reactor content is then treated in batch mode with sodium metabisulphite ($Na_2S_2O_5$ or SMBS) as the SO_2 source and air to reduce the CN_{WAD} concentrations to low levels. The oxidation reduction potential (ORP) of the pulp is monitored with a Pt/Ag/AgCl combination electrode, while the residual CN_{WAD} concentration in the solution phase is analyzed during the test determined using the Modified Potentiometric Titration method. Initial target batch retention times are between 30 and 60 minutes. The batch test serves to produce treated material with low residual CN_{WAD}, the product is used as starting feed material for the initial continuous test. Final solutions are submitted for analysis at the completion of each test or run.

A 0.9 L reactor was used for both batch and continuous tests. For the continuous tests, an overflow nozzle on the reactor transferred treated slurry to a storage tank.

13.3.9.2 Goliath Master Composite Cyanide Destruction Testing

The results of the Goliath master composite cyanide destruction testing are presented in Table 13-31. All tests were conducted at a pulp density of 40 wt% solids. Air and or oxygen was added to maintain a target dissolved concentration of 8.0 mg/L.

Table 13-31: Goliath Master Composite Cyanide Destruction Testing Results

Test	Objective	Retention Time	Reactor Chemistry (Solution)					Reagent Addition (g/g CN_{WAD})		
			pH	CN_t mg/L	CN_{WAD} mg/L	Cu mg/L	Fe mg/L	SO_2 Equiv.	Lime	Cu mg/L
Feed	-	-	12.2	287	284	4.78	<1	-	-	-
CND-C1	<5 mg/L WAD	60	9.5	13.9	0.6	0.09	4.74	5.0	0.0	96
CND-C2	<5 mg/L WAD	60	10.5	55.3	0.5	0.31	< 19.61	5.0	0.0	25
CND-C3	<5 mg/L WAD	45	10.9	57.1	2.9	1.68	19.40	5.0	0.0	25
CND-C4	<5 mg/L WAD	30	9.7	62.0	0.2	0.07	22.10	5.0	0.0	25
CND-C5	<5 mg/L WAD	30	9.5	65.1	0.3	0.06	23.20	5.0	0.0	15

The target CN_{WAD} concentration of < 5 mg/L was achieved at the initial test conditions of 60 minutes, SO_2:CN_{WAD} ratio of 10:1 and a copper addition rate of 23 mg/L Cu^{2+}. The SO_2:CN_{WAD} ratio was decreased to 7.5:1 while the copper addition rate was increased to 50 mg/L for test CND-C2. The CN_{WAD} target concentration of < 5 mg/L was not achieved. The SO_2:CN_{WAD} ratio was increased to 10.0:1, while retention time was decreased to 45 minutes. The copper addition rates of the CND-C3 and CND-C4 tests were 50 and 25 mg/L, respectively. The target CN_{WAD} concentration was achieved for all tests with SO_2:CN_{WAD} ratios of 10.0:1. The high initial pH of 12.2 required no additional lime addition during the test and the ideal pH of 8.5 was not achieved. The high SO_2 addition rate was required to provided acidity to lower the pH. Future testing should investigate the economics of using sulphuric acid to reduce the pH and lower the addition of SO_2.

13.3.9.3 Goldlund Master Composite Cyanide Destruction Testing

The results of the Goldlund master composite cyanide destruction testing are presented in Table 13-32.

All tests were conducted at a pulp density of 40 wt% solids. Air and or oxygen was added to maintain a target dissolved concentration of 8.0 ppm.

Table 13-32: Goldlund Master Composite Cyanide Destruction Testing Results

Test	Objective	Retention Time	Reactor Chemistry (Solution)					Reagent Addition (g/g CN_{WAD})		
Feed	-	-	12.9	572	566	2.62	1.94	-	-	-
CND-C1	<5 mg/L WAD	60	8.3	1.0	0.9	0.19	< 1	10.0	1.5	23
CND-C2	<5 mg/L WAD	60	10.4	9.7	9.6	0.20	< 1	7.5	0.0	50
CND-C3	<5 mg/L WAD	45	8.3	0.9	0.7	0.16	< 1	10.0	2.3	50
CND-C4	<5 mg/L WAD	45	8.2	0.8	0.7	0.09	< 1	10.0	2.2	25

The target CN_{WAD} concentration of < 5 mg/L was achieved at the initial test conditions of 60 minutes, SO_2:CN_{WAD} ratio of 10:1 and a copper addition rate of 23 mg/L Cu^{2+}. The SO_2:CN_{WAD} ratio was decreased to 7.5:1 while the copper addition rate was increased to 50 mg/L for test CND-C2. The CN_{WAD} target concentration of < 5 mg/L was not achieved. The SO_2:CN_{WAD} ratio was increased to 10.0:1, while retention time was decreased to 45 minutes. The copper addition rates of the CND-C3 and CND-C4 tests were 50 and 25 mg/L, respectively. The target CN_{WAD} concentration was achieved for all tests with SO_2:CN_{WAD} ratios of 10.0:1. As with the Goliath sample, the high initial pH required minimal addition of lime to achieve the target pH of 8.5.

13.3.9.4 Blend 1 Cyanide Destruction Testing

The results of the Blend 1 cyanide destruction testing are presented in Table 13-33.

Table 13-33: Blend 1 Cyanide Destruction Testing Results

Test	Objective	Retention Time	Reactor Chemistry (Solution)					Reagent Addition (g/g CN_{WAD})		
			pH	CN_t mg/L	CN_{WAD} mg/L	Cu mg/L	Fe mg/L	SO_2 Equiv.	Lime	Cu mg/L
Feed	-	-	12.4	551	548	4.71	0.9	-	-	-
CND-C1	<5 mg/L WAD	60	8.2	0.9	0.7	0.19	< 1	10.0	2.2	31
CND-C2	<5 mg/L WAD	60	8.2	0.0	0.0	0.0	< 1	10.0	3.7	25
CND-C3	<5 mg/L WAD	45	8.2	96.9	96.8	14.5	< 1	10.0	3.2	15
CND-C4	<5 mg/L WAD	45	8.2	1.2	1.0	0.21	< 1	7.5	4.3	25
CND-C5	<5 mg/L WAD	45	8.6	16.6	16.5	8.73	< 1	5.0	0.0	25

The target CN_{WAD} concentration of < 5 mg/L was achieved at the initial test conditions of 60 minutes, SO_2:CN_{WAD} ratio of 10:1 and a copper addition rate of 31 mg/L Cu^{2+}. The high initial pH of 12.4 required excess SO_2 addition to reduce the pH to the target of 8.5 in three of the four tests.

13.3.9.5 Blend 2 Cyanide Destruction Testing

The results of the Blend 2 cyanide destruction testing are presented in Table 13-34.

Table 13-34: Blend 2 Cyanide Destruction Testing Results

Test	Objective	Retention Time	Reactor Chemistry (Solution)					Reagent Addition (g/g CN_{WAD})		
			pH	CN_t mg/L	CN_{WAD} mg/L	Cu mg/L	Fe mg/L	SO_2 Equiv.	Lime	Cu mg/L
Feed	-	-	12.4	540	539	3.70	0.50	-	-	-
CND-C1	<5 mg/L WAD	60	8.0	2.3	0.9	0.11	< 1	10.0	4.4	31
CND-C2	<5 mg/L WAD	45	7.7	2.4	1.0	0.20	< 1	10.0	2.2	25
CND-C3	<5 mg/L WAD	45	8.1	2.8	1.4	0.21	< 1	7.5	2.9	25
CND-C4	<5 mg/L WAD	45	8.9	3.2	1.8	1.18	< 1	5.0	0.0	25
CND-C5	<5 mg/L WAD	45	8.8	3.2	1.8	1.13	< 1	5.0	0.0	15

The target CN_{WAD} concentration of < 5 mg/L was achieved at the initial test conditions of 60 minutes, SO_2:CN_{WAD} ratio of 10:1, lime addition of 4.4 g/g CN_{WAD} and a copper addition rate of 31 mg/L Cu^{2+}. Lime addition, copper addition, SO_2:CN_{WAD} ratios, and residence time were altered to determine optimal cyanide destruction conditions. CN_{WAD} concentrations of below 5 mg/L were achieved in all cases, including cases without lime addition. The high initial pH of 12.4 required excess SO_2 addition to reduce the pH and reduced lime addition to zero in tests CND-C4 and CND-C5.

13.4 Thickening Tests

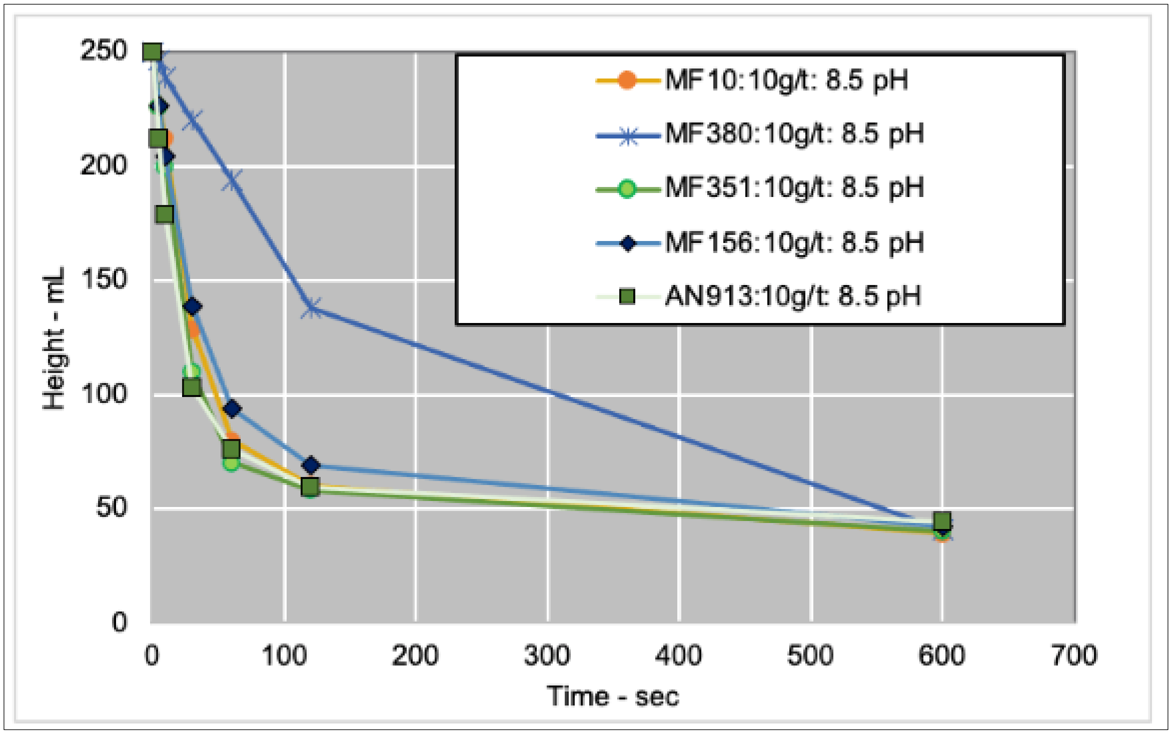
The slurry produced from the Blend 2 cyanide destruction testwork was used to perform solid-liquid separation testing. Flocculant scoping and static testing were completed, followed by bench-scale dynamic thickening testwork.

13.4.1 Flocculant Scoping

Five flocculant reagents were investigated, including Magnafloc 10, 351, 380, and 156, and AN913SH. The flocculant scoping study results are shown in Figure 13-17.

The tests demonstrate that MF351, AN913SH, and MF10 provide the quickest settling times, with AN913SH selected for subsequent testing.

Figure 13-17: Flocculant Scoping Study Results



Source: BaseMet Laboratories, 2022

13.4.2 Static Settling Tests

Static settling tests were performed using AN913SH to determine the free settling velocity of the tailings. The flocculant was adding at dosages of 10, 20, and 40 g/t, reaching final densities of 66.8, 65.5, and 61.8 w/w%, respectively. The results of the static settling tests are summarized in Table 13-35.

Table 13-35: Static Settling Test Results

Test	Sample	Grind (um)	Flocculant		pH	Density (w/w%)		Free Settling
			Type	g/t		Initial	Final	Velocity (m/h)
S1	Blend-2 Detox Tails	85	AN913SH	10	8.5	13.9	66.8	5.6
S2				20		13.9	65.5	8.2
S3				40		13.9	61.8	11.0

An increase in flocculant dosage resulted in an increase in settling velocity but a decrease in final density.

13.4.3 Dynamic Thickening

Dynamic thickening tests were conducted based on the results from the static settling tests. The tests were performed using a feed density of 15 wt% solids, achieving final underflow densities of 50-60 wt% solids. Flocculant dosage ranged between 20 and 40 g/t. Results of the dynamic thickening testing are shown in Table 13-36.

Table 13-36: Dynamic Settling Test Results

Test	Sample	Grind (um)	Density (w/w%)		Flocculant		pH	Rise Rate (m/h)	Loading Rate (t/m²/h)	Turbidity (mg/L)
			Feed	U/F	Type	g/t				
	tox Tails	85	15	60.6	AN913SH	40	8.2	3.1	0.5	44
D1-B			15	58.7		40		4.4	0.7	33
D1-C			15	55.6		40		6.2	1.0	31
D1-D			15	59.0		20		4.3	0.7	84
D1-E			15	50.8		30		4.3	0.7	42

The highest underflow density achieved was 60.6 wt% solids. This was achieved using 40 g/t AN913SH flocculant and a 0.5 t/m^2/h loading rate. An underflow density of 59 wt% solids was also achieved using 20 g/t flocculant at a loading rate of 0.7 t/m^2/h.

13.4.4 Rheology

Rheology tests were performed to determine whether the deformation potential of the settled tailings from the previous test under the influence of imposed stress. Results are summarized in Table 13-37.

Table 13-37: Rheology Data

Test	wt% Solids	Yield Stress (Pa)
V1-A	60.6	73
V1-B	58.7	59
V1-C	55.6	55
V1-D	59.0	41
V1-E	50.8	22

13.5 Deleterious Elements

Metallurgical testing has not identified any deleterious elements that would impair the quality of the doré bullion that will be produced.

13.6 Recovery Models

A combined recovery model was developed for the Goliath/Miller deposit as both pits share similar metallurgical characteristics. The Goldlund pit was divided into several zone classifications as some areas of the deposit are telluride bearing. In each case, a recovery loss of 0.6% was applied to the extracted gold recovery to account for in plant losses such as residual gold in solution after carbon adsorption.

Recovery ranges for each deposit based on the predicted head grades are summarised in Table 13-38.

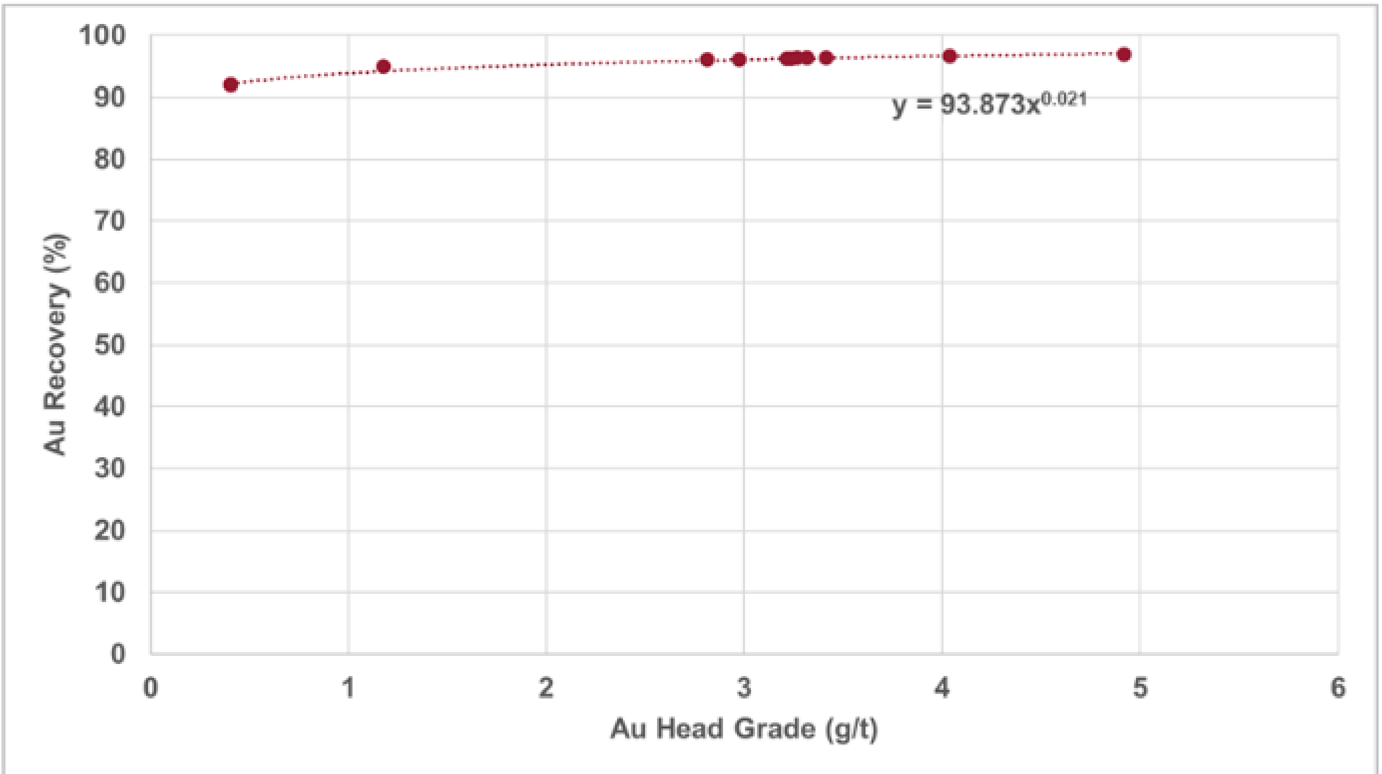
Table 13-38: Summary of Recovery Ranges per Deposit

Deposit	Maximum Head Grade (g/t)	Minimum Head Grade (g/t)	Maximum Gold Recovery (%)	Minimum Gold Recovery (%)
Goliath	3.55	1.39	96.4	94.5
Goldlund, Zone 1	1.83	1.23	92.0	89.6
Remaining Goldlund Zones	1.83	1.23	95.6	93.7
Miller	1.18	0.86	94.2	93.6

13.6.1 Goliath and Miller

A regression of the leaching recovery data for the Goliath deposit is presented below in Figure 13-18.

Figure 13-18: Goliath and Miller Recovery Model



Source: Ausenco, 2023

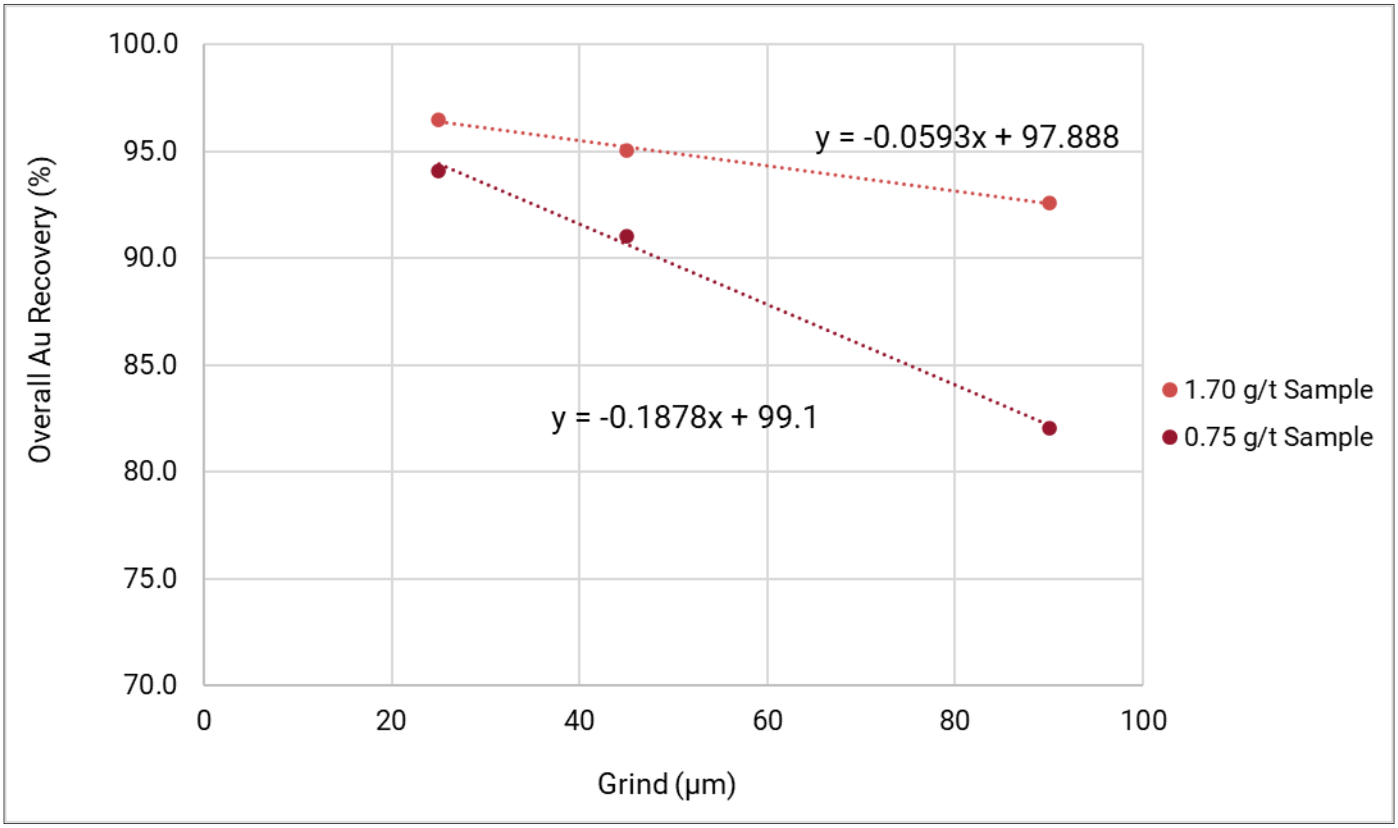
A silver recovery was estimated at 60% for the Goliath deposit based on the results of six leaching tests completed on the which achieved extractions that ranged from 56% to 68% of silver.

13.6.2 Goldlund

13.6.3 Grind Size

Subsequent analysis of the metallurgical testwork presented above indicated that the optimum economic grind for the Goldlund deposit was 85 µm as opposed to the 90 µm grind used in leaching testwork. Two composites from the main Goldlund pit were tested at a series of grinds ranging from 25 to 90 µm as part of the process of optimizing the telluride leaching conditions which found a relationship between recovery and fineness of grind at lower feed grades for these ore types as shown below in Figure 13-19. This opportunity should be confirmed with future testwork for other telluride samples.

Figure 13-19: Goldlund Grind Size vs. Recovery



Source: Ausenco, 2023

13.6.4 Goldlund Zone 1

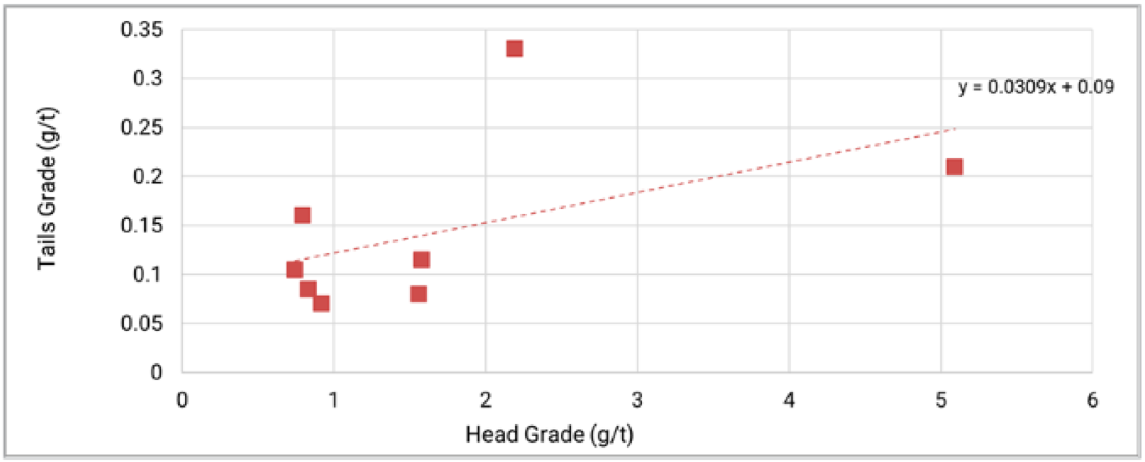
Residue grades for Zone 1 leaching tests indicate a weak relationship with head grade over the range of grades examined as shown in Figure 13-20. An anomalous residue grade in excess of 0.3 g/t was noted for a single sample.

Diagnostic leaching for this test indicated that this residue gold was largely locked in arsenopyrite, which was substantially higher than any other diagnostic leach test conducted on Goldlund. This sample was retained and not excluded until the mineralogical distribution of arsenopyrite is modelled within Zone 1 to confirm it is not expected to be a common feed material.

Based on this regression, Zone 1 recovery is modelled as follows:

$$Recovery = 1 - \frac{0.031 * Au\ Head\ Grade\ \left(\frac{g}{t}\right) + 0.09}{Au\ Head\ Grade}$$

Figure 13-20: Goldlund Zone 1 Residue Prediction

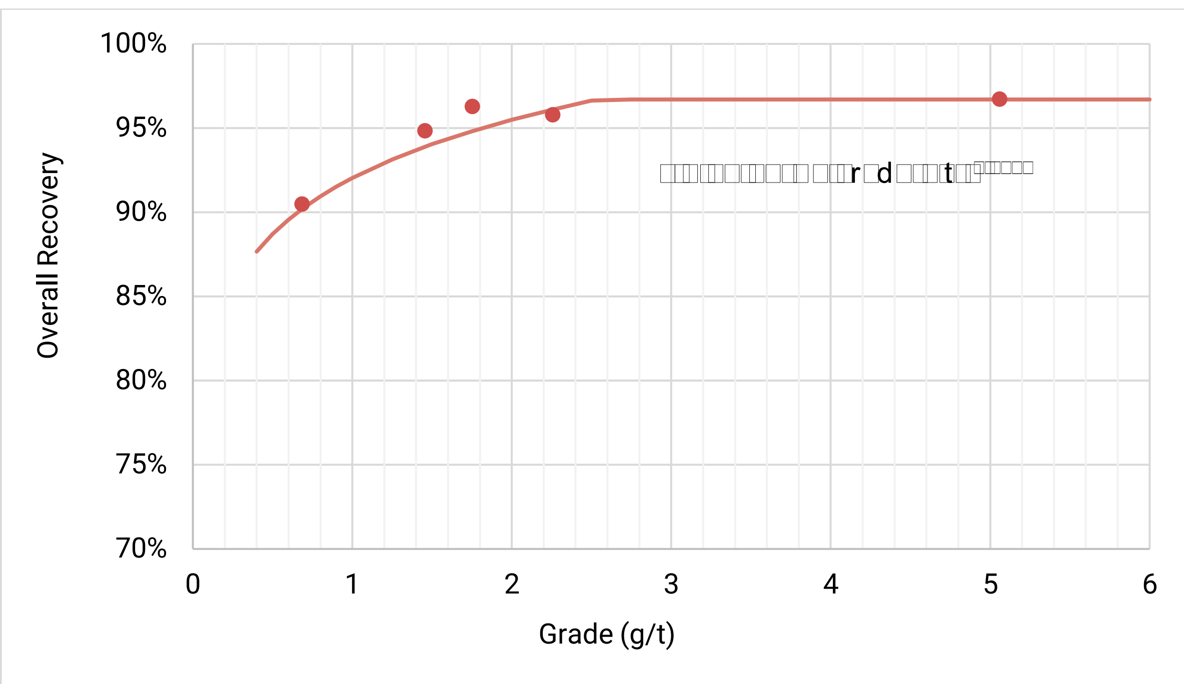


Source: Ausenco, 2023

13.6.5 Other Goldlund Zones

Recovery losses in other zones (Zones 4 and 7) of the Goldlund deposit were modelled as shown in Figure 13-21. These zones do not exhibit telluride mineralogy and leach more readily at conventional cyanide leaching conditions than Zone 1.

Figure 13-21: Other Goldlund Zones Recovery Prediction



Source: Ausenco, 2023

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource estimates presented in this section of the technical report represent an update to the resources presented in the March 10, 2021, technical report for the Goliath Gold Complex. The mineral resources for the Goliath and Goldlund deposits were prepared by Dr. Gilles Arseneau and Ms. Sheila Ulansky of SRK Consulting (Canada) Inc. (SRK) The mineral resources for the Miller deposit were prepared by Dr. Arseneau. Dr. Arseneau is the qualified person for all three mineral resource statements presented in this technical report.

14.2 Data

Treasury Metals provided SRK with project databases for the Goliath, Goldlund and Miller deposits. Each dataset consisted of collar data, down-the-hole survey, logged lithology, assays, and density. The data sets were supplemented with assay certificates, the 2021 resource models and associated wireframes, QA/QC data, and topographic data.

The data were fully validated before being used in the resource estimate. As a final step, drill data were checked for overlapping, missing, and negative length intervals. No erroneous data was detected affecting the primary database table used in the resource estimation.

14.2.1 Goliath

For the Goliath deposit, Goliath and Goliath East, there are 904 core holes existing in the database representing 290,685 m of core. Of these, 772 holes contributed to the grade estimation of the Goliath deposit and 83 holes were used to prepare the Goliath East estimate.

14.2.2 Goldlund

For the Goldlund deposit, the dataset consists of 1,934 core holes representing 250,861 m of core (1,454 surface holes and 480 underground drillholes). In addition, the Goldlund data also includes 246 underground channel samples representing 3,637 m and 188 trenches and one pit for 1,444 m of sampling. Of these, 1,375 core holes contributed to the estimation of mineral resources for Goldlund. The underground channel and trench samples were not considered for grade estimation but were included in the modelling of the mineralized zones.

14.2.3 Miller

There are 61 drillholes in the Miller database totalling 10,370 m of drilling, of these, 49 drillholes, 7,964 m contributed to the Miller resource estimate.

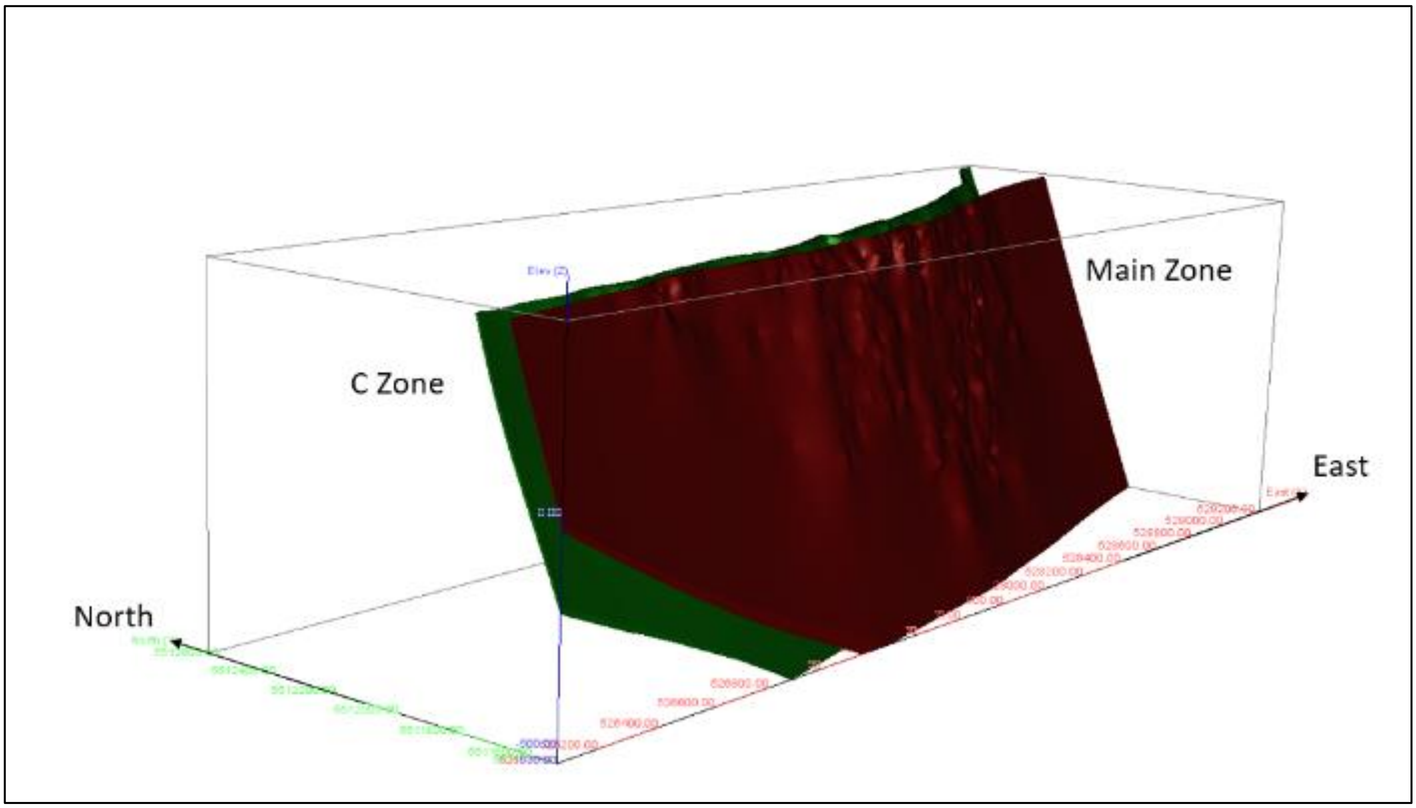
14.3 Geological Models

Geological models for the Goliath Gold Complex deposits were prepared by the QP in conjunction with Treasury Metals geological staff.

14.3.1 Goliath

The mineralization at Goliath occurs in higher grade pyritic muscovite-sericite schist horizons (MSS) intercalated between lower grade to waste biotite-muscovite schists (BMS) with minor metasedimentary rocks (MSED). Alteration consists mainly of sericitization and silicification associated with gold mineralization. The BMS/MSS horizons are variable in thickness and the logged intervals of MSS can be interpreted as containing "mostly" MSS with possibly some BMS. The bulk of the mineralization is located in two principal mineralized corridors namely the Main Zone and the C Zone (refer to Figure 14-1). Other minor zones of mineralization exist on the hanging wall and footwall but are not as well developed, are lower grade, and more discontinuous.

Figure 14-1: Perspective View of Goliath Main and C Zones

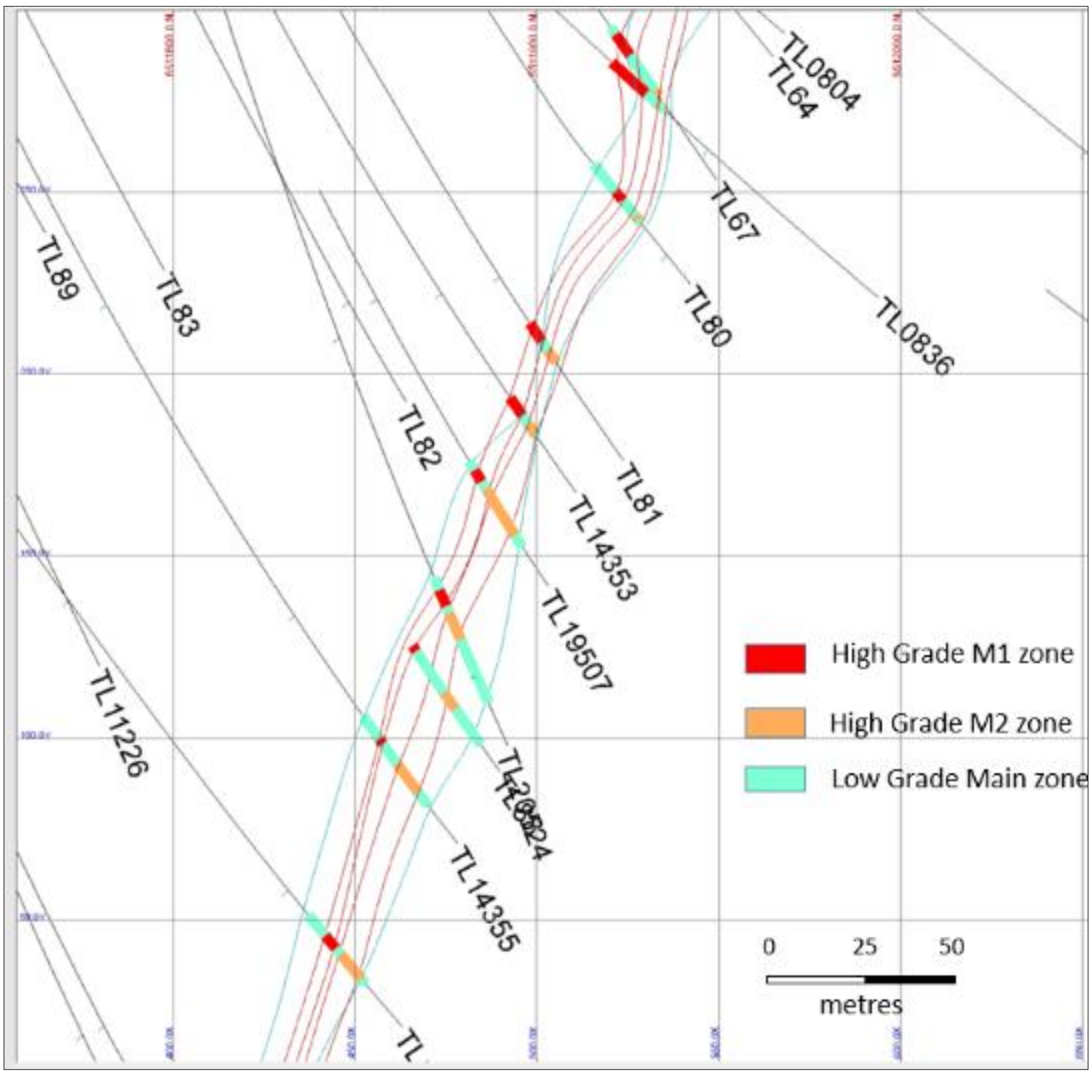


Note: Markers are 200 m apart. Source: SRK, 2023.

14.3.1.1 Main Zone & C Zone

For the Main Zone and C Zone, the mineralization is located within two wide mineralized corridors as defined with assays above 0.2 g/t. Grade tends to be the highest in proximity to the corridor edges within a few meters from the "contact". Internally, there is often a waste/low grade zone, but that can be variable from hole to hole. On some sections, the waste/low grade is well defined but on other sections, that pattern is broken by high-grade holes. The position of the mineralized corridors is predictable as evidenced by the numerous infill holes which intersected the mineralization at the expected location with similar tenors. To capture these features, the hanging wall and footwall extent of the corridors were selected as points along the drillhole trace. Hanging wall and footwall surfaces were created and stitched together to form a 3D solid which was then clipped to the extent of the drilling. The higher-grade cores were modelled using Leapfrog® Geo. (refer to Figure 14-2)

Figure 14-2: Cross-section showing High-Grade Core and Low-Grade Envelop for Main Zone at Goliath



Source: SRK, 2023.

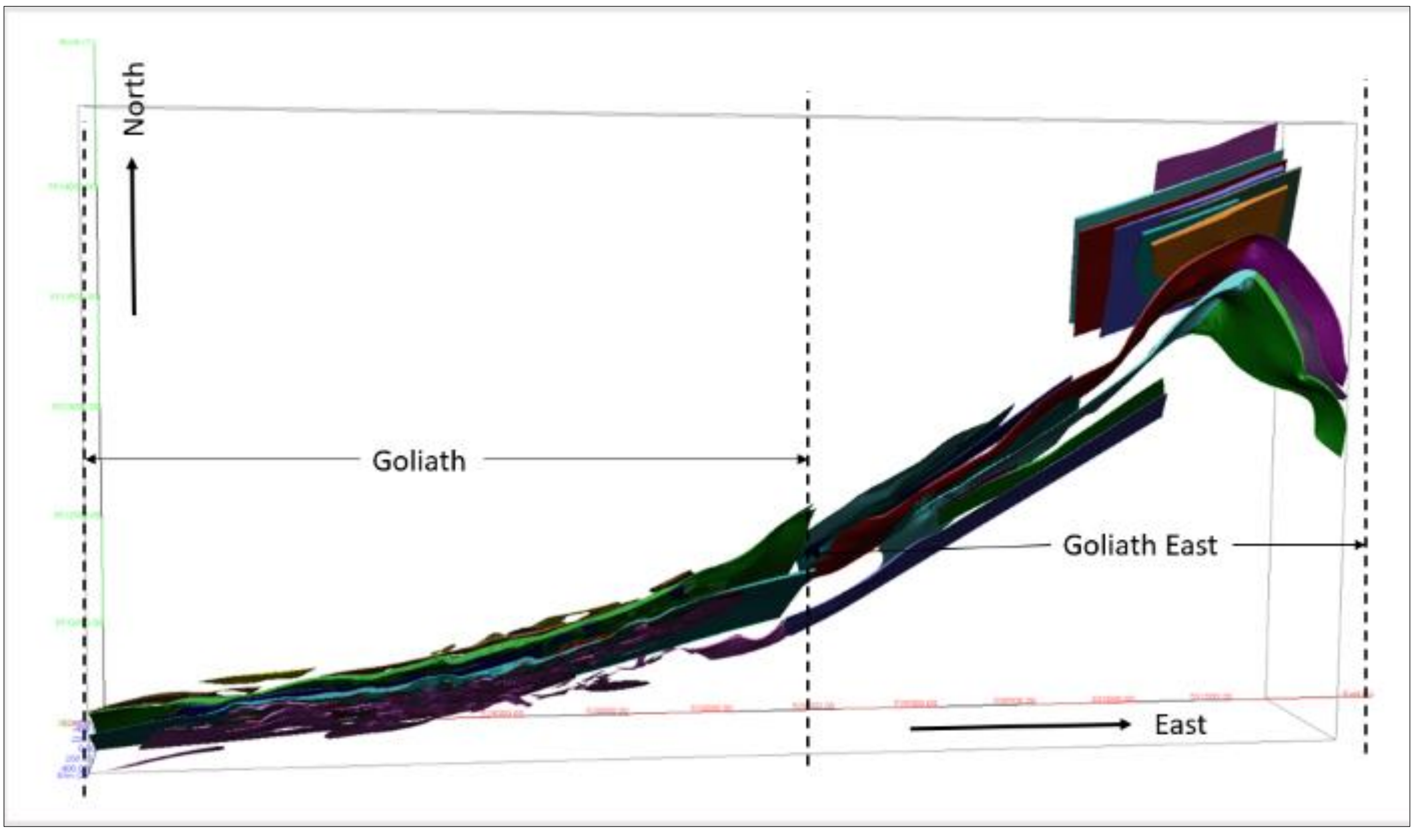
14.3.1.2 B, D, E & H Zones

On the hanging wall of the Main Zone, the H series zone comprised of H1 Zone, H2 Zone, H3 Zone, H4 Zone, and H5 Zone. The B Zone is located between the Main Zone and C Zone and on the footwall of the C Zone there are two more zones namely the D Zone and the E Zone. All these zones were modelled using Leapfrog® Geo. Treasury Metals assisted in assigning intersections to specific zones.

The QP is confident that continuity of the higher-grade zones is well maintained for the Main and C zones. Continuity of grade for the lesser zones is not as good as evidenced by the multiple gaps in the hanging wall zones. The QP recognizes that some gaps in the hanging wall zones could be attributed to the fact that the older drillholes (Teck drillholes) didn't sample these lower-grade zones, thereby generating zero grade values for these intervals in the current database.

In all, 24 narrow higher-grade domains and 2 lower-grade domains were generated to model the mineralization at Goliath and 20 wireframes were used to model the Goliath East deposit (Figure 14-3) (Table 14-1).

Figure 14-3: Perspective View of Goliath and Goliath East Mineralized Zone Wireframes



Note: Markers are 500 m apart. Source: SRK, 2023..

Table 14-1: List of Mineralized Wireframes for Goliath and Goliath East

Domain Name	Volume (M^3)	Domain Code
Goliath Domains		
Main Low-grade	44,183,313	1000
C Zone Low-grade	36,433,281	3000
C Zone High-grade C1_1	274,688	310
C Zone High-grade C1_2	359,786	310
C Zone High-grade C1_3	597,977	310
C Zone High-grade C1_4	591,534	310
C Zone High-grade C2_1	675,066	320
C Zone High-grade C1_2	442,027	320
Main Zone M1 High-grade	1,748,001	100
Main Zone M2 high-grade	1,758,966	110
B 6	416,570	260
B 4	126,565	240
B Footwall	620,275	230
B Hanging wall	637,180	220
B Main	4,236,423	210
D 2	469,402	400
D Main	1,324,776	400
E Main	1,295,436	500
H 1	1,765,241	610
H 2	21,114	620
H A	2,996,253	620
H A2	126,941	630
H B	1,493,716	620
H B2	62,696	620
H C2	45,030	620
H W	3,567,181	620
Goliath Total Volume	106,269,438	
Goliath East Domains		
Zone B	1,949,537	2000
Zone H 2	4,459,746	6100
Zone FW 2	4,620,958	3300
Zone FW C	1,527,520	3100
Zone FW D	2,759,143	3200
Zone HW	149,422	4000
Zone HW 2	1,470,111	4100
Zone N_A	3,077,796	5000
Zone N_B	2,599,365	5100
Zone N_C	6,372,608	5200
Zone N_D	6,430,608	5300
Zone N_E	3,671,954	5400
Zone N_F	1,698,058	5500
Zone N_G	2,795,869	5600
Zone SHW 1	107,432	4200
Zone Main	6,848,195	1100
Zone Main B	1,094,110	1101
Zone Z1	5,410,097	6000
Zone Z1 B	4,444,589	6001
Goliath East Total	61,487,118	

The topographic model was provided from Lidar data.

An overburden surface was developed using the drillhole lithological codes of OVB (overburden) or CAS (casing). The surface was interpolated in Geovia Gems® software using a nearest neighbour algorithm to produce a surface. The resulting surface of the bottom of the overburden was normalized against the topography surface to ensure there was consistency between the two surfaces and that the bottom of the overburden was not above the surface topography. The estimated overburden thickness across the mineralized zones ranges from 0 to 10 m. The same method was used for all three deposits at the Goliath Gold Complex.

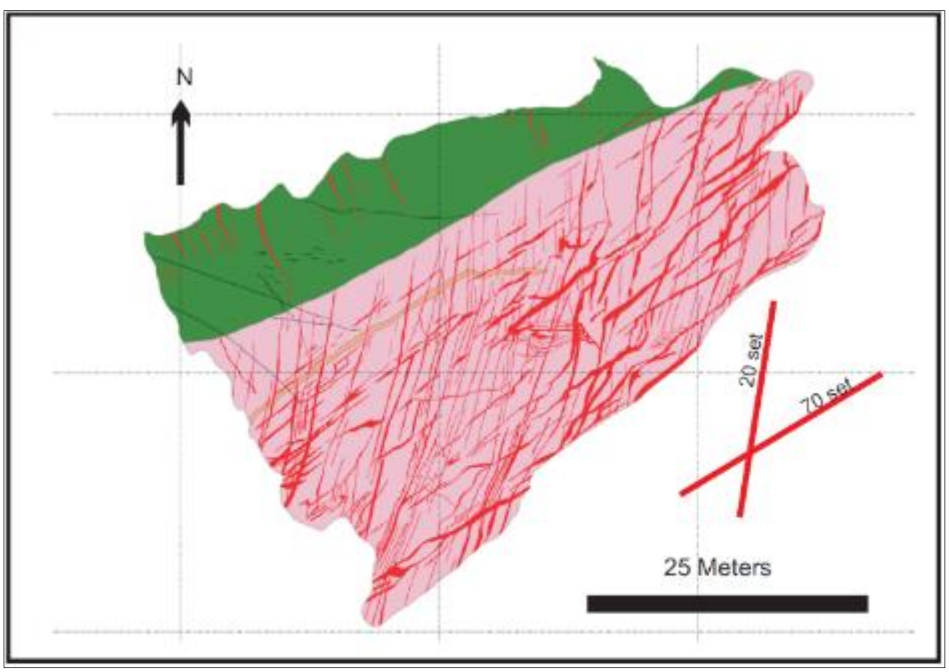
14.3.2 Goldlund

Gold mineralization at Goldlund is associated with quartz vein and stock-work structures situated inside northeast-trending albite-trondhjemite dykes (granodiorite), with lesser amounts in porphyry dykes and metavolcanic rocks.

The mineralized dykes generally strike (065°) and dip steeply to the southeast. The gold-bearing quartz stockwork veins consist of two synchronous sets of veins, referred to as the 20° set (trending 189°/53°W) and 70° set (trending 239°/58°N). The vein structures have developed preferentially in the granodiorite dykes, as they were the most competent (brittle) rock type; however, vein structures do propagate into the surrounding metavolcanic rocks, most often as brittle-ductile, biotite-carbonate-rich shears.

Figure 14-4 displays a map of the historical open pit area showing the 20° set and 70° set veins (red). The veins are hosted in a fine-grained granodiorite (pink), with the footwall gabbroic rocks shown in green, and late “tension veins” shown in orange (Pettigrew, 2012).

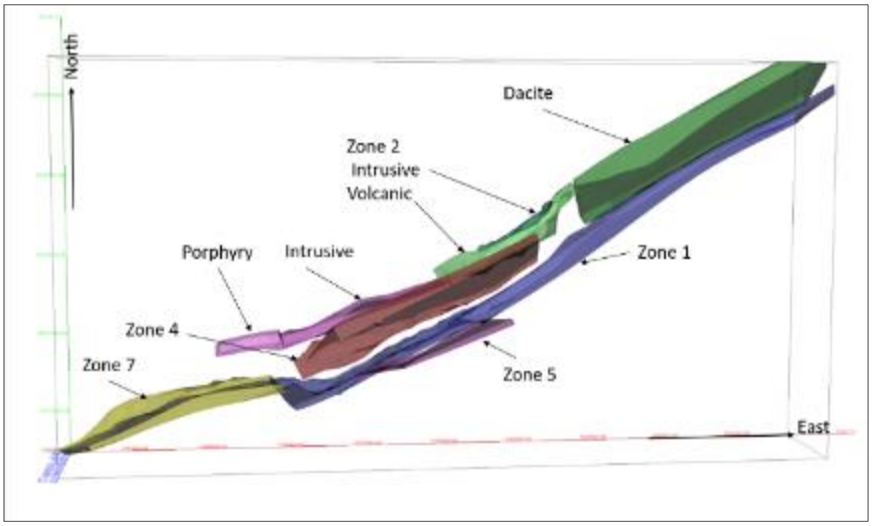
Figure 14-4: Example of 020 and 070 Vein Set in Plan View



Source: Pettigrew, 2012.

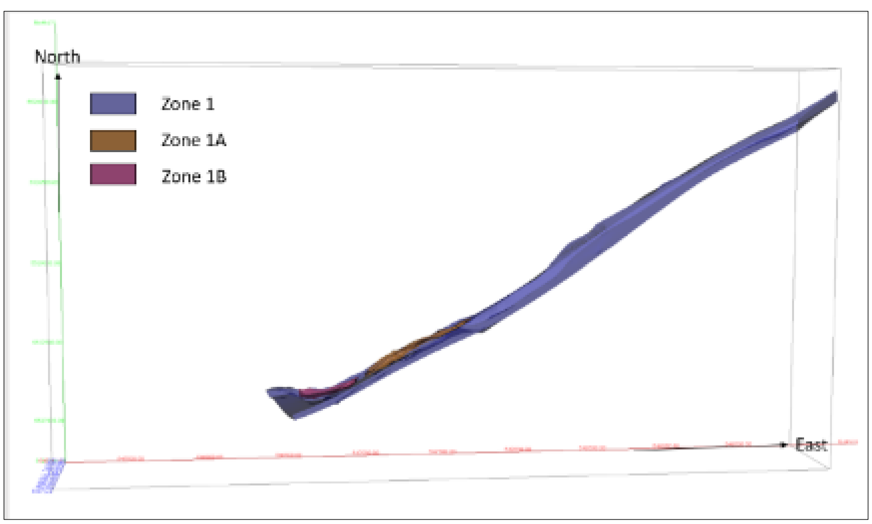
The interpretation of the mineralized zones was based on previous models generated by First Mining and generally based on grouping assays greater than a nominal 0.1 g/t Au cut-off and designed to principally follow the trend of the granodiorite dykes and porphyry dykes (Figure 14-5). The wireframes were modified to include the recent drill results using Leapfrog® Geo by Treasury Metals geological staff. The QP reviewed, validated and accepted the updated wireframes representing the broad mineralization wireframe. After review, the QP decided to modify the low-grade envelops to include a high-grade internal wireframe enclosing Zone 1 to better restrict the influence of the higher-grade assays found within the underground workings at Goldlund (Figure 14-6). Table 14-2 lists the Goldlund mineralized wireframes used to estimate the mineral resources.

Figure 14-5: Perspective View Looking Down of Low-Grade Mineralized Envelops at Goldlund



Note: Markers are 500 m apart. Source: SRK, 2023.

Figure 14-6: Goldlund Zone 1 with High-Grade internal Zones 1A and 1B



Note: Markers are 500 m apart. Source: SRK, 2023.

Table 14-2: List of Mineralized Wireframes for Goldlund

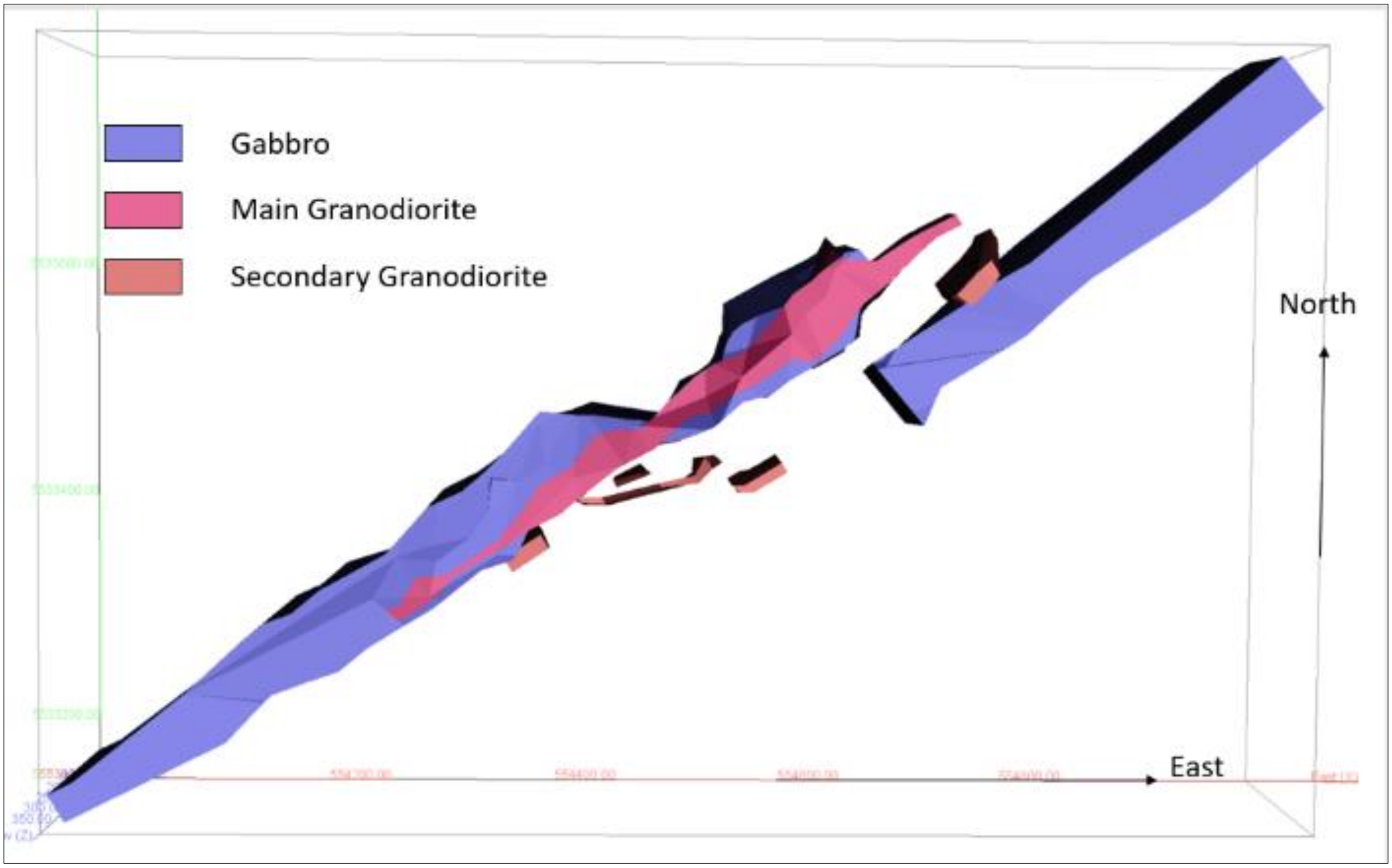
Domain Name	Volume (m³)	Domain Code
Zone 1	212,899,307	1
Zone 2 Intrusive	3,535,592	20
Zone 2 Volcanic	40,586,093	21
Zone 4	99,711,352	4
Zone 5	19,266,012	5
Zone 7	58,415,926	7
Porphyry	11,663,712	40
Intrusive	15,907,117	50
Dacite	106,982,406	30
Zone 1A	1,094,352	10
Zone 1B	3,505,798	11
Total	573,567,667	

14.3.3 Miller

The mineralized granodiorite domains at the Miller deposit were generated using conventional polylines on vertical sections defined along 10 to 25 m spaced sections. The polylines capture the main mineralized granodiorite body and a secondary granodiorite dyke (Figure 14-7). These domains are host to most of the gold mineralization.

The gabbro and andesite lithologies were modelled as separate domains and represent the surrounding country rock. Minor intercepts of dacite, tuff, and diorite were incorporated into the andesite wireframe. The andesite country rock is considered as waste and were not used in the resource estimation. Minor mineralization does occur in the gabbro unit next to the granodiorite.

Figure 14-7: Perspective View of Miller Wireframes



Note: Markers are 500 m apart. Source: SRK, 2023.

Table 14-3 lists the mineralized wireframes for the Miller deposit.

Table 14-3: List of Mineralized Wireframes for Miller Deposit

Domain Name	Volume (m³)	Domain Code
Main Granodiorite	2,901,428	400
Secondary Granodiorite	277,056	401
Gabbro	12,360,175	210
Total	15,538,659	

14.4 Exploratory Data Analysis

Exploratory data analysis is the statistical characterization and statistical behaviour of the data set. In this case, the data evaluated are the gold and silver grades and the objective is to understand the population distribution of the grade elements in the various domains using such tools as histograms, descriptive statistics, and probability plots.

14.4.1 Goliath

The raw assay statistics were evaluated, grouping all assays intersecting the mineralized zones. The assays from all of the zones were back tagged from their corresponding wireframes. Box and whisker plots on the H and B zones indicated that the gold distribution within these zones were sufficiently similar to allow the grouping of those zones for statistical evaluation.

Table 14-4 provides descriptive statistics for raw, uncapped, gold values for the Goliath deposit and Table 14-5 shows the silver data. It should be noted that the number of silver assays are less than the number of gold assays.

Table 14-4: Descriptive Statistical Data for Uncapped Gold Assays at Goliath

Domain	All	Main Zone		C Zone		B Zones	D Zone	E Zone	H Zones
		High Grade	Low Grade	High Grade	Low Grade				
Valid Cases	33,752	2,828	12,932	1,101	8,631	2,744	735	329	4,452
Mean	0.84	5.07	0.3	2.96	0.39	0.64	0.45	0.42	0.47
Maximum	870	870	93.4	152	23.77	286.23	45.17	8.87	53.24
Upper Quartile	0.45	2.9	0.33	2.26	0.32	0.30	0.33	0.34	0.33
Median	0.18	1.31	0.15	1.05	0.14	0.14	0.15	0.12	0.14
Lower Quartile	0.069	0.69	0.05	0.60	0.05	0.07	0.08	0.07	0.07
Standard Deviation	8.53	27.5	1.07	10.03	0.72	6.16	1.91	1.00	1.83
Coefficient of Variance	10.14	5.4	3.54	3.39	2.25	9.59	4.24	2.39	3.88

Table 14-5: Descriptive Statistical Data for Uncapped Silver Assays at Goliath

Domain	All	Main Zone		C Zone		B Zones	D Zone	E Zone	H Zones
		High Grade	Low Grade	High Grade	Low Grade				
Valid Cases	24,719	1,896	,8865	863	6,887	2,124	606	303	3,175
Mean	3.75	16.51	2.63	10.46	1.87	3.44	1.77	1.76	2.29
Maximum	1,214.00	1,214.00	923.00	921.00	257.00	719.00	186.00	22.00	565.00
Upper Quartile	2.00	10.80	2.00	7.80	2.00	1.84	1.00	2.00	1.23
Median	1.00	3.90	1.00	3.00	1.00	1.00	1.00	1.00	0.50
Lower Quartile	0.50	1.00	0.50	1.00	0.50	0.50	0.50	0.50	0.00
Standard Deviation	22.58	64.21	13.21	37.89	5.94	19.59	8.02	2.97	12.68
Coefficient of Variance	6.02	3.89	5.02	3.62	3.18	5.70	4.54	1.68	5.53

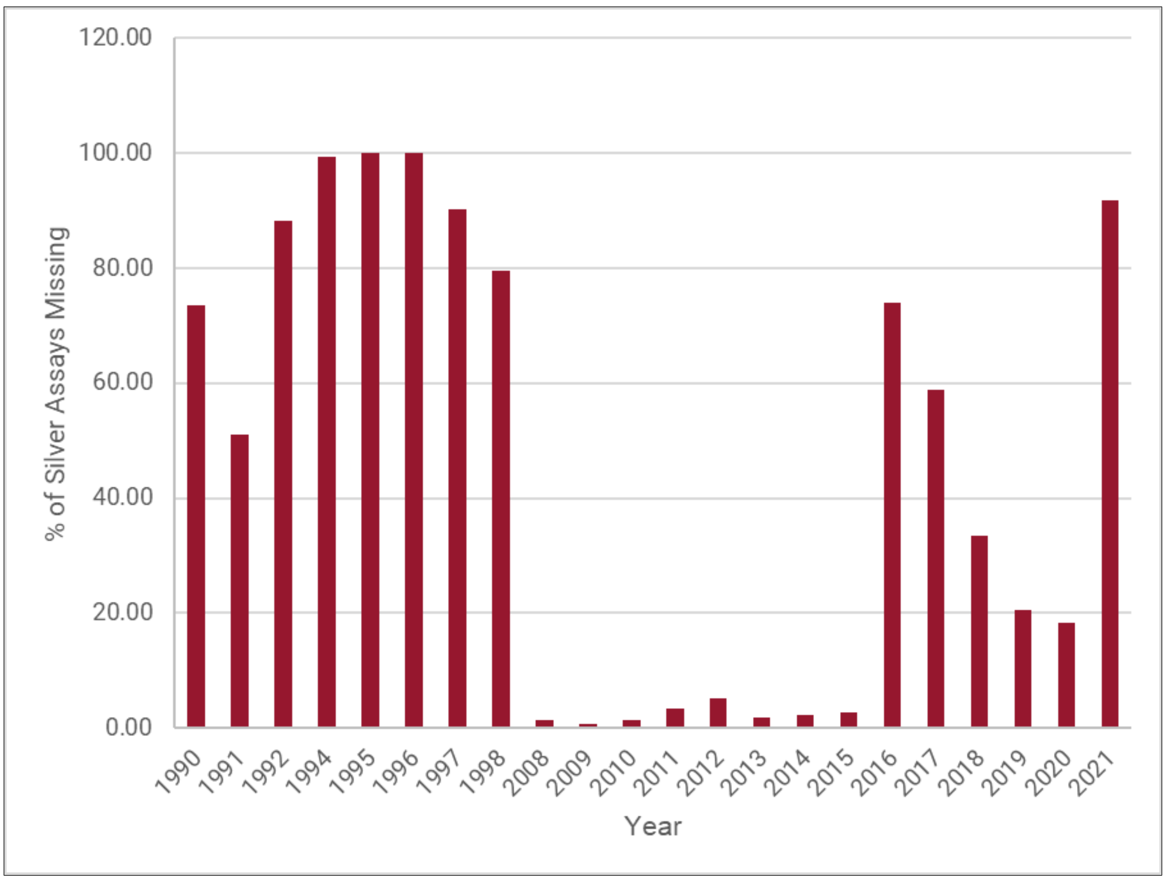
The high-grade Main C zone carries by far the highest gold and silver grades averaging 5.07 and 2.96 g/t Au compared to the remaining zones all averaging less than 1.0 g/t gold. The coefficient of variation (CV) indicates high variability in the assay distribution, indicating that capping of outliers is required.

The correlation between gold and silver shows a correlation of 0.28. A linear regression was attempted and found to be poor with an R-Square of 0.086. For this reason, silver values could not be estimated with confidence from the gold values.

14.4.1.1 Missing Silver Data

A review of the silver assay data showed that only approximately 65% of the gold assay data within the mineralized zones have corresponding silver assays. This is due to limited assaying for silver throughout the years. The silver assaying practice changed over the years. The best data are between 2008 and 2015 where most gold assays also have a silver assay (Figure 14-8). Treasury Metals has initiated an aggressive silver re-assaying program to narrow this gap, however, results from the re-assaying program were not available at the time of this report.

Figure 14-8: Unassayed Silver Intervals by Drill Campaigns



Source: SRK, 2023.

14.4.1.2 Goliath East

The raw assay statistics were evaluated, grouping all assays intersecting the mineralized zones. The assays from all of the zones were back tagged from their corresponding wireframes. Table 14-6 summarizes the descriptive statistics for raw, uncapped, gold values for the Goliath East deposit.

Table 14-6: Descriptive Statistical Data for Uncapped Gold Assays at Goliath East

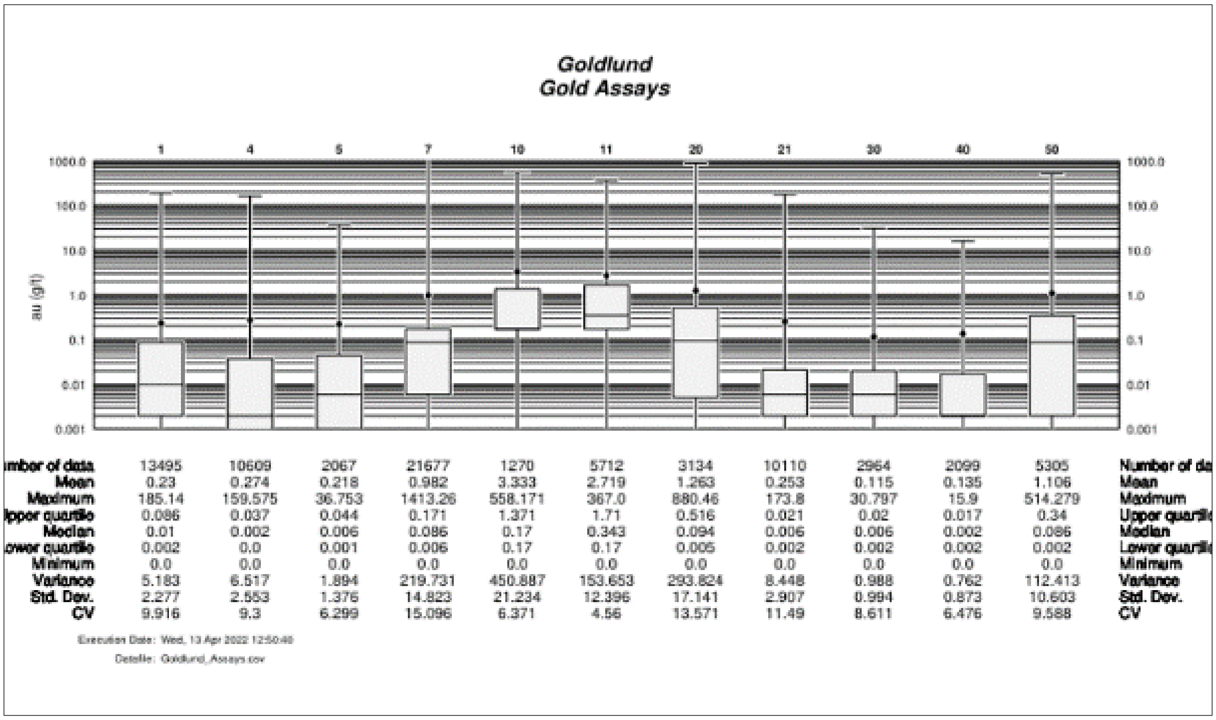
Domain	All	1100	2000	3000	4000	5000	6000
Valid Cases	1,369	369	58	218	56	147	521
Mean	0.37	0.29	0.30	0.54	0.23	0.23	0.43
Maximum	46.74	4.02	2.30	45.37	1.66	3.87	46.74
Upper Quartile	0.28	0.31	0.35	0.35	0.31	0.23	0.25
Median	0.11	0.12	0.19	0.12	0.10	0.10	0.08
Lower Quartile	0.04	0.05	0.10	0.06	0.04	0.04	0.02
Standard Deviation	1.96	0.50	0.42	3.23	0.35	0.48	2.31
Coefficient of Variance	5.18	1.72	1.39	5.96	1.48	2.03	5.35

14.4.2 Goldlund

The sample lengths vary from 0.1 m to 3.0 m with the most common being a 2 m length. Typically, the sample lengths prior to 1977 were less than 1.0 m, while after this period the samples were more than 1.0 m. Therefore, the following summary statistics will be weighted by sample length.

Figure 14-9 displays the boxplots and summary statistics (weighted by sample length) for gold separated by mineralized zone. The highest mean grades are from Zones 10 and 11, the higher-grade sub-zones of Zone 1. Zones 7, 20 and 50 have very similar average grade around 1.0 g/t while the other zones display very similar averages around 0.20 g/t.

Figure 14-9: Boxplot of Gold Assay by Mineralized Zones at Goldlund



Note: Refer to Table 14-2 above for zone codes. Source: SRK, 2023.

14.4.3 Miller

The drillhole database for the Miller deposit consists of 62 drillholes, 49 of which are included in the resource estimate. Any assay values reported below detection limit were assigned half the detection limit for statistical analysis and grade estimation. Any missing values were assigned a zero. Table 14-7 presents the descriptive statistics for the Miller deposit. While gold mineralization occurs in all rock types, the main granodiorite body contains the bulk of the higher-grade values with a mean grade of 0.69 g/t compared with the average grades of less than 0.1 g/t for the other zones.

Table 14-7: Descriptive Statistical Data for Uncapped Gold Assays at Miller

Domain	All	Andesite	Gabbro	Main Granodiorite	Secondary Granodiorite
Valid Cases	6,666	1,927	1,374	3,182	183
Mean	0.35	0.03	0.07	0.69	0.02
Maximum	137.00	7.58	15.33	137.00	1.04
Upper Quartile	0.04	0.00	0.00	0.26	0.00
Median	0.02	0.00	0.00	0.00	0.00
Lower Quartile	0.00	0.00	0.00	0.00	0.00
Standard Deviation	2.86	0.26	0.58	4.08	0.09
Coefficient of Variance	8.05	8.69	7.65	5.91	4.52

14.5 Density Assignment

Bulk density forms an important component of the mineral resource statement. Bulk density is used to convert the volumes estimated for each of the domains into tonnage. Bulk density is normally determined in the field by weighting a small piece of core in air and in water and then density can be determined by using the formula:

$$Density = \frac{(Weight\ dry)}{(Weight\ dry - Weight\ wet)}$$

14.5.1 Goliath

Treasury Metals provided 545 bulk density measurements. Core samples typically measuring 10 cm were analysed at the same laboratory used for the assays.

The 545 samples averaged 2.76 g/cm^3 with a median value of 2.75 g/cm^3. There was a slight increase in density with the average gold grade, but it is very minor. In 3D the bulk density data is well distributed throughout the entire deposit. A base bulk density value was assigned for each domain and then density was interpolated where data permitted the interpolation to better define any local variations in bulk densities. Table 14-8 shows the base bulk density assigned to the domains. The interpolated bulk density relied on an inverse distance squared (ID^2) methodology carried out in a single pass using a minimum of two samples and maximum of 15 samples, and a maximum of three samples originating from a single drillhole.

Table 14-8: Bulk Density Assignment by Domain at Goliath

Domain (Domain Code)	Bulk Density (t/m^3)
Main Zone HG (100, 110)	2.76
Main Zone LG (1000)	2.70
C Zone HG (310, 320)	2.77
C Zone LG (3000)	2.76
All other Zones	2.76
Waste outside the Wireframes	2.75
Overburden	1.75

All domains at the Goliath East deposit were assigned a fixed density of 2.76 t/m^3 as no density data has been collected from any of the zones yet.

14.5.2 Goldlund

The Goldlund drillhole database contains a total of 2,154 bulk density measurements that were collected by both Tamaka and First Mining on representative pieces of drill core. The core samples were weighted in air and then in water, the buoyancy method, using an Acculab VIC-612 electronic balance. Table 14-9 summarizes the average density values by domains for Goldlund.

Table 14-9: Bulk Density Average by Domain at Goldlund

Domain (Domain Code)	Bulk Density (t/m³)
Zone 1 (1)	2.76
Zone 4 (4)	2.74
Zone 5 (5)	2.77
Zone 7 (7)	2.74
Zone 1 High-grade A (10)	2.72
Zone 1 Hight Grade B (11)	2.72
Zone 2 Intrusive (20)	2.74
Zone 2 Volcanic (21)	2.79
Dacite (30)	2.74
Porphyry (40)	2.83
Intrusive (50)	2.75
Waste (99)	2.83

14.5.3 Miller

Bulk density measurements were collected by First Mining during the 2019 drill program. A total of 389 core samples across the Miller deposit were selected for density determination by water immersion method. All the density measurements were collected by First Mining staff at the Goldlund exploration camp. The core samples tested were generally whole core pieces ranging in length from approximately 10 to 15 cm. Core samples were then weighed in air and in water. The mean value was assigned to the three interpreted domains. Overburden was assigned a density of 2.2. Table 14-10 shows the descriptive statistics for density used in the Miller deposit by domain.

Table 14-10: Bulk Density by Domain for Miller Deposit

Domain	Granodiorite	Gabbro	Andesite	Overburden
Count	148	88	153	
Minimum	2.62	2.74	2.71	
Maximum	3.03	3.12	2.08	
Mean	2.82	2.93	2.83	2.20 (assigned)
Median	2.83	2.91	2.83	
Standard Deviation	0.07	0.08	0.08	
CV	0.02	0.03	0.03	

No bulk density measurements were collected during the 2021 drill program by Treasury Metals.

14.6 Grade Capping/Outlier Restrictions

Capping of high-grade values is carried out to prevent the over-smearing of outlier values throughout the resource model. While capping is not an exact science, several methods have been developed to determine the most appropriate capping level. Capping is best achieved on uncomposited assay data but where short composite lengths are selected, capping of composited is also acceptable. A combination of probability plots and degradation analysis are generally applied to determine the potential risk of grade distortion from higher grade assays.

14.6.1 Goliath

A review of the Goliath and Goliath East assay data showed that the uncapped assays displayed some high coefficient of variations and that capping of individual assays was likely appropriate.

The grade capping strategy used has the benefit of limiting the influence of extreme outliers while restricting the range of influence of the “mild” high-grade outliers to prevent them from influencing blocks further away than the first pass search ellipsoid.

Tables 14-11 and 14-12 show a summary of the treatment of high-grade outliers for gold and silver. The cap value selected for gold and silver was generally above the 99th percentile of the raw assay distribution. The raw assay capping scenario for gold reduced the CV by approximately 50% on average for the Main Zone and C Zone. Once that data was composited at 1.5 m, the CV was further reduced.

Table 14-11: Goliath and Goliath East Gold Capping Levels

Domain (Domain Code)	Cap Level Au (g/t)	Number of Assay Affected	Total Number of Assays	Assays Capped (%)	High Grade Restriction Au (g/t)	CV of Uncapped Assays	CV of Capped Assays
Main High Grade (100, 110)	125	11	2,828	0.38	45	5.42	2.88
B Zone (210, 220, 230, 240, 250)	20	10	2,744	0.36	NA	9.58	3.43
C Zone high Grade (310, 320)	30	9	1,095	0.82	NA	3.88	1.75
D Zone (400)	20	1	735	0.14	NA	4.24	2.88
E Zone (500)	20	0	329	0	NA	2.39	2.39
H Zone (610, 620, 630, 640)	20	8	4,435	0.18	NA	3.88	2.96
Main Low Grade (1000)	20	2	12,910	0.01	NA	3.53	2.14
C Zone Low Grade (3000)	15	4	8,678	0.04	NA	2.44	2.27
All Goliath East Domains	20	2	1,295	0.15	NA	5.27	3.48
Waste (99)	20	6	87,969	0.006	NA	8.72	4.86

Table 14-12: Goliath and Goliath East Silver Capping Levels

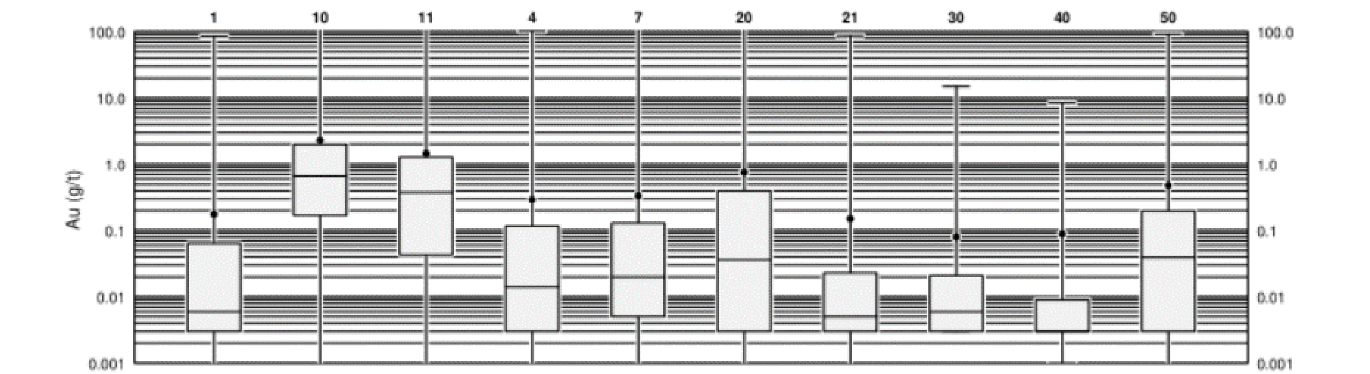
Domain (Domain Code)	Cap Level Au (g/t)	Number of Assay Affected	Total Number of Assays	Assays Capped (%)	CV of Uncapped Assays	CV of Capped Assays
Main High Grade (100, 110)	240	19	1,896	1.0	3.89	2.41
B Zone (210, 220, 230, 240, 250)	100	6	2,124	0.28	3.44	2.99
C Zone high Grade (310, 320)	60	26	862	3.0	3.62	1.62
D Zone (400)	40	1	606	0.16	4.54	2.08
E Zone (500)	40	0	303	0	1.68	1.68
H Zone (610, 620, 630, 640)	100	5	3,175	0.15	5.53	3.28
Main Low Grade (1000)	100	8	8,865	0.09	5.02	2.63
C Zone Low Grade (3000)	100	4	6,904	0.05	3.17	2.59
All Goliath East Domains	100	0	581	0	1.67	1.67
Waste (99)	100	14	59,041	0.02	8.26	3.67

14.6.2 Goldlund

Grade capping is often carried out prior to compositing to limit any possible smearing of high-grade assays inside a composite. Statistical analysis and grade capping for Goldlund was carried out on the 2 m composites due to the use of variable sample length intervals and due to the shorter samples being taken in the strongly altered and mineralized drill core. This creates a selection bias in the sampling and capping the assays would result in a reduction of too much metal from the composite data.

The side-by-side boxplots in Figure 14-10 show that for each zone there are some very high-grade gold assays even after compositing to a regular length. The highest average grade can be found in Zone 10, at 354.5 g/t Au. This appears to be an extreme grade that requires adjustment or grade capping prior to block model grade estimation. Compositing the assays to 2 m composites has not changed the average but has reduced the variability as the CV is reduced for each of the zones. For example, the Zone 7 assays have a CV of 15.09, as shown in Figure 14-9, while the 2 m composites have a CV of 7.57, as shown in Figure 14-10, which is a reduction in variability of almost half. The maximum gold grade for the Zone 10 assays is 1,413 g/t Au, while the 2 m composites for Zone 10 have a maximum grade of 354.52 g/t Au. While the CV for each of the zones has been reduced by compositing, they are still very high, due to some high-grade outliers. Adjustment of these outliers by grade capping is required.

Figure 14-10: Side-by-Side Boxplot of Composited Au (g/t) by Zones



	1	10	11	4	7	20	21	30	40	50	
Number of data	13066	3099	7529	5235	17348	1917	6365	2243	1448	3040	Number of data
Mean	0.176	2.307	1.458	0.293	0.337	0.754	0.152	0.08	0.089	0.481	Mean
Maximum	85.715	354.523	192.765	101.09	148.456	137.66	86.904	14.898	8.427	91.639	Maximum
Upper quartile	0.064	1.984	1.287	0.117	0.13	0.4	0.023	0.021	0.009	0.198	Upper quartile
Median	0.006	0.66	0.379	0.014	0.02	0.036	0.005	0.006	0.003	0.039	Median
Lower quartile	0.003	0.17	0.043	0.003	0.005	0.003	0.003	0.003	0.003	0.003	Lower quartile
Minimum	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.003	0.001	0.0	Minimum
Variance	1.548	95.501	23.406	4.648	6.523	29.139	1.744	0.261	0.223	6.977	Variance
Std. Dev.	1.244	9.772	4.838	2.156	2.554	5.398	1.32	0.511	0.472	2.641	Std. Dev.
CV	7.073	4.235	3.319	7.351	7.577	7.158	8.665	6.412	5.286	5.486	CV

Execution Date: Tue, 03 May 2022 09:39:13
Datafile: COMPS_RC_AU_AUCAP.csv

Source: SRK, 2023.

To determine if a composite grade was an outlier and should be capped, a series of graphical and statistical summaries were considered, including log-probability plots, cutting statistics plots and degradation analysis plots. Table 14-13 displays the summary statistics for the uncapped (AUGPT) and capped (CAPAU) gold grades. Capping of the 2 m composites has reduced the metal in the composites by approximately 6% with a capping of 70 composites. The capping grades range from no capping in Zone 5 up to 90 g/t Au for Zone 1.

Capping the outlier grades has also reduced the variability with a change in the global CV from 8.103 for the uncapped 2 m composites to 5.640 for the capped 2 m composites. While capping has helped reduce the CV values, they are still very high, ranging from 2.797 for Zone 10 to 6.634 for Zone 7. These high CV values indicate that the 2 m composites for the mineralized zones should be separated into more stable statistical groups and reduce the CV values for each zone prior to block grade estimation.

Table 14-13: Summary Statistics for 2 m Composites Uncapped & Capped Gold Grades

Zone	No of 2 m AUGPT	Ave. of AUGPT	CV of AUGPT	No. of 2 m CAPAU	Ave of CAPAU	CV of CAPAU	Capping Grade	No. Capped	Mean Ratio	CV Ratio
1	13,066	0.176	7.073	13,066	0.165	4.899	21	5	0.938	0.693
10	3,099	2.307	4.235	3,099	2.141	2.797	90	6	0.928	0.660
11	7,529	1.458	3.319	7,529	1.431	2.901	75	5	0.981	0.874
20	1,917	0.754	7.158	1,917	0.588	3.547	30	5	0.780	0.496
21	6,365	0.152	8.665	6,365	0.14	5.352	15	4	0.921	0.618
4	5,235	0.293	7.35	5,235	0.273	5.342	35	2	0.932	0.727
5	1,602	0.157	4.038	1,602	0.157	4.038	no cap	0	1.000	1.000
7	17,348	0.337	7.577	17,348	0.327	6.634	60	7	0.970	0.876
30	2,243	0.08	6.412	2,243	0.077	5.728	9.5	2	0.963	0.893
40	1,448	0.089	5.286	1,448	0.086	4.917	5	2	0.966	0.930
50	3,040	0.482	5.486	3,040	0.455	4.359	35	2	0.944	0.795
100	31,192	0.039	11.152	31,192	0.033	6.812	5	30	0.846	0.611
Total	94,084	0.356	8.103	94,084	0.336	5.640	-	70	0.914	0.722

14.6.3 Miller

A review of the Miller assay data indicated that capping or high-grade outliers was necessary. Capping was applied to the assay data prior to compositing. Assays were capped at 35 g/t gold. A total of eight of 6,652 assays were affected by capping, all were from the main granodiorite domain. Table 14-14 summarizes the basic statistical information for the capped assays for the Miller deposit.

Table 14-14: Miller Deposit Basic Statistical Data for Capped Assays

Domain	All	Andesite	Gabbro	Main Granodiorite	Secondary Granodiorite
Valid Cases	6666	1927	1374	3182	183
Mean	0.314	0.03	0.07	0.61	0.02
Maximum	35.0	7.58	7.0	35.0	1.04
Upper Quartile	0.04	0.00	0.00	0.26	0.00
Median	0.00	0.00	0.00	0.02	0.00
Lower Quartile	0.00	0.00	0.00	0.00	0.00
Standard Deviation	1.76	0.26	0.45	2.49	0.09
Coefficient of Variance	5.63	8.69	6.46	4.10	4.52

14.7 Composites

Assay data are composited to assure equal weighting during the grade interpolation. Selecting an appropriate composite length is a function of the individual sample lengths and the width of the mineralized zones and anticipated mining method.

14.7.1 Goliath

The drill core was preferentially sampled in either 0.5, 1.0, or 1.5 m intervals. Within the mineralized domains, the core length average was 1.08 m.

Based on an analysis of the sample length statistics, the QP selected a composite length of 1.5 m. The composite size selected is above the third quartile to allow grade variations to be represented while reducing the variance.

Assays were length-weight averaged, and any grade capping was applied to the raw assay data prior to compositing. True gaps in sampling were composited at zero grade.

The 1.5 m composite intervals were created moving downward from the collar of the holes toward the hole bottoms. Composite lengths are automatically adjusted by the software to leave no remnants. The adjustment resulted in composite lengths ranging between 0.77 and 2.25 m, with a mean and median of 1.5 m, and a standard deviation of 0.08. Table 14-15 and 14-16 show the descriptive statistics for gold and silver capped composites within the various domains for Goliath and Goliath East.

Table 14-15: Descriptive Statistical Data for Capped Gold Composites at Goliath

Domain	Main High-Grade	Main Low-Grade	C High-Grade	C Low-Grade	B Zone	D Zone	E Zone	H Zone	Goliath East
Valid Cases	2,042	9,532	803	6,255	1,997	526	226	3,331	1,180
Mean	3.83	0.28	2.13	0.28	0.45	0.36	0.38	0.43	0.32
Maximum	125.00	13.76	30.0	12.63	20.00	8.45	4.50	20.0	20.00
Upper Quartile	3.00	0.34	2.15	0.33	0.34	0.36	0.37	0.36	0.31
Median	1.54	0.18	1.08	0.16	0.15	0.16	0.13	0.16	0.12
Lower Quartile	0.83	0.07	0.66	0.07	0.09	0.09	0.08	0.09	0.05
Standard Deviation	9.30	0.46	3.18	0.48	1.37	0.78	0.70	1.07	1.03
Coefficient of Variance	2.43	1.64	1.49	1.74	3.02	2.14	1.87	2.53	3.22

Table 14-16: Descriptive Statistical Data for Silver Composites at Goliath

Domain	Main High-Grade	Main Low-Grade	C High-Grade	C Low-Grade	B Zone	D Zone	E Zone	H Zone	Goliath East
Valid Cases	1,289	6,118	612	4,871	1,484	414	204	2,257	503
Mean	12.09	2.29	7.40	1.71	2.69	1.34	1.69	1.83	1.27
Maximum	240.00	100.00	60.00	69.75	100.00	18.00	18.00	90.06	31.50
Upper Quartile	11.25	2.26	8.00	1.97	1.88	1.37	2.00	1.66	1.33
Median	4.38	1.00	3.43	1.00	1.00	0.82	1.00	0.50	0.50
Lower Quartile	1.41	0.50	1.09	0.50	0.50	0.50	0.50	0.09	0.50
Standard Deviation	24.83	5.01	10.96	3.26	7.51	2.05	2.51	4.69	2.44
Coefficient of Variance	2.05	2.18	1.48	1.90	2.79	1.53	1.49	2.56	1.92

14.7.2 Goldlund Compositing

The drillhole data was composited to 2 m down-the-hole composites, without consideration of any geological boundaries. The 2 m composite length was selected because it is approximately half of the block height used for the block model and is the next most common sample length used.

The unsampled intervals were assigned background gold grades prior to compositing. Some of these unsampled intervals were long and this would generate multiple 2 m composites. Therefore, the treatment of the unsampled intervals will have a greater impact on the number of composites and on the average grade of those composites.

Figure 14-10 (above) shows side-by-side boxplots and length weighted summary statistics for the 2 m composites based on sample assays that have the unsampled intervals assigned to a background value.

Compositing the assays to 2 m composites has not changed the average but has reduced the variability as the CV is reduced for each of the zones. For example, the Zone 1 assays have a CV of 9.93, as shown in Figure 14-9, while the 2 m composites have a CV of 4.36, as shown in Figure 14-10, which is a reduction in variability of almost half. The maximum gold grade for the Zone 1 assays is 1,413 g/t Au, while the 2 m composites for Zone 1 have a maximum grade of 354.57 g/t Au. While the CV for each of the zones has been reduced by compositing, they are still very high, due to some high-grade outliers. Adjustment of these outliers by grade capping is required.

14.7.3 Miller

Most of the Miller drill core was sampled in 1.0 m intervals, 96 percent of the assay lengths were 1.0 m or less in length. A composite length of 2.0 m was selected to reduce the variance and allow for a better block grade definition.

Assays were length-weight averaged, and any grade capping was applied to the raw assay data prior to compositing. True gaps in sampling were composited at zero grade.

The 2.0 m composite intervals were created moving downward from the collar of the holes toward the hole bottoms. Composite lengths are automatically adjusted by the software to leave no remnants. The adjustment resulted in composite lengths ranging between 1.16 and 2.81 m, with mean and median of 2.0 m, and a standard deviation of 0.05. Table 14-17 shows the descriptive statistics for gold capped composites within the various domains for the Miller deposit.

Table 14-17: Descriptive Statistical Data for Capped Composites at Miller

Domain	Main Granodiorite	Secondary Granodiorite	Gabbro	Andesite
Valid Cases	1,600	90	1,197	2,028
Mean	0.56	0.02	0.04	0.01
Maximum	17.63	0.76	3.68	2.66
Upper Quartile	0.42	0.01	0.00	0.00
Median	0.07	0.00	0.00	0.00
Lower Quartile	0.00	0.00	0.00	0.00
Standard Deviation	1.57	0.08	0.25	0.12
Coefficient of Variance	2.82	3.90	6.08	8.66

14.8 Block Model Definitions

14.8.1 Goliath and Goliath East

The block models were constructed using GEMS 6.8™. An elongated block size of 5 m along the strike of the deposit (horizontally) x 5 m vertically x 2 m across was selected for the Goliath model based on the shape of the mineralized zones, mining selectivity considerations and the density of the dataset. This block matrix size assumed a mid-size open pit operation that would also be suitable for long hole underground operation.

The Goliath East model size was set at 5 by 5 by 5 m because of the lower density sampling. The block size was chosen to reflect a potential selective mining unit (SMU) of 5 m x 5 m x 5 m, given the anticipated open-pit mining scenario.

The block models were defined on the project coordinate system with a 0-degree rotation.

Table 14-18 lists the block model origins for the Goliath model and Table 14-19 lists the origin of the Goliath East block model.

Table 14-18: Goliath Block Model Matrix

Coordinates	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	Number of Blocks
Easting	526,050	529,240	3,190	5	638
Northing	5,511,500	5,512,736	1,236	2	618
Elevation	410	500	910	5	182

Table 14-19: Goliath East Block Model Matrix

Coordinates	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	Number of Blocks
Easting	529,240	531,700	2,460	5	492
Northing	5,511,800	5,514,500	2,700	5	540
Elevation	410	500	910	5	182

14.8.2 Goldlund

The block model was constructed using MineSight™ 15.80.5 software using block model definition as outlined in Table 14.20. The block size was chosen to reflect a potential selective mining unit (SMU) of 5 m x 5 m x 5 m, given the anticipated open-pit mining scenario. The block model covers an area of approximately 4.7 by 2.5 km in plan view, and approximately 800 m vertically. The block model coordinates are in the NAD83 UTM Zone 15 grid system.

Table 14-20: Goldlund Block Model Matrix

Coordinates	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	Number of Blocks
Easting	545,000	549,700	4,700	5	940
Northing	5,265,000	5,529,000	2,500	5	500
Elevation	350	460	810	5	162

14.8.3 Miller

The Miller block model was constructed using GEMS 6.8™ software and a block model definition as outlined in Table 14-21. The block size was chosen to reflect a potential selective mining unit (SMU) of 5 m x 5 m x 5 m, given the anticipated open-pit mining scenario. The block model coordinates are in the NAD83 UTM Zone 15 grid system.

Table 14-21: Miller Block Model Matrix

Coordinates	Minimum (m)	Maximum (m)	Extent (m)	Block Size (m)	Number of Blocks
Easting	553,800	555,200	1,400	5	280
Northing	5,533,000	5,534,000	1,000	5	200
Elevation	-135	420	285	5	57

14.9 Variography

14.9.1 Goliath

Geostatisticians use a variety of tools to describe the pattern of spatial continuity, or strength of the spatial similarity, of a variable with separation distance and direction. If we compare samples that are close together, it is common to observe that their values are quite similar. As the distance between samples increases, there is likely to be less similarity in the values. The experimental variogram mathematically describes this process. It is commonly represented as a graph that shows the variance in measurements with distance between all pairs of sampled locations.

In all semi-variograms, the distance where the model first flattens out is known as the range. Sample locations separated by distances closer than the range are believed to be spatially auto-correlated. The sill is the value on the Y-axis where the model attains the range, while the nugget is the value at the location where the model intercepts the Y-axis. The nugget typically represents variation at a micro scale that can be attributed to measurement errors, sources of variation at distances smaller than the sampling interval, or both. Therefore, the shape of the semi-variogram describes the pattern of spatial continuity. A very rapid decrease near the origin indicates short-scale variability. A more gradual decrease moving away from the origin suggests longer-scale continuity.

Various semi-variogram types exist. Using SAGE™ software, experimental correlograms for gold and silver were computed for the various domains.

The resulting anisotropy models generated were visually inspected in GEMS™ to ensure the ellipsoid model corresponded well with the expected orientation of the deposit.

Variograms were developed for the Main and C zones are they offer the most short-range sampling. The Main zone variograms were used to estimate zones B, D, E and H as no robust variograms could be generated from these secondary zones. For gold, the maximum apparent range plunges steeply to the west in the plane of the mineralization. The nugget is very high, between 70 to 84% of the sill value. At 100% of the sill, the maximum range is estimated to be between 150 and 170 m. The direction and plunge represented by the variogram coincide with the known interpreted plunge of the mineralization.

Table 14-22 lists the variogram parameters used in the model for gold and silver. The variograms were fitted using the GEMS™ "Z-X-Z" righthand rule rotation method which is dependent of the block model orientation.

Table 14-22: Goliath Variogram Parameters

Domain Code	Element	Model	Nugget	C1	ZXZ (degree)	C1 Range (m)
Main Zone and (B, D, E, H)	Gold	Exponential	0.700	0.300	13, 72, -8	10, 58, 4
C Zone (High and Low Grade)	Gold	Exponential	0.845	0.155	19, 80, 59	51, 18, 3
Main Zone and (B, D, E, H)	Silver	Exponential	0.794	0.206	2, 72, -6	45, 143, 13
C Zone (High and Low Grade)	Silver	Exponential	0.807	0.193	12, 85, -68	59., 215, 21

Because of the relatively wider-spaced sampling in the Goliath East zones, no variograms could be constructed for any of the Goliath East zones. The model was interpolated using inverse distance square interpolant (ID^2).

14.9.2 Goldlund

Variography is a study of the spatial continuity of an attribute. The variography study for Goldlund consisted of two parts: indicator semi-variograms to estimate the proportion of high-grade in a block, and correlograms of the gold grades for the estimation of low-grade (LG) and high-grade (HG) block grade estimates.

14.9.2.1 CV Partitioning

The high CV values observed for each of the mineralized zones requires some additional effort to separate the 2 m composite grades into more stable statistical groups. One approach is to use a CV partitioning methodology to separate the composite grades.

The concept of CV partitioning is to find a grade threshold that can separate the composites into two groups with the lowest CV. This concept is based on a paper by H. Parker, "Statistical Treatment of Outlier Data in Epithermal Gold Deposit Reserve Estimation" (1991). The concept is to calculate the CV of the composite grades starting with all the data, and then leave one out at a time to examine how the CV changes as composite grades are excluded.

Table 14-23 shows the thresholds used to divide the gold grades into separate domains that have minimum CV values. This separation creates more statistically stable groups of composite data that are suitable for block grade estimation.

Zones 10 and 11, which are high-grade domains within an overall low-grade zone (Zone 1), were not estimated with indicator kriging, and therefore were not evaluated for CV partitioning.

Table 14-23: CV Partitioning of Gold Grades by Zones

Zone	Threshold	No. of LG Data	LG Ave CAPAU	LG CV	Threshold	No. of HG Data	HG Ave. CAPAU	HG CV
1	≤ 0.245	11,592	0.030	1.732	>0.245	1,474	1.224	1.735
4	≤ 0.600	4,703	0.064	1.710	>0.600	456	2.512	1.726
5	≤ 0.130	1,292	0.016	1.681	>0.130	230	0.821	1.657
7	≤ 1.000	16,452	0.090	1.845	>1.000	923	4.535	1.832
20	≤ 0.650	1,343	0.100	1.552	>0.650	327	2.763	1.565
21	≤ 0.140	5,296	0.014	1.698	>0.140	733	1.090	1.701
30	≤ 0.180	1,564	0.020	1.660	>0.180	104	0.907	1.691
40	≤ 0.075	1,265	0.007	1.622	>0.075	181	0.634	1.638
50	≤ 0.750	2,397	0.095	1.540	>0.750	319	3.199	1.541
100	≤ 0.09	31,583	0.008	1.550	>0.090	2,081	0.646	1.563

14.9.2.2 Indicator Variography

For each zone, down-the-hole indicator semi-variograms were computed to determine the nugget effect and directional indicator semi-variograms were computed in multiple directions using Datamine Supervisor® software. The indicator thresholds are listed in Table 14-23. The indicator semi-variograms were then fitted manually, considered a nugget effect and two spherical structures. The models were adjusted, if required, to better match the controls on the gold mineralization, which is in part interpreted from the historical stopes in Zone 1 that display a low angle plunge to the west.

Table 14-24 shows a listing of the indicator semi-variogram models used for kriging the HG proportion (HGIND) by zone for blocks in the model.

Table 14-24: Indicator Semi-Variogram Models for Kriging the HG Proportion in the Block Model

Zone	1	20	21	4	5	7	30	40	50
Cc0	0.345	0.372	0.349	0.323	0.275	0.388	0.438	0.198	0.362
C1	0.547	0.572	0.453	0.521	0.601	0.577	0.412	0.546	0.495
C2	0.108	0.0559	0.198	0.156	0.124	0.034	0.15	0.256	0.143
C1: Range 1 - Z	22	25	30	47	40	20	45	50	25
C1: Range 2 - X	20	20	30	20	22	20	45	50	25
C1: Range 3 - Y	10	10	10	10	10	15	15	15	10
Rotation 1	-126.9	50	60	-109.1	60	80	-103.3	-105	60.9
Rotation 2	-24.1	0	0	10.0	0	0	58.5	-40	23.4
Rotation 3	73.5	-80	-75	-95.1	-70	-75	70.6	-90	-68.1
C2: Range 1 - Z	60	70	80	150	90	60	100	80	50
C2: Range 2 - X	35	50	60	85	50	50	80	80	40
C2: Range 3 - Z	15	20	30	20	20	25	40	25	15
Model Type 1-Spherical	1	1	1	1	1	1	1	1	1

14.9.2.3 Gold Grade Variography

The spatial continuity of the 2 m composite gold grades was assessed using Datamine Supervisor® software. The correlogram was selected as the methodology to study the spatial continuity of the gold grades and to build models of the spatial continuity for block grade estimation using ordinary kriging.

Down-the-hole correlograms were calculated to determine the nugget effect for each zone and directional correlograms were computed in multiple directions for each zone to determine the anisotropy of the gold mineralization. The experimental correlogram of the gold grades were then fitted using manual fitting methodology that considered a nugget effect and two spherical structures. The models were adjusted, if required, to better match the controls on the gold mineralization, which is in part interpreted from the historical stopes in Zone 1 that display a low angle plunge to the west.

Table 14-25 presents a summary of the model parameters used for kriging the LG gold and the HG gold grades into the blocks in the model. The nugget effects are typically between 0.2 and 0.5, which indicates a significant level of short-scale variability that is typical for Archean lode-gold deposits. As well, the first structure ranges of the spatial models are short, with ranges of 15 to 40 m. This is a further indication of the significant level of short-scale variability. The second structure ranges are much longer, 100 to 160 m.

Table 14-25: Gold Correlograms Parameters for Goldlund

Zone	1	10	11	4	5	7	20	21	30	40	50	1
C0	0.45	0.368	0.488	0.563	0.593	0.505	0.275	0.542	0.438	0.288	0.465	0.424
C1	0.432	0.499	0.393	0.388	0.385	0.476	0.61	0.427	0.412	0.655	0.47	0.42
C2	0.118	0.133	0.12	0.0486	0.0217	0.0191	0.124	0.0307	0.15	0.0576	0.062	0.156
C1: Range 1 - Z	40	25	30	20	20	40	40	15	45	40	25	60
C1: Range 2 - X	20	20	15	20	15	40	22	15	45	40	25	30
C1: Range 3 - Y	10	10	10	10	10	10	10	15	15	15	10	30
Rotation 1	-126.9	-126.9	-126.9	50	60	-109.1	60	80	-103.3	-105	60.9	65
Rotation 2	-24.1	-24.1	-24.1	0	0	10.0	0	0	58.5	-40	23.4	0
Rotation 3	73.5	73.5	73.5	-80	-75	-95.1	-70	-75	70.6	-90	-68.1	115
C2: Range 1 - Z	100	110	90	50	50	160	90	85	100	80	50	100
C2: Range 2 - X	75	80	85	50	45	120	50	85	80	50	40	90
C2: Range 3 - Z	15	15	20	10	30	20	20	40	40	25	15	45
Model type 1-spherical	1	1	1	1	1	1	1	1	1	1	1	1

14.9.3 Miller

Attempts to construct variograms for the Miller deposit were not successful because of the wider spaced-drill mesh. The resulting variogram reflected the drillhole orientation rather than the orientation of the mineralized domains, for this reason, the model was interpolated using an inverse distance square interpolant instead of ordinary kriging.

14.10 Estimation/Interpolation Methods

14.10.1 Goliath

The Goliath model was constructed in multiple steps with each zone being interpolated in separate models and a final model was prepared by assembling the various component using weighted averages of each of the sub-model. Rock coding in the final model was based on the lithology with the highest percentage contained within the block.

All grades were estimated using ordinary kriging.

The Goliath model was interpolated in three consecutive passes using the following parameters:

- Pass 1 used an ellipsoid search with 8 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of three holes to be used in the search.
- Pass 2 used an ellipsoid search with 6 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of two holes to be used in the search.
- Pass 3 used an ellipsoid search with 3 minimum and 15 maximum samples. A maximum of three samples per hole was imposed on the data selection, allowing a block to be interpolated with composites originating from one hole.

The wireframe boundaries were considered hard for all zones.

While the geological domains at Goliath generally strike east-northeast, the strike of the zones generally change gradually from east to west. The search ellipsoid orientation was adjusted via the use of three subdomains to optimize the alignment of the search volume with the mineralization. Table 14-26 lists the final values used in the resource model for the range of the major, semi-major, and minor axes. Rotation angles are based on the GEMS ZXZ methodology, which uses a conventional right-hand rule. The search ellipsoids dimension and orientation were applied for both the gold and silver interpolation plan.

The Goliath East model was created in GEMS 6.8™ with a single folder setup, using inverse distance (ID^2) for interpolating the gold and silver grades. The interpolation for both the Goliath and Goliath East models was carried out in a multi-pass approach, with an increasing search dimension coupled with decreasing sample restrictions.

Similarly, the Goliath East model was divided into two subdomains to better orient the search ellipse with the mineralized zones.

The Goliath East model was interpolated in four consecutive passes using the following parameters:

- Pass 1 used an ellipsoid search with 3 minimum and 12 maximum samples. A maximum of two samples per hole was imposed on the data selection, forcing a minimum of three holes to be used in the search.
- Pass 2 used an ellipsoid search with 4 minimum and 12 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of two holes to be used in the search.
- Pass 3 used an ellipsoid search with 1 minimum and 12 maximum samples. A maximum of three samples per hole was imposed on the data selection, allowing a block to be interpolated with composites originating from one hole.
- Pass 4 used an ellipsoid search with 1 minimum and 12 maximum samples. A maximum of three samples per hole was imposed on the data selection, allowing a block to be interpolated with composites originating from one hole.

Table 14-26: Goliath Interpolation Parameters

Zone	Sub-Domain	Pass	Right Hand Rule Rotation			Range (m)			Minimum No. of Composites	Maximum No. of Composites	Maximum per Drill Hole
			Z	X	Z	X	Y	Z			
M1	East	1	30	70	50	75	31	10	8	15	3
M1	East	2	30	70	50	135	55	15	6	15	3
M1	East	3	30	70	50	229	94	25	3	15	3
M1	Central	1	17	70	75	75	31	10	8	15	3
M1	Central	2	17	70	75	135	55	15	6	15	3
M1	Central	3	17	70	75	229	94	25	3	15	3
M1	West	1	0	70	75	75	31	10	8	15	3
M1	West	2	0	70	75	135	55	15	6	15	3
M1	West	3	0	70	75	229	94	25	3	15	3
M2	East	1	30	70	50	75	31	10	8	15	3
M2	East	2	30	70	50	135	55	15	6	15	3
M2	East	3	30	70	50	229	94	25	3	15	3
M2	Central	1	17	70	75	75	31	10	8	15	3
M2	Central	2	17	70	75	135	55	15	6	15	3
M2	Central	3	17	70	75	229	94	25	3	15	3
M2	West	1	0	70	75	75	31	10	8	15	3
M2	West	2	0	70	75	135	55	15	6	15	3
M2	West	3	0	70	75	229	94	25	3	15	3
C1	East	1	30	70	50	75	31	10	8	15	3
C1	East	2	30	70	50	135	55	15	6	15	3
C1	East	3	30	70	50	229	94	25	3	15	3
C1	Central	1	17	70	75	75	31	10	8	15	3
C1	Central	2	17	70	75	135	55	15	6	15	3
C1	Central	3	17	70	75	229	94	25	3	15	3
C1	West	1	0	70	75	75	31	10	8	15	3
C1	West	2	0	70	75	135	55	15	6	15	3
C1	West	3	0	70	75	229	94	25	3	15	3
C2	East	1	30	70	50	75	31	10	8	15	3
C2	East	2	30	70	50	135	55	15	6	15	3
C2	East	3	30	70	50	229	94	25	3	15	3
C2	Central	1	17	70	75	75	31	10	8	15	3
C2	Central	2	17	70	75	135	55	15	6	15	3
C2	Central	3	17	70	75	229	94	25	3	15	3
C2	West	1	0	70	75	75	31	10	8	15	3
C2	West	2	0	70	75	135	55	15	6	15	3
C2	West	3	0	70	75	229	94	25	3	15	3
B to H	East	1	30	70	50	75	31	10	8	15	3
B to H	East	2	30	70	50	135	55	15	6	15	3
B to H	East	3	30	70	50	229	94	25	3	15	3
B to H	Central	1	17	70	75	75	31	10	8	15	3
B to H	Central	2	17	70	75	135	55	15	6	15	3
B to H	Central	3	17	70	75	229	94	25	3	15	3
B to H	West	1	0	70	75	75	31	10	8	15	3
B to H	West	2	0	70	75	135	55	15	6	15	3
B to H	West	3	0	70	75	229	94	25	3	15	3
Low Grade	East	1	30	70	50	75	31	10	8	15	3
Low Grade	East	2	30	70	50	135	55	15	6	15	3
Low Grade	East	3	30	70	50	229	94	25	3	15	3
Low Grade	Central	1	17	70	75	75	31	10	8	15	3
Low Grade	Central	2	17	70	75	135	55	15	6	15	3
Low Grade	Central	3	17	70	75	229	94	25	3	15	3
Low Grade	West	1	0	70	75	75	31	10	8	15	3
Low Grade	West	2	0	70	75	135	55	15	6	15	3
Low Grade	West	3	0	70	75	229	94	25	3	15	3

The wireframe boundaries were considered hard for all zones. Table 14-27 summarizes the interpolation parameters used for the Goliath East block model. The same parameters were used for both gold and silver.

Table 14-27: Goliath East Interpolation Parameters

Zones	Pass	Right Hand Rule Rotation			Range (m)			Minimum No. of Composites	Maximum No. of Composites	Maximum per Drill Hole
		Z	Y	Z	X	Y	Z			
1101, 4200, 6001	1	20	-55	0	75	31	10	3	12	2
1101, 4200, 6001	2	20	-55	0	135	55	35	4	12	3
1101, 4200, 6001	3	20	-55	0	229	95	35	4	12	3
1101, 4200, 6001	4	0	0	0	15	15	15	1	12	3
All Other Zones	1	-48	68	0	75	31	10	3	12	2
All Other Zones	2	-48	68	0	135	55	35	4	12	3
All Other Zones	3	-48	68	0	229	95	35	4	12	3
All Other Zones	4	0	0	0	15	15	15	1	12	3

14.10.2 Goldlund

Two high grade zones (Zone 10 and 11), which occur within a lower grade envelope (Zone 1), were modelled and estimated with ordinary kriging only. The remainder of the zones, which have mixed populations of high and low grades that could not be effectively separated, were estimated using an indicator kriging approach. The method involved defining the proportion of high-grade in a block and then ordinary kriging to estimate gold grades for the low-grade and high-grade domains separately. The final block grade is then a proportional weighted average grade of the low-grade and high-grade kriged estimates. This combined kriging methodology will be referred to as probability assisted kriging or PAK.

There are five steps to the PAK procedure:

- Define the HG/LG threshold using CV partitioning to find the lowest CV for the HG/LG domains for each zone
- Evaluate indicator value and kriging of the indicator to define the proportion of HG/LG in each block in the model for each zone
- Prepare ordinary kriging using the LG composites for each block in the model for each zone
- Prepared ordinary kriging using the HG composites for each block in the model for each zone
- Combine the LG and HG block grade estimates using the indicator proportion to build the final block grade estimates

The approach described above is based on two papers by Dr. Isobel Clark, “Practical Reserve Estimation in a Shear-Hosted Gold Deposit, Zimbabwe” (1993) and in “Geostatistical Modelling for Realistic Mine Planning” (1999).

The grade block model estimation methodology considered the domains to be the principal control, with the secondary control by the mineralized zone wireframes for the estimation of the gold grades. The density item in the block model was assigned the average density of the drill core measurements by Zone.

Block model gold grades were also estimated using NN, ID2, and OK with each zone estimated independently using hard boundaries; that is, there was no sharing of composites across zone boundaries. These three additional models were used to validate the PAK methodology and to ensure that it was working as intended.

14.10.2.1 IK Estimation Parameters

The following is a summary of the parameters used to estimate the high-grade proportion in the block model (HGIND):

- Two-meter composites, assigned with an indicator value using the indicator thresholds listed in Table 14-33, were used for ordinary kriging of the high-grade proportion in the blocks for Zones 1, 4, 5, 7, 20, 21, 30, 40, 50, and 100.
- Geological zone boundaries based on the mineralized zone wireframes were used to control the selection of the 2 m composites and the blocks to be estimated in the model. There was no sharing of composite grades across the zone boundaries.
- Spatial 3D mathematical models were fitted to the experimental indicator semi-variograms for each of the zones and used for ordinary kriging of the indicator variable for each of the zones in the model (refer to Table 14-34).
- Ordinary kriging was used to interpolate the high-grade block indicator proportions using a block discretization of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the semi-variogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (refer to Table 14-37).
- A minimum of four and maximum of 16 composites were required to make a block estimate, with a maximum of four composites allowed from a single drillhole (refer to Table 14-38).

Table 14-28: Kriging Parameters for High-grade Domain Indicator Proportion (HGIND)

Zone	1	4	5	7	20	21	30	40	50
MIN-COMP	4	4	4	4	4	4	4	4	4
MAX-COMP	16	16	16	16	16	16	16	16	16
MAX-PER-DH	4	4	4	4	4	4	4	4	4
MAJOR-SRCH	120	225	135	90	105	120	150	120	75
MINOR-SRCH	70	127.5	75	75	75	90	120	120	60
VERT-SRCH	30	30	30	37.5	30	45	60	37.5	22.5
ROTATION1	-126.9	-109.1	60	80	50	60	-103.3	-105	60.9
ROTATION2	-24.1	10.0	0	0	0	0	58.5	-40	23.4
ROTATION3	73.5	-95.1	-70	-75	-80	-75	70.6	-90	-68.1
SEMI-VAR-MODEL	z01ind.var	z04ind.var	z05ind.var	z07ind.var	z20ind.var	z21ind.var	z30ind.var	z40ind.var	z50ind.var
BLK-CODE Zone	1	4	5	7	20	21	30	40	50
CMP Zone-CODE	1	4	5	7	20	21	30	40	50

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards.

14.10.2.2 Gold Grade Estimation Parameters LG Domains

The following is a summary of the parameters used to estimate the LG domain block gold grades in the block model (LGZN):

- Capped gold grade composites (CAPAU) of 2 m were used for ordinary kriging of the LG gold composites into the blocks in the model for Zones 1, 4, 5, 7, 20, 21, 30, 40, 50, and 100.
- Geological zone boundaries based on the mineralized zone wireframes and the domain codes were used to control the selection of the 2 m composites and the blocks to be estimated in the model. There was no sharing of composite grades across the zone or the domain boundaries.
- Spatial 3D mathematical models were fitted to the experimental semi-variograms of capped gold composites for each of the zones and used for ordinary kriging of the block estimates in the model (refer to Table 14-35).
- Ordinary kriging was used to interpolate the LG gold grade estimates using a block discretization of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the correlogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (refer to Table 14.38).
- A minimum of four and maximum of 12 composites were required to make a block estimate, with a maximum of four composites allowed from a single drillhole (refer to Table 14-29).

Table 14-29: Kriging Parameters for Low-Grade Domains (LGZN)

Zone	1	4	5	7	20	21	30	40	50	100
MIN-COMP	4	4	4	4	4	4	4	4	4	4
MAX-COMP	12	12	12	12	12	12	12	12	12	12
MAX-PER-DH	4	4	4	4	4	4	4	4	4	4
MAJOR-SRCH-LG	200	480	270	255	150	150	300	240	150	100
MINOR-SRCH-LG	150	360	150	255	150	135	240	150	120	90
VERT-SRCH-LG	30	60	60	120	30	90	120	75	45	45
ROTATION1-LG	-126.9	-109.1	60	80	50	60	-103.3	-105	60.9	65
ROTATION2-LG	-24.1	10.0	0	0	0	0	58.5	-40	23.4	0
ROTATION3-LG	73.5	-95.1	-70	-75	-80	-75	70.6	-90	-68.1	115
SPATIAL-MODEL	z01c.var	z04c.var	z05c.var	z07c.var	z20c.var	z21c.var	z30c.var	z40c.var	z50c.var	z100c.var
BLK-CODE Zone	1	4	5	7	20	21	30	40	50	100
CMP Zone-CODE	1	4	5	7	20	21	30	40	50	100

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards

14.10.2.3 Gold Grade Estimation Parameters for HG Domain

The following is a summary of the parameters used to estimate the HG domain block gold grades in the block model (HGZN):

- Capped gold grade composites (CAPAU) of 2 m were used for ordinary kriging of the HG gold composites into the blocks in the model for Zones 1, 4, 5, 7, 20, 21, 30, 40, 50, and 100.

- Geological zone boundaries based on the mineralized zone wireframes and the domain codes were used to control the selection of the 2 m composites and the blocks in the model for grade estimation. There was no sharing of composite grades across the zone or domain boundaries.
- Spatial 3D mathematical models were fitted to the experimental semi-variograms of capped gold 2 m composites for each of the zones and used for ordinary kriging of the block estimates in the model (refer to Table 14-35).
- Ordinary kriging was used to interpolate the HG gold grade estimates using a block discretization of 5 x 5 x 3.
- A single-pass search strategy was used, with the ranges based on the semi-variogram models. For some zones, the search ellipsoid was expanded to ensure that a reasonable amount of the zone was estimated (refer to Table 14-39).
- A minimum of four and maximum of 12 composites were required to make a block estimate, with a maximum of four composites allowed from a single drillhole (refer to Table 14-30).

Table 14-30: Kriging Parameters for HG Gold Grades (HGZN)

Zone	1	4	5	7	20	21	30	40	50	100
MIN-COMP	4	4	4	4	4	4	4	4	4	4
MAX-COMP	12	12	12	12	12	12	12	12	12	12
MAX-PER-DH	4	4	4	4	4	4	4	4	4	4
MAJOR-SRCH-HG	200	480	270	255	150	150	300	240	150	100
MINOR-SRCH-HG	150	360	150	255	150	135	240	150	120	90
VERT-SRCH-HG	30	60	60	120	30	90	120	75	45	45
ROTATION1-HG	-126.9	-109.1	60	80	50	60	-103.3	-105	60.9	65
ROTATION2-HG	-24.1	10.0	0	0	0	0	58.5	-40	23.4	0
ROTATION3-HG	73.5	-95.1	-70	-75	-80	-75	70.6	-90	-68.1	115
SPATIAL-MODEL	z01c.var	z04c.var	z05c.var	z07c.var	z20c.var	z21c.var	z30c.var	z40c.var	z50c.var	Z100c.var
BLK-CODE Zone-HG	1	4	5	7	20	21	30	40	50	100
CMP-Zone-CODE-HG	1	4	5	7	20	21	30	40	50	100

Note: The rotation convention is (ZXY, LRL) in degrees and follows the order Z, X, Y using the left, right, left rotation directions. Dips are negative downwards

14.10.2.4 Block Grade Estimation

The final block gold grade estimate (BKAU) is a probability weighted combination of the LGZN kriged gold grade estimates with the HGZN kriged gold grade estimates using the HGIND as the proportion of high-grade in each block. If a block does not have a HGZN estimate or the probability of high-grade is zero, then the block is assigned the LGZN grade. It should be noted that Zones 10 and 11 were estimated exclusively with Ordinary Kriging methodology, and this estimate was merged into the final block model grade item (BKAU).

The equation used to calculate the BKAU using the HGIND block proportion and the two kriged gold grades is shown below.

$$BKAU = (HGZN * HGIND) + (LGZN * (1 - HGIND))$$

14.10.3 Miller

The Miller model was created in GEMS 6.8™ with separate model setup for each the granodiorite and gabbro units. The final model was assembled using weighted averages of each zone. Grades were interpolated using inverse distance (ID^2). The interpolation for the Miller model was carried out in a multi-pass approach, with an increasing search dimension coupled with decreasing sample restrictions as outlined in Table 14-31.

Pass 1 used an ellipsoid search with 4 minimum and 20 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of three holes to be used in the search.

Pass 2 used an ellipsoid search with 4 minimum and 20 maximum samples. A maximum of three samples per hole was imposed on the data selection, forcing a minimum of two holes to be used in the search.

Pass 3 used an ellipsoid search with 4 minimum and 20 maximum samples. A maximum of three samples per hole was imposed on the data selection, allowing a block to be interpolated with composites originating from one hole.

Table 14-31: Miller Interpolation Parameters

Zones	Pass	Right Hand Rule Rotation			Range (m)			Minimum No of Composites	Maximum No of Composites	Maximum per Drill Hole
		Z	X	Z	X	Y	Z			
All Zones	1	45	85	-50	25	50	10	4	20	3
All Zones	2	45	85	-50	50	100	20	4	20	3
All Zones	3	45	85	-50	100	120	25	4	20	3

14.11 Block Model Validation

14.11.1 Goliath

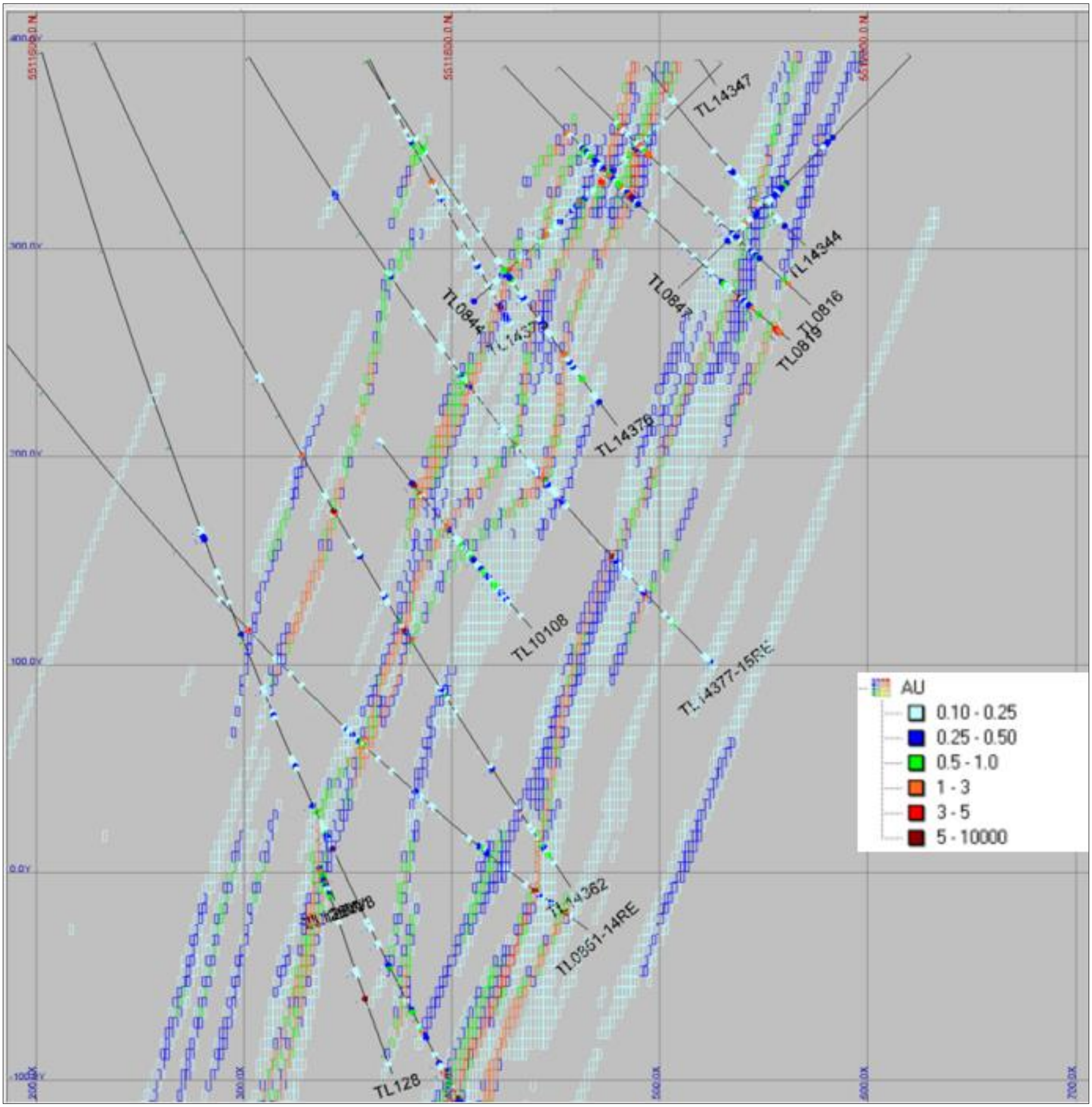
The Goliath deposit grade models were validated by four methods:

- visual comparison of colour-coded block model grades with composite grades on sections and plans
- local comparison using swath plots
- comparison of the resource estimate against the Teck Exploration underground bulk sample
- comparison of the global results with the previous estimate prepared for the PEA.

14.11.1.1 Visual Comparison

The visual comparison of block model grades on sections and plans indicated a general good correlation between drillhole composited grades and the estimated resource model grades (Figure 14-11).

Figure 14-11: Section 527500E Comparing Drill Hole Composites and Estimated Gold Grades

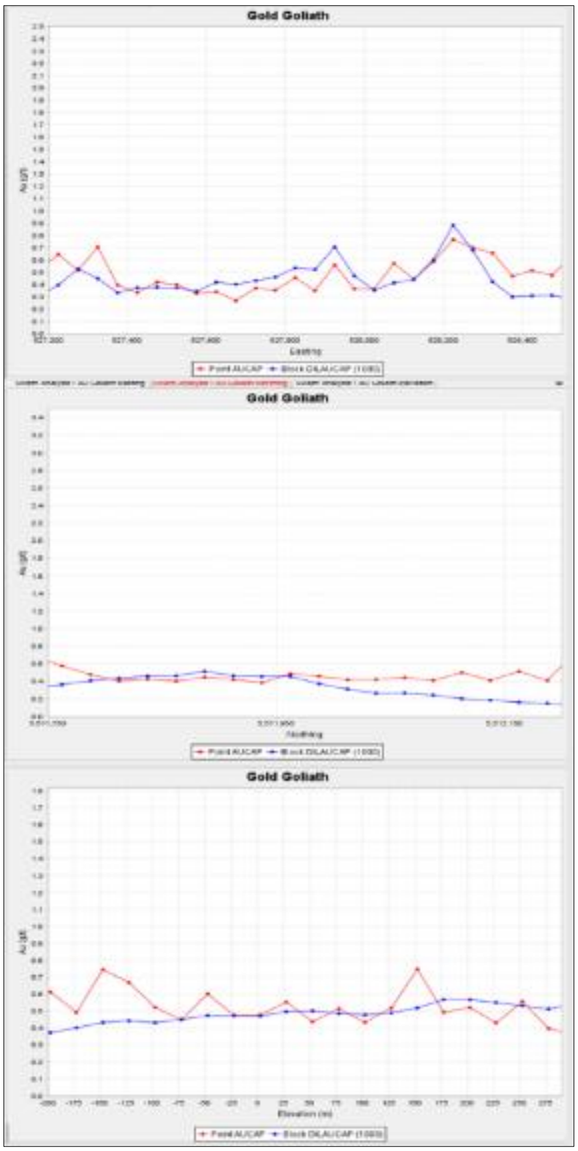


Note: Section is looking west and grid lines are 100 m apart. Source: SRK, 2023.

14.11.1.2 Local Comparison using Swath Plots

Comparison of the grade profiles (swath plots) of the composites and estimated grades allow for a visual verification of an over or under estimation of the block grades at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of swath plots, generated at 50 m intervals in the X and Y direction and 25 m vertically (Figure 14-12).

Figure 14-12: Swath Plot Comparing Estimated Gold Values and Composted Assays



Source: SRK

14.11.1.3 Comparison with Teck Underground Bulk Sample

Teck conducted an underground exploration and bulk sampling program in 1998. The result of the program was documented in a report by Stewart and Galway (1989). Teck collected the bulk samples in areas believed to exceed 3.0 g/t gold. On the west side of the ramp, three areas were included in the bulk sample. On the east side, three low-grade areas, one high-grade area, and one take-down-back were included in the bulk sample. Table 14-32 shows a summary of the bulk sample tonnage and grade recovered from Teck.

Table 14-32: Teck Bulk Sample Results

Drift & Rounds	Calculated Volumes (m^3)	Average SG	Calculated Tonnes	Face Sample Grade PMA Au (g/t)	Actual Measured Tonnes	Final Grade Au (g/t)
B-West	216	2.75	594	4.61	636	3.57
A-East LG	291	2.88	837	6.30	865	7.46
A-East HG	171	2.85	488	35.10	447	16.80
A-East TDB	111	3.00	333	23.00	388	12.70
Total	789	2.85	2,252	14.12	2,336	9.05

The excavated bulk sample areas were digitized and evaluated against the resource model. The PFS resource model reported 2,120 tonnes grading 8.47 g/t gold. The block model reported a bulk density of 2.76 while Teck used a density of 2.85 t/m^3. Increasing the density in the block model increases the tonnage of the bulk tonnage volume to 2,190 tonnes which is comparable to Teck's measured tonnage of 2,252 tonnes. The block model grade of 8.47 g/t is also very close to final grade calculated by Teck at 9.05 g/t.

14.11.1.4 Global Comparison with 2020 Resource Estimate

As a final comparison, the resource model was compared on a global basis with the 2021 mineral resource estimate prepared by AGP in March of 2021. Table 14-33 compares the current block model results with the AGP model with the current model. The reader is cautioned that the table is used only to compare the current block model estimate with the previous estimate for validation only and the numbers do not represent a mineral resource statement. As can be seen, the PFS model reports higher tonnage in the measured category reflecting the addition of infill drilling since 2021. On a global basis, the measured plus Indicated classes compare reasonably well with the previous estimate, reporting slightly higher tonnages at a slightly lower grade for slight increase of contained metal indicating that there are no major global biases with the PFS block model estimate.

Table 14-33: Global Comparison Between the PEA and PFS Resource Models

	Class	Tonnes	Au (g/t)	Au (Oz)	Ag (g/t)	Ag (Oz)
2021 PEA	Measured	1,569,000	2.09	105,300	7.57	382,000
	Indicated	29,548,000	1.07	1,020,100	3.39	3,224,100
	Measured + Indicated	31,116,000	1.13	1,128,700	3.61	3,608,000
	Inferred	4,348,000	1.02	142,200	2.69	375,900
2022 PFS	Measured	6,635,000	1.31	279,000	5.07	1,081,000
	Indicated	28,541,000	0.95	873,000	2.79	2,564,000
	Measured + Indicated	35,176,000	1.02	1,152,000	3.22	3,645,000
	Inferred	1,065,000	0.54	18,500	1.67	57,200

Note: Mineral resources are reported at a 0.25 g/t gold within the PEA optimized pit and at a 1.6 g/t cut-off below the pit shell.

14.11.2 Goldlund

The block model gold grade estimates (AUGPT) were validated using a series of statistical and graphical methods. These include a check of the global average using the NN model, a comparison with the ID2 and OK block grade estimates, a check of the global trends using swath plots, and visual validation in plan and section to confirm that the estimates honoured the composite grades, domain and zone boundary conditions and the kriging plan.

Table 14-34 shows summary statistics for the NNAU and AUGPT block grade estimates. Overall, the average block grade estimates are similar: 0.167 g/t Au for NNAU and 0.149 g/t for AUGPT. While there are differences for certain zones between NNAU and AUGPT average grades, the most important zones, Zone 10 and Zone 11, show good agreement.

Figure 14-13 displays grade-tonnes curves that compare the AUGPT block grade estimates and the NNAU and IDAU block grade estimates. Overall, the AUGPT estimates (shown in black) are less variable than the other two estimates. That is, for the range of likely mining cut-offs, the AUGPT block grade estimate predicts more tonnes at a lower grade than the other two methodologies.

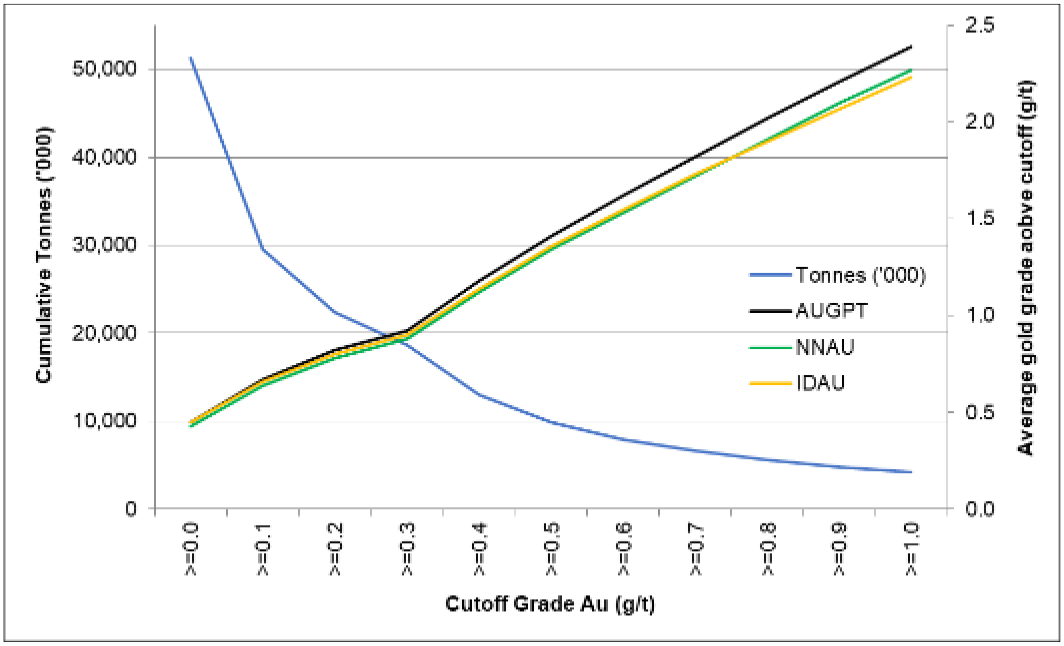
Table 14-34: Summary Statistics for Block Grade Estimates NNAU & AUGPT (Class 1 and 2)

Zone	No. of Blocks NNAU	Average NNAU	Std. Dev. NNAU	No. of Blocks AUGPT	Average AUGPT	Std. Dev. AUGPT
1	597,321	0.091	0.454	789,557	0.064	0.138
10	8,787	1.341	4.252	8,787	1.484	2.099
11	27,708	0.902	3.329	27,819	0.900	1.321
4	451,964	0.270	1.510	457,449	0.252	0.399
5	106,443	0.118	0.523	106,888	0.103	0.176
7	378,413	0.184	1.444	382,142	0.196	0.421
20	32,221	0.367	1.395	33,168	0.417	0.468
21	293,660	0.112	0.643	299,183	0.119	0.154
30	146,121	0.109	0.581	146,124	0.103	0.157
40	54,495	0.089	0.445	54,495	0.084	0.140
50	155,332	0.148	0.776	182,406	0.122	0.252
100	93,800	0.078	0.422	128,666	0.061	0.170
1 to 100	2,346,265	0.167	0.933	2,616,684	0.149	0.262

Swath plots were generated to determine if the block model gold grade estimates honoured the local trends in gold grade. A swath is the average of the NNAU, IDAU and AUGPT block grade estimates for collections of blocks. The swath width is 20 m or four blocks in easting, 20 m or four blocks in northing, and 10 m or two blocks in elevation. The average swath grade is then plotted versus the easting, northing, and elevation coordinates. There should be reasonable agreement between the trends of the two block grade estimates.

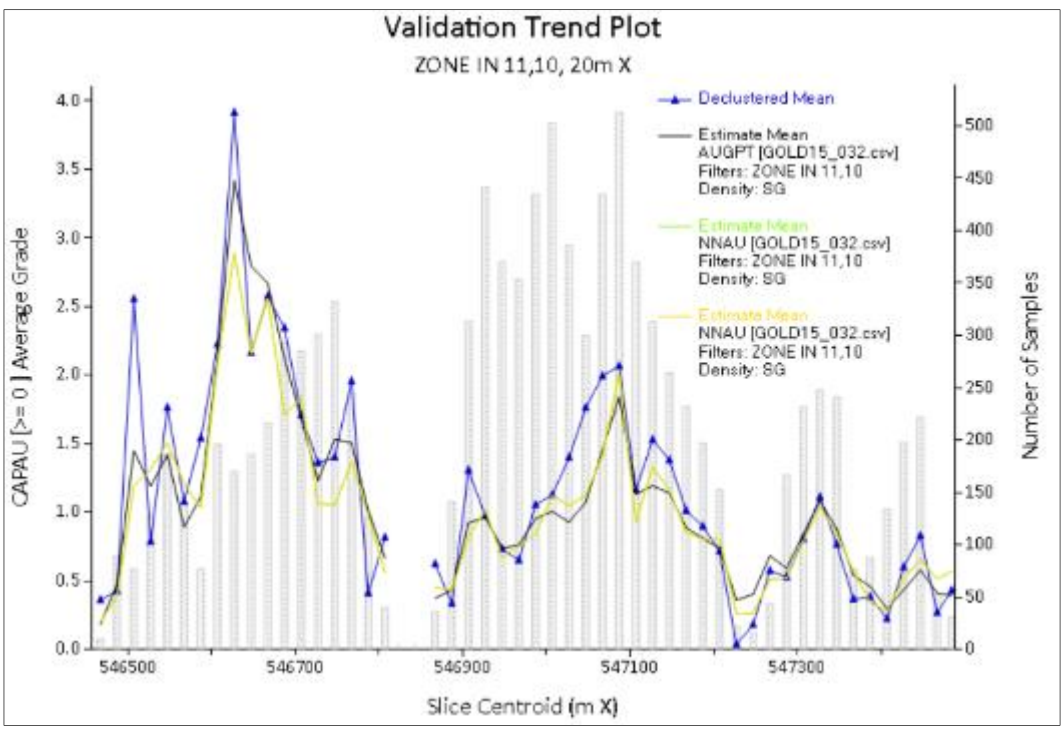
Figure 14-14, Figure 14-15, and Figure 14-16 display swath plots of the AUGPT, NNAU, and IDAU grade estimates within the combined Zone 1, 10 and 11 zones. For all three directions, there is acceptable agreement between the three block grade estimation results. That is, the AUGPT block grade estimates honour the gold grade trends as modelled by the NNAU and IDAU block grade estimates.

Figure 14-13: Grade-Tonnage Curve for AUGPT, NNAU &IDAU Block Grade Estimates in Zones 1, 10 & 11



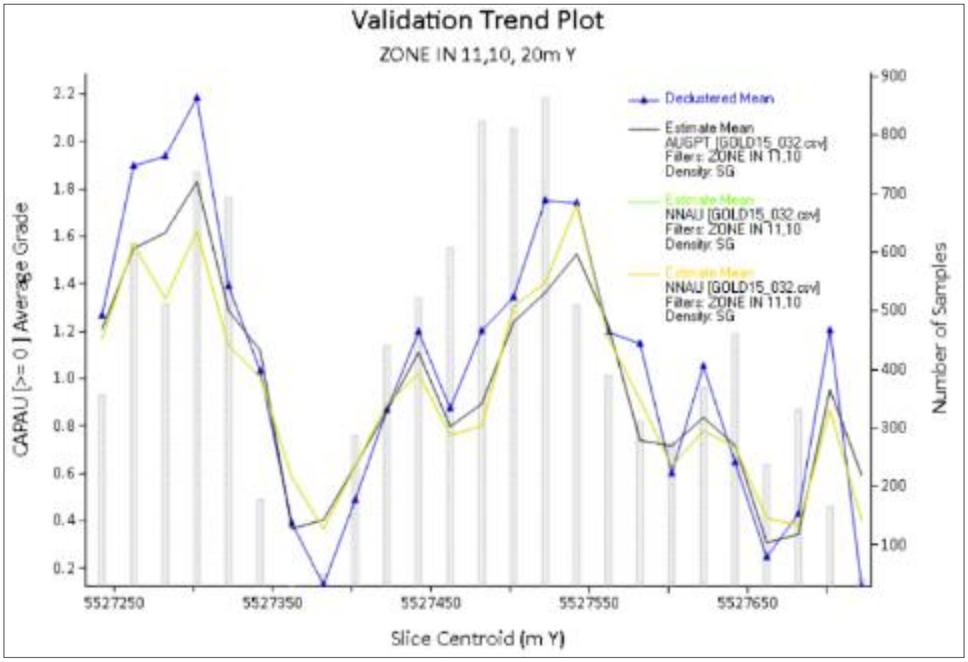
Source: SRK

Figure 14-14: Swath Plot in East Direction – AUGPT, IDAU & NNAU for Zones 1, 10 & 11



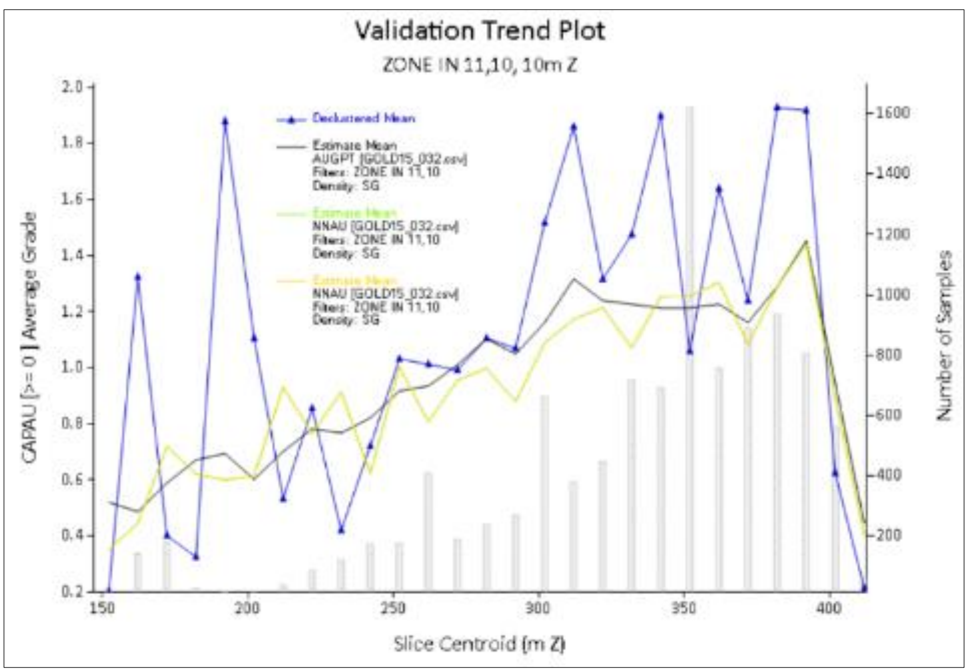
Source: SRK

Figure 14-15: Swath Plot in North Direction – AUGPT, IDAU & NNAU for Zones 1, 10 & 11



Source: SRK

Figure 14-16: Swath Plot in Vertical Direction – AUGPT, IDAU & NNAU for Zones 1, 10 & 11

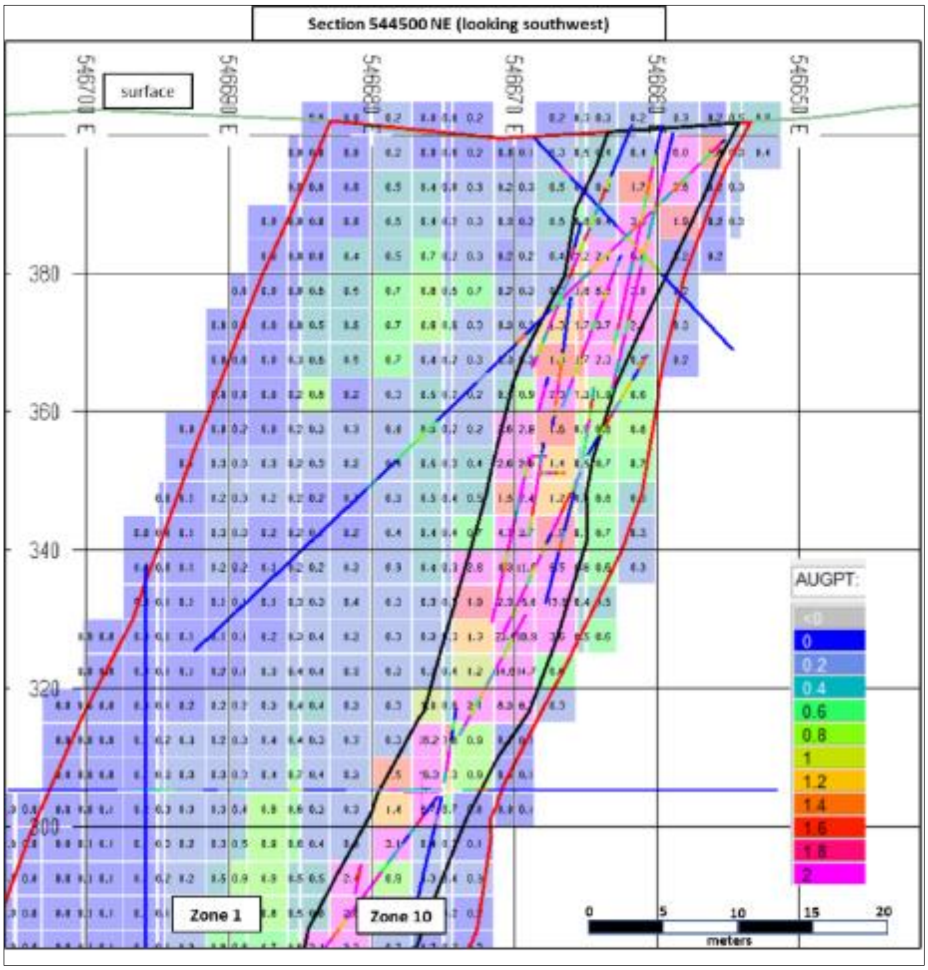


Source: SRK

Detailed visual inspection of the block grade estimates (AUGPT) were conducted in both plan and section, to ensure that the interpolation results honoured the geological boundaries and the drillhole data. This validation included confirmation of the proper coding of blocks for each of the mineralized zones and the distribution of block gold grade estimates relative to the 2 m drillhole composites, to ensure that the drillhole data were properly represented in the model.

Figure 14-17 displays a cross-section of the block gold grade estimates (AUGPT) for Zone 1 and 10 at 544500 NE, looking to the southwest. There appears to be good agreement between the 2 m composite gold grades and the estimated block model gold grade estimates. There is a marked break between the block model gold grade estimates between Zone 1 and Zone 10, which confirms the use of "hard" boundary requirements in the kriging plan.

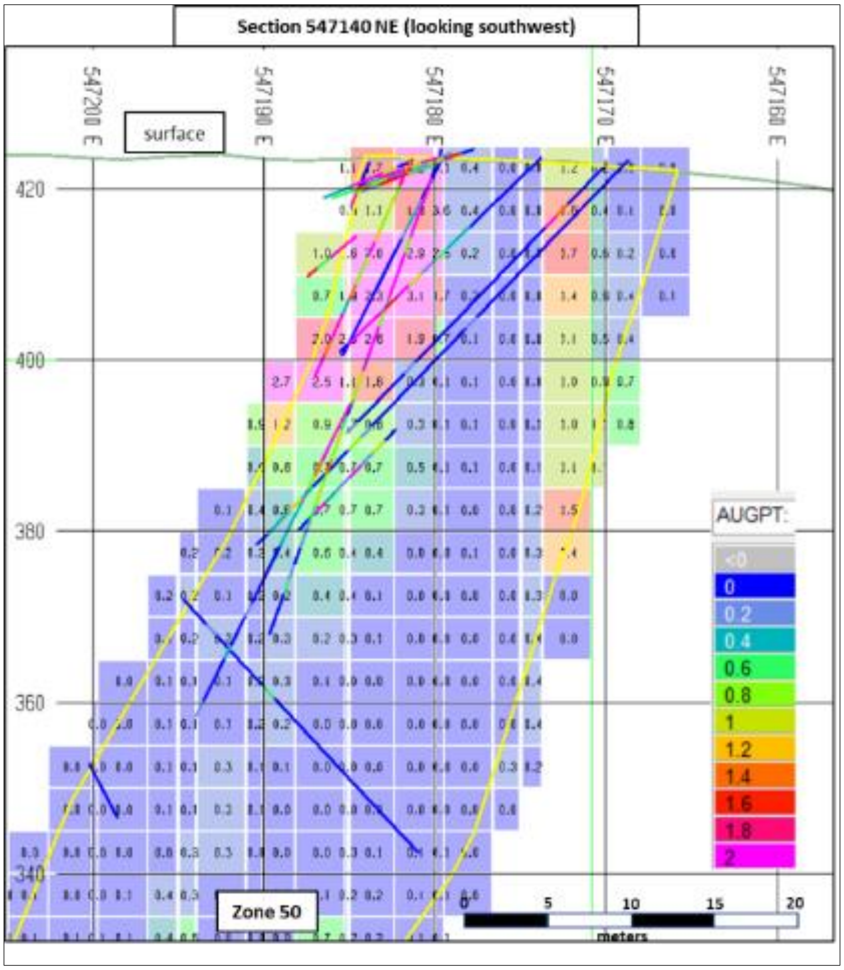
Figure 14-17: Section 544500 NE Showing Estimated Gold Grades and Drill Hole Composites (Zones 1 & 10)



Source: SRK

Figure 14-18 displays a cross-section of the block gold grade estimates (AUGPT) for Zone 50 at 547140 NE, looking to the southwest. There also appears to be good agreement between the 2 m composite gold grades and the estimated block model gold grade estimates for this zone.

Figure 14-18: Section 547140 NE Showing Estimated Gold Grades and Drill hole Composites (Zone 50)



Source: SRK

The results of the various validation statistical and graphical summaries show that the kriging plan and block model gold grade estimates are working as intended. Based on the validation results, the Qualified Person for this section of the report believes that the block model grade estimates (AUGPT) are suitable for the estimation of mineral resources at Goldlund.

14.11.3 Miller

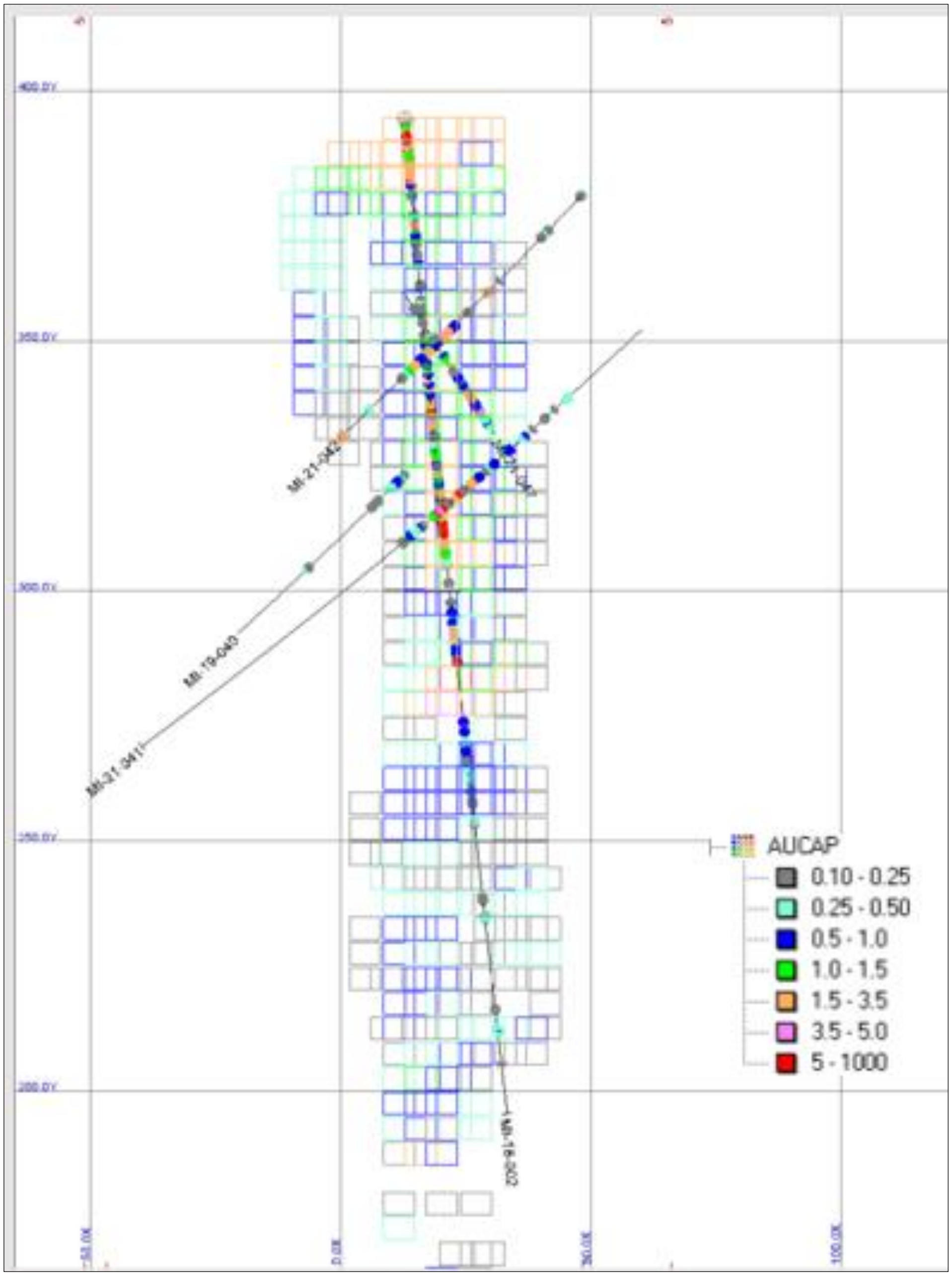
The Miller deposit grade model was validated by three methods:

- visual comparison of colour-coded block model grades with composite grades on sections and plans
- local comparison using swath plots
- comparison of the grades of blocks pierced by drillholes (well informed blocks) with the composites used to estimate the model.

14.11.3.1 Visual Comparison

The visual comparison of block model grades on sections and plans indicated a general good correlation between drillhole composited grades and the estimated resource model grades (Figure 14-19).

Figure 14-19: Section 775 E Comparing Estimated Gold Grades and Drill Hole Composite Grades



Note: Section is looking northeast and gridlines are 50 m apart. Source: SRK, 2023.

14.11.3.2 Local Comparison using Swath Plots

Comparison of the grade profiles (swath plots) of the composites and estimated grades allow for a visual verification of an over or under estimation of the block grades at the global and local scales. A qualitative assessment of the smoothing and variability of the estimates can also be observed from the plots. The output consists of swath plots, generated at 25 m intervals in the X, Y and Z directions. (Figure 14-20).

Figure 14-20: Swath Plot Comparing Estimated Gold Values and Composted Assays for Miller

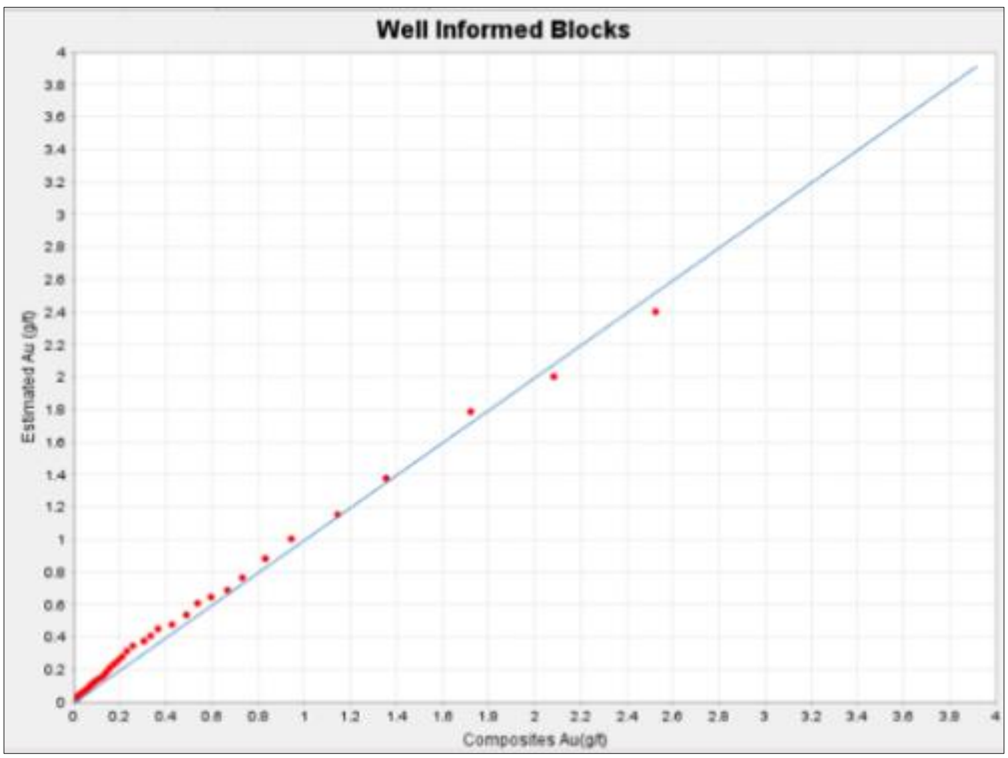
Source: SRK

On average, the estimated block grades agree well on a local basis with the composites being slightly higher and more erratic than the estimated values which show slight smoothing.

14.11.3.3 Well-Informed Bocks

A comparison of the grades of blocks pierced by drillholes with the average grades of the drillholes piercing the blocks can provide a local assessment of the general degree of smoothing and over or under estimation of the block grades with respect to the drillhole grades. Generally, interpolated grades will display a slight over estimation of lower grade values and a slight under estimation of higher-grade values when compared to the drill data. Figure 14-21 compares the estimated block grades for the Miller deposit against the composite grades piercing the blocks. As can be seen, the estimated block grades agree well with the composited data with only slight over estimation of the lower grades and moderate under estimation of the higher-grade values.

Figure 14-21: Comparison of Estimated Gold Grades with Composite Grades for Blocks Pierced by Drill Holes



Source: SRK

14.12 Classification of Mineral Resources

Mineral resources were estimated in conformity with generally accepted CIM "Estimation of Mineral Resource and Mineral Reserve Best Practices" Guidelines (2019). Mineral resources are not mineral reserves and do not have demonstrated economic viability.

Mineral resources are classified into three confidence categories: measured, indicated, and inferred. These terms are defined under the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014). Since each mineral resource estimate contains its own unique set of conditions, the selection of the criteria by which the mineral resource is assigned to each category relies on the judgement and experience of the Practitioners. Mineral resources for the Goliath, Goldlund and Miller deposits were classified by Dr. Gilles Arseneau, P. Geo. (APEGBC#23474) an "independent qualified person" as defined by NI 43-101.

Mineral resource classification is typically a subjective concept, industry best practices suggest that resource classification should consider both the confidence in the geological continuity of the mineralized structures, the quality and quantity of exploration data supporting the estimates and the geostatistical confidence in the tonnage and grade estimates. Appropriate classification criteria should aim at integrating both concepts to delineate regular areas at similar resource classification.

The QP is satisfied that the geological modelling for all deposits honours the current geological information and knowledge. The location of the samples and the assay data are sufficiently reliable to support resource evaluation.

The QP is unaware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, or other relevant issues that may currently affect the estimate of mineral resources presented in this report.

14.12.1 Goliath

The sampling information was acquired primarily by core drilling on sections spaced at approximately 12.5 to 35-meter spacing for the central portions of the Main and C zones and approximately 50 to 100 m spacing for the remainder of the Goliath deposit. Data for the Goliath East deposit is at approximately 35 to 75 m spacing for the central parts of the Goliath East zones and 75 to 120 m for the edges of the mineralized domains. At the current stage of drilling, the QP considers that the mineralization at Goliath satisfies the definition of measured, indicated and inferred mineral resource as defined by CIM and the wider spaced drilled Goliath East deposit satisfies the definition of inferred mineral resource.

The estimated blocks were classified according to the following:

- confidence in interpretation of the mineralized zones
- number of drillholes and composites used to estimate a block
- average distance to the composites used to estimate a block.

The classification parameters applied for the Goliath and Goliath East deposits are outlined in Table 14-35.

Table 14-35 : Goliath and Goliath East Classification Parameters

Class	Criteria
Measured	Estimation Pass 1 and average distance to the nearest composites < 25 m
Indicated	Pass 1 not Classified as Measured and Pass 2 with average distance < 120 m
Inferred	Pass 2 not classified as Indicated and Pass 3 with at least 2 DDH within 120 m

The classification was then evaluated in plan and section for consistency and was smoothed using a script to downgrade or upgrade block with inconsistent classification based on the surrounding blocks.

14.12.2 Goldlund

The classification parameters considered for the mineral resources at Goldlund are based on the proximity to and quantity of data used to estimate the block grade. The distance parameters used for the classification were based on a geostatistical method proposed by Davis (1997) that defines confidence limits using large sample normal theory. The confidence limit analysis considers the drillhole spacing, the variability of the data from the correlogram model and the planned production rate. For this study, measured material is considered known within ±15% 90% of the time for a quarterly production period, and indicated material is considered known within ±15% 90% of the time for an annual production period.

The methodology considers an idealized block representing a one-month production period, and a series of grids of different drillhole spacings are used to estimate the block to calculate the kriging variance. The nominal one-month production period is approximately a 110 m x 110 m x 5 m panel. The kriging variance needs to be adjusted by the square of the CV to obtain a relative variance as correlogram models were used to estimate the panel.

Based on the current drillhole spacing at Goldlund, the QP estimated that blocks could be classified as indicated if estimated with at least three drillholes within a 25 m spacing or by at least two drillholes with an average spacing of 25 m if one hole was within 12.5 m of the estimated block.

All other estimated blocks within the mineralized domains were classified as inferred mineral resource.

While there are some areas in Zone 1 that have sufficient drillhole data to support measured resources, these areas have been downgraded to indicated material due to the historical nature of the drillhole data. Therefore, there are only indicated and inferred mineral resources for Goldlund.

14.12.3 Miller

The sampling information was acquired primarily by core drilling on sections spaced at approximately 20 to 50-meter spacing for most of the main granodiorite body. At the current stage of drilling, the QP considers that the mineralization at the Miller deposit satisfies the definition of indicated and inferred mineral resource as defined by CIM.

The estimated blocks were classified according to the following:

- confidence in interpretation of the mineralized zones
- number of drillholes and composites used to estimate a block
- average distance to the composites used to estimate a block.

Based on the above, the QP classified all blocks estimated during pass one and two with at least two drillholes within a 50 m down dip and 25 m as indicated mineral resources and all blocks estimated during pass 3 with at least two drillholes were classified as inferred mineral resource.

14.13 Reasonable Prospects for Eventual Economic Extraction

CIM Definition Standards for Mineral Resources and Mineral Reserves (May 2014) defines a mineral resource as:

> *"A Mineral Resource is a concentration or occurrence of solid material of economic interest in or on the Earth's crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction. The location, quantity, grade or quality, continuity and other geological characteristics of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge, including sampling."*

The "material of economic interest" refers to diamonds, natural solid inorganic material, or natural solid fossilized organic material including base and precious metals, coal, and industrial minerals.

The "reasonable prospects for economic extraction" requirement generally implies that the quantity and grade estimates meet certain economic thresholds and that the mineral resources are reported at an appropriate cut-off grade taking into account extraction scenarios and processing recoveries. To meet this requirement, the QP evaluated the Goliath and Goldlund deposits as having potential for both open pit and underground mining operation and the Miller deposit as having potential for open pit extraction only.

14.13.1 Goliath

The parameters used to determine the quantities of material offering "reasonable prospects for eventual economic extraction" by open pit or underground mining methods for the Goliath deposits were derived from similar projects and from actual costs estimated as part of this study as outlined in Table 14-36.

Table 14-36: Parameters Used to Determine Reasonable Prospect of Economic Extraction for Goliath

Item	Value	Units
Currency Conversion	1.33	CAD/USD
Pit Slope	43.60	Degree
Gold Recovery	93.87	Percent
Silver Recovery	60.00	Percent
Open Pit Mining Cost	2.5	C$/t mined
Process & G&A	13.42	C$/t processed
Transport and Refining (Gold)	5.00	C$/oz
Transport and Refining (Silver)	0.26	C$/oz
Payable Gold	99.80	Percent
Payable Silver	97.00	Percent
Gold Price	1,700.00	US$/oz
Silver Price	23.00	US$/oz
Open Pit Cut-off	0.20	g/t Au
Underground Mining Costs	106.48	C$/t mined
Underground Cut-off	1.71	g/t Au

The reader is cautioned that these costs are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" by a potential open pit or underground mining operation and do not represent an attempt to estimate mineral reserves. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

The open pit parameters are then used to derive an optimized resource shell using Whittle™ pit optimizer. All blocks being captured by the optimized pit and above the cut-off grade value for open pit are considered part of the mineral resource. All blocks below the resource shell were evaluated with Deswik™ mining stope optimizer (MSO) and all blocks captured by the stope optimizer were classified as being part of the underground mineral resource. Blocks that were not inside the resource shell or within the MSO shapes were excluded from the mineral resource statement and tabulation.

14.13.2 Goldlund

The parameters used to determine the quantities of material offering "reasonable prospects for eventual economic extraction" by open pit or underground mining methods for the Goldlund deposit were derived from similar projects and from actual costs estimated as part of this study as outlined in Table 14-37.

Table 14-37: Parameters Used to Determine Reasonable Prospect of Economic Extraction for Goldlund

Item	Value	Units
Currency Conversion	1.33	CAD/USD
Pit Slope	48.00	Degree
Gold Recovery	90.30	Percent
Silver Recovery	NA	Percent
Open Pit Mining Cost	2.50	C$/t mined
Process & G&A	13.42	C$/t processed
Transport and Refining (Gold)	5.00	C$/oz
Transport and Refining (Silver)	NA	C$/oz
Payable Gold	99.80	Percent
Payable Silver	NA	Percent
Gold Price	1,700.00	US$/oz
Silver Price	23.00	US$/oz
Open Pit Cut-off	0.23	g/t Au
Underground Mining Costs	106.48	C$/t mined
Underground Cut-off	1.78	g/t Au

Note: NA = not applicable.

The reader is cautioned that these costs are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" by a potential open pit or underground mining operation and do not represent an attempt to estimate mineral reserves. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

The open pit parameters are then used to derive an optimized resource shell using Whittle™ pit optimizer. All blocks being captured by the optimized pit and above the cut-off value for open pit are considered part of the mineral resource. All blocks below the resource pit shell were evaluated with Deswik™ mining stope optimizer (MSO) and all blocks captured by the stope optimizer were classified as being part of the underground mineral resource. Blocks that were not inside the resource shell or within the MSO shapes were excluded from the mineral resource statement and tabulation.

14.13.3 Miller

The parameters used to determine the quantities of material offering "reasonable prospects for eventual economic extraction" by open pit or underground mining methods for the Miller deposit were derived from similar projects and from actual costs estimated as part of this study as outlined in Table 14-38.

Table 14-38: Parameters Used to Determine Reasonable Prospect of Economic Extraction for Goldlund

Item	Value	Units
Currency Conversion	1.33	CAD/USD
Pit Slope	48	Degree
Gold Recovery	93.8	Percent
Silver Recovery	0	Percent
Open Pit Mining Cost	2.5	C$/t mined
Process & G&A	13.42	C$/t processed
Transport and Refining (Gold)	5	C$/oz
Transport and Refining (Silver)	NA	C$/oz
Payable Gold	99.8	Percent
Payable Silver	NA	Percent
Gold Price	1700	US$/oz
Silver Price	23	US$/oz
Open Pit Cut-off	0.21	g/t Au
Underground Mining Costs	NA	C$/t mined
Underground Cut-off	NA	g/t Au

Note: NA = not applicable.

The reader is cautioned that these costs are used solely for the purpose of testing the "reasonable prospects for eventual economic extraction" by a potential open pit or underground mining operation and do not represent an attempt to estimate mineral reserves. The results are used as a guide to assist in the preparation of a mineral resource statement and to select an appropriate resource reporting cut-off grade.

The open pit parameters are then used to derive an optimized resource shell using Whittle™ pit optimizer. All blocks being captured by the optimized pit and above the cut-off value for open pit are considered part of the mineral resource. All blocks below the resource pit shell were excluded from the mineral resource statement and tabulation.

14.14 Mineral Resource Statement

Mineral resources for the Goliath Gold Complex are reported as being potentially extractable by open pit and underground operations. The mineral resources statements were prepared by Dr. Gilles Arseneau, P. Geo., associate consultant with

SRK. The mineral resources are based on a total of 2,899 drillholes measuring 551,916 meters for the Goliath, Goldlund and Miller deposits, incorporating 176 drillholes and 41,072 meters from the 2021 drilling campaign.

The mineral resources are prepared in accordance with NI 43-101 and the "CIM Definition Standards for Mineral Resources and Mineral Reserves" (2014) and the "CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (2019). The estimated mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The effective data of the mineral resource statement is January 17, 2022.

14.14.1 Goliath

The Goliath open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.25 g/t Au that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of $93.873*Au(g/t)^{0.021}$ and 60% respectively.

The underground mineral resources are reported inside shapes generated from Deswik™ Mining Stope Optimizer (MSO) at a cut-off grade of 2.2 g/t Au that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of $93.873*\text{ Au }(g/t)^{0.021}$ and 60%, respectively. Table 14-39 summarizes the mineral resources for the Goliath and Goliath East deposits.

Table 14-39: Mineral Resource Statement, Goliath and Goliath East, SRK, Effective on January 17, 2022

Type	Classification	Cut-off Grade	Tonnes	Au (g/t)	Au (Oz)	Ag (g/t)	Ag (oz)
Open Pit	Measured	0.25	6,223,000	1.20	239,500	4.70	940,600
	Indicated	0.25	23,081,000	0.75	559,400	2.53	1,878,500
	Measured + Indicated	0.25	29,304,000	0.85	798,900	2.99	2,819,100
	Inferred	0.25	3,330,000	0.66	70,200	0.80	85,200
Underground	Measured	2.20	170,000	6.24	34,100	22.34	122,100
	Indicated	2.20	2,550,000	3.55	291,000	7.08	580,800
	Measured + Indicated	2.20	2,720,000	3.72	325,100	8.04	702,900
	Inferred	2.20	48,000	2.95	4,600	4.06	6,300
Total	Measured		6,393,000	1.33	273,600	5.17	1,062,700
	Indicated		25,631,000	1.03	850,400	2.98	2,459,300
	Measured + Indicated		32,024,000	1.09	1,124,000	3.42	3,522,000
	Inferred		3,378,000	0.69	74,800	0.84	91,500

Notes: 1. Mineral Resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc., Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral resource effective date January 17, 2022. 3. Goliath open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.25 g/t Au that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60%, respectively. 4. Goliath underground mineral resources are reported inside shapes generated from Deswik Mining Stope Optimizer (DSO) at a cut-off grade of 2.2 g/t Au that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60%, respectively. 5. Gold and silver assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.5 m for Goliath. 6. Excludes unclassified mineralization located within mined-out areas. 7. Silver grade and ounces are derived from the Goliath tonnage only. 8. Goliath open pit cut-off grade is 0.25 g/t. 9. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

14.14.2 Goldlund

The Goldlund open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344x Au (g/t)$^{0.0527}$.

Underground mineral resources are reported inside MSO shapes at a cut-off grade of 2.2 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344x Au (g/t)$^{0.0527.}$ Table 14-40 summarizes the mineral resources for the Goldlund deposit.

Table 14-40: Mineral Resource Statement, Goldlund Deposit, SRK, Effective on January 17, 2022

Type	Classification	Cut-off Grade	Tonnes	Au (g/t)	Au (oz)
Open Pit	Measured	0.30	0	0.00	0
	Indicated	0.30	33,353,000	0.85	911,000
	Measured +Indicated	0.30	33,353,000	0.85	911,000
	Inferred	0.30	28,833,000	0.73	680,200
Underground	Measured	2.20	0	0.00	0
	Indicated	2.20	222,000	4.06	29,000
	Measured +Indicated	2.20	222,000	4.06	29,000
	Inferred	2.20	222,000	3.26	23,300
Total	Measured		0	0.00	0
	Indicated		33,575,000	0.87	940,000
	Measured +Indicated		33,575,000	0.87	940,000
	Inferred		29,055,000	0.75	703,500

Notes: 1. Mineral resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc. Mineral resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral resource effective date January 17, 2022. 3. Goldlund open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344x Au (g/t)^0.0527. 4. Goldlund underground mineral resources are reported inside DSO shapes at a cut-off grade of 2.2 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344x Au (g/t)^0.0527. 5. Gold assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 2.0 m for Goldlund. 6. Excludes unclassified mineralization located within mined-out areas. 7. Goldlund open pit cut-off grade is 0.30 g/t. 8. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

14.14.3 Miller

The Miller open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 93.873* Au (g/t)$^{0.021}$. Table 14-41 summarizes the mineral resources for the Goldlund deposit.

Table 14-41: Mineral Resource Statement, Miller Deposit, SRK, Effective on January 17, 2022

Type	Classification	Cut-off Grade	Tonnes	Au (g/t)	Au (oz)
Open Pit	Measured	0.30	0	0	0
	Indicated	0.30	2,112,000	1.10	74,600
	Measured +Indicated	0.30	2,112,000	1.10	74,600
	Inferred	0.30	138,000	1.01	4,500

Notes: **1.** Mineral resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc. Mineral resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. **2.** Mineral resource effective date January 17, 2022. **3.** Miller open pit mineral resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t Au that is based on a gold price of US$1,700/oz and a gold processing recovery of 93.873*Au(g/t)^0.021. **4.** Gold assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.0 m for Miller. **5.** Miller cut-off grade is 0.30 g/t. **6.** All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

14.15 Sensitivity to Cut-off Grade

14.15.1 Goliath

The mineral resources at Goliath are susceptible to cut-off grade selection. To highlight this sensitivity, the mineral resources are presented at various cut-off for the in-pit and below pit in Tables 14-42 and 14-43 for the Goliath deposits. It should be noted that the inferred mineral resources combine the Goliath and Goliath East deposits. The underground resource is not as susceptible to cut-off grade selection as the in-pit mineral resource. The underground resource does not vary much between 1.0 and 2.5 g/t Au cut-off grade.

14.15.2 Goldlund

The mineral resources at Goldlund are also susceptible to cut-off grade selection. To highlight this sensitivity, the mineral resources are presented at various cut-off grades for the in-pit and below pit in Tables 14-44 and 14-45 for the Goldlund deposit.

14.15.3 Miller

The mineral resources at Miller are also susceptible to cut-off grade selection. To highlight this sensitivity, the mineral resources are presented at various cut-off grades for the in-pit in Tables 14-42 and 14-46 for the Miller deposit. The cut-off grade used for the mineral resources are highlighted in each of the tables.

Table 14-42: Sensitivity of Open Pit Resource to Cut-off Grade Selection for Goliath Deposits

Category	Cut-off Grade (g/t Au)	Tonnes	Au (g/t)	Gold (oz)	Ag (g/t)	Silver (oz)
Measured	>2.0	909,000	4.53	132,200	13.36	390,200
	>1.5	1,223,000	3.81	149,800	11.64	457,700
	>1.0	1,799,000	2.98	172,300	9.59	554,900
	>0.9	1,987,000	2.79	178,000	9.09	580,800
	>0.8	2,211,000	2.59	184,100	8.54	607,300
	>0.7	2,489,000	2.38	190,800	7.96	636,700
	>06	2,824,000	2.18	197,800	7.39	671,000
	>0.5	3,321,000	1.93	206,600	6.70	715,700
	>0.4	4,122,000	1.65	218,100	5.92	784,500
	>0.3	5,397,000	1.34	232,200	5.09	883,200
	>0.25	6,223,000	1.20	239,500	4.70	940,600
	>0.2	7,092,000	1.08	245,800	4.37	997,500
Indicated	>2.0	1,290,000	3.47	144,100	6.26	259,800
	>1.5	2,286,000	2.71	199,000	5.38	395,400
	>1.0	4,540,000	1.97	287,300	4.49	655,100
	>0.9	5,243,000	1.83	308,700	4.29	722,900
	>0.8	6,067,000	1.70	331,200	4.10	799,300
	>0.7	7,103,000	1.56	356,100	3.89	888,600
	>06	8,441,000	1.41	384,000	3.66	993,100
	>0.5	10,369,000	1.25	417,800	3.40	1,132,700
	>0.4	13,452,000	1.07	462,000	3.09	1,335,000
	>0.3	18,966,000	0.86	523,200	2.72	1,656,700
	>0.25	23,081,000	0.75	559,400	2.53	1,878,500
	>0.2	28,168,000	0.66	596,100	2.36	2,137,700
Inferred	>2.0	101,000	2.54	8,300	0.75	2,400
	>1.5	244,000	2.07	16,300	0.79	6,200
	>1.0	583,000	1.54	28,900	0.71	13,200
	>0.9	725,000	1.43	33,300	0.74	17,200
	>0.8	836,000	1.35	36,300	0.73	19,700
	>0.7	986,000	1.26	39,900	0.74	23,400
	>06	1,185,000	1.16	44,000	0.73	28,000
	>0.5	1,477,000	1.04	49,200	0.73	34,400
	>0.4	2,003,000	0.88	56,700	0.71	45,600
	>0.3	2,785,000	0.73	65,500	0.75	67,300
	>0.25	3,330,000	0.66	70,300	0.80	85,100
	>0.2	4,095,000	0.58	75,700	0.83	109,000

Table 14-43: Sensitivity of Underground Resource to Cut-off Grade Selection for Goliath Deposits

Category	Cut-off Grade (g/t Au)	Tonnes	Au (g/t)	Gold (oz)	Ag (g/t)	Silver oz)
Measured	>10.0	25,000	13.42	10,600	33.18	26,300
	>5.0	92,000	8.62	25,400	27.00	79,700
	>2.5	156,000	6.59	33,100	22.98	115,300
	>2.2	170,000	6.24	34,100	22.34	122,100
	>2.0	181,000	5.99	34,800	21.84	127,100
	>1.8	193,000	5.73	35,600	21.28	132,300
	>1.6	203,000	5.53	36,100	20.80	136,000
	>1.5	207,000	5.46	36,300	20.64	137,100
	>1.0	214,000	5.33	36,600	20.29	139,300
Indicated	>10.0	36,000	12.81	14,800	7.53	8,700
	>5.0	252,000	7.38	59,900	8.56	69,500
	>2.5	2,060,000	3.83	254,000	7.42	491,200
	>2.2	2,550,000	3.55	291,000	7.08	580,800
	>2.0	2,851,000	3.40	311,400	6.93	635,300
	>1.8	3,129,000	3.26	328,300	6.81	685,400
	>1.6	3,344,000	3.16	340,100	6.71	721,900
	>1.5	3,427,000	3.12	344,200	6.67	735,000
	>1.0	3,669,000	3.00	354,200	6.52	768,700
Inferred	>10.0	0	12.52	100	8.96	100
	>5.0	2,000	7.20	500	11.21	700
	>2.5	25,000	3.53	2,800	5.72	4,600
	>2.2	48,000	2.95	4,600	4.06	6,300
	>2.0	58,000	2.81	5,300	3.88	7,300
	>1.8	69,000	2.66	5,900	3.67	8,100
	>1.6	76,000	2.58	6,300	3.62	8,800
	>1.5	78,000	2.55	6,400	3.64	9,100
	>1.0	88,000	2.41	6,800	3.60	10,200

Table 14-44: Sensitivity of Open Pit Resource to Cut-off Grade Selection for Goldlund Deposit

Category	Cut-off Grade (g/t Au)	Tonnes	Au (g/t)	Gold (oz)
Indicated	>2.0	2,060,000	3.68	243,400
	>1.8	2,534,000	3.34	272,300
	>1.6	3,136,000	3.03	305,100
	>1.5	3,527,000	2.86	324,600
	>1.0	7,009,000	2.04	459,700
	>0.9	8,296,000	1.87	499,000
	>0.8	10,046,000	1.69	546,600
	>0.7	12,348,000	1.52	602,000
	>0.6	15,489,000	1.34	667,400
	>0.5	19,594,000	1.17	739,500
	>0.4	25,261,000	1.01	820,900
	>0.3	33,353,000	0.85	911,000
	>0.25	38,706,000	0.77	958,100
	>0.2	45,218,000	0.69	1,005,000
Inferred	>2.0	662,000	2.83	60,200
	>1.8	932,000	2.56	76,600
	>1.6	1,410,000	2.27	102,800
	>1.5	1,790,000	2.11	121,700
	>1.0	5,409,000	1.50	261,700
	>0.9	6,858,000	1.39	305,800
	>0.8	8,716,000	1.27	356,500
	>0.7	10,960,000	1.17	410,600
	>0.6	13,903,000	1.06	471,800
	>0.5	17,956,000	0.94	542,800
	>0.4	22,850,000	0.83	613,300
	>0.3	28,833,000	0.73	680,200
	>0.25	32,137,000	0.69	709,300
	>0.2	35,569,000	0.64	734,100

Table 14-45: Sensitivity of Underground Resource to Cut-off Grade Selection for Goldlund Deposit

Category	Cut-off Grade (g/t Au)	Tonnes	Au (g/t)	Gold (oz)
Indicated	>10.0	12,000	15.64	6,000
	>5.0	30,000	10.47	10,100
	>2.5	182,000	4.44	26,000
	>2.2	222,000	4.06	29,000
	>2.0	246,000	3.87	30,600
	>1.8	300,000	3.51	33,900
	>1.6	335,000	3.32	35,800
	>1.5	346,000	3.27	36,400
	>1.0	394,000	3.03	38,300
Inferred	>10.0	0	0.00	0
	>5.0	0	0.00	0
	>2.5	193,000	3.40	21,100
	>2.2	222,000	3.26	23,300
	>2.0	267,000	3.07	26,400
	>1.8	304,000	2.93	28,600
	>1.6	329,000	2.83	30,000
	>1.5	332,000	2.82	30,100
	>1.0	359,000	2.71	31,200

Table 14-46: Sensitivity of Open Pit Resource to Cut-off Grade Selection for Miller Deposit

Category	Cut-off Grade (g/t Au)	Tonnes	Au (g/t)	Gold (oz)
Indicated	>2.0	263,000	3.29	27,800
	>1.8	316,000	3.06	31,000
	>1.6	397,000	2.78	35,400
	>1.5	440,000	2.66	37,600
	>1.0	753,000	2.06	49,800
	>0.9	845,000	1.94	52,700
	>0.8	953,000	1.81	55,600
	>0.7	1,107,000	1.67	59,300
	>0.6	1,286,000	1.52	63,000
	>0.5	1,510,000	1.38	66,900
	>0.4	1,794,000	1.23	71,100
	>0.3	2,112,000	1.10	74,600
	>0.25	2,302,000	1.03	76,300
	>0.2	2,503,000	0.97	77,800
Inferred	>2.0	13,000	3.83	1,700
	>1.8	14,000	3.71	1,700
	>1.6	16,000	3.47	1,800
	>1.5	17,000	3.36	1,900
	>1.0	36,000	2.21	2,600
	>0.9	45,000	1.97	2,800
	>0.8	56,000	1.74	3,100
	>0.7	70,000	1.55	3,500
	>0.6	80,000	1.43	3,700
	>0.5	94,000	1.30	3,900
	>0.4	112,000	1.17	4,200
	>0.3	138,000	1.01	4,500
	>0.25	151,000	0.95	4,600
	>0.2	163,000	0.89	4,700

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The QPs accepting the professional responsibility for the open pit and underground mineral reserve estimates section are Ms. Colleen MacDougall, P.Eng. (PEO#100530936) and Mr. Sean Kautzman, P.Eng. (PEO#100159892), respectively. Ms. MacDougall undertook open pit mine planning work supporting the preparation of the mineral reserve statement for the Goliath, Goldlund and Miller open pits. Mr. Kautzman undertook the underground mine planning work supporting the preparation of the mineral reserve statement for the Goliath underground project. Mineral reserves are derived from measured and indicated mineral resources after applying economic parameters and other modifying factors following with the "CIM Definition Standards for Mineral Resources & Mineral Reserves (May 10, 2014) and the "CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (Nov 29, 2019). Inferred mineral resources were treated as waste in the life-of-mine plan. Mineral Reserves are classified using the following criteria:

- Proven mineral reserves are the measured mineral resources where development work for mining and information on processing, metallurgy and other relevant factors demonstrate that economic extraction is achievable. A proven mineral reserve implies a high degree of confidence in the modifying factors.
- Probable mineral reserves are those measured and indicated mineral resources where development work for mining and information on processing/metallurgy and other relevant factors demonstrate that economic extraction is achievable. The confidence in the modifying factors applying to a probable mineral reserve is lower than that applying to a proven mineral reserve.

The mineral reserves for the Goliath Complex consist of open pit mineral reserves at Goliath, Goldlund and Miller, and underground mineral reserves at Goliath, with an effective date of December 31, 2022 are founded on and included within the mineral resource estimates with an effective date of January 17, 2022. The reference point at which the mineral reserve is identified is where the ore is delivered to the processing plant referred to as plant feed.

Project base case economic analysis presented in Section 22 shows that the LOM plan founded on the mineral reserve estimates in Table 15-1 provides a positive present value of the net cash flow, confirming that the mineral reserves are economically viable, and that economic extraction can be justified.

The QPs are not aware of any additional mining, metallurgical, infrastructure, permitting, or other factors not presented in this report that could materially affect the mineral reserve estimate.

15.2 Estimation Procedure

15.2.1 Open Pit Mineral Reserves

The open pit mineral reserves were estimated by Ms. Colleen MacDougall using the following methodology:

- reviewed geological information and resource block model, and estimation of the mining modifying factors
- reviewed commodity price consensus forecasts

- reviewed proposed operating costs and processing parameters for input into the pit optimization process
- reviewed geotechnical slope parameters
- undertook pit optimizations and the selection of optimal pit shells for the basis of the pit designs
- engineered ultimate and phased pit designs for each deposit
- estimated mineral reserves from the ultimate pit inventory
- generated mine sequence and production schedule
- estimated open pit mining costs for the life of mine
- prepared the mineral reserve statement.

The open pit mineral reserves are based on and are part of the mineral resources presented in Section 14 of this report. The open pit mineral reserves are reported based on open pit mining within the engineered pit designs presented in Section 16.4.3.2 of this report. The effective date of the mineral reserves is December 31, 2022.

The following methodology was used to convert mineral resources within the open pit area to mineral reserves:

- The Goldlund and Miller partial block resource models were converted to full block models (5 x 5 x 5 m). The Goliath partial resource block model was converted to a full block model and then regularized to 5 x 4 x 5 m. Mining dilution of 10%, 7%, and 9% were applied to Goliath, Goldlund and Miller, respectively at a diluting grade of 0 g/t Au and Ag. Ore loss of 1% was applied to Goliath and Miller, with no loss applied to Goldlund.
- Cut-off values of C$15.22/t, C$16.00/t and C$23.63/t were applied to Goliath, Goldlund and Miller, respectively for reserve estimation. These values are based on the following:
    - long-term gold price of US$1,550/oz Au, silver price of US$22/oz Ag and exchange rate of C$1.30/US$1.00
    - transportation costs of C$5/oz Au
    - payabilities of 99% Au and 97% Ag
    - variable gold processing recoveries, derived with Equation 1 to 3 in Section 16.4.2.1 of this report, averaged over the life of mine: 94.2% for Goliath, 94.3% for Goldlund, and 94.0% for Miller. A silver recovery of 60% was assumed for Goliath.

Further details on dilution, mining loss, and cut-off grade estimation are presented in Section 16.4.2 of this report. The mineral reserves are located within designed pits at an average waste-to-ore strip ratio of 3.1:1.

15.2.2 Underground Mineral Reserves

The underground mineral reserves were estimated by Mr. Sean Kautzman, PEng, using the following methodology:

- reviewed geological information and resource block model, and estimation of the mining modifying factors

- reviewed commodity price consensus forecasts
- reviewed proposed operating costs and processing parameters for input into the stope optimization process
- reviewed the geotechnical parameters pertaining to stope dimensions, stope sequencing, capital development dimensions and siting (e.g., physical offsets from other infrastructure items), and the crown pillar offset under the Goliath open pit
- undertook stope optimizations
- estimated mineral reserves from the ultimate underground inventory
- assessed the economics of specific stoping areas to identify and remove stopes incapable of supporting the development required to access them
- generated mine sequence and production schedule
- estimated underground mining costs for the life of mine
- prepared the mineral reserve statement.

The Goliath underground mineral reserves are based on and are part of the mineral resources presented in Section 14 of this report. The Goliath underground mineral reserves are reported based on underground mining within the engineering design presented in Section 16.5 of this report. The effective date of the mineral reserves is December 31, 2022.

The following methodology was used to convert mineral resources within the Goliath underground area to mineral reserves:

- The resource model was already in a sub-celled format, so no further conversion or formatting of the model was required. Mining dilution of 15% and 5% was applied to stope and development shapes, respectively, at diluting grades of 0 g/t Au and Ag. Ore recovery values of 90% and 80% were applied to downhole and uphole stopes, respectively, with no loss applied to development activities.
- A mill feed cut-off value of C$107.66/t was applied to Goliath for reserve estimation; this equates to approximately $124/t based on the estimated 15% stope dilution. This value uses the same assumptions for long-term gold price, silver price, transportation costs, payabilities, and gold processing recovery as the reserve estimation for the open pits (as described in Section 15.2.1). An exchange rate of C$1.33/US$1.00 was utilized in the underground reserve estimation. The QP has reviewed the results of using the C$1.30/US$1.00 exchange rate and concluded that the variance is not material.

Further details on dilution, mining loss, and cut-off grade estimation are presented in Section 16.5.2 of this report.

15.3 Mineral Reserves Statement

The mineral reserve estimate is presented in Table 15-1.

Table 15-1: Mineral Reserves Estimate

Classification	Quantity (kt)	Au (g/t)	Au (koz)	Ag (g/t)	Ag (koz)
Open Pit – Goliath					
Proven	3,969	1.05	134	3.22	410
Probable	5,580	0.67	119	2.20	395
Proven & Probable	9,549	0.83	254	2.62	805
Open Pit – Goldlund					
Proven	-	-	-	-	-
Probable	16,256	1.19	621	-	-
Proven & Probable	16,256	1.19	621	-	-
Open Pit – Miller					
Proven	-	-	-	-	-
Probable	738	1.03	24	-	-
Proven & Probable	738	1.03	24	-	-
Underground – Goliath					
Proven	596	3.96	76	16.73	321
Probable	3,180	2.85	292	5.85	598
Proven & Probable	3,776	3.03	368	7.56	918
Total					
Proven	4,565	1.43	210	4.98	731
Probable	25,754	1.28	1,057	1.20	993
Proven & Probable	30,319	1.30	1,267	1.77	1,724

Notes: **1.** Mineral reserves with an effective date of December 31,2022, are founded on and included within the mineral resource estimates, with an effective date of January 17, 2022. **2.** Mineral reserves were developed in accordance with CIM Definition Standards (2014). **3.** Open pit mineral reserves incorporate 10%, 7% and 9% dilution for Goliath, Goldlund and Miller, respectively. Open pit mineral reserves include 1% loss for Goliath and Miller, no losses are included for Goldlund. Goliath underground mineral reserves include 5% dilution and 0% loss for development. For stopes at Goliath underground, the mineral reserves include 15% dilution (both downhole and uphole stopes) and 90% (downhole) and 80% (uphole) recovery. **4.** Open pit mineral reserves are reported based on open pit mining within designed pits above cut-off values of C$15.22/t, C$16.00/t and C$23.63/t for Goliath, Goldlund and Miller, respectively. Goliath underground mineral reserves are reported based on underground mining within designed underground stopes above a mill feed cut-off value of C$107.66/t (inclusive of 15% mining dilution). The cut-off values are based on a gold price of US$1,550/oz Au, a silver price of US$22, transportation costs of C$5/oz Au, payabilities of 99% Au and 97% Ag, LOM average gold recoveries of 94.2% for Goliath, 94.3% for Goldlund and 94.0% for Miller, and a silver recovery of 60% for Goliath. **5.** Underground mineral reserves following Year 13 have been removed from the LOM plan and thus are excluded in the mineral reserve table above. Some low grade Goldlund material above cut-off is not fed to the plant and therefore not included in the mineral reserves. **6.** The Qualified Person for the open pit mineral reserve estimate is Colleen MacDougall, PEng; and the Qualified Person for the underground mineral reserve estimate is Sean Kautzman, PEng, both are employees of SRK Consulting (Canada) Inc. **7.** Rounding may result in apparent summation differences between tonnes, grade and contained metal.

15.4 Factors that May Affect the Mineral Reserves

Project risks and opportunities are summarized in Section 25.15 of this report.

The QPs are not aware of mining, metallurgical, infrastructure, permitting, or other factors not presented in this report that could materially affect the mineral reserve estimate.

16 MINING METHODS

16.1 Introduction

The open pit and underground mines conceived as part of the Goliath Gold Complex will be developed using conventional, modern methodologies and technologies, with 13 years of planned mill feed. Summaries of the life-of-mine designs and schedules for each operation, along with data and assumptions used to support them, are presented below.

16.2 Geotechnical Considerations

The following sub-sections describe geotechnical site characterization. Geotechnical design constraints have been defined by (MDEng 2014, RockEng 2021, RockEng 2022a, RockEng 2022b) for the underground and open pit mine plans at Goliath, Goldlund and Miller.

16.2.1 Goliath

The Goliath geotechnical site characterization has incorporated geological and geotechnical drill core logs and laboratory strength testing data. Review and analyses of the available data has led to the following conclusions:

Joint-scale structural trends are listed below and agree well with project structural geology interpretations:

- foliation parallel jointing (dips to the SSE at about 74°)
- a sub-horizontal jointing
- steep north-dipping set (may be bedding plane parallel, occurs sub-parallel to foliation)
- steep northeast-dipping set (common trend with brittle faults, intensity likely to vary by proximity to faulting)
- steep west-dipping set (common trend with brittle faults, intensity likely to vary by proximity to faulting).

Intact strength and rock mass quality varies modestly by lithological domains:

- BMS: Q'=15.9 to 24.6 (good), UCS = 67 MPa
- MSS: Q'=11.7 to 16.3 (good), UCS = 76 MPa
- MSED (MS & CSZ): Q'=7.8 to 16.4 (fair to good), UCS = 71 MPa
- QP/QFP: Q'=6.1 to 11 (fair to good), UCS = 85 MPa.

Table 16-1: Summary of Joint Sets Identified from Oriented Geotechnical Boreholes

Joint Set	All Data			Hangingwall	
		BMS	MSS	MSED	QFP
Foliation	**74/165 (14)**	**74/164 (14)**	**74/167 (14)**	**71/168 (14)**	73/176 (13)
J1	**4/315 (18)**	**2/115 (17)**	**12/290 (15)**	11/009 (15)	3/326 (17)
J2				**80/007 (16)**	
J3					**81/310 (15)**
J4					**83/266 (14)**

Note: Orientations listed are recorded in dip and dip direction format with the first standard deviation of the variability limit quoted in brackets. Bold font indicates major features and italic font indicates minor features. Source: MDEng (2014).

16.2.2 Goldlund

The Goldlund geotechnical site characterization has incorporated geological and geotechnical drill core logs, televiewer data and laboratory strength testing data. Detailed review and analyses of all available data has led to the following conclusions:

- Broad trends in joint orientation are as follows (see Table 16-2):
  - sub-vertical northeast-southwest-striking joint trend which is parallel to regional foliation trend
  - sub-horizontal joint trend
  - north-south (to NNW-SSE)-striking joint trend that is sub-vertical and follows the general orientation of north-south-striking veins (this trend locally rotates within the lithology domains)
  - three global joint trends appear in all rock types, and there is no considerable spatial change in the structural regime; the dominance of each trend varies by rock type.
- The intact rock strength is very strong to extremely strong for all geological units with average UCS values as follows:
  - andesite: 154 MPa
  - granodiorite: 308 MPa
  - basalt: 200 MPa
  - mafic volcanic: 146 MPa
  - porphyry: 162 MPa.
- All lithological domains are characterized as fair to good rock with median Q' values as follows:
  - andesite: 8.3 to 12.5
  - granodiorite: 5.6 to 8.3
  - basalt/mafic volcanic: 10.8 to 16.2
  - porphyry: 5.6 to 8.3.

Table 16-2: Goldlund Global Joint Set Orientations

JSET 1 Foliation Parallel (NE-SW Striking)	JSET 2 (Sub-Horizontal)	JSET 3 Vein Parallel	
		N-S-Striking	NW-SE-Striking
79/152	11/077	72/273	86/070

16.2.3 Miller

The Miller geotechnical site characterization has been based core photo-logging and comparison to the Goldlund property which has a similar geological setting.

- Based on the similar geological setting, and fault-scale trends, this study assumes that Miller will have similar joint trends as Goldlund (see Section 16.2.2). It is noted that there may be jointing sub-parallel to the fault which transects the Southwest pit wall; however, dip is unknown.
- All lithological domains are characterized as fair-to-good rock with Q' values as follows:
    - felsic Intrusive: 6.6 to 11.5
    - felsic extrusive: 5.3 o 11.2
    - mafic intrusive: 5.2 to 8.3
    - mafic extrusive: 5.3 to 8.3
    - intermediate intrusive: 4.9 to 7.8
    - intermediate extrusive: 5.4 to 8.3
    - porphyry: 5.4 to 8.3.

16.3 Hydrogeological Considerations

Limited or no hydrogeological information is available at the time of writing. Refer to Section 18.3.

16.4 Open Pit Mining

16.4.1 Modifying Factors

16.4.1.1 Resource Model

The Goliath resource model framework is shown in Table 16-3. The Goliath resource model was converted from a partial block model and regularized to 5 x 4 x 5 m. The Goliath model contains gold and silver grades.

Table 16-3: Goliath Resource Model Framework

Type	Units	X	Y	Z
Model Origin	m	526050	5511500	410
Parent Cell Size	m	5	2	5
Number of Cells	#	638	618	182
Rotation	°	-	-	-

The Goldlund resource model framework is shown in Table 16-4. The Goldlund resource model was converted from a partial block model to a regular model at 5 x 5 x 5 m. The Goldlund model only contains gold grades.

Table 16-4: Goldlund Resource Model Framework

Type	Units	X	Y	Z
Model Origin	m	545000	5526500	-350
Parent Cell Size	m	5	5	5
Number of Cells	#	940	500	162
Rotation	°	-	-	-

The Miller resource model framework is shown in Table 16-5. The Miller resource model was converted from a partial block model to a regular model at 5 x 5 x 5 m. The Miller model only contains silver grades.

Table 16-5: Miller Resource Model Framework

Type	Units	X	Y	Z
Model Origin	m	553800	5533000	135
Parent Cell Size	m	5	5	5
Number of Cells	#	280	200	57
Rotation	°	-	-	-

16.4.1.2 Loss & Dilution Assessment

A loss and dilution assessment was undertaken for each deposit based on a specified cut-off applied to the regular block model. Internal dilution (isolated waste blocks) and external loss (isolated ore blocks) were identified, and a dilution skin applied to evaluate the amount of expected dilution and loss. The dilution skin was estimated 0.5 m. The diluting grades were based on the surrounding below cut-off material. Only measured and indicated classified mineral resources were considered in the assessment. The results are presented in Table 16-6 to Table 16-8 within an optimized pit shell. The cut-off values applied in the assessment were:

- Goliath: C$15.22/t
- Goldlund: C$16.00/t
- Miller: C$23.63/t

Table 16-6: Goliath Block Assessment Results

Parameter	Quantity (kt)	Au (g/t)	Au (koz)
Regularized Model			
No Boundary Dilution	867	1.86	1,611
1 Block	1,617	1.72	2,783
2 Blocks	1,386	1.09	1,518
3 Blocks	247	0.13	31
Isolated	348	0.69	240
Total	4,465	1.38	6,183
Internal Dilution & External Loss			
Internal Dilution	81	0.29	23
External Loss	348	0.69	240
Total	4,198	1.42	5,966
Dilution Skin			
Dilution Skin	680	0.24	164
Total	**4,878**	**1.26**	**6,129**

Source: SRK, 2023

Table 16-7: Goldlund Block Assessment Results

Parameter	Quantity (kt)	Au (g/t)	Au (koz)
Regularized Model			
No Boundary Dilution	303	1.46	14.3
1 Block	163	0.98	5.2
2 Blocks	216	0.84	5.8
3 Blocks	62	0.87	1.7
Isolated	46	0.65	1.0
Total	790	1.10	28.0
Internal Dilution & External Loss			
Internal Dilution	13	0.22	0.1
External Loss	46	0.65	1.0
Total	757	1.11	27.1
Dilution Skin			
Dilution Skin	94	0.18	0.6
Total	**851**	**1.01**	**27.7**

Source: SRK, 2023

Table 16-8: Miller Block Assessment Results

Parameter	Quantity (kt)	Au (g/t)	Au (koz)
Regularized Model			
No Boundary Dilution	11,514	1.41	16,254
1 Block	5,757	0.83	4,750
2 Blocks	5,179	0.64	3,310
3 Blocks	1,461	0.50	727
Isolated	464	0.42	197
Total	24,375	1.04	25,239
Internal Dilution & External Loss			
Internal Dilution	556	0.39	219
External Loss	464	0.42	197
Total	24,467	1.03	25,261
Dilution Skin			
Dilution Skin	2,050	0.18	368
Total	**26,517**	**0.97**	**25,630**

Source: SRK, 2023

The loss and dilution results are shown in Table 16-9. The dilution values have been calculated assuming a zero diluting grade will be applied. Some loss values resulted in negative numbers, as there was more metal introduced through dilution than was lost through external loss. These loss and dilution values have been applied in the LOM plan and mineral reserves. For Goldlund 0% loss was assumed.

Table 16-9: Loss & Dilution Results

Deposit	Loss (%)	Dilution (%)
Goliath	1	10
Goldlund	-2*	7
Miller	1	9

Note: *0% loss was applied. Source: SRK, 2023

16.4.2 Pit Optimization

16.4.2.1 Pit Optimization Parameters

The pit optimization parameters are shown in Table 16-10 and have been based on PEA (Ausenco, 2021) and some preliminary strategic assessments completed by SRK in early 2022. The gold processing recoveries are shown in Equations 1 to 3. Only measured and indicated classified mineral resources were considered in the pit optimization.

Table 16-10: Pit Optimization Parameters

Parameters	Units	Goliath	Goldlund Zone 1 to 6	Goldlund Zone ≥7	Miller
Production					
Production Rate	Mt/a	1.8	1.8	1.8	0.4
Discount Rate	%	5	5	5	5
Geotechnical					
Overall Slope Angle	°	43 - 46	44 / 50	44 / 50	39
Mining Factors					
Dilution	%	10	7	7	9
Loss	%	1	0	0	1
Limits		Waste storage	-	-	-
Processing Recovery					
Recovery Au	%	Equation 1 Min. 75% Max. 98%	Equation 2 Min. 75% Max. 98%	Equation 3 Min. 75% Max. 98%	Equation 1 Min. 75% Max. 98%
Recovery Ag	%	60	-	-	-
Costs					
Base Mining	C$/t mined	3.50	3.50	3.50	3.50
Incremental Mining	C$/t/10 m	0.035	0.035	0.035	0.035
Reference Bench	mRL	410	410	410	395
Processing	C$/t ore	11.76	11.76	11.76	11.76
Run-of-Mine Transport	C$/t ore	-	-	0.41	7.84
Tailings & Water Management	C$/t ore	0.41	0.41	0.41	0.41
G&A	C$/t ore	1.67	1.67	1.67	1.67
Royalty	%	-	-	-	-
Product Transportation	C$/oz Au	5.00	5.00	5.00	5.00
Revenue					
Exchange Rate	CAD:USD	1.30	1.30	1.30	1.30
Selling Price Au	US$/oz	1,550	1,550	1,550	1,550
Selling Price Ag	US$/oz	21	21	21	21
Payability Au	%	99	99	99	99
Payability Ag	%	97	-	-	-
Cut-offs					
Marginal Costs (Cut-off Value)	C$/t ore	15.22	16.00	16.00	23.63
Cut-off Grade	g/t Au	0.26	0.33	0.31	0.40

Source: SRK, 2023.

$$Equation\ 1 = \frac{93.873\ x\ Au^{0.021}}{100}$$

$$Equation\ 2 = 1 - \left(\frac{(0.0309x\ Au + 0.09024)}{Au}\right) - 0.006$$

$$Equation\ 3 = \left(\frac{92.638\ x\ Au^{0.0528}}{100}\right) - 0.006$$

There is limited waste storage available at the Goliath deposit. For the purposes of the pit optimization, the following assumptions were made:

- External waste storage capacity: 40 Mt
- Backfill capacity: 1.9 Mt
- Underground mining waste storage requirements: 2.3 Mt
- Waste deviation allowance from pit shell to pit design: 5%

The pit shell selection at Goliath was therefore limited to 37.7 Mt of waste. To ensure the main pit at Goliath was maximized, a pit limit constraint was applied to exclude the western satellite pits. Blocks west of 526,970E were excluded from the pit optimization.

SRK notes that some changes were made to the project assumptions after the pit optimizations were completed, most notably:

- The truck size was reduced from 91 t class to 63 t class, which resulted in steeper slope angles. This is discussed in Section 16.4.3.
- Transportation of run-of-mine material (ROM) from Goldlund to the Goliath plant was assumed to be by a rail conveyor during the pit optimization stage of study. This was changed to truck haulage based on preliminary economic evaluations. The operating costs for truck haulage are significantly higher than the rail conveyor costs and could increase the cut-off value by 70%. This is discussed further in Section 16.4.2.3.

16.4.2.2 Pit Optimization Results

16.4.2.2.1 Approach

The pit optimizations were undertaken in NPV Scheduler (NPVS). A discounted cashflow (DCF) analysis was undertaken on the optimization results to identify an optimal pit shell for each deposit. NPVS applies discounting based on the selected production rate. Three scenarios are evaluated:

- Best Case: Mine sequence based on mining each revenue factor (RF) pit shell as a sequential pushback.
- Worst Case: Mine sequence based on a bench-by-bench mining approach with no pushbacks.
- Average Case: The average DCF between the Best and Worst cases. This generally provides a more realistic assessment of the DCF.

16.4.2.2.2 Goliath

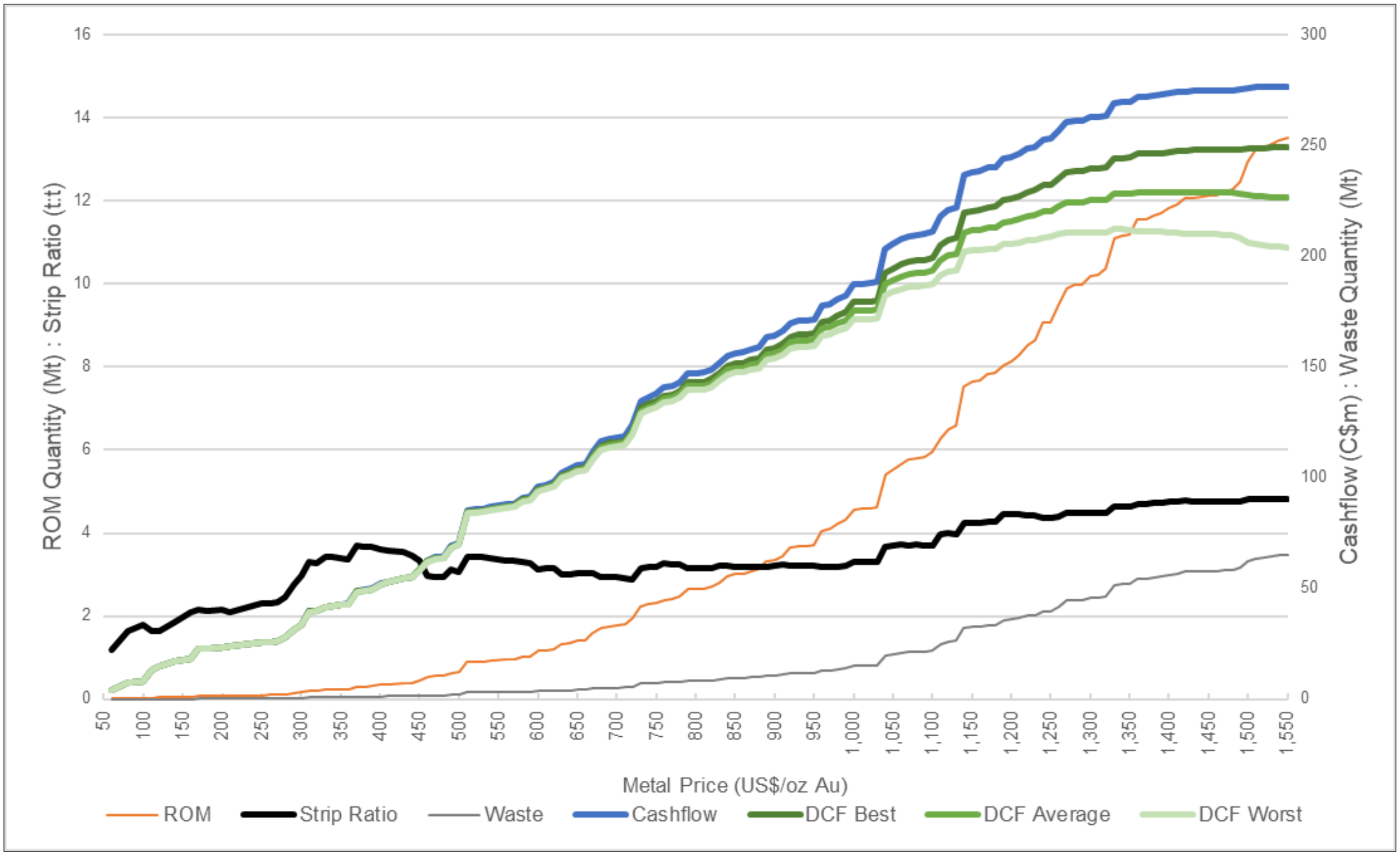
The pit optimization results are shown Table 16-11 for selected RFs. The pit optimization price sensitivity results are shown in Figure 16-1. The pit optimization results show that RF0.86 (C$ 1,330/oz) represents the highest 'worst case' DCF, while RF1.0 (C$ 1,550/oz) represents the highest 'best case' DCF. RF0.92 (C$ 1,420/oz) produces the highest 'average case' DCF. RF0.79 (C$ 1,220/oz) pit shell was selected as the basis for the pit design as it is within the waste limit requirement.

Table 16-11: Selected Goliath Pit Optimization Results

Optimization Results	Units	1,140 US$/oz	1,220 US$/oz	1,270 US$/oz	1,300 US$/oz	1,330 US$/oz	1,360 US$/oz	1,420 US$/oz	1,550 US$/oz
		RF0.74	RF0.79	RF0.82	RF0.84	RF0.86	RF0.88	RF0.92	RF1.0
Inventory									
Total	Mt	39.6	46.3	54.3	55.9	62.6	66.0	69.6	78.6
Waste	Mt	31.4	37.0	43.5	44.8	50.5	53.4	56.5	63.9
Strip Ratio	t:t	3.8	4.0	4.0	4.0	4.2	4.2	4.3	4.3
ROM	Mt	8.2	9.3	10.8	11.1	12.1	12.6	13.1	14.7
Au Grade	g/t	0.97	0.95	0.91	0.90	0.89	0.88	0.88	0.85
Au Contained	koz	256	284	316	323	346	358	370	400
Ag Grade	g/t	2.91	2.86	2.80	2.79	2.77	2.77	2.79	2.71
Ag Contained	koz	769	855	972	996	1,073	1,123	1,178	1,282
Recovered									
Au Average Recovery	%	95.0	95.0	94.9	94.8	94.8	94.8	94.8	94.7
Au Recovered	koz	244	269	300	306	328	339	351	379
Ag Average Recovery	%	60.0	60.0	60.0	60.0	60.0	60.0	60.0	60.0
Ag Recovered	koz	461	513	583	598	644	674	707	769
Operating Costs									
Mining Cost	C$M	146.5	171.7	201.8	207.7	233.4	246.0	259.7	293.7
	C$/t mined	3.70	3.71	3.71	3.72	3.73	3.73	3.73	3.74
	C$/t ore	19.43	20.11	20.38	20.36	21.06	21.28	21.55	21.75
	US$/oz Au	463	491	518	523	547	558	570	597
Processing Cost	C$M	113.6	128.7	149.2	153.8	167.1	174.2	181.7	203.6
	C$/t ore	13.84	13.84	13.84	13.84	13.84	13.84	13.84	13.84
	US$/oz Au	359	368	383	387	392	395	399	414
Total Selling Cost	C$M	6.5	7.2	8.0	8.2	8.8	9.1	9.4	10.2
	US$/oz Au	27	27	27	27	27	27	27	27
Cash Cost	US$/oz Au	849	885	928	936	966	980	996	1,037
Cashflow									
Revenue	C$M	503.1	556.2	619.5	632.2	678.5	701.5	725.2	783.8
UDCF	C$M	236.5	248.6	260.5	262.6	269.2	272.1	274.4	276.3
Best Case DCF	C$M	219.6	229.0	237.9	239.5	244.1	246.1	247.7	249.0
Average Case DCF	C$M	210.7	218.1	224.2	225.1	228.1	228.7	229.0	226.4
Worst Case DCF	C$M	201.9	207.2	210.5	210.7	212.1	211.3	210.3	203.8
Profit Margin	US$/oz Au	741	705	663	655	625	611	596	555
Mine Life	yrs	4.6	5.2	6.0	6.2	6.7	7.0	7.3	8.2

Notes: RF: revenue factor; ROM: run of mine; UDCF: undiscounted cashflow; DCF: discounted cashflow. Source: SRK, 2023

Figure 16-1: Goliath Pit Optimization Results



Notes: ROM: run of mine; DCF: discounted cashflow. Source: SRK, 2023

16.4.2.2.3 Goldlund

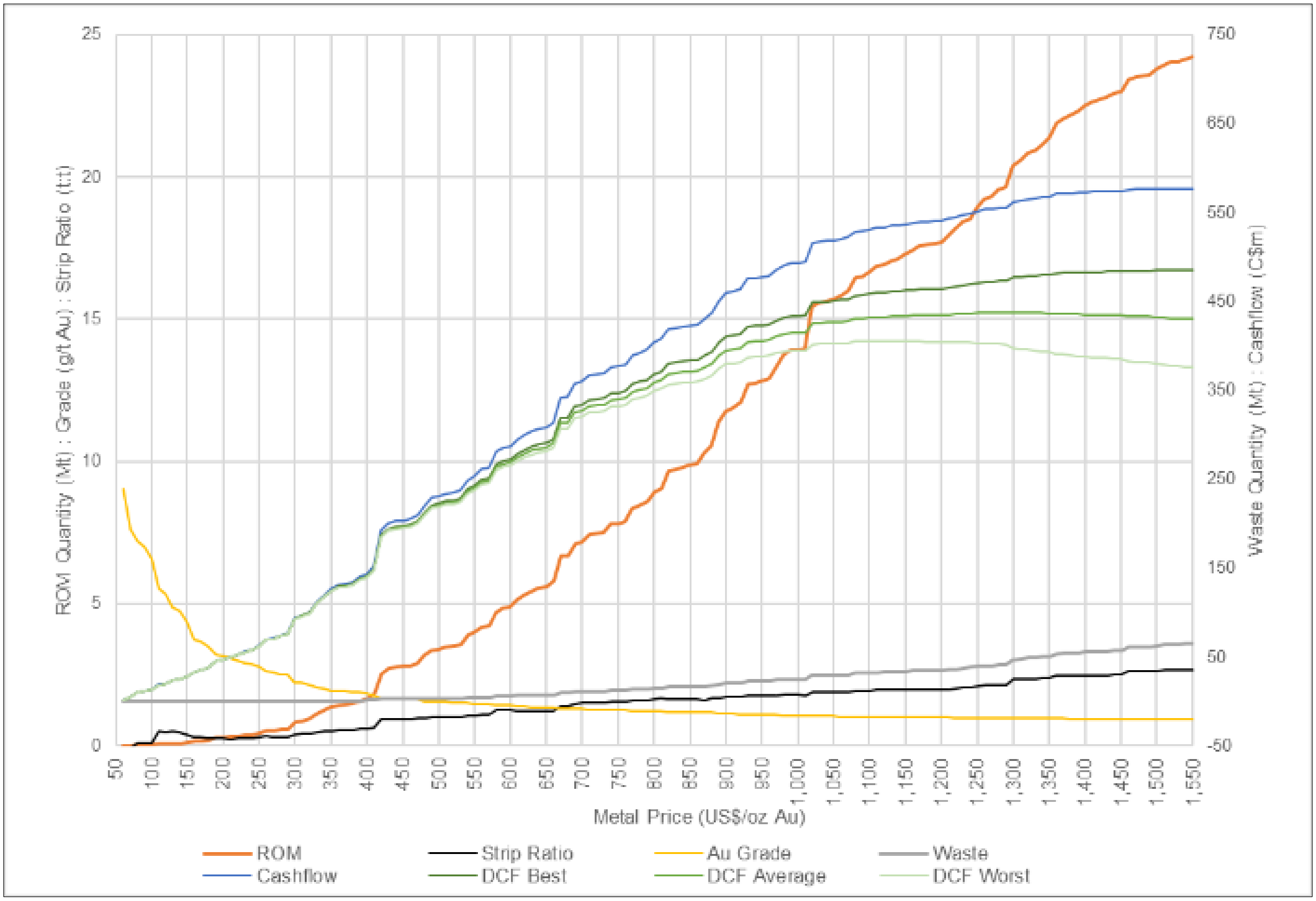
The pit optimization results are shown in Table 16-12 for selected RFs. The pit optimization price sensitivity results are shown in Figure 16-2. The pit optimization results show that RF0.83 (C$ 1,130/oz) represents the highest 'worst case' DCF, while RF1.0 (C$ 1,550/oz) represents the highest 'best case' DCF. RF0.83 (C$ 1,280/oz) produces the highest 'average case' DCF. RF0.88 (C$ 1,370/oz) pit shell was selected as the basis for the pit design. The difference in DCF between RF0.83 and RF0.88 is minimal.

Table 16-12: Selected Goldlund Pit Optimization Results

Optimization Results	Units	1,130 US$/oz	1,260 US$/oz	1,280 US$/oz	1,300 US$/oz	1,350 US$/oz	1,370 US$/oz	1,460 US$/oz	1,550 US$/oz
		RF0.73	RF0.81	RF0.83	RF0.84	RF0.87	RF0.88	RF0.94	RF1.0
Inventory									
Total	Mt	50.5	59.9	61.2	67.9	72.3	76.4	84.7	89.3
Waste	Mt	33.5	40.6	41.6	47.5	50.9	54.4	61.3	65.1
Strip Ratio	t:t	2.0	2.1	2.1	2.3	2.4	2.5	2.6	2.7
ROM	Mt	17.1	19.2	19.6	20.4	21.4	22.1	23.4	24.2
Au Grade	g/t	1.01	0.98	0.97	0.97	0.95	0.95	0.94	0.93
Au Contained	koz	553	604	610	635	657	674	706	723
Recovered									
Au Average Recovery	%	88.6	88.4	88.3	88.3	88.2	88.2	88.1	88.1
Au Recovered	koz	490	533	539	561	580	594	622	637
Operating Costs									
Mining Cost	C$M	184.5	219.0	223.8	247.8	264.2	279.4	310.2	327.2
	C$/t mined	3.65	3.66	3.66	3.65	3.65	3.66	3.66	3.66
	C$/t ore	11.58	12.18	12.24	13.00	13.21	13.55	14.16	14.46
	US$/oz Au	290	316	320	340	351	362	384	396
Processing Cost	C$M	254.9	287.6	292.4	304.9	319.9	329.9	350.4	361.9
	C$/t ore	14.95	14.95	14.95	14.95	14.95	14.95	14.95	14.95
	US$/oz Au	400	415	417	418	425	428	434	438
Total Selling Cost	C$M	12.3	13.4	13.6	14.1	14.6	14.9	15.6	16.0
	US$/oz Au	19	19	19	19	19	19	19	19
Cash Cost	US$/oz Au	710	751	756	778	795	809	837	852
Cashflow									
Revenue	C$M	986.9	1,073.7	1,085.4	1,129.5	1,166.8	1,196.0	1,252.3	1,282.0
UDCF	C$M	535.2	553.7	555.7	562.8	568.1	571.7	576.0	576.9
Best Case DCF	C$M	460.8	471.9	473.1	477.0	480.0	481.9	484.2	484.6
Average Case DCF	C$M	433.2	437.4	437.6	437.1	436.5	436.1	433.1	430.1
Worst Case DCF	C$M	405.6	402.9	402.1	397.1	392.9	390.3	382.1	375.5
Profit Margin	US$/oz Au	840	799	794	772	755	741	713	698
Mine Life	yrs	9.5	10.7	10.9	11.3	11.9	12.3	13.0	13.4

Notes: RF: revenue factor; ROM: run of mine; UDCF: undiscounted cashflow; DCF: discounted cashflow. Source: SRK, 2023

Figure 16-2: Goldlund Pit Optimization Results



Notes: ROM: run-of-mine; DCF: Discounted Cashflow. Source: SRK, 2023.

16.4.2.2.4 Miller

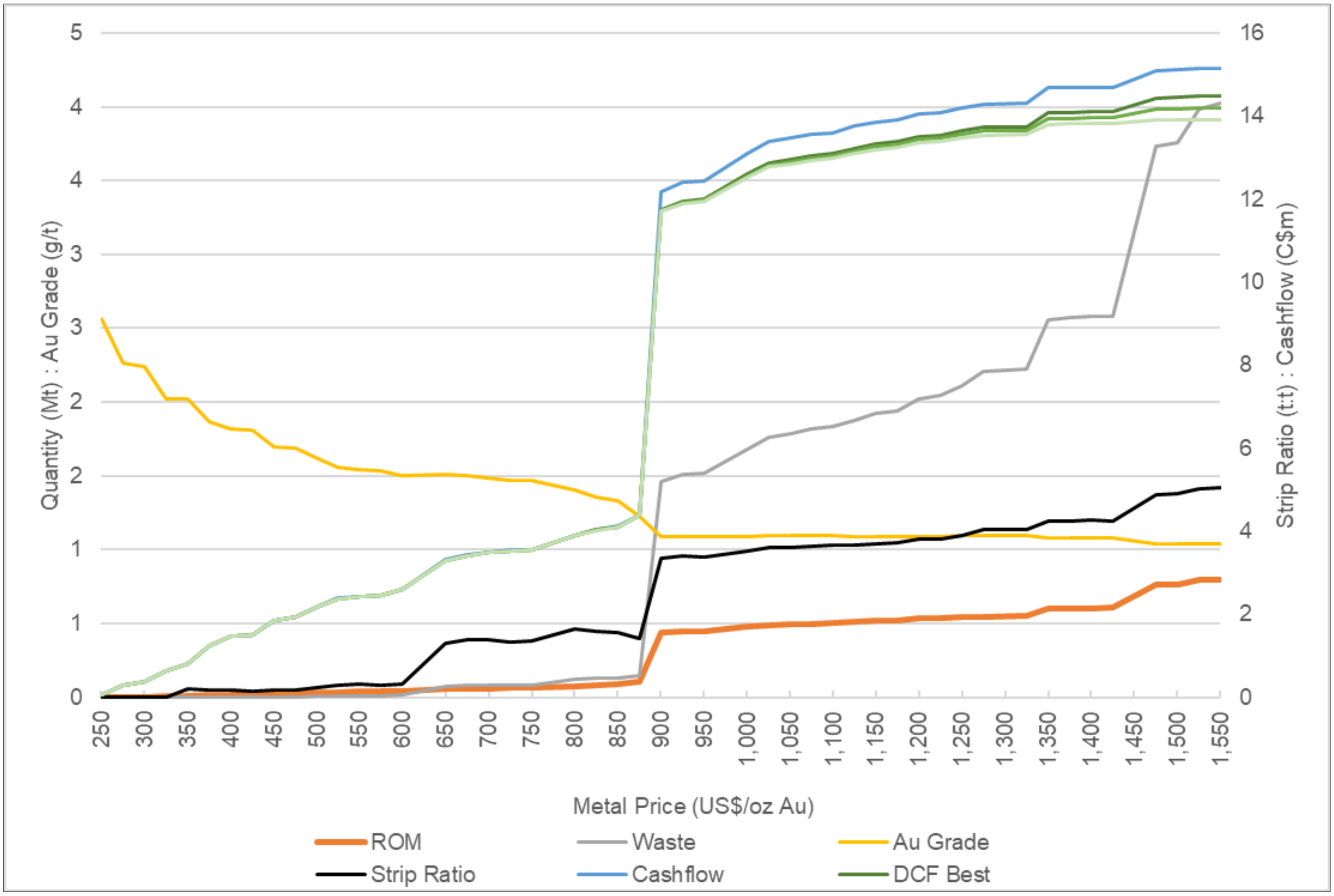
The pit optimization results are shown Table 16-13 for selected RFs. The pit optimization price sensitivity results are shown in Figure 16-3. The pit optimization results show that RF0.95 (C$ 1,475/oz) represents the highest 'worst case' DCF, while RF1.0 (C$ 1,550/oz) represents the highest 'best case' DCF. RF0.98 (C$ 1,525/oz) produces the highest 'average case' DCF. RF0.95 (C$ 1,475/oz) pit shell was selected as the basis for the pit design, as Miller will be mined as a single pit with no pushbacks.

Table 16-13: Selected Miller Pit Optimization Results

Optimization Results	Units	900 US$/oz	1,250 US$/oz	1,350 US$/oz	1,475 US$/oz	1,525 US$/oz	1,550 US$/oz
		RF0.58	RF0.81	RF0.87	RF0.95	RF0.98	RF1.0
Inventory							
Total	Mt	1,902	2,655	3,162	4,494	4,779	4,821
Waste	Mt	1,464	2,114	2,560	3,731	3,986	4,023
Strip Ratio	t:t	3.3	3.9	4.3	4.9	5.0	5.0
ROM	Mt	438	541	602	763	793	797
Au Grade	g/t	1.09	1.09	1.08	1.04	1.04	1.04
Au Contained	koz	15	19	21	26	26	27
Recovered							
Au Average Recovery	%	94.8	94.8	94.7	94.7	94.6	94.6
Au Recovered	koz	14	18	20	24	25	25
Operating Costs							
Mining Cost	C$M	6.4	9.0	10.7	15.3	16.3	16.5
	C$/t mined	3.65	3.66	3.66	3.68	3.69	3.69
	C$/t ore	14.69	16.63	17.82	20.09	20.58	20.65
	US$/oz Au	342	385	417	488	502	504
Processing Cost	C$M	8.6	10.6	11.8	14.9	15.5	15.6
	C$/t ore	19.54	19.54	19.54	19.54	19.54	19.54
	US$/oz Au	455	452	457	475	476	477
Total Selling Cost	C$M	0.4	0.5	0.5	0.6	0.6	0.6
	US$/oz Au	19	19	19	19	19	19
Cash Cost	US$/oz Au	816	857	893	982	998	1,000
Cashflow							
Revenue	C$M	27.54	34.21	37.65	45.95	47.60	47.84
UDCF	C$M	12.18	14.18	14.67	15.10	15.15	15.16
Best Case DCF	C$M	11.75	13.63	14.08	14.44	14.48	14.48
Average Case DCF	C$M	11.73	13.55	13.94	14.18	14.19	14.19
Worst Case DCF	C$M	11.70	13.46	13.80	13.91	13.90	13.89
Profit Margin	US$/oz Au	647	607	570	481	466	464
Mine Life	yrs	1.1	1.4	1.5	1.9	2.0	2.0

Notes: RF: revenue factor; ROM: run of mine; UDCF: undiscounted cashflow; DCF: discounted cashflow. Source: SRK, 2023

Figure 16-3: Miller Pit Optimization Results



Notes: ROM: run-of-mine; DCF: Discounted Cashflow. Source: SRK, 2023.

16.4.2.3 Cut-off Values

Cut-off values (COV), sometimes referred to as net smelter return (NSR), were used as the cut-offs for each deposit. The mineralization was generally classified into high-grade (HG) and low-grade (LG) bins, which allowed lower grade material to be stockpiled and fed when there was plant capacity or at the end of the mine life.

Goldlund includes a medium-grade (MG) bin which reflects the elevated cut-off due to the change in ROM transportation cost. Goldlund LG was only fed to top up the plant feed until the end of the mine life (Year 13). The remaining Goldlund LG was not fed to the plant. Only a single cut-off was applied at Miller as it is mined towards the end of the mine life during a two-year period. The cut-off values are shown in Table 16-14.

Table 16-14: Open Pit Cut-off Values

Cut-off Values (C$/t)	LG	MG	HG
Goliath	15.22	-	46.50
Goldlund	16.00	27.56	41.00
Miller	-	-	23.63

Source: SRK, 2023

16.4.3 Mine Design

16.4.3.1 Design Parameters

The geotechnical pit design parameters are shown in Table 16-15 (note that berm widths exclude blast damage allowances; it is assumed that this can be managed by operations. All roads have been designed at 21 m width at a maximum gradient of 10%. A maximum of four benches at the pit bottom have been designed for single lane access at 15 m width.

Table 16-15: Geotechnical Pit Design Parameters

Design Parameters	Bench Height (m)	Face Angle (°)	Berm Width (m)	OSA (°)	Maximum Slope Height (m)	Maximum Stack Height (m)	Geotechnical Berm width (m)
Goliath Top 10 m	10	75	6.5	-	-	-	-
Goliath	20	75	8.0	57	165	100	16
Goldlund Top 10 m	10	75	6.5	-	-	-	-
Goldlund (355° - 085° and 130° - 265°)	20	85	8.5	55-59	175	100	16
Goldlund (85° - 130°)	20	76	8.5	59	175	100	16
Goldlund (265° - 310°)	20	72	8.5	59	175	100	16
Goldlund (310° - 355°)	20	74	8.5	55	175	100	16
Miller Top 10 m	10	70	6.5	-	-	-	-
Miller (154°, 334°)	20	70	8.5	52	90	100	16
Miller (062°, 242°)	20	70	8.5	56	90	100	16

Source: SRK, 2023

16.4.3.2 Pit Design

The Goliath pit design inventory is shown in Table 16-16 along with a comparison to the selected pit shell. The ultimate pit design is shown in Figure 16-4. The ramp width was decreased after the pit optimizations were completed, which reduced the slope angle and would result in less waste stripping. Understanding that the Goliath pit shell was selected based on a maximum waste storage limitation, the pit was designed larger, following some slightly larger shells, with the understanding that there would be less waste than in the shells, due to the shallower slope angle. The design inventory compared to the selected shell therefore shows an increase in ROM material and waste tonnage and a drop in Au grade, as the additional ROM material is lower in grade. The drop in Au grade is also attributed to higher grades at the bottom of the shell, which could not be included in the pit design, due to minimum mining width requirements.

Table 16-16: Goliath Ultimate Pit Design Inventory

Inventory	Units	Shell	Design	Delta	Delta
Total	kt	46,336	47,810	1,474	3%
Waste	kt	37,039	38,258	1,219	3%
Strip Ratio	t:t	3.98	4.01	0.02	1%
ROM	kt	9,297	9,548	251	3%
Au Grade	g/t	0.95	0.83	-0.12	-13%
Au Contained	koz	284	254	-30	-11%
Ag Grade	g/t	2.86	2.62	-0.24	-8%
Ag Contained	koz	855	805	-49	-6%

Source: SRK, 2023

Figure 16-4: Goliath Ultimate Pit Design

Source: SRK, 2023

The Goliath open pit has been divided into six stages, as shown in Figure 16-5. The stage inventories are shown in Table 16-17. Goliath Stage 5 pit is used as access to the Goliath underground mine and as such is mined first in the mine sequence.

Table 16-17: Goliath Open Pit Stage Inventory

Inventory	Units	Total	1	2	3	4	5	6
Total	kt	47,806	14,205	14,678	10,367	7,172	982	401
Waste Rock	kt	33,206	9,568	8,898	8,729	5,293	523	197
Overburden	kt	5,052	1,337	1,964	419	897	303	132
Strip Ratio	t:t	4.0	3.3	2.8	7.5	6.3	5.3	4.6
Total ROM	kt	9,548	3,301	3,817	1,219	983	156	72
Au Grade	g/t	0.83	0.88	0.70	1.03	0.94	0.51	0.63
Au Contained	koz	254	93	86	41	30	3	1
Ag Grade	g/t	2.62	3.17	2.02	3.52	2.22	1.73	1.42
Ag Contained	koz	805	337	248	138	70	9	3
High Grade	kt	3,207	1,010	1,297	424	417	35	23
Au Grade	g/t	1.70	2.02	1.32	2.24	1.64	0.92	1.07
Au Contained	koz	175	65.6	55.1	30.5	22.0	1.0	0.8
Ag Grade	g/t	4.02	5.28	2.84	5.99	2.92	2.75	1.58
Ag Contained	koz	415	171.6	118.2	81.7	39.2	3.1	1.2
Low Grade	kt	6,342	2,291	2,520	795	566	121	49
Au Grade	g/t	0.39	0.38	0.38	0.39	0.42	0.39	0.41
Au Contained	koz	79	27.7	31.2	10.0	7.6	1.5	0.6
Ag Grade	g/t	1.91	2.25	1.61	2.20	1.71	1.43	1.34
Ag Contained	koz	390	165.4	130.2	56.1	31.1	5.6	2.1

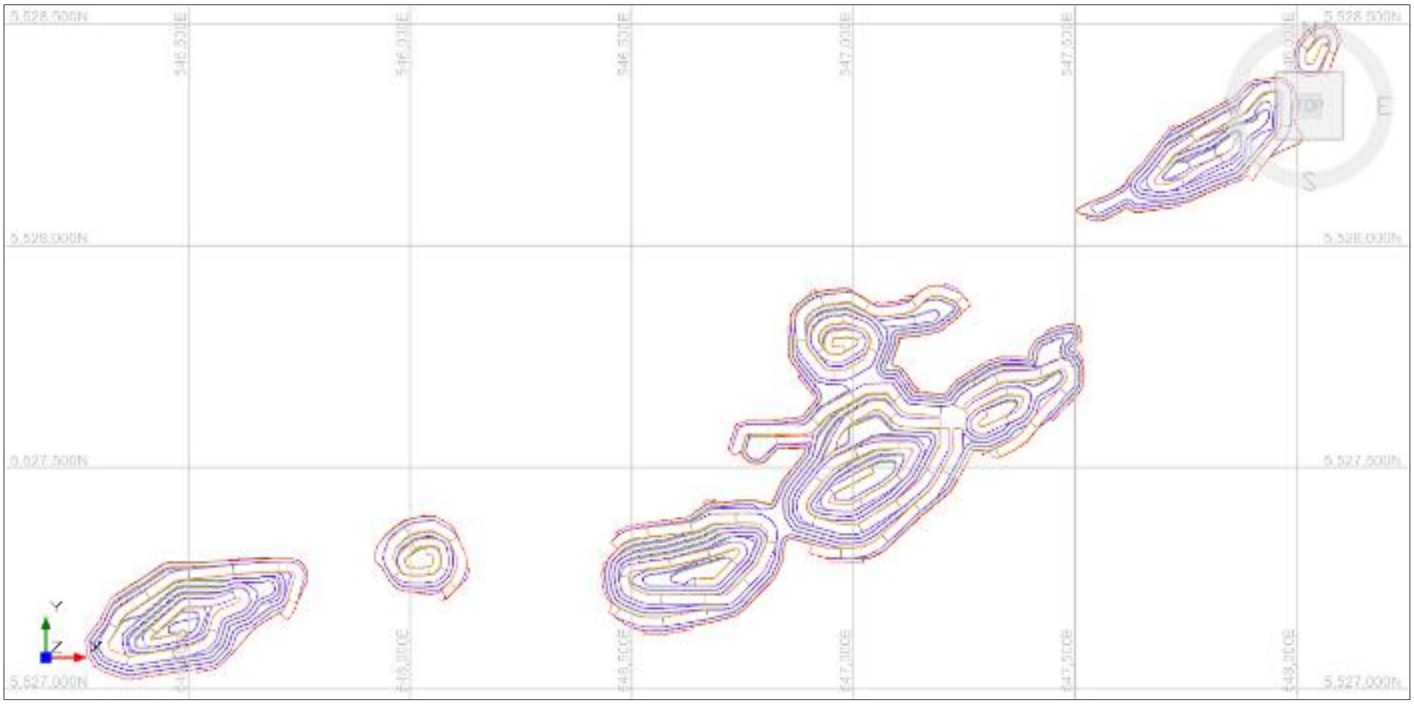
Source: SRK, 2023

Figure 16-5: Goliath Open Pit Stages

Source: SRK, 2023

The Goldlund ultimate pit design is shown in Figure 16-6. The inventory is shown in Table 16-18 along with a comparison to the selected pit shell. There is a reduction in waste as the ramp width was decreased from original assumptions, which resulted in a steeper slope angle. Some additional LG ore was also lost mainly on the outer walls; however, this had a negligible impact on contained metal. The Goldlund deposit has been divided into 11 stages, with 7 stages in the main central pit, as shown in Figure 16-7. The stage inventory is shown in Table 16-19.

Figure 16-6: Goldlund Ultimate Pit Design



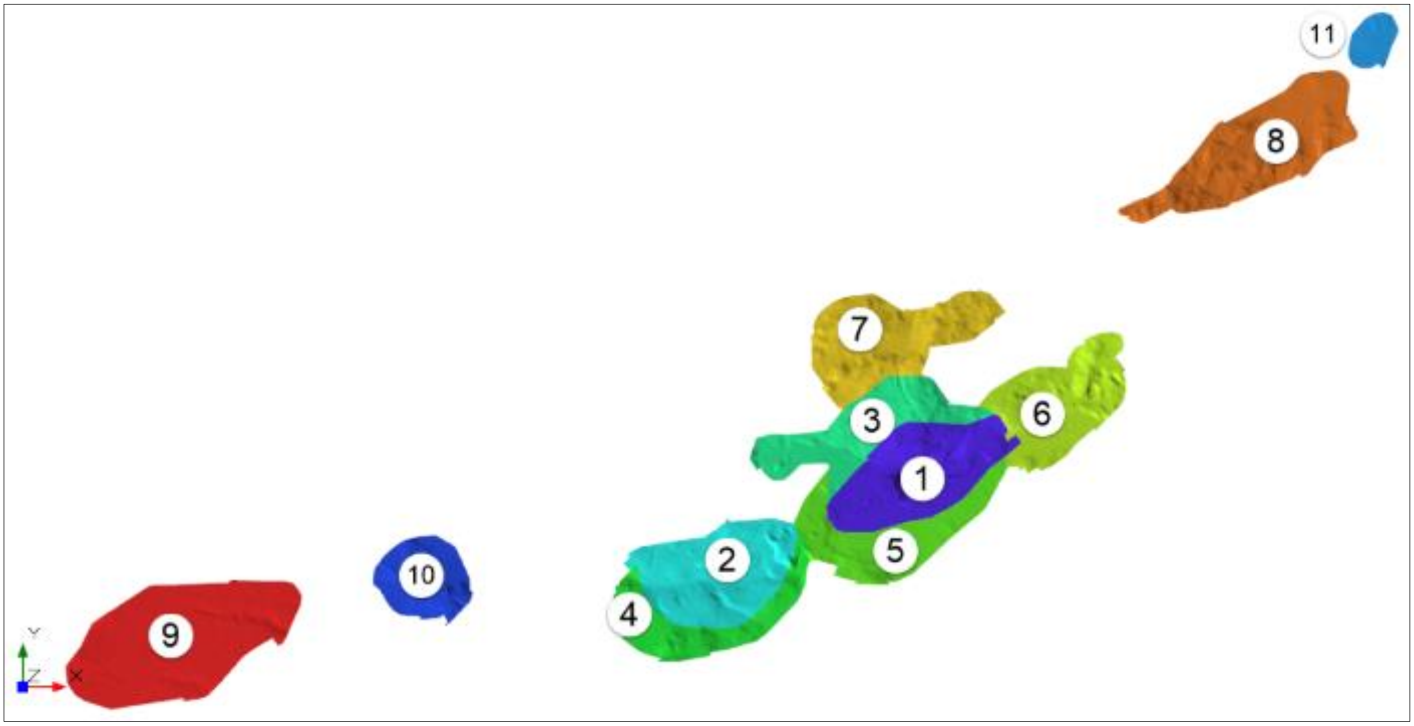
Source: SRK, 2023

Table 16-18: Goldlund Ultimate Pit Design Inventory

Inventory	Units	Shell	Design	Delta	Delta
Total	kt	76,438	72,488	-3,950	-5%
Waste	kt	54,369	52,405	-1,963	-4%
Strip Ratio	t:t	2.46	2.61	0.15	6%
ROM	kt	22,070	20,083	-1,987	-9%
Au Grade	g/t	0.95	1.04	0.09	9%
Au Contained	koz	674	669	-5	-1%

Source: SRK, 2023

Figure 16-7: Goldlund Stages



Source: SRK, 2023

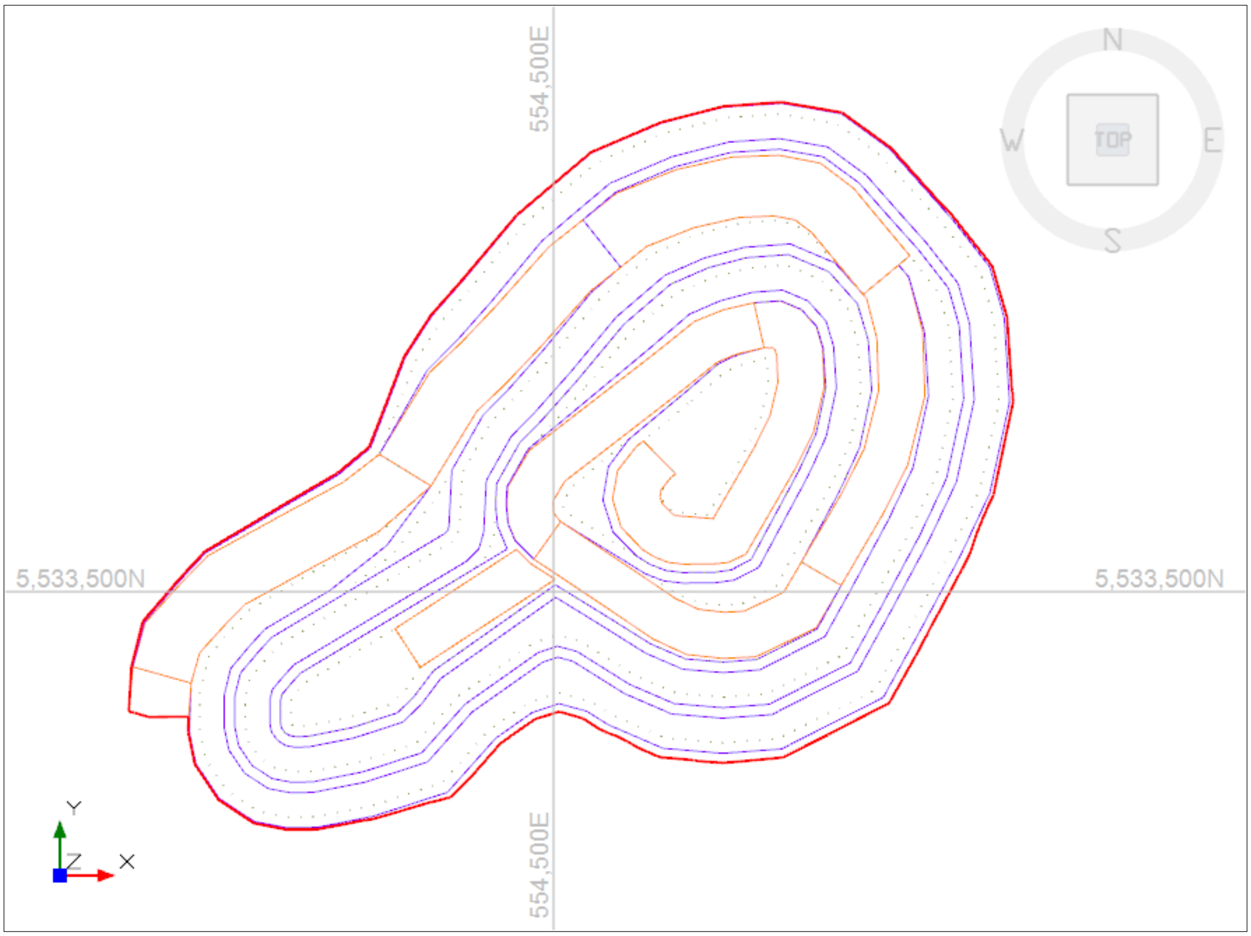
Table 16-19: Goldlund Stage Inventory

Inventory	Units	Total	1	2	3	4	5	6	7	8	9	10	11
Total	kt	72,488	6,265	6,853	7,165	7,161	12,170	5,187	6,415	6,591	12,267	2,116	298
Waste Rock	kt	45,574	2,847	3,918	4,909	4,711	8,831	3,593	5,462	3,666	6,151	1,446	40
Overburden	kt	6,831	795	643	380	583	693	456	392	444	2,014	308	124
Strip Ratio	t:t	2.6	1.4	2.0	2.8	2.8	3.6	3.6	10.4	1.7	2.0	4.8	1.2
Total ROM	kt	20,083	2,623	2,293	1,876	1,867	2,646	1,138	561	2,481	4,102	363	134
Au Grade	g/t	1.04	1.44	1.49	0.69	1.32	1.02	0.91	1.09	0.78	0.76	0.93	0.48
Au Contained	koz	669	122	110	42	79	87	33	20	62	101	11	2
High Grade	kt	10,423	1,714	1,597	569	1,197	1,564	536	330	1,043	1,687	174	10
Au Grade	g/t	1.56	1.95	1.93	1.26	1.79	1.39	1.41	1.53	1.22	1.19	1.44	0.78
Au Contained	koz	523	107.2	99.1	23.0	68.9	70.1	24.3	16.2	40.9	64.7	8.1	0.3
High Grade	kt	3,989	420	317	380	276	477	228	92	648	1,031	75	45
Au Grade	g/t	0.59	0.60	0.59	0.59	0.60	0.60	0.60	0.58	0.58	0.58	0.58	0.56
Au Contained	koz	75	8.1	6.1	7.2	5.3	9.2	4.4	1.7	12.1	19.1	1.4	0.8
Low Grade	kt	5,671	488	379	927	394	605	374	139	790	1,385	113	78
Au Grade	g/t	0.39	0.41	0.40	0.39	0.40	0.40	0.40	0.37	0.38	0.38	0.37	0.39
Au Contained	koz	71	6.4	4.9	11.6	5.0	7.8	4.8	1.6	9.6	16.7	1.3	1.0

Source: SRK, 2023

The Miller pit design is shown in Figure 16-8. The inventory is shown in Table 16-20 along with a comparison to the selected pit shell. There is less waste than expected due to the change in ramp width, which resulted in steeper slope angles. The differences are well within acceptable tolerances.

Figure 16-8: Miller Ultimate Pit Design



Source: SRK, 2023

Table 16-20: Miller Ultimate Pit Design Inventory

Inventory	Units	Shell	Design	Delta	Delta
Total	kt	4,494	4,382	-111	-2%
Waste	kt	3,731	3,644	-87	-2%
Strip Ratio	t:t	4.89	4.94	0.05	1%
ROM	kt	763	738	-25	-3%
Au Grade	g/t	1.04	1.03	-0.01	-1%
Au Contained	koz	26	24	-1	-5%

Source: SRK, 2023

16.4.3.3 Waste Storage Areas

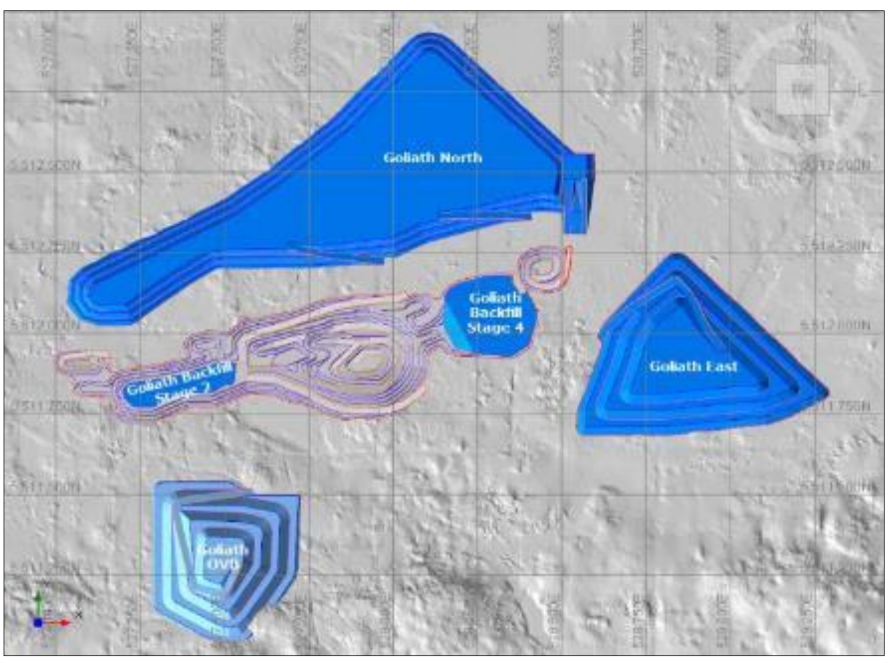
Waste storage areas (WSAs) have been divided into external rock WSAs, external overburden WSAs and backfill WSAs. The design criteria applied to the WSAs are shown in Table 16-21 and have been based on an overall slope angle of 2.5H:1V. The roads have been designed at 21 m width at a maximum gradient of 10%. The Goliath, Goldlund and Miller WSAs are shown in Figure 16-9 to Figure 16-11. The capacities of the WSAs are shown in Table 16-22. The WSAs have been designed to have additional capacity than required to allow for waste destination scheduling to choose the shortest haulage path.

Table 16-21: Waste Storage Design Parameters

Waste Storage Area	Material	Lift Height (m)	Rill Angle (°)	Berm Width (m)
Goliath North	Waste Rock	10	35	11
Goliath East	Waste Rock	20	35	22
Goliath OVB	Overburden	20	35	22
Goliath Backfill Stage 2	Waste Rock	35	35	-
Goliath Backfill Stage 4	Waste Rock	30	35	-
Goldlund Waste	Waste Rock	20	35	22
Goldlund OVB	Overburden	20	35	22
Goldlund Backfill Stage 4	Waste Rock	40	35	-
Goldlund Backfill Stage 6	Waste Rock	20 / 40	35	-
Miller Waste	Waste Rock	10	35	11
Miller OVB	Overburden	10	35	11

Source: SRK, 2023

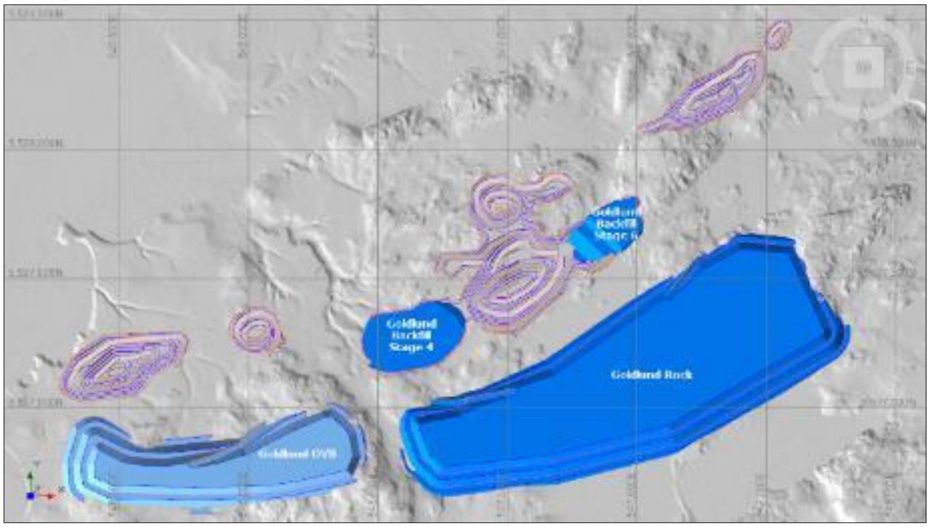
Figure 16-9: Goliath Waste Storage Areas



Source: SRK, 2023

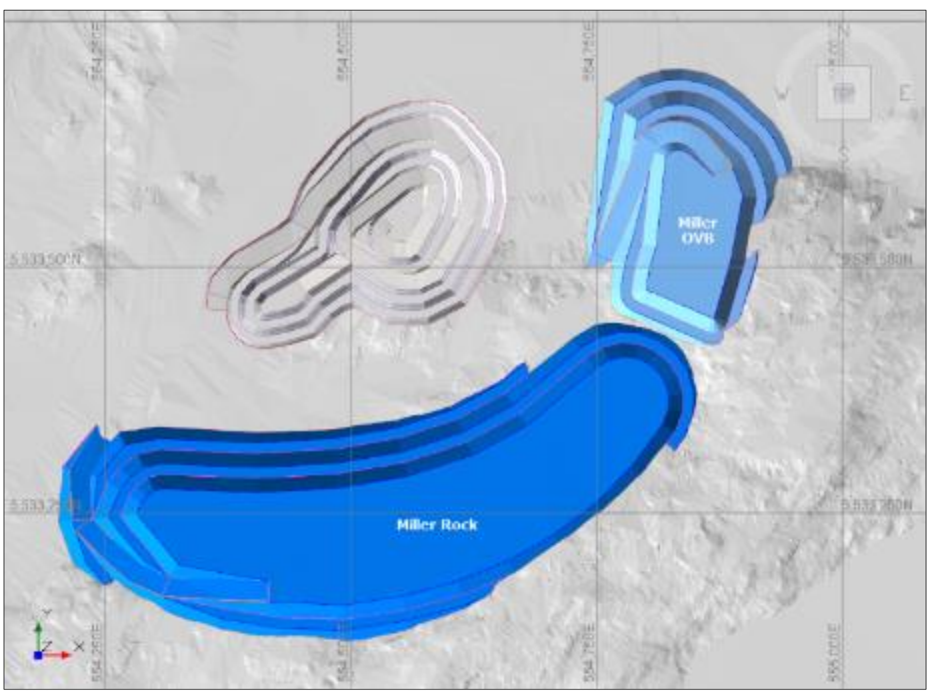
The Goliath North WSA was limited to three lifts to reduce visibility form nearby dwellings. The Goliath North WSA will be completed in Year 2 and therefore rehabilitation of the WSA will commence then.

Figure 16-10: Goldlund Waste Storage Areas



Source: SRK, 2023

Figure 16-11: Miller Waste Storage Areas



Source: SRK, 2023

Table 16-22: Waste Storage Area Capacities

Waste Storage Area	Capacity (lcm)
Goliath North	11,525,531
Goliath East	8,310,818
Goliath OVB	4,924,967
Goliath Backfill Stage 2	513,212
Goliath Backfill Stage 4	2,224,004
Goldlund Waste	23,517,836
Goldlund OVB	6,699,414
Goldlund Backfill Stage 4	4,785,592
Goldlund Backfill Stage 6	1,491,127
Miller Waste	1,635,540
Miller OVB	486,762
Total	**66,114,804**

Note: lcm: loose cubic meter; Source: SRK, 2023.

16.4.4 Mine Schedule

The mine schedule is based on the following criteria:

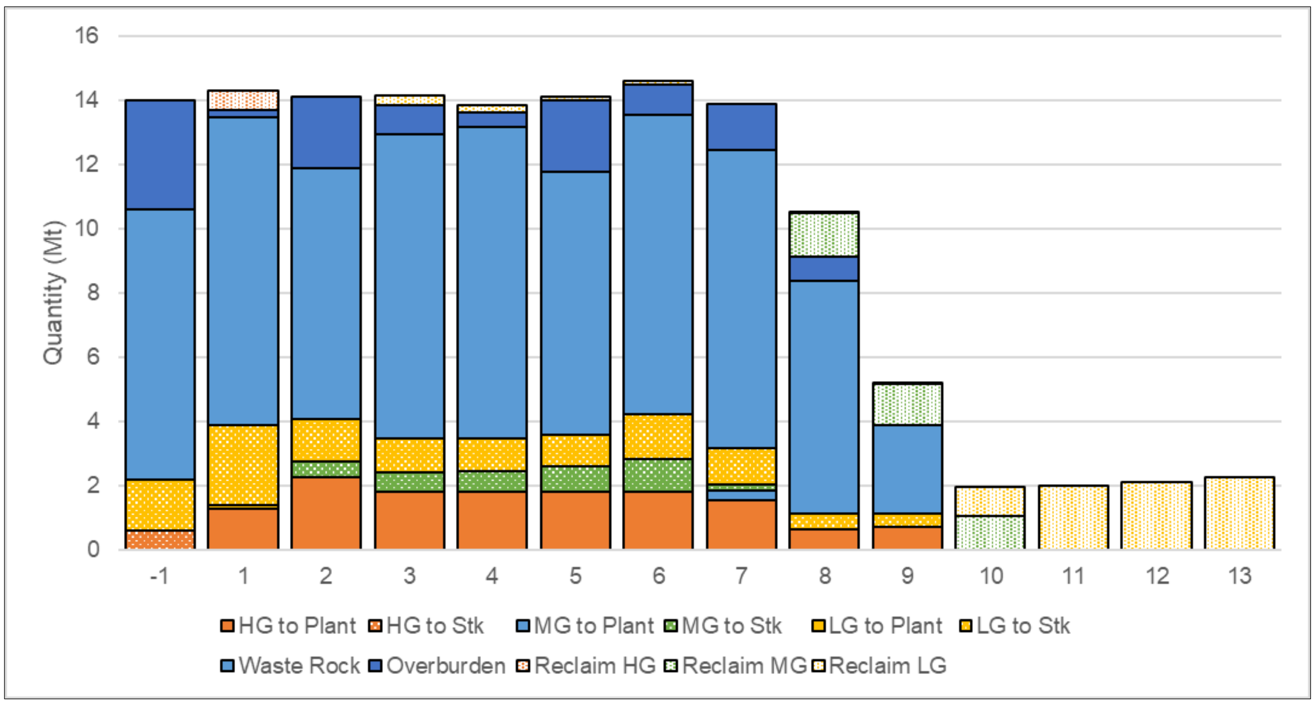
- ore production rate of 6,460 t/d (2.36 Mt/a), with 85% production in Year 1
- LG is stockpiled to be fed when there is plant capacity and at the end of the mine life
- Goliath underground ROM to be fed preferentially, as it has higher Au grade
- open pit mining to commence at Goliath.

The open pit mine schedule is shown in Figures 16-12 to 16-14 and Table 16-23 and has one year of pre-production and 9 years of production from the pits. Mining begins at Goliath in the Stage 5 pit to open access for Goliath underground mining. Goliath Stage 5 will be completed in Year -1. Goliath Stage 1 and 2 are mined in Year -1 to 2. Mining moves to Goldlund in Year 2, where the pits are mined until the beginning of Year 7. Mining then shifts back to Goliath Stage 3 to 6 in Year 7. Miller is mined in Year 8 and 9.

The vertical advance rate is limited to six 10 m benches per year.

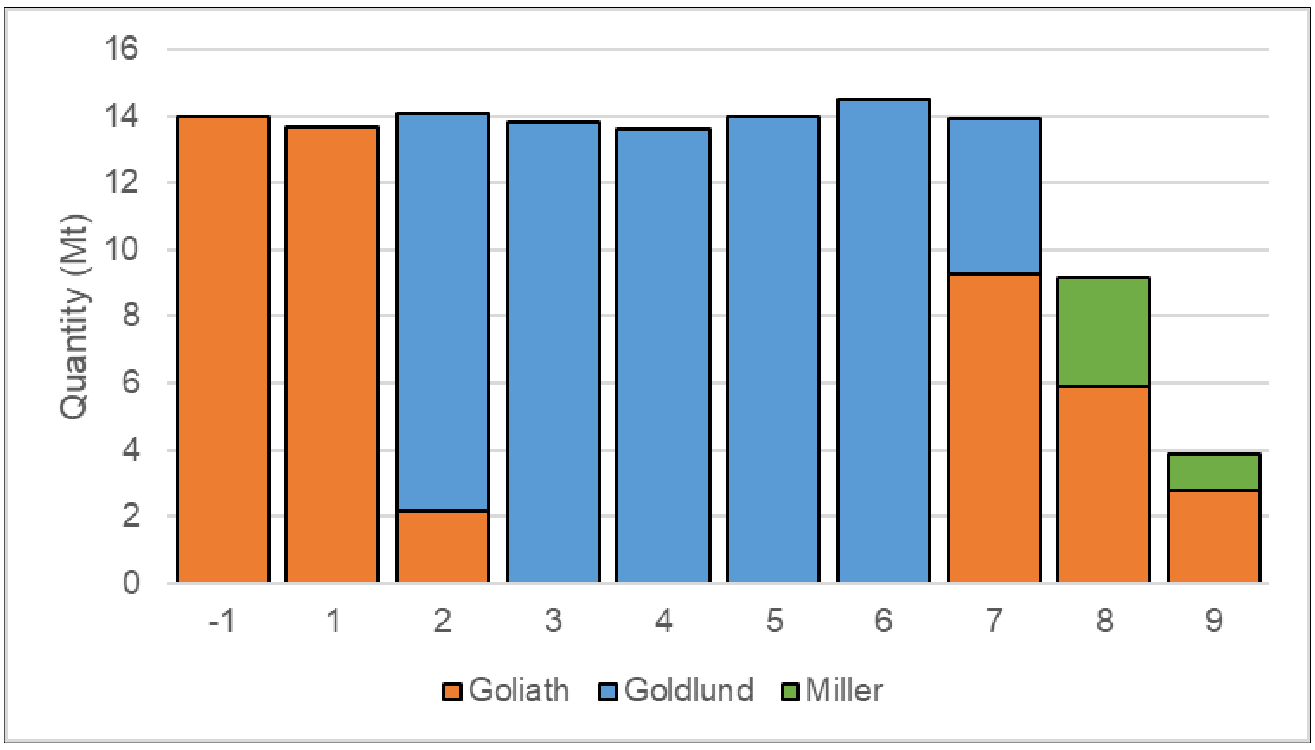
All Goliath HG material is stockpiled in Year -1 and then fed to the plant in Year 1. Most MG and LG is stockpiled and fed when there is capacity or at the end of the mine life. All Goldlund and Miller ROM is stockpiled at Goldlund prior to being hauled to Goliath. The material is hauled to the Goliath crusher when required as plant feed.

Figure 16-12: Open Pit Mining Schedule



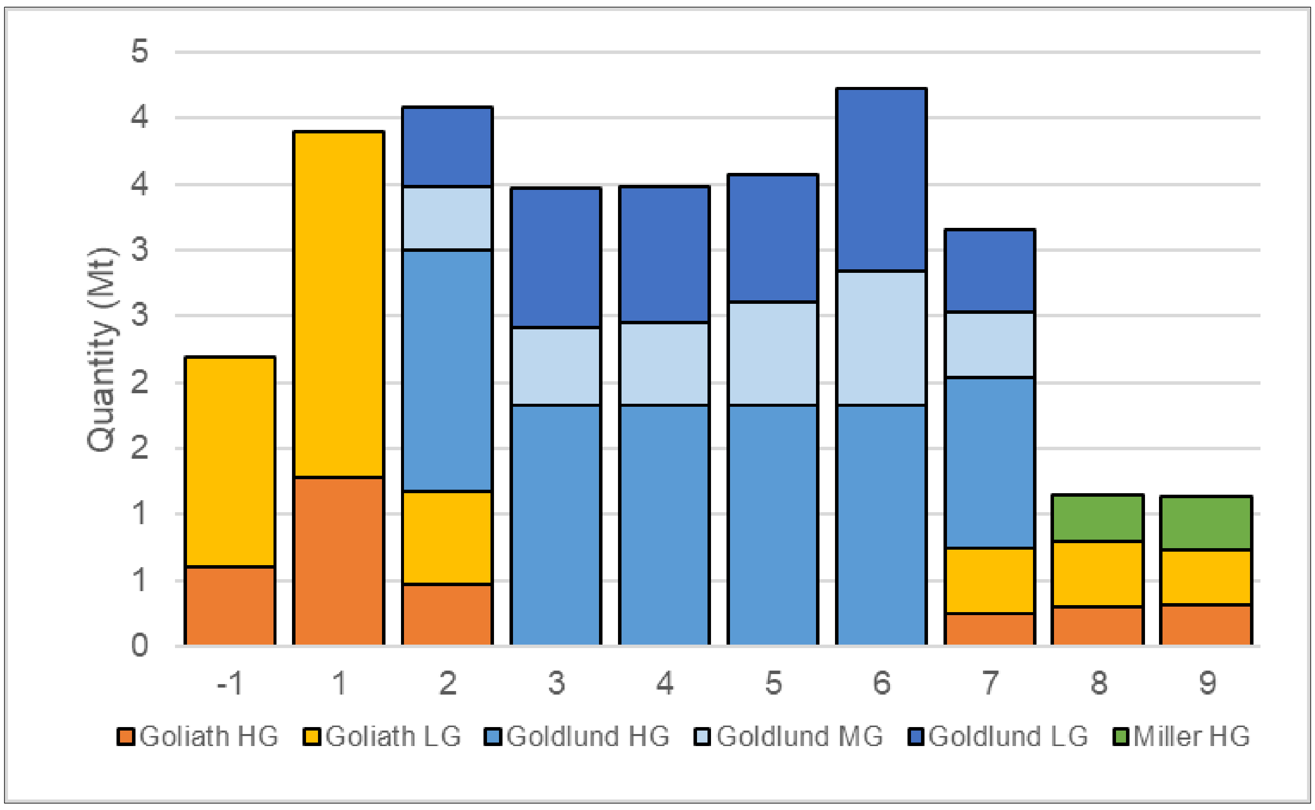
Notes: HG: high grade; MG: medium grade; LG: low grade; Stk: Stockpile. Source: SRK, 2023

Figure 16-13: Open Pit Mining Schedule by Deposit



Source: SRK, 2023

Figure 16-14: Open Pit Ex-Pit ROM Schedule



Notes: HG: high grade; MG: medium grade; LG: low grade; Source: SRK, 2023

Table 16-23: Open Pit Mine Schedule

Mine Schedule	Units	Total	-1	1	2	3	4	5	6	7	8	9
Total Ex-Pit												
Total	kt	124,677	14,000	13,685	14,094	13,836	13,635	13,981	14,492	13,900	9,150	3,904
Waste	kt	94,307	11,804	9,783	10,008	10,368	10,146	10,404	10,270	10,743	8,008	2,773
Rock	kt	81,792	8,401	9,582	7,801	9,481	9,690	8,199	9,317	9,303	7,244	2,773
Overburden	kt	12,515	3,403	201	2,207	887	456	2,204	953	1,440	764	-
Total ROM	kt	30,370	2,196	3,902	4,086	3,468	3,489	3,577	4,222	3,158	1,142	1,130
Au Grade	g/t	0.97	0.68	0.79	1.20	1.18	1.13	0.92	0.79	0.90	0.90	1.25
Ag Grade	g/t	0.82	2.38	2.57	0.79	-	-	-	-	0.49	1.79	2.65
High Grade	kt	14,368	598	1,279	2,286	1,830	1,825	1,825	1,825	1,548	645	707
Au Grade	g/t	1.56	1.47	1.64	1.78	1.83	1.73	1.35	1.23	1.38	1.28	1.76
Ag Grade	g/t	0.90	3.72	3.88	0.84	-	-	-	-	0.44	1.67	2.89
Medium Grade	kt	3,989	-	-	489	581	624	778	1,020	496	-	-
Au Grade	g/t	0.59	-	-	0.60	0.59	0.60	0.59	0.58	0.58	-	-
Ag Grade	g/t	0.00	-	-	-	-	-	-	-	-	-	-
Low Grade	kt	12,012	1,598	2,623	1,311	1,057	1,040	974	1,377	1,113	497	423
Au Grade	g/t	0.39	0.38	0.38	0.39	0.39	0.40	0.38	0.38	0.39	0.40	0.39
Ag Grade	g/t	1.01	1.87	1.94	0.98	-	-	-	-	0.77	1.94	2.25
Strip Ratio	t:t	3.10	5.38	2.51	2.45	2.99	2.91	2.91	2.43	3.40	7.01	2.45
Goliath Ex-Pit												
Total	kt	47,805	14,000	13,685	2,181	-	-	-	-	9,269	5,875	2,796
Waste	kt	38,258	11,804	9,783	1,004	-	-	-	-	8,529	5,076	2,061
Rock	kt	33,206	8,401	9,582	1,004	-	-	-	-	7,213	4,944	2,061
Overburden	kt	5,052	3,403	201	-	-	-	-	-	1,316	132	-
Total ROM	kt	9,548	2,196	3,902	1,176	-	-	-	-	740	799	736
Au Grade	g/t	0.83	0.68	0.79	0.92	-	-	-	-	0.74	0.91	1.29
Ag Grade	g/t	2.62	2.38	2.57	2.73	-	-	-	-	2.08	2.56	4.07
High Grade	kt	3,207	598	1,279	465	-	-	-	-	251	302	312
Au Grade	g/t	1.70	1.47	1.64	1.75	-	-	-	-	1.39	1.75	2.51
Ag Grade	g/t	4.02	3.72	3.88	4.14	-	-	-	-	2.70	3.58	6.54
Low Grade	kt	6,342	1,598	2,623	711	-	-	-	-	490	497	423
Au Grade	g/t	0.39	0.38	0.38	0.38	-	-	-	-	0.41	0.40	0.39
Ag Grade	g/t	1.91	1.87	1.94	1.81	-	-	-	-	1.76	1.94	2.25
Strip Ratio	t:t	4.01	5.38	2.51	0.85	-	-	-	-	11.52	6.36	2.80
Goldlund Ex-Pit												
Total	t	72,484	-	-	11,913	13,835	13,633	13,980	14,492	4,631	-	-
Waste	kt	52,405	-	-	9,003	10,368	10,146	10,404	10,270	2,214	-	-
Rock	t	45,574	-	-	6,796	9,481	9,690	8,199	9,317	2,090	-	-
Overburden	t	6,831	-	-	2,207	887	456	2,204	953	124	-	-
Total ROM	t	20,083	-	-	2,910	3,468	3,489	3,577	4,222	2,417	-	-
Au Grade	g	1.04	-	-	1.31	1.18	1.13	0.92	0.79	0.96	-	-
High Grade	kt	10,423	-	-	1,821	1,830	1,825	1,825	1,825	1,297	-	-
Au Grade	kt	1.56	-	-	1.79	1.83	1.73	1.35	1.23	1.37	-	-
Medium Grade	g/t	3,989	-	-	489	581	624	778	1,020	496	-	-
Au Grade	kt	0.59	-	-	0.60	0.59	0.60	0.59	0.58	0.58	-	-
Low Grade	g/t	5,671	-	-	599	1,057	1,040	974	1,377	624	-	-
Au Grade	g/t	0.39	-	-	0.41	0.39	0.40	0.38	0.38	0.38	-	-
Strip Ratio	t:t	2.61	-	-	3.09	2.99	2.91	2.91	2.43	0.92	-	-
Miller Ex-Pit												
Total	t	4,382	-	-	-	-	-	-	-	-	3,275	1,107
Waste	kt	3,644	-	-	-	-	-	-	-	-	2,932	712
Rock	t	3,012	-	-	-	-	-	-	-	-	2,300	712
Overburden	t	632	-	-	-	-	-	-	-	-	632	-
Total ROM	g/t	738	-	-	-	-	-	-	-	-	343	395
Au Grade	t:t	1.03	-	-	-	-	-	-	-	-	0.86	1.18
Strip Ratio	t:t	4.94	-	-	-	-	-	-	-	-	8.54	1.80
Goliath Stockpiles												
HG to Stockpile	t	598	598	-	-	-	-	-	-	-	-	-
Au Grade	g/t	1.47	1.47	-	-	-	-	-	-	-	-	-
Ag Grade	g/t	3.72	3.72	-	-	-	-	-	-	-	-	-
LG to Stockpile	t	6,233	1,598	2,514	711	-	-	-	-	490	497	423
Au Grade	g/t	0.38	0.38	0.38	0.38	-	-	-	-	0.41	0.40	0.39
Ag Grade	g/t	1.92	1.87	1.95	1.81	-	-	-	-	1.76	1.94	2.25
Goldlund Stockpiles												
MG to Stockpile	t	3,667	-	-	489	581	624	778	1,020	174	-	-
Au Grade	g/t	0.59	-	-	0.60	0.59	0.60	0.59	0.58	0.58	-	-
LG to Stockpile	t	5,671	-	-	599	1,057	1,040	974	1,377	624	-	-
Au Grade	g/t	0.39	-	-	0.41	0.39	0.40	0.38	0.38	0.38	-	-

Source: SRK, 2023

16.4.5 Operating Strategy

The open pit operations will be a conventional drill, blast, truck and shovel operation. The mining operation work pattern will be two 12-hour shifts with four operating and maintenance crews. The mine is planned to be in operation for 365 days per year, with an assumed five days of weather days per year.

All open pit material will require drilling and blasting, which will be undertaken on 10 m benches. The ultimate pit walls will be pre-split on 20 m benches. Infill drilling will be undertaken on 10 m benches with 20 m spacing, which will be undertaken by a contractor. Grade control drilling will be undertaken on 10 m benches with 20 m spacing and samples taken every meter.

Loading will be undertaken on 10 m benches with two flitches. There will be one 11 m^3 excavator and along with two 6 m^3 excavators which will paired be used with 63 t haul trucks in the pit. A 6 m^3 front-end loader (FEL) will be used at Goliath on the stockpiles. All Goldlund and Miller ROM will be rehandled with the FEL.

16.4.6 Equipment & Labour Requirements

The following equipment list (Table 16-24) specifies the planned open pit mining equipment. Although this study has used various makes and models of equipment, this report does not recommend one particular manufacturer or equipment model over any others. Where specific equipment models or manufacturers have been referred to, it is merely to acknowledge where information has been derived, or to provide the reader with an example of the type of equipment being discussed.

The mining equipment operating time has been developed from first principles, based on mechanical losses, operating standby and operational delays and is shown in Table 16-25.

Table 16-24: Open Pit Mining Equipment List

Equipment	Make	Model	Description
Primary Shovel	Komatsu	PC2000-11	Diesel Hydraulic Excavator
Secondary Shovel	Komatsu	PC1250LC-11	Diesel Hydraulic Excavator
Primary Loader	Komatsu	WA600-8	Front-end-loader
Primary Truck	Komatsu	HD605-8	Rigid Dump Truck
Primary Drill	Epiroc	FlexiROC D65	Down the hole drill 110-203 mm
Track Dozer	Komatsu	D155AX-8	Track Dozer 350hp
Wheel Dozer	CAT	824	Wheel Dozer 350hp
Wheel Loader	Komatsu	WA380-8	
Tire Handler	Komatsu	WA480-8	
Grader	Komatsu	GD655-7	
Backhoe & Hammer	Komatsu	PC650LC-11	

Source: SRK, 2023

Table 16-25: Open Pit Mining Equipment Operating Time

Equipment List	Availability (%)	Operating Time (h/y)	Use of Availability (%)	Direct Operating Time (h/y)	Operating Efficiency (%)	Effective Utilization (%)
Primary Shovel	85	6,713	90	5,253	78	60
Secondary Shovel	85	6,713	90	5,253	78	60
Primary Loader	85	6,713	90	5,071	76	58
Primary Truck	85	6,713	90	5,436	81	62
Primary Drill	80	6,318	90	5,041	80	58
Track Dozer	85	6,093	82	4,816	79	55
Wheel Dozer	85	5,472	73	4,195	77	48
Wheel Loader	85	4,231	57	3,319	78	38
Tire Handler	85	4,231	57	3,502	83	40
Grader	85	5,472	73	4,743	87	54
Backhoe & Hammer	85	5,472	73	4,743	87	54
Fuel/Lube Truck	85	5,472	73	4,743	87	54
Water Truck	85	865	12	865	100	10
Snow Plow	85	865	12	865	100	10
Low Boy Trailer	85	508	7	508	100	6
Lighting Plant	85	3,301	44	2,936	89	34
Light Vehicle	85	4,231	57	3,867	91	44

Source: SRK, 2023

The loading productivities are shown in Table 16-26. Haulage travel times were estimated in Deswik's Landform and Haulage (LHS) module. A haulage network consisting of design strings was used to represent in-pit haulage, pit ramps, ex-pit or backfill haulage, and on-dump haulage. This network was used to estimate haulage distances and travel times between mining solids, crusher, stockpiles, external waste storage blocks, and backfill blocks. The maximum haulage gradient was limited to 10%. The rolling resistance was estimated at 3.0%.

The drill and blast parameters used to estimate the drill and blast requirements are shown in Table 16-27.

Table 16-26: Open Pit Mining Loading Productivities

Loading Parameters	Units	ROM	Waste	ROM	Waste	Stockpile Reclaim	ROM Reclaim
Loading Unit		Primary Shovel	Primary Shovel	Secondary Shovel	Secondary Shovel	Primary Loader	Primary Loader
Bucket Size	m^3	10.8	10.8	6.0	6.0	6.3	6.3
Truck Payload	t	63.0	63.0	63.0	63.0	63.0	0.0
Loading Cycle Time	min.	2.27	2.87	4.07	4.07	3.73	1.47
In-Situ Density	t/bcm	2.76	2.56	2.76	2.56	2.76	2.76
Swell Factor	lcm/bcm	1.40	1.40	1.40	1.40	1.40	1.40
Loose Dry Density	dt/lcm	1.97	1.83	1.97	1.83	1.97	1.97
Moisture Factor	%	5.00	5.00	5.00	5.00	5.00	5.00
Passes	#	3	4	6	6	5	1
Loader Productivity	dt/dop-h	1,588	1,256	885	885	964	460
	dt/op-h	1,243	983	693	693	728	347
	Mt/a	8.3	6.6	4.7	4.7	4.9	2.3

Notes: bcm: bank cubic meter; lcm: loose cubic meter; dop-h; direct operating hour; op-h: operating hour. Source: SRK, 2023

Table 16-27: Open Pit Mining Drill & Blast Parameters

Drill & Blast Parameters	Units	ROM	Waste
Bench Height	m	10	10
Hole Diameter	mm	152	152
Subdrill	m	1.00	1.00
Spacing	m	4.50	4.50
Burden	m	3.88	3.88
Stemming Height	m	3.00	3.00
Penetration Rate	m/h	30.0	30.0
Productivity per meter	m/dop-h	25.5	25.5
Productivity per tonne	t/dop-h	1,118	1,037
Powder Factor	kg/m^3	0.79	0.79
Powder Factor	kg/t	0.29	0.31

Notes: dop-h; direct operating hour; op-h: operating hour. Source: SRK, 2023.

The open pit mining equipment estimated are shown in Table 16-28. The open pit mining labour requirements are shown in Table 16-29.

Table 16-28: Open Pit Mining Equipment Requirements

Equipment	Max.	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Primary Shovel	1	1	1	1	1	1	1	1	1	1	-	-	-	-	-
Secondary Shovel	2	2	2	2	2	2	2	2	2	2	2	-	-	-	-
Primary Loader	1	-	1	1	1	1	1	1	1	1	1	1	1	1	1
Primary Truck	12	10	11	11	11	11	12	12	12	9	5	1	1	1	1
Primary Drill	4	4	4	4	4	4	4	4	4	3	2	-	-	-	-
Track Dozer	3	2	3	3	3	3	3	3	3	3	2	-	-	-	-
Wheel Dozer	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Wheel Loader	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Tire Handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Backhoe & Hammer	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Fuel/Lube Truck	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Water Truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Snow Plow	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Low Boy Trailer	1	1	1	1	1	1	1	1	1	1	1	-	-	-	-
Lighting Plant	13	10	13	13	13	13	13	13	13	11	8	2	2	2	2
Light Vehicle	16	16	16	16	16	16	16	16	16	15	14	11	11	11	11

Source: SRK, 2023

Table 16-29: Open Pit Mining Labour Requirements

Labour Requirements	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13
Labour Requirements	198	178	190	190	190	190	194	198	198	174	134	64	64	64	64
Mine Operations	135	119	131	131	131	131	135	135	135	119	91	39	39	39	39
Mine Operations Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Operations Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Truck Operators	44	36	40	40	40	40	44	44	44	32	20	4	4	4	4
Shovel Operators	12	12	12	12	12	12	12	12	12	12	8	-	-	-	-
Loader Operators	4	-	4	4	4	4	4	4	4	4	4	4	4	4	4
Track Dozer Operators	12	8	12	12	12	12	12	12	12	12	8	-	-	-	-
Ancillary Operators	16	16	16	16	16	16	16	16	16	16	16	4	4	4	4
Dispatch Operators	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Drillers	16	16	16	16	16	16	16	16	16	12	8	-	-	-	-
Blast Crew	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Blasting Assistant	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Dewatering Crew	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Operations Coverage	8	8	8	8	8	8	8	8	8	8	4	4	4	4	4
Safety Officer	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Maintenance	46	42	42	42	42	42	42	46	46	38	26	10	10	10	10
Maintenance Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Crew	40	36	36	36	36	36	36	40	40	32	20	4	4	4	4
Technical Services	17	17	17	17	17	17	17	17	17	17	17	15	15	15	15
Technical Services Manager	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Planning Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologist	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Sampler	4	4	4	4	4	4	4	4	4	4	4	4	4	4	4
Mine Surveyor	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Surveyor Assistant	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1
Administrative Assistant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Consultant	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Source: SRK, 2023

16.5 Underground Mining

Of the three deposits, only Goliath features a high potential for extraction of gold-bearing mineralization via underground mining methods. Additional information regarding Goldlund's underground potential can be found in Section 24: Other relevant data and information.

16.5.1 Mining Method Selection

The process of selecting the underground mining method(s) to apply to the resource considered a number of factors, including:

- deposit size and geometry
- geotechnical and hydrogeological parameters
- proximity to surface
- desired production rate
- economic assumptions
- available resources (e.g., mobile equipment, labour)
- mitigation of risks (e.g., safety, cost, production).

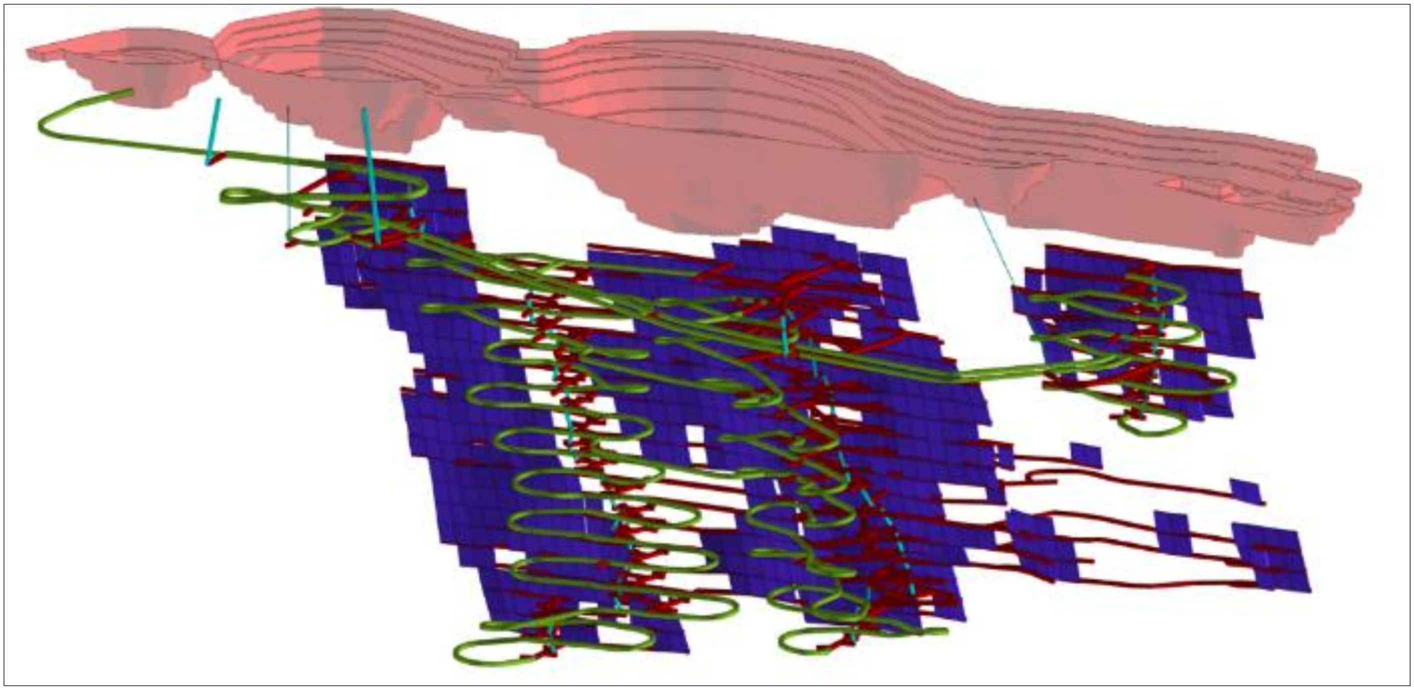
In the PEA (2021), the longhole open stoping (LHOS) mining was preferred primarily due to its compatibility with the deposit geometry and the surrounding host rock. After reviewing potential mining methods to apply in this study, the decision was made to utilize a similar narrow-vein, LHOS method. This choice was largely based on the following drivers:

- The 2021 diamond drilling campaign yielded an updated mineralized resource that bore similarities to the previous model (i.e., continuous, narrow, and steeply dipping), with the inclusion of additional above-COG material located along strike
- No new geotechnical investigation was completed that would result in a change to the understanding of the competency of the host rock
- Excess tailings are required to be deposited in the Goliath open pit once mining of it has ceased due to capacity limitations at the planned tailings storage facility (TSF).

16.5.2 Design Assumptions and Design Criteria

The criteria and assumptions utilized to design the Goliath underground mine (shown with the open pit for reference in Figure 16-15) are based on data gathered during previous field studies, recommendations from geotechnical engineering analysis, first principles calculations and industry experience, and are summarized in the sections below.

Figure 16-15: Isometric View of Goliath Open Pit and Underground Designs (Looking Southeast)



Source: SRK, 2023

16.5.2.1 Throughput Rate and Supporting Assumptions

The mine's production rate was derived through a cycle time analysis of the processes that primarily compose the stoping activity. Table 16-30 presents the process rates and fixed duration assumptions used when creating the life of mine schedule.

Table 16-30: Average Stope Cycle Time

Process	Avg. Amount per Stope	Rate	Duration (days)
Stope Prep			3.0
Production Drilling	900	240 m/d	3.8
Loading & Blasting			6.0
Mucking	4,360	466 t/d	9.4
Backfill Barricade Construction			4.0
Paste Backfill Plug	1,030 t	800 t/d	1.3
Paste Backfill Plug Cure			4.0
Paste Backfill Body	2,400 t	800 t/d	3.0
Paste Backfill Body Cure			14.0
Total Stope Cycle Time			48.5

Note: Values for Amount and Duration have been rounded for reporting purposes; Source: SRK, 2023

16.5.2.2 Opening Sizing

Following the geotechnical considerations addressed above, the mines underground lateral and vertical excavations are listed below. SRK applied the development profiles found in Table 16-31 and Table 16-32 to development designs strings.

Table 16-31: Mine Design Parameters – Lateral Development

Lateral Development Type	Section Profile
Ramp	5.0 mW x 5.0 mH / 6.0 mW x 6.0 mH / 6.0 mW x 7.0 mH
Level Access	5.0 mW x 5.0 mH
Crosscuts	5.0 mW x 5.0 mH
Ventilation Drives	5.0 mW x 5.0 mH
Ore Drives	4.0 mW x 4.5 mH
Paste Fill Crosscut	5.0 mW x 5.0 mH / 5.0 mW x 6.0 mH
Sump	5.0 mW x 5.0 mH
Escapeway Access	5.0 mW x 5.0 mH
Refuge Station	5.0 mW x 5.0 mH
Electrical Substation	5.0 mW x 5.0 mH

Source: SRK, 2023

Table 16-32: Mine Design Parameters – Vertical Development

Vertical Development Type	Dimensions (Diameter)
Primary Fresh Air Raise	4.0 m x 4.0 m
Primary Return Air Raise	4.0 m x 4.0 m
Internal Return Air Raise	3.5 m x 3.5 m
Escape Raise	1.2 mD

Source: SRK, 2023

16.5.2.3 Stope Sizing

The stopes are planned to be 20 m long (along strike) and roughly 29.5 m high (measured on the incline according to typical 70-75° orebody dip). The longitudinal stopes HW to FW width are typically in the order of 4 m with larger stopes having spans of 8 to 15 m (these larger stopes are rare). The following generalized conclusions are made for the PFS:

- Stope backs have design HR of 1.7 m (typical) to 4.3 m (maximum – rare), back performance is expected to be good with rare utilization of cable bolting. Cable bolts will be utilized where the HW to FW spans exceed roughly 7.3 m (HR = 2.7 m, suitable for the BMS and MSS). Later phases of study may optimize expectations for cable bolting once more detailed lithological modelling is available.

- Stope endwall HR are expected to be 1.8m (typical) to 5.0m (maximum – rare). Stope endwalls will be expected to perform very well.

- Stope HW and FW HR will be 6.1 m for average HW dip angles. Stope FWs are expected to perform well, and stope HWs will require cable bolting at depth (below -150L) in the BMS and MSS, and at all mining depths when MSED or QFP are exposed. 7 m cables on a 2 m square (toe spacing) pattern can be fan drilled from the ore sills to manage stability and dilution. Consideration may also be given to reducing stope strike length (to 14 m) to reduce cable bolting needs.

16.5.2.4 Dilution and Mining Recovery

An external dilution value of 5% has been applied to lateral development in mineralized ground, with an assumed mining recovery of 100% regardless of the type of development. No background grade has been assigned to the dilution material.

Mining recovery for stopes varies depending on the drilling direction. For downhole stopes, which account for 78% of the total stope tonnage, the recovery is assumed to be 90%. The uphole stopes carry a recovery factor of 80%. For both down- and uphole stopes the external mining dilution is set at 15%.

The dilution and recovery values applied to the development and stope shapes is based on experience with mines of similar setting and consider factors such as excavation dimensions and orebody dip.

16.5.2.5 Net Smelter Return

A Net Smelter Return was utilized to incorporate the Au and Ag metals found within the Goliath Model. SRK derived the Net Smelter Return calculation to code into the resource model to be utilized for Deswik Stope Optimizer as the optimization field. The parameters are based on the PEA (Ausenco, 2021) and some preliminary strategic assessments completed by SRK in early 2022. The parameters utilized for the NSR calculation are listed in Table 16-33.

Table 16-33: Net Smelter Return Parameters

Parameter for NSR	Unit	Value
NSR		
Transport & Refining Charge	C$/oz Au	5.00
Au Royalty	%	-
Ag Royalty	%	-
Au Payability	%	99
Ag Payability	%	97
Au Price	US$/oz	1,550
Ag Price	US$/oz	21
Au Recovery		$\frac{93.873\ x\ Au^{0.021}}{100}$ (Min. 75%, Max. 98%)
Ag Recovery	%	60

Source: SRK, 2023

A metal value was calculated for each of the respective metal utilizing the payability, royalties and selling costs. The NSR Formula accounts for the variable Au recovery shown below and a fixed Ag recovery at 60%.

$$Au\ Recovery\ Factor = \frac{93.873\ x\ Au^{0.021}}{100}$$

$$NSR\ Value = 2035.0706 \times ((Au\ Recovery\ Factor) \div 31.10348\ ) \times Au\ Grade\ (In\ Model) + 0.52262 \times Ag\ Grade\ (In\ Model)$$

16.5.2.6 Cut-off Criteria

SRK reviewed existing operations and projects during the benchmarking exercise for the Goliath underground mining cost, focusing on analogues with similar a mining method, planned production rate, mine access, and deposit geometry. The cut-off value (COV) chosen for the Goliath underground mine design was based on an estimated production rate of approximately 1,100 tonnes per day (t/d). While the LOM plan reaches a peak of slightly less than 1,400 t/d in Year 7, the average mining rate during the period of sustained production (Years 4 through 11) is similar to the rate chosen during the COV derivation process.

The total operating cost estimate formed the basis of the diluted COV (shown in Table 16-34). Based on an average external stope dilution of 15%, an in-situ COV of C$124/t (rounded) was selected for the stope optimization process and mineral reserve determination.

Table 16-34: Initial Estimate – Operating Cost

Cost Category	Value (C$/t ore)
Mine Operating Cost	85.59
Sustaining Capital	8.53
Mill Operating Cost	11.76
Tailings & Water Management	0.41
G&A	1.67
Total	107.96

Source: SRK, 2023

An NSR value for each block was calculated utilizing the NSR Value formula noted above and coded into the resource model. The underground reserves were identified using Deswik.SO (Stope Optimizer, or DSO), a strategic planning tool that assists with stope design. The remaining processes used to develop the underground LOM design and schedule were also completed using the Deswik® suite of mine planning software. DSO first requires the input of several key mining and economic parameters, which it in turn uses to interrogate the resource model to generate simplified mining shapes. The software creates stopes with maximum material above cut-off grade based on user inputs. DSO optimizes stopes in one dimension and requires set parameters for the other two dimensions. For this work, stopes were optimized in thickness (hanging wall to footwall), and the stope height and length were fixed. These shapes represent the potentially economic mineral resource at that COV.

The most significant DSO inputs are detailed in Table 16-35.

Table 16-35: Key DSO Input Parameters

Parameter	Unit	Value
NSR		
Cut-off Value	C$/t	124.00
Transport & Refining Charge	C$/oz Au	5.00
Au Royalty	%	-
Ag Royalty	%	-
Au Payability	%	99
Ag Payability	%	97
Au Price	US$/oz	1,550
Ag Price	US$/oz	21
Au Recovery		$\frac{93.873\ x\ Au^{0.021}}{100}$ (Min. 75%, Max. 98%)
Ag Recovery	%	60
Stope Design		
Sublevel Spacing	m	25
Minimum Stope Width	m	3
Stope Length	m	20
Minimum Stope Pillar Width	m	5
Side Ratio		2
Minimum Dip	°	55
Maximum Dip	°	135
Minimum Strike Direction Change	°	-40
Maximum Strike Direction Change	°	40

Source: SRK, 2023

The stope shapes were reviewed to ensure the DSO software applied the parameters correctly. Some of the stope shapes were removed from the LOM plan if they:

- were located within the open pit mining limits or the crown pillar beneath the pit
- were isolated from other mining areas and could not justify the development required to access them
- were considered "marginal" as the NSR value of the stope is very close to the COV
- dominantly contained material classified as inferred resource.

16.5.3 Primary Access

The conceptual underground mine design for the Goliath deposit is based on a mine that comes online while the Goliath open pit is in operation. As discussed in Section 16.4.3.2, the decline used to access the underground workings (the East Ramp) will be collared from within the Goliath Stage 5 pit on the northeastern limit of the deposit. The bottom of the pit will be backfilled with waste material and compacted to provide a larger, more stable working floor from which to collar the East Ramp portal. The floor of the East Ramp will be at the 370 elevation at its collar location.

The East Ramp will measure 6.0mW x 6.0mH in its initial sinking phase (the first 70m) and decrease to 5.0 m W x 5.0 m H for the subsequent ~285 meters. At this point, it will be fed by the primary fresh air raise (FAR), and the decline is enlarged to 6.0 m W x 7.0 m H as part of the design of the ventilation system (Section 16.5.9). The ramp again reverts to its nominal 5.0 m W x 5.0 m H dimensions once it has reached 250 Level.

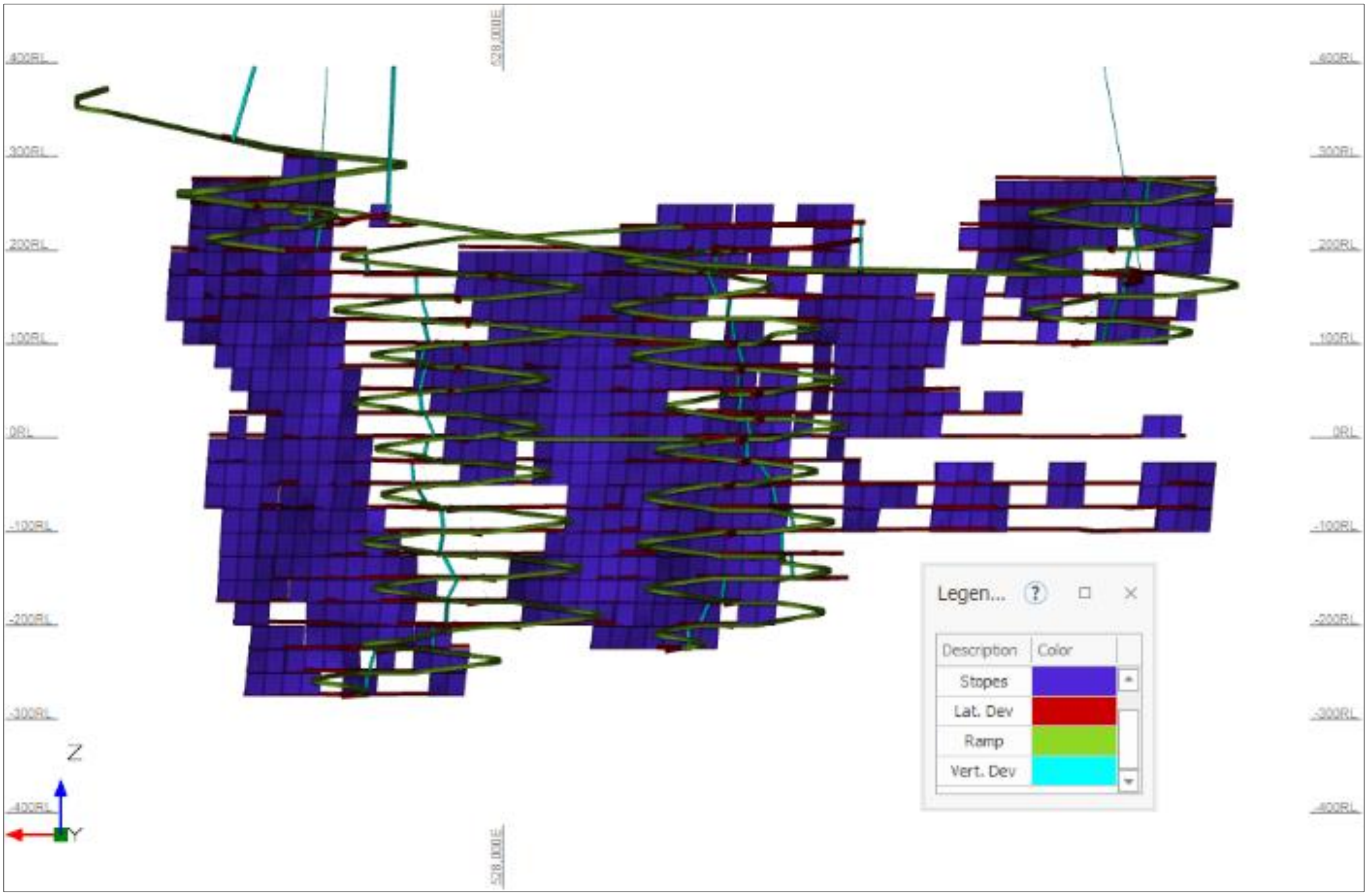
Once it has reached 250 Level the East Ramp splits into two segments; the primary section continues to the west while a secondary branch is used to access stopes in the Main Zone East on the 225 and 200 levels. The primary East Ramp continues to the western extent of the deposit, with two additional segments (Central Ramp and West Ramp) branching off to continue the internal ramping system. The ore zones served by the ramps are as follows:

- East Ramp – used to access the Main Zone East and the eastern section of the Main Zone Central
- Central Ramp – provides access to the remainder of the Main Zone Central, as well as the C Zone Central and Main Zone West (lower portion)
- West Ramp – the smallest of the three ramps, and serves only the upper part of the Main Zone West

Excluding the initial sections of East Ramp documented above, all three ramps will measure 5.0 m W x 5.0 m H. The average design gradient is 15.0% to accommodate the rubber-tired mobile equipment that will traverse the system, with some local deviations to align with the stoping horizons and other infrastructure.

Figure 16-16 presents a longitudinal section view of the underground mine.

Figure 16-16: Long Section of Goliath Underground Mine Design (Looking South)



Source: SRK, 2023

16.5.4 Pillar Design

There are four types of pillars that must be considered for the Goliath underground project:

- Sill pillars: The shallowest three sill pillars (at 275, 125 and 0 levels) are all considered low risk as stress induced damaged within the horizons is predicted to be negligible to minor. The deepest sill pillar is numerically predicted to see more significant stress induced damage potential. The risks associated with stress loading in a sill can be mitigated through strategic mine planning such as an inclined mining front to reduce stress loading in the sill and planning for heavier support systems.

- Inter-lens pillars: As an ore lens is extracted, the rock mass in the immediate HW and FW of the stope becomes relaxed (stress shadowing) which results in conditions that are prone to gravity fall. For this reason, it is favourable to extract FW lens stopes ahead of HW lens stopes and the extraction lag (vertical and horizontal) between FW and HW stopes be minimized where lenses are stacked to minimize the severity of rock mass relaxation prior to extraction of the lagging mining front.

- Mining zone pillars: There are no concerns with mining zone interaction contributing to large-scale pillar instability or adverse stress conditions within the pillars between mining zones.

- Crown pillars: For open stopes below the pit and topographic surface, the minimum crown pillar thickness recommended at this stage of study is 20m, and HW to FW spans should not exceed 10 m for upper most stoping level.

16.5.5 Level Design

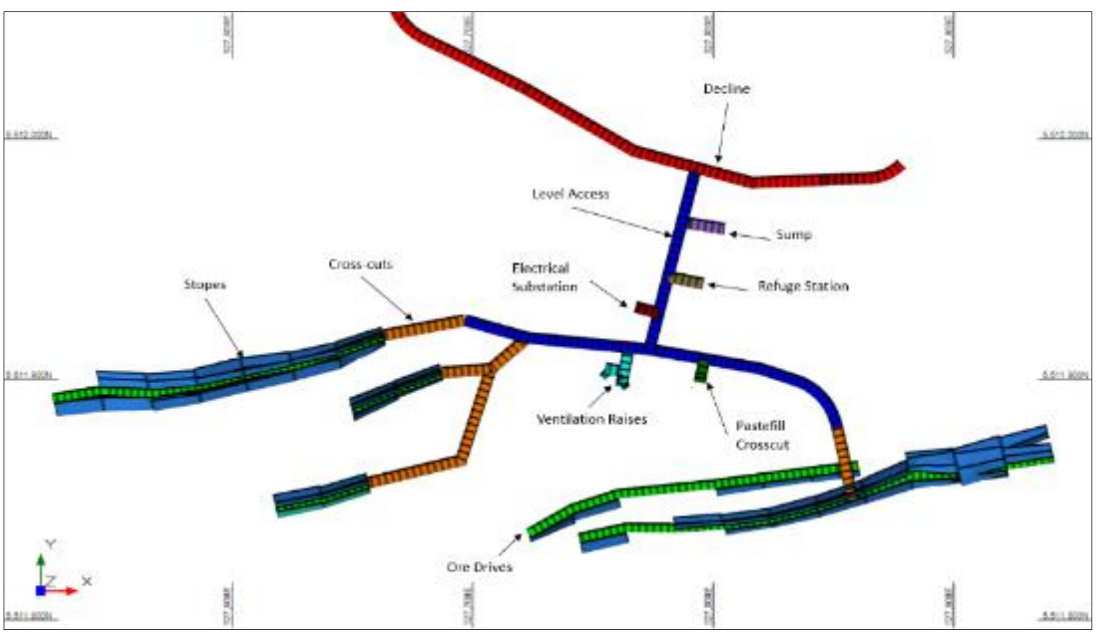
The levels were designed based on a longitudinal retreat mining method, therefore enabling the level designs to be simplified to a degree. As the ore zones are steeply dipping and continuous, it enables level accesses to be designed symmetrically from one level to the next. Since the number of stoping cross cuts can be reduced, it creates opportunities for the level access to house most of the required infrastructure.

Each level access will be equipped with a ventilation raise access, emergency escape way access, paste fill station, de-watering sump, and electrical sub-station cut-out. Each of the infrastructure aspects has been designed such that there is a minimum 10 m offset from one infrastructure element to another.

The footwall drives are oriented in a sub-east-west direction and follow the contour of the orebody, with a 20 m standoff distance allowing for safe passage when stoping activities are occurring nearby. Along the footwall drives, re-muck bays will be placed strategically to support the material handling activities. All crosscuts are designed to intersect the stopes perpendicularly while the ore drives follow the trend of mineralization.

A plan view of a typical level in the Central Zone is depicted below in Figure 16-17.

Figure 16-17: Typical Level Design (Central Zone)



Source: SRK, 2023

16.5.6 Production Schedule

The strategy used for scheduling mine development and stoping includes targeting stoping fronts and working levels for the Main Zone – East and Central. The ventilation circuit is of high priority since it controls the production start; therefore, the schedule ensures that sufficient balance between ramp, level, infrastructure, and ore development is completed.

Once a stoping front has been established, the next priority is to continue developing downwards and towards the west to complete the next ventilation circuit leg. Once full production is achieved in the first two mining blocks (Main Zone – East and Central), the ramp development continues downwards with production following closely. The goal is to maintain steady production while working bottom up, ensuring sequence and geotechnical constraints are followed.

16.5.6.1 Ore and Waste Schedule

The mine plan includes a total run-of-mine extraction of 3.78 Mt, with a peak annual tonnage of 494 kt in Year 7. Years 1 and 2 are ramp-up years, with commercial production achieved in Year 3. Production steadily declines following the peak in Year 7 due to a lower number of active workplaces, with ramp down beginning in Year 12. Closure of the underground mine is scheduled to occur in Year 13.

The underground mine is scheduled to operate 365 days per year with two 12-hour shifts per day. Given the absence of significant underground infrastructure and a relatively short mine life, no planned maintenance periods (i.e., shutdowns) have been built into the schedule.

Table 16-36 through Table 16-38 provide summaries of the annual production schedule during the life of mine period.

16.5.6.2 Development Schedule

The development activities are largely front-loaded in the LOM schedule to activate as many mining faces as possible and thus enable a higher production rate sooner. During the peak development phase (Years 3 through 5), five development crews will be active. To promote efficiency, each crew will have its own dedicated jumbo, LHD, and bolter, with the crews sharing emulsion loading units. When not active in new development headings, these crews will be used for wall and back slashes and rehabilitation efforts as required.

Within the development plan, a growth allowance has of 5% has been determined to account for capital infrastructure excavations that have not been included in the design, such as safety bays, re-mucks, storages, and backslashes for truck loading (at ramp or re-muck intersections).

The annual development requirements are depicted below in Table 16-39 (annual advance per zone) and Table 16-40 (annual advance by type).

Table 16-36: Goliath Underground – Life-of-Mine Total Material Excavation per Annum (in kt)

Mine Schedule	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Total Tonnage	6,681.3	230.4	412.8	729.0	863.7	902.0	851.0	711.1	413.2	368.3	485.8	367.9	246.9	99.3
Total ROM	3,775.8	17.6	72.2	219.5	323.3	409.4	410.6	494.4	390.1	368.1	374.5	357.9	245.0	93.4
C Zone Central	410.4	-	-	17.8	5.5	13.8	6.8	40.7	41.4	33.7	61.1	71.4	86.7	31.4
Main Zone Central	1,571.7	-	11.9	94.3	145.3	213.0	189.5	238.9	138.4	188.8	148.2	97.3	94.9	11.2
Main Zone East	1,276.0	17.6	60.3	92.8	134.6	106.2	177.2	142.9	142.9	133.3	113.3	98.3	40.0	16.7
Main Zone West	517.7	-	-	14.5	37.9	76.4	37.1	71.8	67.4	12.3	51.9	91.0	23.4	34.1
Total Waste	2,905.5	212.8	340.6	509.6	540.4	492.6	440.4	216.7	23.0	0.3	111.3	9.9	2.0	5.9
C Zone Central	301.2	-	27.8	39.4	32.5	30.0	35.2	103.4	9.5	0.2	14.5	6.7	2.0	-
Main Zone Central	970.0	-	79.8	205.0	199.6	235.9	169.2	75.7	3.1	-	1.7	-	-	-
Main Zone East	1,265.2	212.8	231.5	244.5	122.4	199.5	225.1	20.7	2.8	-	-	-	-	5.9
Main Zone West	369.0	-	1.5	20.6	185.9	27.3	10.9	17.0	7.6	0.1	95.1	3.2	-	-

Notes: ROM: run-of mine. Source: SRK, 2023.

Table 16-37: Goliath Underground – Life-of-Mine Run-of-Mine per Annum (in kt) by Source and Zone

Mine Schedule	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Total ROM	3,775.8	17.6	72.2	219.5	323.3	409.4	410.6	494.4	390.1	368.1	374.5	357.9	245.0	93.4
Stopes	3,258.8	-	43.1	139.0	255.5	313.6	322.9	393.6	372.9	368.1	358.2	353.6	245.0	93.4
C Zone Central	342.9	-	-	17.8	3.2	-	3.8	3.6	40.0	33.7	54.6	68.1	86.7	31.4
Main Zone Central	1,356.8	-	-	47.8	111.7	166.7	159.1	193.3	138.0	188.8	148.2	97.3	94.9	11.2
Main Zone East	1,097.0	-	43.1	58.9	122.3	77.7	126.2	130.4	136.9	133.3	113.3	98.3	40.0	16.7
Main Zone West	462.1	-	-	14.5	18.3	69.2	33.9	66.3	58.0	12.3	42.2	89.9	23.4	34.1
Development	517.0	17.6	29.1	80.5	67.8	95.8	87.7	100.8	17.2	-	16.2	4.4	-	-
C Zone Central	67.5	-	-	-	2.3	13.8	3.0	37.1	1.4	-	6.5	3.3	-	-
Main Zone Central	214.9	-	11.9	46.5	33.6	46.3	30.4	45.6	0.5	-	-	-	-	-
Main Zone East	179.0	17.6	17.2	34.0	12.3	28.5	51.0	12.5	6.0	-	-	-	-	-
Main Zone West	55.7	-	-	-	19.5	7.1	3.3	5.5	9.4	-	9.7	1.1	-	-

Notes: ROM: run-of mine. Source: SRK, 2023.

Table 16-38: Goliath Underground – Life-of-Mine Gold Production by Zone

Mine Schedule	Units	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Total ROM	kt	3,775.8	17.6	72.2	219.5	323.3	409.4	410.6	494.4	390.1	368.1	374.5	357.9	245.0	93.4
Au Grade	g/t	3.03	2.92	3.50	3.08	2.84	3.15	3.45	3.08	3.04	2.91	2.69	2.87	2.97	3.02
Au Mined	koz	367.6	1.7	8.1	21.8	29.6	41.5	45.6	49.0	38.1	34.4	32.4	33.0	23.4	9.1
C Zone Central	kt	410.4	0.0	0.0	17.8	5.5	13.8	6.8	40.7	41.4	33.7	61.1	71.4	86.7	31.4
Au Grade	g/t	2.47	0.00	1.81	0.00	1.99	2.06	2.22	0.00	2.32	2.61	2.23	2.28	2.66	2.45
Au Mined	koz	32.6	0.0	0.0	1.1	0.3	1.1	0.5	2.9	3.4	2.9	5.2	6.1	6.7	2.4
Main Zone Central	kt	1,571.7	0.0	11.9	94.3	145.3	213.0	189.5	238.9	138.4	188.8	148.2	97.3	94.9	11.2
Au Grade	g/t	2.80	0.00	3.44	2.92	3.43	2.75	2.57	2.61	2.93	2.48	2.89	2.78	2.45	2.90
Au Mined	koz	141.5	0.0	1.2	9.0	11.9	20.0	19.8	20.6	11.9	14.6	12.9	9.0	9.5	1.0
Main Zone East	kt	1,276.0	17.6	60.3	92.8	134.6	106.2	177.2	142.9	142.9	133.3	113.3	98.3	40.0	16.7
Au Grade	g/t	3.56	2.93	3.26	3.65	2.90	4.52	4.42	4.15	3.39	3.16	2.93	2.38	3.22	3.73
Au Mined	koz	146.0	1.7	7.0	10.4	13.7	13.3	21.1	18.9	16.8	15.9	10.4	10.3	4.7	1.9
Main Zone West	kt	517.7	0.0	0.0	14.5	37.9	76.4	37.1	71.8	67.4	12.3	51.9	91.0	23.4	34.1
Au Grade	g/t	2.85	0.00	0.00	2.77	3.08	2.94	2.91	1.94	3.02	2.41	2.18	2.39	3.31	3.33
Au Mined	koz	47.5	0.0	0.0	1.2	3.6	7.2	4.2	6.6	5.9	1.0	3.9	7.5	2.5	3.8

Notes: ROM: run-of mine. Source: SRK, 2023

Table 16-39: Goliath Underground – Annual Development Advance per Zone (meters)

Zone	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
C Zone Central	6,310	0	400	560	570	790	580	2,620	210	0	350	170	40	0
Main Zone Central	19,830	0	1,360	4,140	3,880	4,690	3,380	2,290	70	0	20	0	0	0
Main Zone East	22,600	2,980	3,930	4,420	2,160	3,660	4,600	600	170	0	0	0	0	90
Main Zone West	7,430	0	20	300	3,430	820	260	440	330	0	1,750	80	0	0
Total	56,180	2,980	5,720	9,410	10,040	9,960	8,810	5,950	780	10	2,130	260	40	90

Source: SRK, 2023

Table 16-40: Goliath Underground – Annual Development Advance by Type (meters)

Zone	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Capital														
Lateral	16,130	780	2,120	3,070	3,260	2,360	3,130	680	0	0	610	50	0	70
Ramp	15,260	1,590	2,450	3,220	2,930	2,930	1,660	0	0	0	480	0	0	0
Vertical	2,690	120	410	400	670	490	460	50	0	0	90	0	0	0
Capital Subtotal	34,080	2,500	4,980	6,690	6,860	5,780	5,240	730	0	0	1,180	50	0	70
Operating	22,090	490	740	2,720	3,180	4,190	3,570	5,210	780	10	950	200	40	20
Total	56,170	2,990	5,720	9,410	10,040	9,970	8,810	5,940	780	10	2,130	250	40	90

Source: SRK, 2023

16.5.7 Mining Sequence

The following section provides details of the Underground mine sequencing of the Life of Mine plan that has been developed to support the mineral reserve estimate. Due to the nature of the deposit, the mine is separated into three distinct mining sectors: Main Zone East, Main Zone Central, and C Zone Central. Each zone is separated by different mining blocks where sub-level open stoping mining method is utilized.

The extraction of each zone is based on the higher-grade areas and the proximity to the main ramp. A mining block within a zone consists of approximately four sublevels that are comprised of stopes that will be mined via holes drilled in a downward direction, with the top sublevel of the mining block mined via up holes.

The sublevel open stoping mining method at Goliath is executed as follows:

- Top and bottom sublevels are established in 25m intervals for each mining zone.
- All support infrastructure for the mining block must be in place for the mining block to begin production.
- Stope preparation activities (e.g., installation of air and water lines, brow markers, remote stands, etc.) are completed to all of for the slot raise to be drilled, followed by production drilling. Drilling and blasting progresses using a longitudinal retreat methodology from the farthest stope on the sublevel.
- After production drilling is completed along with any hole cleaning or re-drills, blasting preparation will take place. The slot raise will be brought up first to provide void space necessary to blast adjacent rings.
- LHDs will muck the broken ore from the bottom sublevel and load the 45 t class haul trucks, which will haul the material to the crusher on the surface.
- When the stope is completely mucked out, a backfill barricade will be installed and the backfill plug will be poured (approximately 30% of the stope by volume).
- After the backfill plug has cured, the remainder of the backfill body will be poured.
- Upon completion and curing of the backfill plug, the adjacent stope in the mining sequence can begin stope preparation and production drilling. Blasting can only occur after the predecessor stope is completely backfilled and cured.

16.5.8 Material Handling

Mineralized material produced from stoping and development processes will be trammed to level re-mucks for short-term storage or directly side-loaded into haulage trucks at the ramp intersection. All mineralized material will be hauled to the surface ROM pad, where it will be dumped into the hopper above the conveyor that will feed the mill.

The waste created during the development process will be brought to surface and stockpiled with the waste generated during open pit mining. When possible, development waste will remain underground to be used as backfill to minimize operating costs.

All underground supplies and materials will be stored in the level storage bays or other designated locations. As mining progresses, the storages should be regularly evaluated to ensure they are still crucial for operations and are re-purposed as necessary to minimize infrastructure costs.

16.5.9 Ground Support

Based on empirical and kinematic analyses it has been concluded that standard drift support of 2.4 m rebar in the back and 1.8 m rebar in the walls on a 1.2 m square pattern with weld wire mesh be suitable. Support should be installed to within 1.5 m of the floor. It is assumed that shotcrete will be required in 2% of standard development. For intersections (up to 8 m span) it is appropriate to assume 4 m long 25 tonne swellex or single strand cables on a 1.5 x 1.5 m pattern OR 4 m double strand cable bolts on a 2 m x 2 m pattern for the PFS study. Shotcrete is planned for all intersections below 400 m depth and in ore sill intersections below 0L.

The drawpoint support will consist of standard ground support with additional 3 rings of 3 m long single strand cable bolts or 25 tonne inflatable bolts with 2 holes per ring across the drift back.

16.5.10 Backfill

Once a stope has been fully mucked and declared empty, it will be backfilled such that mining activities in adjacent stopes can continue. A cemented rock fill (CRF) plant was selected in the Goliath Gold Complex Preliminary Economic Assessment (PEA) (Ausenco, 2021) for the purposes of supplying backfill for stope voids in order to maintain ground stability and to promote higher mining recovery. As well, the CRF plant option carries with it a much lower capital expenditure compared to a hydraulic or paste backfill plant.

This option was carried forward into the early stages of the PFS but was later replaced by paste backfill (pastefill) following a high-level trade-off study. Other options considered in the trade-off study included hydraulic and paste backfills, as well as cemented aggregate fill (CAF). CAF is similar to CRF except that rather than utilizing waste material sourced directly from underground development or open pit stripping activities, the waste is conditioned/sized to provide a more ideal gradation curve.

Pastefill was the chosen alternative for multiple reasons, chief among them:

- It is easier to manage from an operational standpoint (as compared to CAF and CRF),
- It requires a lower unconfined compressive strength (UCS) to achieve the same HW/FW span than CAF or CRF, and
- It allows for a portion of the mill tailings to be pumped back underground, reducing the amount of environmental risk posed by long-term surficial storage.

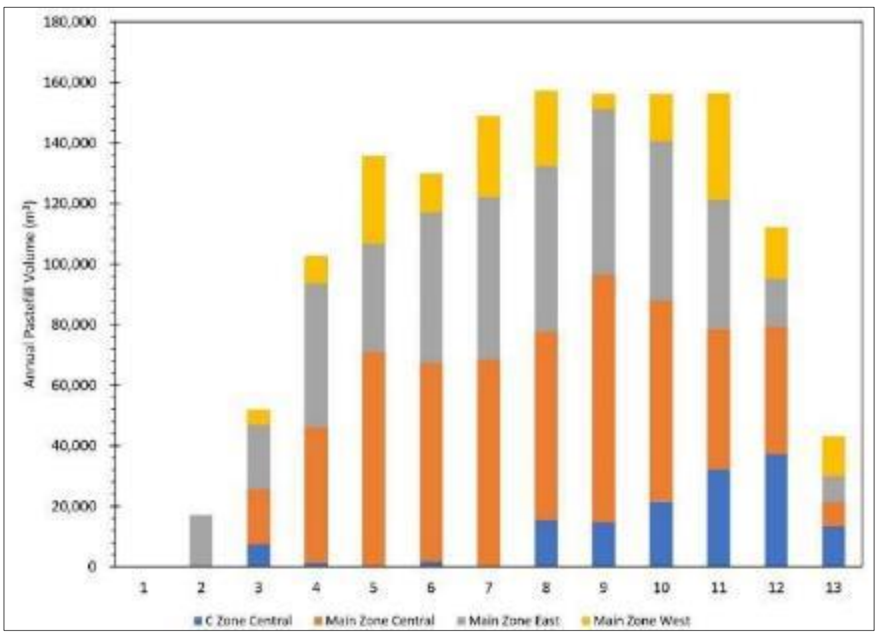
Two operating scenarios were studies for cost estimation purposes. The first would be a traditional owner-operated model, with the project responsible for the capital expenditures associated with the plant. To align with the project's goals, a second option was reviewed in which the backfilling operation would be carried out by a contractor, with the capital expenditure significantly reduced. When comparing the discounted capital and operating costs over the 13-year life of mine period, the contracted backfill option provides better economics and thus that option has been used in this study.

The stope void will be filled in two stages: the initial plug portion, followed by the body portion. A delay of four days has been assumed between when the plug finishes and the curing of the plug section. For scheduling purposes, it has been assumed that 30% of the mined stope volume (i.e., considering the effects of external dilution and mining recovery on the shape) will constitute the plug with the remaining 70% forming the body. An average binder content of 4.5% has been assumed for cost estimation purposes and will be confirmed with further testing in later stages of study. Strength targets paste backfill are 225 kPa for vertical wall exposures and 1000 kPa for undercut exposures in typical 4 m stope spans.

When analyzing the placement of the fill plant, two general areas were analyzed. The first location was adjacent to the Goliath Gold Complex processing plant; the second was just outside of the Goliath open pit northwest wall, essentially directly above the underground workings. The second site was chosen as it improves underground hydraulics; compared to the first location, lower pumping pressures at the second location are required to transport pastefill of the same solids concentration to the stopes.

The pastefill plant capacity has been designed with an annual utilization of 45%, which will allow for the downtime required for plant maintenance, underground pipe changes, waiting for stopes to be ready underground, etc. Figure 16-18 depicts the annual volumes of pastefill required for each of the mining zones.

Figure 16-18: Annual Pastefill Volume Requirements per Zone



Source: SRK, 2023

16.5.11 Ventilation

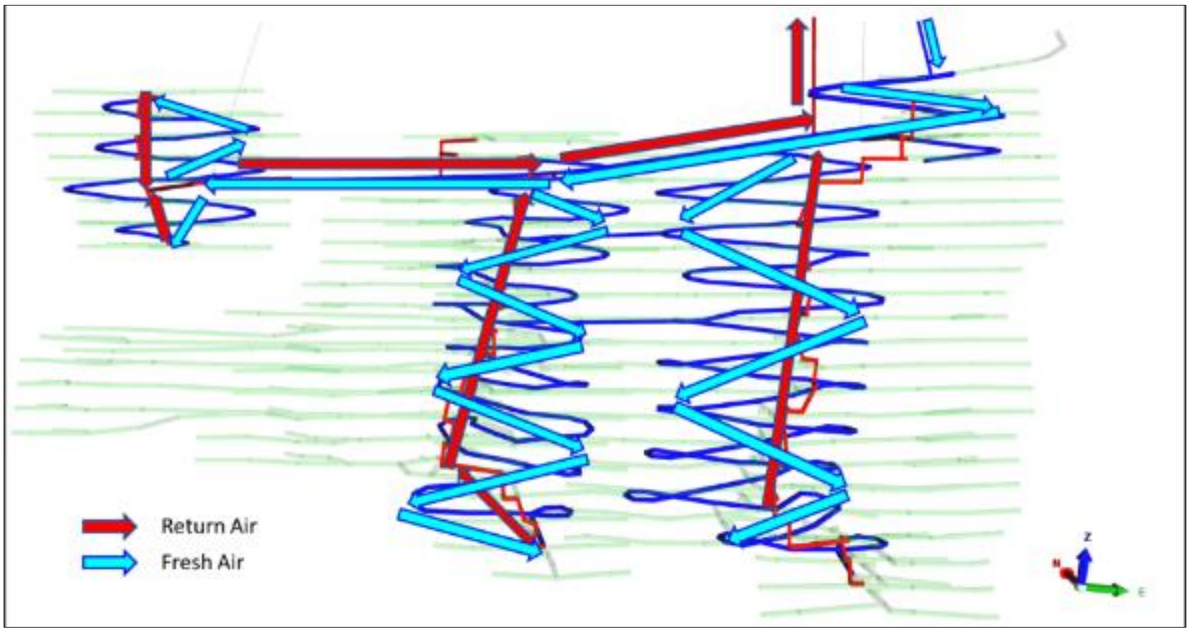
16.5.11.1 Introduction

Ventilation system analysis and network development were completed as part of the underground mine design. This set of design requirements were developed for the ventilation system such that it could adequately support the proposed equipment load and planned operating activities operating during the life of mine period.

16.5.11.1.1 Mine Layout

The primary ventilation system is driven by an underground main fan which is connected to an intake raise to surface. This intake raise will have heaters installed on surface at the raise collar. From the intake fan, air is supplied to the main decline and flows to each active area via the ramp system. A series of exhaust drop raises connect to every level and extract dust and particulates from the working areas. Exhaust raises connect to a primary ventilation drive which directs air to the primary exhaust raise to surface. Figure 16-19 depicts the layout of the primary ventilation system.

Figure 16-19: Primary Ventilation Circuit



Source: SRK, 2023

16.5.11.2 Design Criteria

The ventilation system has been designed based on the requirements of the mobile equipment as it evolves over the mine's life. The airflow requirements were evaluated at multiple stages, with assumptions made for machine availability and utilization. A contingency of 25% was applied to the airflow volumes to allow for potential fleet growth and leakage. Table 16-41 presents the airflow requirements based on the peak period (Q3 of Year 6).

Table 16-41: Goliath Underground – Airflow Requirements

Equipment Type	Diesel Engine Power (kW)	Unadjusted Airflow (0.06 m³/s/kW) (m³/s)	Availability (%)	Utilization (%)	Minimum Required Airflow (m³/s)	# of Eq. Y6 2031 Q3 (Max. Eq. Load)	Total Minimum Required Airflow (m³/s)
Development LHD	265	15.9	100%	80%	12.7	5	63.6
Production LHD	265	15.9	100%	80%	12.7	3	38.2
Haul Truck	405	24.3	100%	70%	17.0	4	68.0
Bolter	-	-	100%	10%	-	5	-
Wet Shotcrete Sprayer	-	-	100%	25%	-	2	-
Transmixer	-	-	100%	50%	-	1	-
Grader	-	-	100%	25%	-	1	-
Mobile Rock Breaker	-	-	100%	25%	-	1	-
Emulsion Loader	-	-	100%	25%	-	3	-
Scissor Lift	-	-	100%	10%	-	2	-
Boom Truck	-	-	100%	10%	-	1	-
Service Truck	-	-	100%	50%	-	1	-
2 Boom Jumbo	110	6.6	100%	10%	0.7	5	3.3
Production Drill	74	4.4	100%	10%	0.4	2	0.9
Skidsteer	73	4.4	100%	50%	2.2	2	4.4
Personnel Carrier	95	5.7	100%	30%	1.7	10	17.1
Forklift	64	3.8	100%	60%	2.3	1	2.3
Underground Explosive Magazines							24.3
Leakage & Fleet Growth (25%)							43.0
Total Minimum Airflow							236.6

Note: Values may not sum due to rounding. Source: SRK, 2023.

16.5.11.2.1 Ventilation Raises and Vertical Development

Two types of ventilation raise designs were used in developing the underground ventilation - system conventional drop raises and raise bores. It was assumed that conventional drop raises were used for all raises in the mine. The primary exhaust raise and the primary intake raise from surface were designed to 4.0 m by 4.0 m across. Internal exhaust raises (i.e., from one production level to another) were designed at 3.5 m by 3.5 m and have Safescape-style escapeways in them. Escapeway raisebores, were designed to 1.2 m in diameter, and assumed to have Safescape style escapeways in them. These raises were primarily located on each level near the connections with the ramp.

Table 16-32, shown earlier in Section 16.5.2.2, lists the various types of vertical development present in the underground mine design and their nominal diameters.

16.5.11.2.2 Air Velocities

Air velocity limitations vary according to airway type. In areas such as return airways and shafts where personnel are not expected to work, higher velocities are acceptable. Airway velocities typically used by SRK for various airway types are shown in Table 16-42. Air velocity limits and recommended values for travelways are established to accommodate work and travel by personnel and equipment, optimization of dust entrainment, and temperature regulation.

Table 16-42: Recommended maximum air velocities for various airway types

Airway Type	Maximum Air Velocity (m/s)
Travelways	6
Primary Ventilation Intake and Exhaust Entries	10
Primary Ventilation Shaft or Raise	20
Ventilation Shaft with Conveyance Or Escape	10

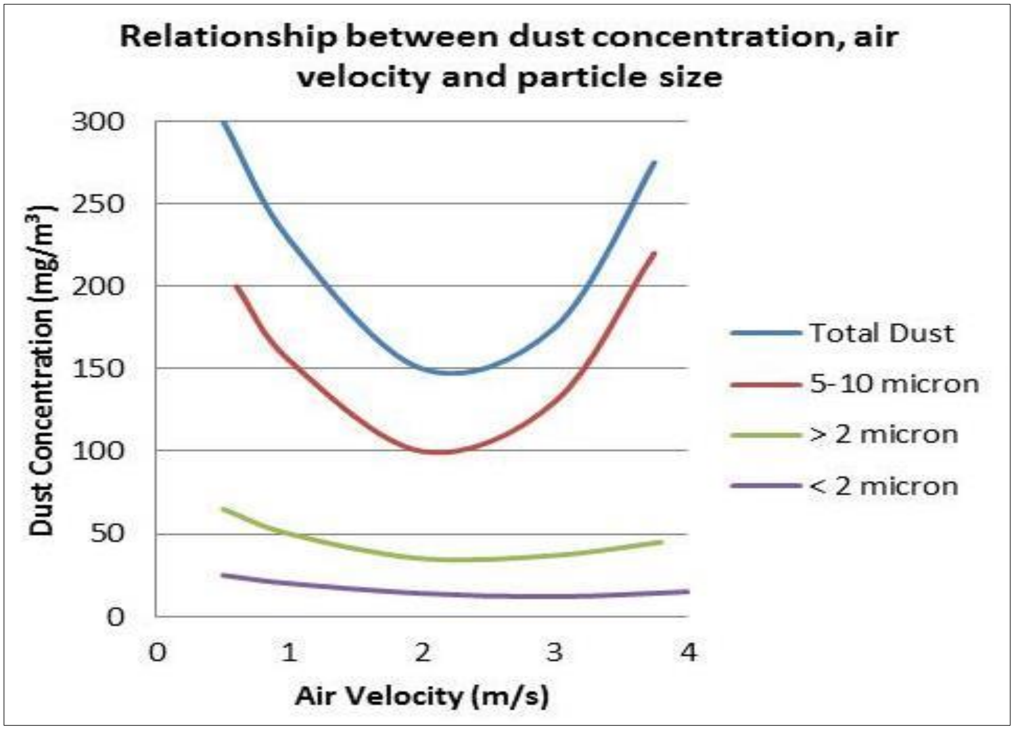
Source: SRK, 2023

Low airflow volumes may insufficiently dilute/remove airborne dust, but high air velocities will entrain larger dust particles, resulting in a potentially hazardous environment for personnel. An air velocity between 1.5 m/s and 2.5 m/s should be maintained to minimize dust in areas affected by dust generation. Air velocities in this range represent the provision of sufficient airflow to dilute the dust, without excessive air velocity to re-entrain dust (Figure 16-20).

In general, the minimum air velocity in a heading (without diesel equipment in operation) is based on the perceptible movement of airflow, which is between 0.3 m/s and 0.5 m/s. The higher value of 0.5 m/s was used for this analysis.

Air velocities in upcasting shafts should be maintained outside of the range of 7 m/s to 12 m/s to avoid water blanketing. Variability of the number of equipment and mining locations throughout the mine life makes this difficult to plan for in advance by manipulating the size of raises. A solution to the problem may be to slightly increase or decrease flow in problematic shafts. This may require some shifting of mining activities. Modeling parameters for different types of airways are shown in Table 16-43, while infrastructure resistances used are shown in Table 16-44.

Figure 16-20: Generalized Relationship between Air Velocity and Airborne Dust Concentration



Source: Vutukuri and Lama, 2023.

Table 16-43: Ventilation modelling parameters by airway type

Airway Type	Diameter (m)	Width (m)	Height (m)	Profile	Friction Factor (kg/m³)	Min-Max Velocity (m/s)
Primary Ventilation Decline	N/A	6	7	Arched	0.0120	0.5 – 7.0
Standard Decline	N/A	5	5	Arched	0.0120	0.5 – 6.0
Vent Drift	N/A	5	5	Arched	0.0120	0.0 – 10.0
Level Access	N/A	5	5	Arched	0.0120	0.0 – 6.0
Level to Level Drop Raise	N/A	3.5	3.5	Square	0.0150	0.0 – 20.0
Surface Raise	N/A	4.0	4.0	Square	0.0150	0.0 – 20.0
Escape Raise	1.2	N/A	N/A	Round	0.0085	0.0 – 10.0

Source: SRK, 2023

Table 16-44: Ventilation Infrastructure Fixed Resistances

Description	Resistance (Ns^2/m^8)
Bulkhead	500
Airlock Equipment Doors	20
Single Equipment Door	10

Source: SRK, 2023

16.5.11.2.3 Escapeways

Two primary escapeways to surface are planned to be separate from the main ventilation system; therefore, minimal airflow was modeled through these raises. Other escapeways are planned as Safescape ladderways located within the level-to-level drop raises. Due to the higher resistances through the escapeways, it is expected that the air velocity will be slower through the escapeways than through the larger surrounding vent raises. In the even that velocities through the escapeways are too high there are options to isolate them from the rest of the raise (see Figure 16-21).

Figure 16-21: Safescape Zipper Jacket at Base of Ladderway

Source: SRK, 2023

16.5.11.2.4 Shock Losses

Additional resistances need to be added to the model for airways which have sharp bends, merge, or diverge. These losses, in which mechanical energy is consumed by eddies created in a change in airflow momentum, are referred to as "shock losses", and they are expressed by a dimensionless value (X) and are based on standard charts (Table 16-45).

Table 16-45: Shock Loss Values

Configuration	Shock Loss (X)
Exit	1.0
Entrance	0.5
90° Bend	1.2

Source: SRK, 2023

Although shock losses could be applied to all branches in the network, they are only significant in high-velocity main airways such as fresh air and exhaust raises, fan installations, and ventilation transfer levels.

16.5.11.3 Decline Development Auxiliary Ventilation Design

At certain points in the mine development schedule, decline development headings will require long lengths of auxiliary ventilation. For this reason, a hardline/plastic duct system is recommended for the decline development. The benefit of hardline ducting is reduced total resistance and leakage through the system and allowance for the system to withstand negative pressures when using multiple fans in series. Hardline ducting does, however, have some drawbacks such as higher unit costs, space requirements for shipping, and longer installation time. Traditional lay flat ducting is not recommended for the declines because it would require several fans running in series which increases the risk of negative pressure on the ducting. A negative pressure on lay flat or fabric ducting will cause it to collapse and make the auxiliary system inoperable.

SRK proposes using plastic hardline duct to reduce storage and shipping requirements over more traditional fiberglass duct. The inlet of the duct should be positioned at least 10 m upstream of the flow through ventilation system. The fan needs to have an inlet bell to reduce shock losses. Hardline duct is then installed at the exhaust of the fan. A fan needs to be installed approximately every 400 m of duct to meet the minimum airflow requirements.

Based on the drift profile and the largest piece of equipment (e.g., haul truck) in a decline, the maximum duct diameter is 1.52 m. A profile drawing of the duct in the mine airway ventilating for an LHD and Haul Truck is shown in Figure 16-22.

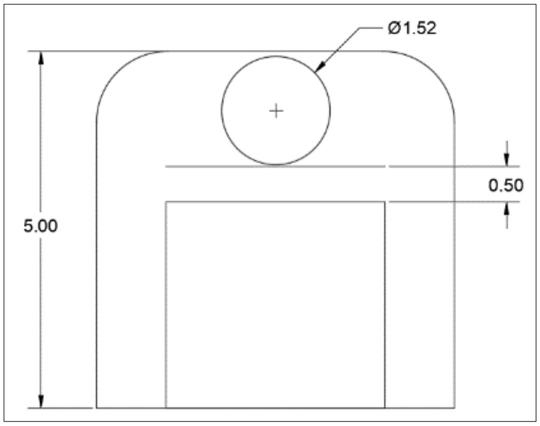
It is estimated that four 100 kW fans paired with a single run of auxiliary duct will be able to supply enough air for each decline auxiliary heading. This assumes layflat ducting will be used with moderate leakage characteristics. Ducting with excessive leakage will need to be repaired to deliver the minimum airflow for LHDs working on the level. Table 16-46 below summarizes auxiliary ventilation system characteristics for one of the longest headings planned the mine.

Table 16-46: Summary of Level Auxiliary Ventilation System

Length of Duct	Airflow at First Fan (m^3/s)	Airflow at Face (m^3/s)	Pressure on Each Fan (kPa)	Number of Fans	Air Power per Fan (kW)	Motor Power per Fan (kW)
Up to 550 meters (LHD)	56.0	40.2	1.33	4	75	100

Source: SRK, 2023

Figure 16-22: Duct in Standard Decline (Dimensions in Meters)

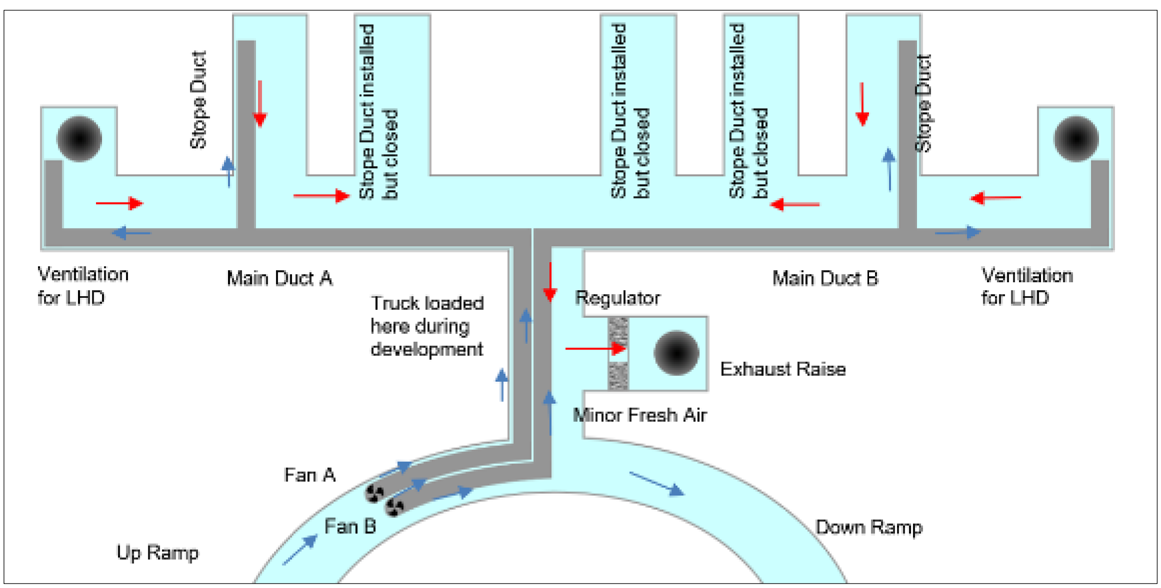


Source: SRK, 2023

16.5.11.4 Level Auxiliary Ventilation Design

For each heading, air enters the level from the ramp. Airflow is drawn through the auxiliary ventilation system (duct line) to the face. The exhaust air courses back across the level toward the exhaust raise. There is a regulator on the exhaust raise that is sized for the level airflow plus some additional airflow to be drawn directly from the ramp/spiral to ventilate the loading of the haul truck (prior to ore pass development). This also keeps the exhaust air segregated from the ramp. An example layout of the auxiliary system on a dead-end level is provided in Figure 16-23. In the figure, air is shown to intake the level via ducting and fans in the ramps. These fans/ducts can also be located on the level, but before the entrance to the exhaust raise depending on the available space.

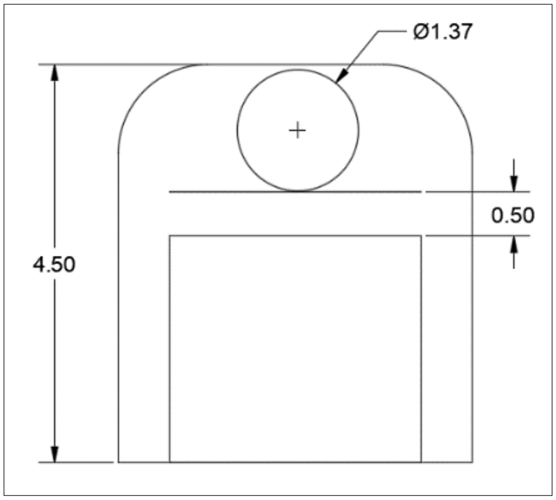
Figure 16-23: Example Dead-End Level Auxiliary Ventilation



Source: SRK, 2023

Based on the drift profile and the largest piece of equipment (LHD) on an active production level, the maximum duct diameter is 1.37 m. A profile drawing of the duct in the mine airway ventilating for an LHD is shown in Figure 16-24.

Figure 16-24: Duct in Mine Airway Profile (Dimensions in Meters)



Source: SRK, 2023

It is estimated that a single 25 kW fan paired with a single auxiliary duct will be able to supply enough air for each auxiliary heading. This assumes layflat ducting will be used with moderate leakage characteristics. Ducting with excessive leakage will need to be repaired to deliver the minimum airflow for LHDs working on the level. Table 16-47 summarizes auxiliary ventilation system characteristics for one of the longest headings planned in the mine.

Table 16-47: Summary of Level Auxiliary Ventilation System

Length of Duct	Airflow at First Fan (m³/s)	Airflow at Face (m³/s)	Pressure on Each Fan (kPa)	Number of Fans	Air Power per Fan (kW)	Motor Power per Fan (kW)
Up to 550 meters (LHD)	19.0	16.0	0.7	1	14	25

Source: SRK, 2023

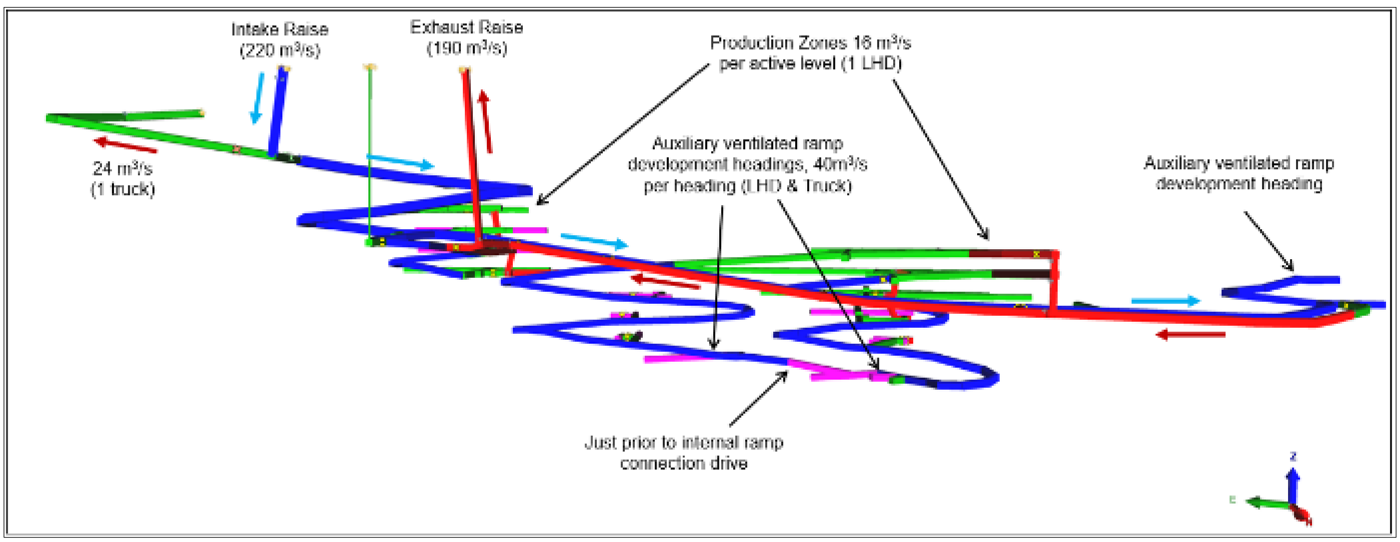
16.5.11.5 Primary Ventilation Model Development

Four ventilation stages were developed at key points in the mine life. Each stage represents a different time period with different equipment requirements and mining development stages based on the mine schedule. Four staged models were completed: Year 3 – 2028 Q3, Year 6 – 2031 Q6, Year 10 – 2035 Q1, and Year 12 – 2037 Q2. For each model, the ventilation system was set up based on the maximum number of equipment that may be in the mine during that stage. For each stage the actual amount of equipment in the mine may vary; however, the maximum amount of equipment for each stage was used to represent potential extreme conditions.

16.5.11.5.1 Year 3 – 2028 Q3: Prior to First Ramp Intersection

The first stage represents a snapshot of the period of mining and development during Year 3 – 2028 Q3. The models shows the extent of mine development just prior to completion of the connection drift between the East Ramp and Central Ramp. After this point, in time airflow will be able to flow more liberally between sets of internal ramps to active areas. At this stage, through flow ventilation is established through the surface intake and exhaust raises but airflow throughout the mine is still largely driven by auxiliary fans to active mining and development areas. Development of the East, Central, and West ramps is largely ventilated from longer auxiliary fan drives during this period of the mine life. Production is occurring at the tops of the East and Central Ramp. The maximum number of haul trucks planned for this stage is 3 while the maximum number of LHDs is 6. The total mine airflow is approximately 220 m^3/s. The early stages of development generally require the greatest demand for auxiliary ventilation systems. displays the ventilation model for Year 3, Q3 with planned scheduled activities for this stage. The main fan operating points for this stage are summarized in Table 16-48.

Figure 16-25: Scheduled Mining and Development Activities for Year 3, Q3



Source: SRK, 2023

Table 16-48: Year 3, Q3 Main Fan Operating Point

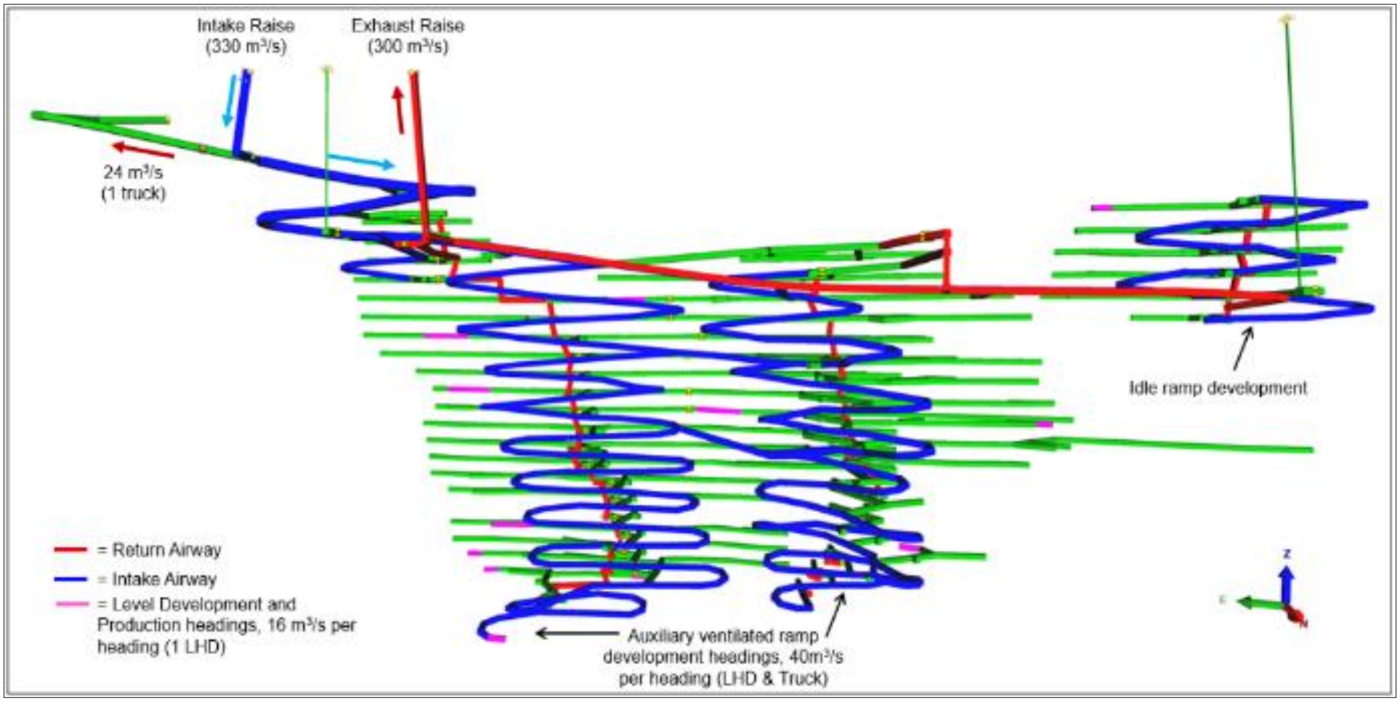
Description	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)[1]	Inlet Density (kg/m^3)
Primary Intake Fan	1.7	220	370	530	1.13

Notes: [1]Assumes 70% fan efficiency. Source: SRK, 2023.

16.5.11.5.2 Year 6 – 2031 Q3: Prior to First Ramp Intersection

The second stage represents a snapshot of the period of mining and development during Year 6 – 2031 Q3. The model shows the extent of mine development at the maximum airflow demand throughout the LOM. At this stage the peak number of LHDs and trucks are operating in the mine (4 trucks and 6 LHDs). The total mine airflow is approximately 330 m^3/s. Active development areas of the mine include the lower East Ramp, lower Central Ramp and lower levels off both the Central and East Ramps. Production areas include upper and central levels off the East and Central Ramp and central levels off the West Ramp. Figure 16-26 displays the ventilation model for Year 6, Q3 with planned scheduled activities for this stage. The main fan operating points for this stage are summarized in Table 16-49.

Figure 16-26: Scheduled Mining and Development Activities for Year 6, Q3



Source: SRK, 2023

Table 16-49: Year 6, Q3 Main Fan Operating Point

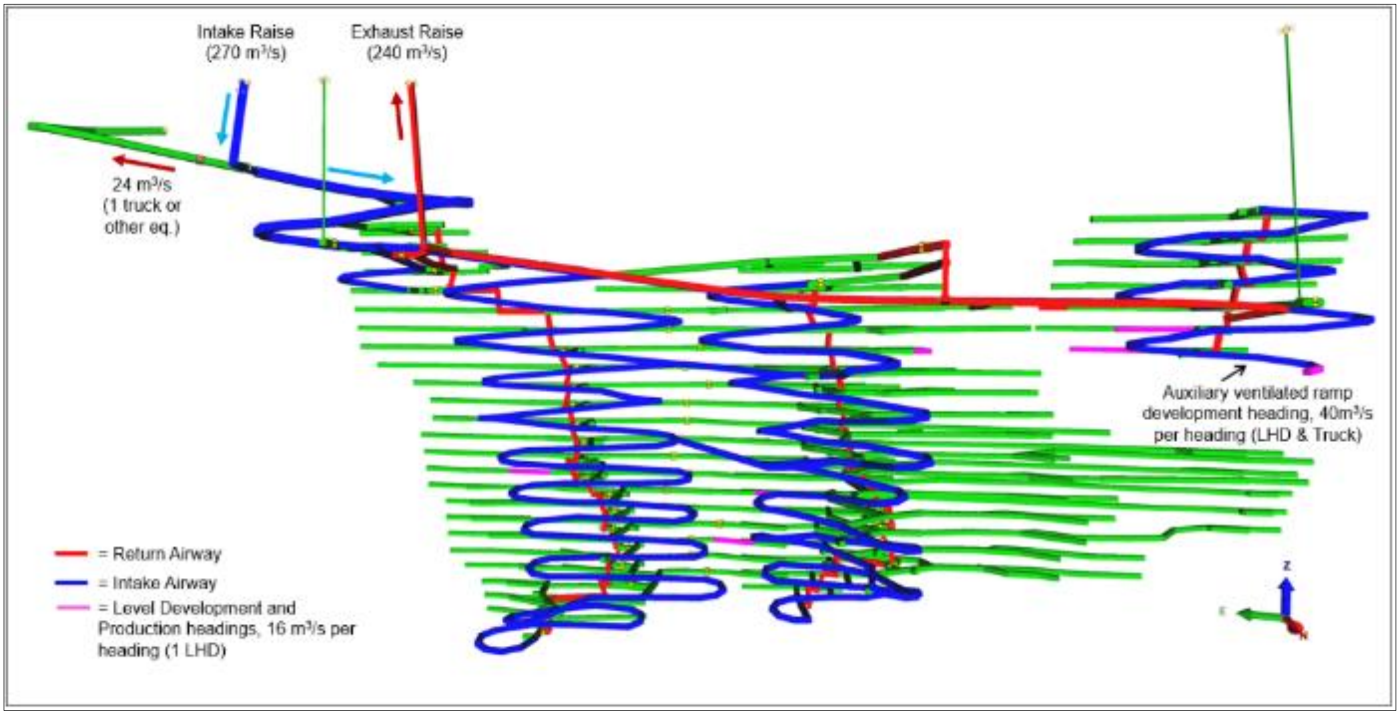
Description	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)[1]	Inlet Density (kg/m^3)
Primary Intake Fan	3.6	330	1,190	1,700	1.13

Notes: [1]Assumes 70% fan efficiency. Source: SRK, 2023.

16.5.11.5.3 Year 10 – 2035 Q1: Development in Lower West Ramp Resumes

The third stage represents the period of mining and development during Year 10 – 2035 Q1. This model shows the mine at a period of relatively high airflow demand. Prior to this stage, lateral and decline development had stopped for 2 years. During this stage, lateral and decline development resumes for approximately a year and a half to finish development of the lower portion of the West Ramp. The maximum number of haul trucks planned for this stage is 2 while the maximum number of LHDs is 5. The total mine airflow is approximately 270 m^3/s. Active development areas of the mine include the lower West Ramp and adjacent levels. Production areas include many different levels throughout the East and Central ramps. Figure 16-27 displays the ventilation model for Year 10, Q1 with planned scheduled activities for this stage. The main fan operating points for this stage are summarized in Table 16-50.

Figure 16-27: Scheduled Mining and Development Activities for Year 10, Q1



Source: SRK, 2023

Table 16-50: Year 10, Q1 Main Fan Operating Point

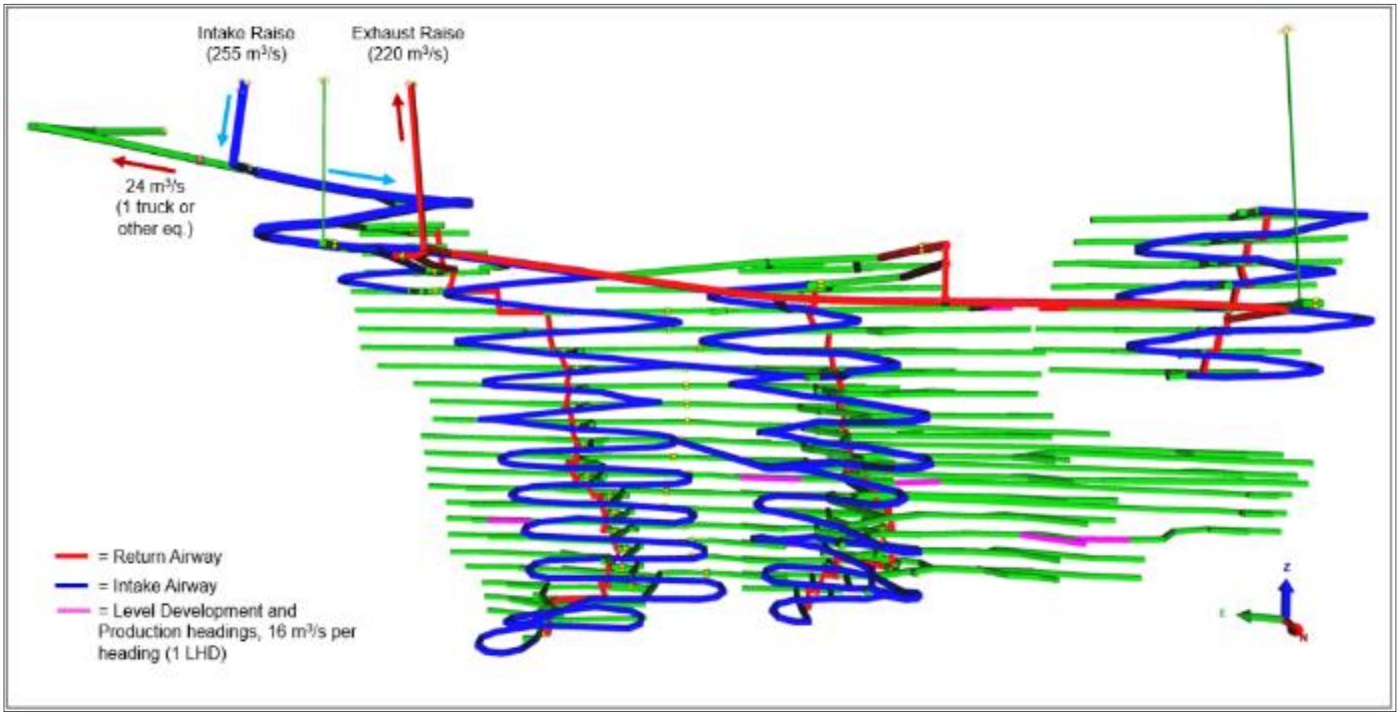
Description	Pressure (kPa)	Quantity (m³/s)	Air Power (kW)	Motor Power (kW)[1]	Inlet Density (kg/m³)
Primary Intake Fan	2.7	270	730	1,040	1.13

Note: [1]Assume 70% fan efficiency. Source: SRK, 2023.

16.5.11.5.4 Year 12 – 2037 Q2: Development in Lower West Ramp Resumes

The fourth stage represents the period of mining and development during Year 12 – 2037 Q2. This model shows the mine at a period of relatively high airflow demand just prior to the production ramp down period at the end of the mine life. After this point in time airflow demands decrease rapidly as the mine finishes production to close the mine. At this point in time, development is complete. The maximum number of haul trucks planned for this stage is 2 while the maximum number of LHDs is 4. The total mine airflow is approximately 255 m^3/s. Production areas include many levels throughout the Central Ramps, one level in the lower East Ramp, and a few levels in the West Ramp. Figure 16-28 displays the ventilation model for Year 12, Q2 with planned scheduled activities for this stage. The main fan operating points for this stage are summarized in Table 16-51.

Figure 16-28: Scheduled Mining and Development Activities for Year 12, Q2



Source: SRK, 2023

Table 16-51: Year 12, Q2 Main Fan Operating Point

Description	Pressure (kPa)	Quantity (m^3/s)	Air Power (kW)	Motor Power (kW)[1]	Inlet Density (kg/m^3)
Primary Intake Fan	2.9	255	740	1,060	1.13

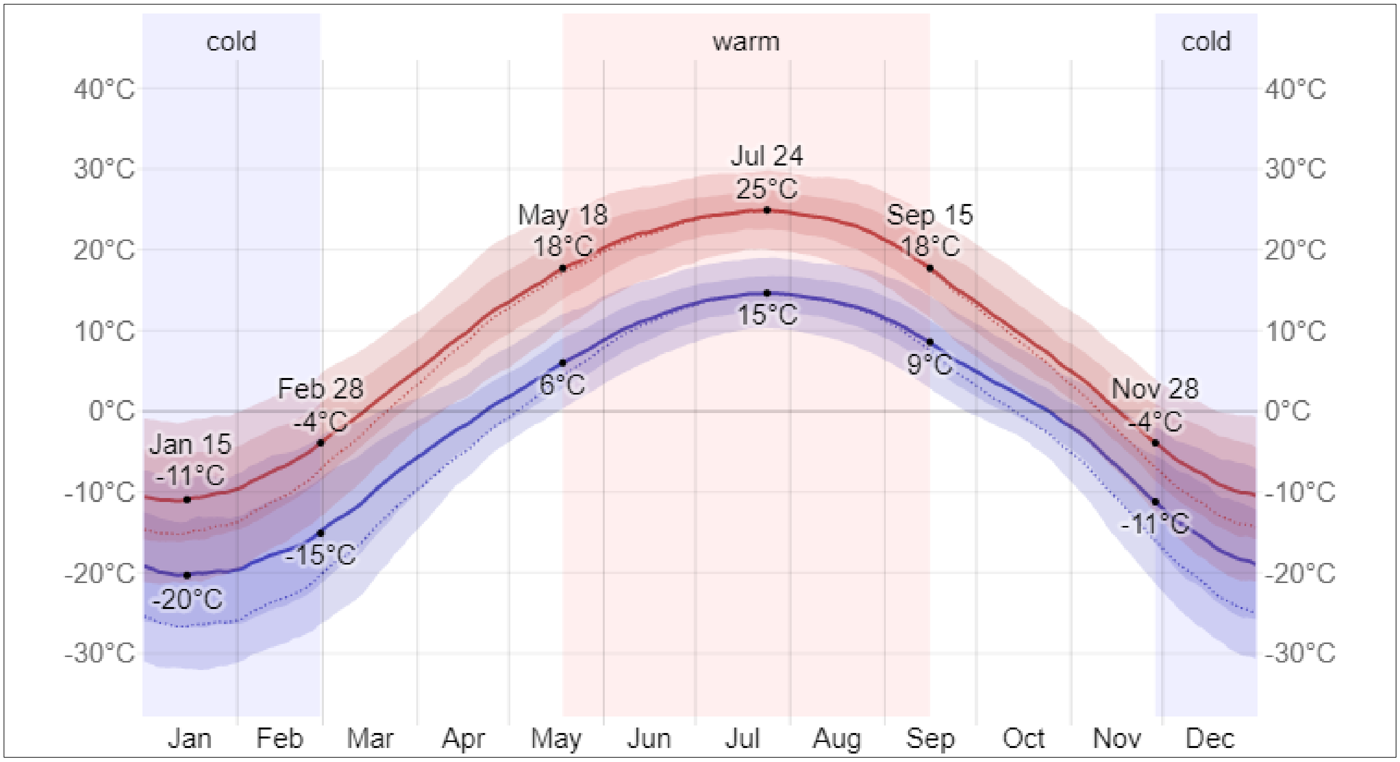
Note: [1]Assume 70% fan efficiency. Source: SRK, 2023.

16.5.11.6 Surface Air Heating

The project site located near Dryden, Ontario, typically experiences freezing temperatures during the months of October through April. A graph of average temperatures at Dryden Airport is provided in Figure 16-29. Intake air should be heated during freezing temperatures to prevent freeze/thaw damage to drifts, prevent icing of travel ways, and protect utilities and plant equipment from freezing. No requirement for air cooling is expected at the Goliath project.

The primary fresh air surface raise will require a direct-fired natural gas heater to heat the intake air to just above freezing (+5°C). Air heating assumed piped natural gas fueling direct-combustion air heaters to the collar of the surface intake raise. Table 16-52 shows the heating degree days and corresponding energy per unit airflow of heating calculated for each month of the year, using historical climate data for Dryden.

Figure 16-29: Average High and Low Temperatures at Mine Site in Dryden, ON



Source: Weatherspark.com via Google Search, 2023.

Table 16-52: Degree Days and Expected Energy Use per Unit Airflow for Surface Air Heaters

Month	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec
Degree Days to 5°C	589	598	289	141	25	0	0	0	3	89	277	515
Air Heating to 5°C ($GJ/m^3/s$) per mo.	62.0	62.9	30.4	14.9	2.6	0.1	0.0	0.0	0.3	9.3	29.2	54.2

Note: Temperature data from Dryden, ON, Canada (WMO: 715270) (92.74W,49.83N). Source: SRK, 2023.

16.5.11.7 Opportunities for Future Studies

- Consider eliminating longer auxiliary ventilated headings by developing drop raises with ramp development.
- Determine the final locations of fixed facilities (magazines, etc.) and determine best ventilation strategies for them.

16.5.12 Underground Infrastructure Facilities

The quantities and distribution of the mine's underground infrastructure are dictated by the life of mine production schedule and are typical of other LHOS mines. Infrastructure planned for the mine includes the following items:

- refuge stations (both permanent and portable)
- latrines
- emergency escapeways
- explosives magazines
- detonator magazines
- shotcrete storages
- definition diamond drilling stations
- fuel bays
- ventilation fans and controls (e.g., bulkheads, regulators, and equipment doors)
- paste backfill receiving and distribution stations
- electrical switch stations and underground power distribution
- automation and communication system hardware
- boreholes for dewatering and distribution of compressed air, water, power, and backfill.

Given the shallow nature of the mine and its ramp access design, it has been assumed that repair and maintenance of the mobile equipment fleet will take place on surface, sharing the maintenance infrastructure with the Goliath open pit mine.

16.5.13 Equipment & Labour Requirements

16.5.13.1 Mobile Equipment

Mining activities at Goliath will be undertaken with a fleet comprised of traditional underground equipment, based on typical productivity values for such equipment and the number of active workplaces at a given time. As the planned mining at Goliath will only reach a depth of approximately 650 m from surface, the potential for concerns with elevated air temperatures is considered low. However, a hybrid fleet of diesel-powered and battery electric vehicles (BEVs) has been proposed for this study. Utilizing BEVs will assist in minimizing the project's scope 1 greenhouse gas (GHG) emissions and provide a healthier environment for underground personnel.

It has been assumed in this study that underground mining will be completed by a contractor that will furnish their own mobile equipment and labour force. The estimated underground mobile equipment fleet is shown in Table 16-53. The fleet peaks in size in Years 4 and 5 as production is ramping up while development activity is at its maximum. The fleet size has been estimated based on the amount of production and development activity in a given quarter, with the values shown below representing the requirements in the year's fourth quarter.

Table 16-53: Goliath Underground – Underground Equipment Fleet per Annum

Equipment Type	1	2	3	4	5	6	7	8	9	10	11	12	13
Two-boom Jumbo	3	4	5	5	5	4	3	1	1	1	1	1	-
Emulsion Loader	2	3	4	4	4	3	3	2	2	2	2	2	1
Development LHD	3	4	5	5	5	4	3	1	1	1	1	1	-
Bolter	3	4	5	5	5	4	3	1	1	1	1	1	-
Production LHD	-	2	3	3	3	3	3	3	3	3	3	3	1
45 t Haulage Truck	1	2	2	4	4	4	2	2	2	2	2	1	1
Production Drill	-	1	2	2	2	2	2	2	2	1	1	1	1
Scissor Lift	1	2	2	2	2	2	2	2	2	2	2	1	1
Boom Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Forklift	-	1	1	1	1	1	1	1	1	1	1	1	1
Shotcrete Sprayer	1	2	2	2	2	2	1	1	1	1	1	1	1
Personnel Carrier	8	8	8	8	8	8	8	8	8	8	8	4	3
Grader	1	1	1	1	1	1	1	1	1	1	1	1	1
Service Truck	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Rock Breaker	-	1	1	1	1	1	1	1	1	1	1	1	1
Skidsteer	1	2	2	2	2	2	2	2	2	2	2	1	-
Total	**26**	**39**	**45**	**47**	**47**	**43**	**37**	**30**	**30**	**29**	**29**	**22**	**14**

Source: SRK, 2023

16.5.13.2 Labour

At peak production, the workforce is anticipated to total approximately 150 people, including roles for mine operations and maintenance, supervision and management, and technical services personnel. Some duties will be shared by the Goliath open pit and underground mines (e.g., mine captain, technical services superintendent, trainer, etc.) with the headcount and related salaries being carried by the open pit.

For most operating labour, the shift schedule will be a 12-hour shift schedule, operating two shifts per day, seven days per week. Direct labour counts have been determined based on the equipment and process performances. Other labour has been based on common practice within the Ontario hard rock underground mining industry, adapted to the mining method, general layout, and infrastructure requirements of the mine plan.

The labour headcounts, segregated by role type, are shown in Table 16-54 through Table 16-56. As with the mobile equipment, the values shown represent the requirements in the year's fourth quarter.

Table 16-54: Goliath Underground – Life of Mine Operating Labour Profile

Role	1	2	3	4	5	6	7	8	9	10	11	12	13
Direct Operating													
Development Miner	6	8	10	10	10	8	6	2	2	2	2	2	-
Development Leaders (Jumbo)	3	4	5	5	5	4	3	1	1	1	1	1	-
Longhole Driller	-	2	4	4	4	4	4	4	4	2	2	2	2
Production Loader/Blaster	-	2	4	4	4	4	4	4	4	2	2	2	1
Development LHD Operator	6	8	10	10	10	8	6	2	2	2	2	2	-
Production LHD Operator	-	6	9	9	9	9	9	9	9	9	9	9	3
Haulage Truck Operator	3	6	6	12	12	12	6	6	6	6	6	3	3
Direct Operating Subtotal	18	36	48	54	54	49	38	28	28	24	24	21	9
Indirect Operating													
Shotcrete Operator	2	4	4	4	4	4	2	2	2	2	2	2	2
Construction Leader	3	4	5	5	5	4	3	1	1	1	1	1	-
Supplies Handling	2	2	2	2	2	2	2	2	2	2	2	2	2
Construction Miner	6	8	10	10	10	8	4	2	2	2	2	2	-
Support Miner	6	8	10	10	10	8	6	2	2	2	2	2	-
Rockbreaker Operator	-	1	1	1	1	1	1	1	1	1	1	1	1
Grader Operator	2	2	2	2	2	2	2	2	2	2	2	2	1
Indirect Operating Subtotal	21	29	34	34	34	29	20	12	12	12	12	12	6
Operating Total	39	65	82	88	88	78	58	40	40	36	36	30	15

Source: SRK, 2023

Table 16-55: Goliath Underground – Life of Mine Maintenance Labour Profile

Equipment Type	1	2	3	4	5	6	7	8	9	10	11	12	13
Fixed Plant Mechanic I	2	4	4	4	4	4	4	4	4	4	4	2	1
Fixed Plant Mechanic II	2	4	4	4	4	4	4	4	4	4	4	2	1
Mobile Mechanic I	2	4	4	4	4	4	4	4	4	4	4	2	1
Mobile Mechanic II	4	8	8	8	8	8	8	8	8	8	8	4	1
Electrician I	2	4	4	4	4	4	4	4	4	4	4	2	1
Electrician II	2	4	4	4	4	4	4	4	4	4	4	2	1
Instrumentation Tech	1	2	2	2	2	2	2	2	2	2	2	1	1
Maintenance Total	15	30	30	30	30	30	30	30	30	30	30	15	7

Source: SRK, 2023

Table 16-56: Goliath Underground – Life of Mine Supervision and Staff Labour Profile

Equipment Type	1	2	3	4	5	6	7	8	9	10	11	12	13
Mine Captain	1	1	1	1	1	1	1	1	1	1	1	1	1
Shift Boss	2	4	4	4	4	4	4	4	4	4	4	4	1
Safety Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1
Yard Attendant	2	2	2	2	2	2	2	2	2	2	2	2	-
Maintenance Superintendent	-	1	1	1	1	1	1	1	1	1	1	1	1
Fixed Plant Mechanical Supervisor	-	4	4	4	4	4	4	4	4	4	4	4	1
Mobile Mechanical Supervisor	2	4	4	4	4	4	4	4	4	4	4	4	-
Electrical Supervisor	2	4	4	4	4	4	4	4	4	4	4	4	-
Senior Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Engineer	-	1	1	1	1	1	1	1	1	1	1	1	-
Rock Mechanics/Ventilation Engineer	2	2	2	2	2	2	2	2	2	2	2	2	2
Mine Technician	1	2	2	2	2	2	2	2	2	2	2	1	1
Ventilation Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Rock Mechanics Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Mine Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Geology Technician/Grade Control	1	2	2	2	2	2	2	2	2	2	2	2	1
Mine Staff Total	18	32	32	32	32	32	32	32	32	32	32	31	13

Source: SRK, 2023

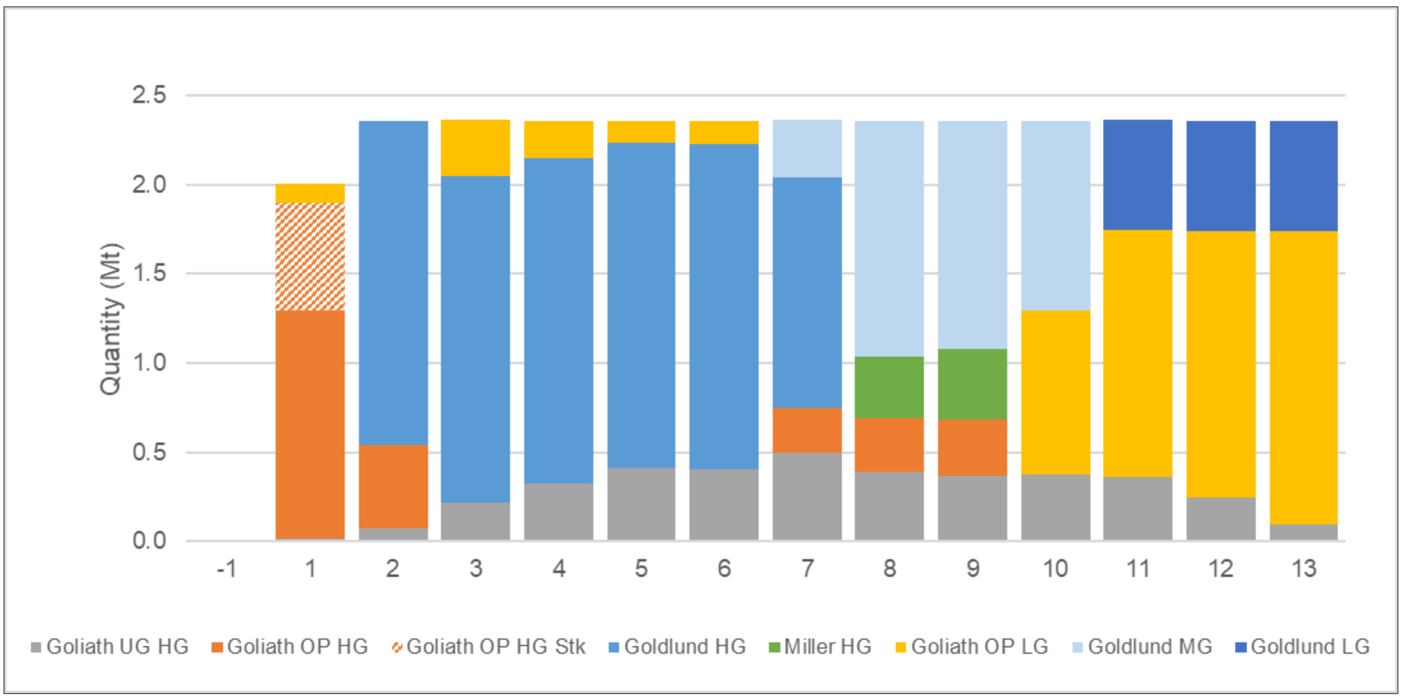
16.5.14 Grade Control

As a routine part of the mining process, TML will utilize a grade control program to measure the quality of the extraction. The specifics of the controls will be developed in further stages of study.

16.6 Plant Feed Schedule

The combined open pit and underground plant feed schedule is shown in Figure 16-30 and Table 16-57. Goliath underground HG is fed preferentially, followed by open pit HG. Goliath LG is used to fill the plant when there is insufficient HG, until Year 7. In Years 7 to 10 Goldlund MG is used to top up the plant feed, until the Goldlund MG stockpile is depleted, with Goliath LG making up the remainder in Year 10. From Year 11 to 13, a mixture of Goliath LG and Goldlund LG is fed along with Goliath UG. Not all Goldlund LG is fed to the plant with some remaining on the stockpile at the end of the mine life.

Figure 16-30: Plant Feed Schedule



Notes: UG: underground; OP: open pit; HG: high grade; MG: medium grade; LG: low grade; Stk: Stockpile. Source: SRK, 2023.

Table 16-57: Plant Feed Schedule

Plant Feed	Units	Total	1	2	3	4	5	6	7	8	9	10	11	12	13
Total Feed	kt	30,318	2,004	2,358	2,364	2,358	2,358	2,358	2,364	2,358	2,358	2,358	2,364	2,358	2,358
Au Grade	g/t	1.30	1.53	1.84	1.75	1.76	1.62	1.57	1.63	1.18	1.30	0.84	0.76	0.65	0.49
Ag Grade	g/t	1.77	3.80	1.36	1.33	1.28	1.34	1.46	1.70	1.80	2.05	1.71	2.01	1.86	1.58
Goliath UG	kt	3,776	18	72	219	324	413	407	494	390	368	374	358	245	93
Au Grade	g/t	3.03	2.93	3.51	3.08	2.84	3.15	3.47	3.08	3.04	2.91	2.69	2.87	2.97	3.02
Ag Grade	g/t	7.56	14.43	17.98	11.59	8.09	7.08	7.87	6.78	8.12	7.58	6.02	5.81	6.19	6.00
Goliath OP HG	kt	3,207	1,877	465	0	-	-	-	251	302	312	-	-	-	-
Au Grade	g/t	1.70	1.58	1.75	3.55	-	-	-	1.39	1.75	2.51	-	-	-	-
Ag Grade	g/t	4.02	3.83	4.14	7.25	-	-	-	2.70	3.58	6.54	-	-	-	-
Goliath OP LG	kt	6,342	109	-	315	209	120	126	-	0	-	922	1,392	1,498	1,650
Au Grade	g/t	0.39	0.41	-	0.38	0.38	0.38	0.38	-	0.65	-	0.39	0.39	0.39	0.39
Ag Grade	g/t	1.91	1.62	-	1.91	1.91	1.91	1.91	-	1.63	-	1.92	1.92	1.92	1.92
Goldlund HG	kt	10,423	-	1,821	1,830	1,825	1,825	1,825	1,297	-	-	-	-	-	-
Au Grade	g/t	1.56	-	1.79	1.83	1.73	1.35	1.23	1.37	-	-	-	-	-	-
Goldlund MG	kt	3,989	-	0	-	-	-	-	322	1,323	1,283	1,061	-	-	-
Au Grade	g/t	0.59	-	0.56	-	-	-	-	0.58	0.59	0.59	0.59	-	-	-
Goldlund LG	kt	1,844	-	-	-	-	0	-	-	-	-	-	615	615	615
Au Grade	g/t	0.39	-	-	-	-	0.30	-	-	-	-	-	0.39	0.39	0.39
Miller HG	kt	738	-	-	-	-	-	-	-	343	395	-	-	-	-
Au Grade	g/t	1.03	-	-	-	-	-	-	-	0.86	1.18	-	-	-	-

Notes: UG: underground; OP: open pit; HG: high grade; MG: medium grade; LG: low grade; Stk: Stockpile. Source: SRK, 2023.

17 RECOVERY METHODS

17.1 Overall Process Design

Throughout the initial stages of the pre-feasibility study, the results of the available testwork programs were analysed to determine the optimal process route. The unit operations selected are typical for gold recovery and the proposed flowsheet uses standard processes and technologies. Key process design criteria are listed below:

- nominal throughput of 6,460 t/d or 2.36 Mt/a
- crushing plant availability of 67%
- plant availability of 92% for grinding, leach/CIL, and gold recovery operations.

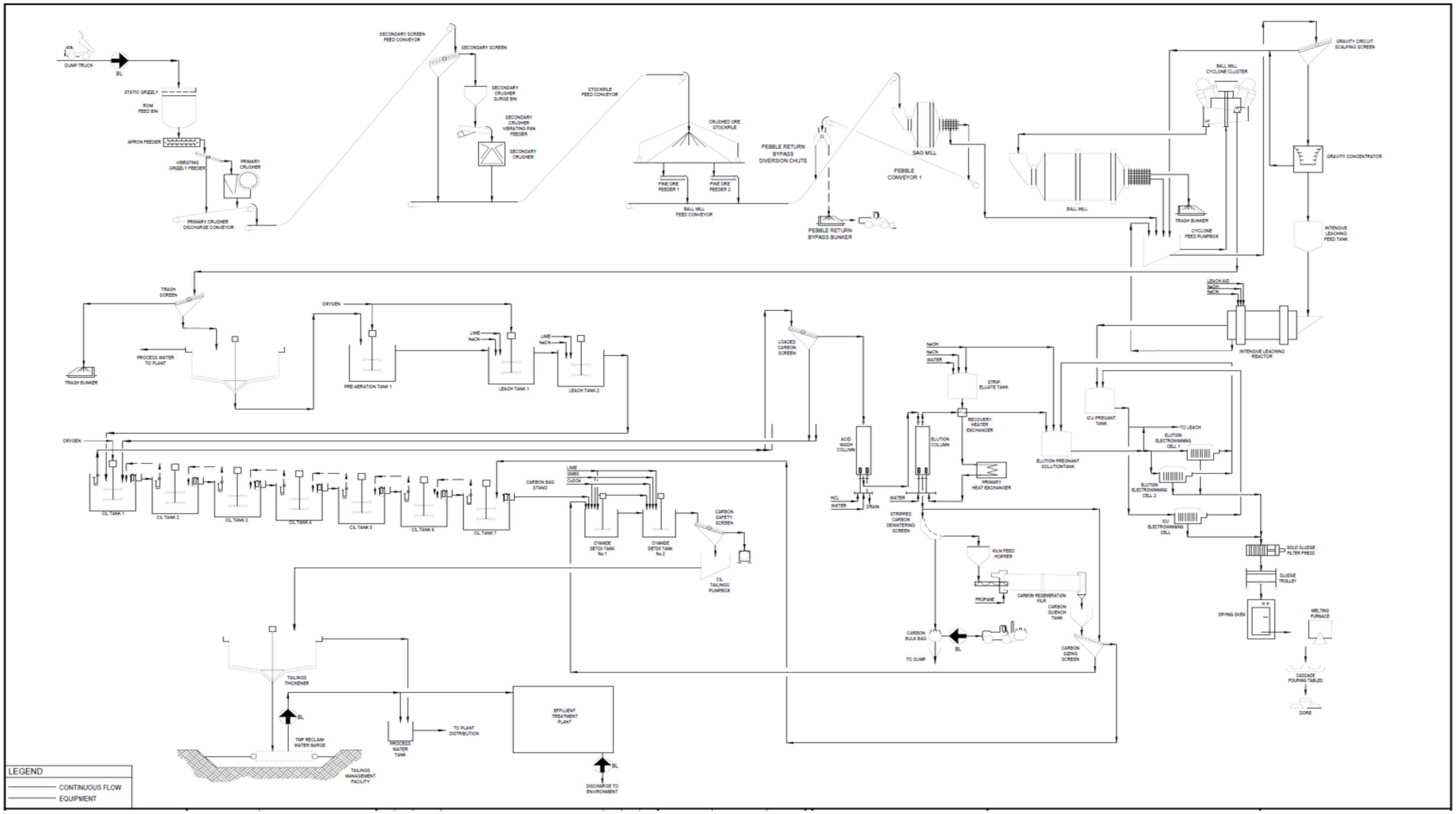
17.2 Mill Process Plant Description

The process plant includes the following:

- two-stage crushing of run-of-mine (ROM) ore
- covered, crushed ore stockpile to provide buffer capacity for the process plant
- SAG mill with trommel screen followed by a ball mill with cyclone classification
- gravity recovery of ball mill discharge by one semi-batch centrifugal gravity concentrator, followed by intensive cyanidation of the gravity concentrate and electrowinning of the pregnant leach solution
- trash screening
- pre-leach thickening
- pre-aeration, leach and carbon-in-leach (CIL) adsorption
- acid washing of loaded carbon and Anglo-American Research Laboratory (AARL) type elution followed by electrowinning and smelting to produce doré
- carbon regeneration
- tailings cyanide destruction using SO_2/air process
- carbon safety screening, tailings thickening and tailings pumping
- reagent storage and distribution
- water services (process water, treated water, fire water, gland water)
- potable water treatment and distribution
- air services.

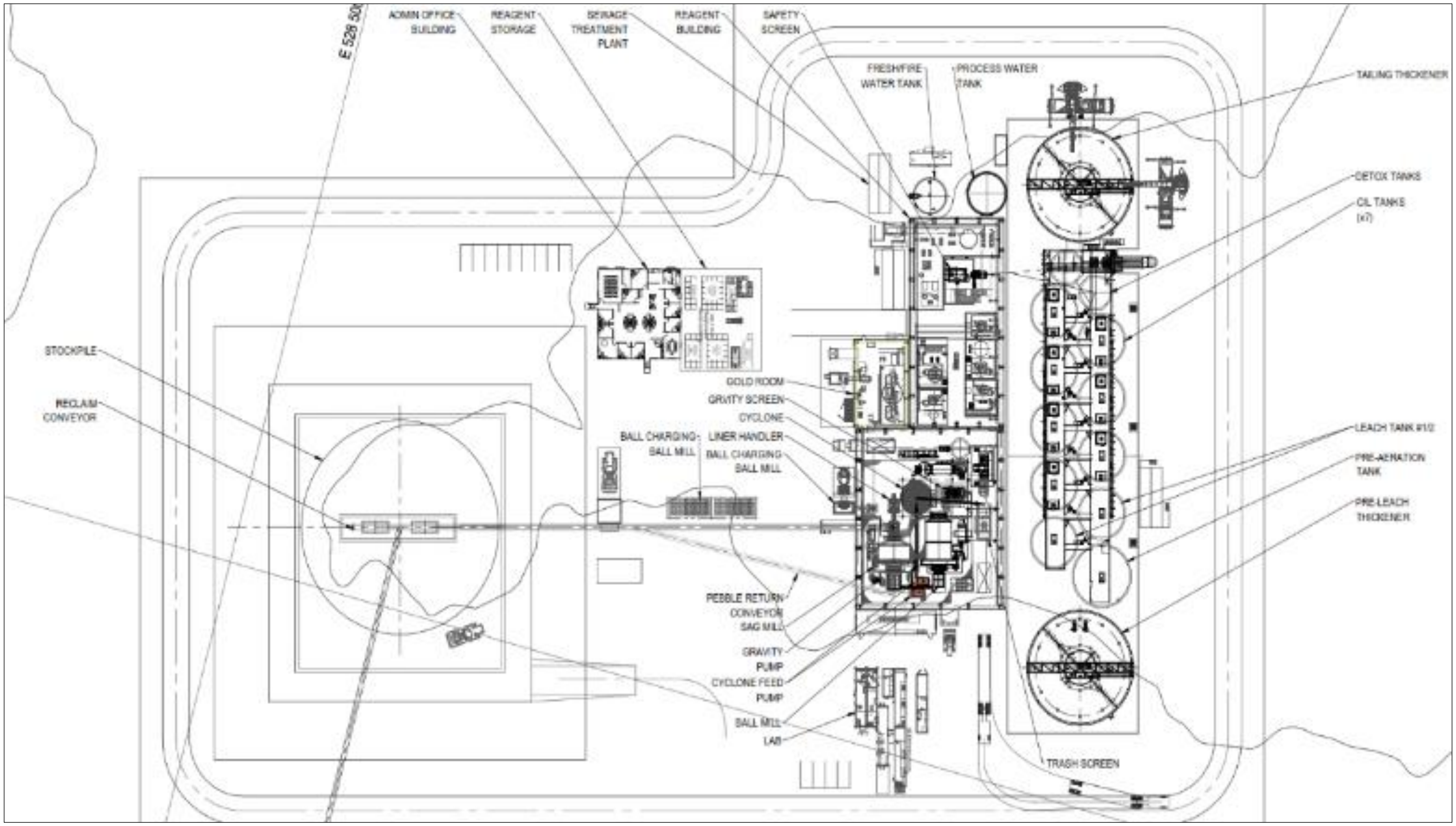
The overall process flow diagram is presented in Figure 17-1 and the general arrangement is presented in Figure 17-3.

Figure 17-1: Overall Process Flow Diagram



Source: Ausenco, 2023

Figure 17-2: Overall Process Plant Layout



Source: Ausenco, 2023

17.2.1 Plant Design Criteria

Key process design criteria for the mill are listen in Table 17-1. Process design criteria for the comminution circuit were selected based on the Goldlund deposit as it is both harder and requires a finer grind than the Goliath deposit. Both deposits exhibited similar leaching and detoxification performance when telluride leaching conditions are used on Zone 1 therefore the design criteria are valid for both deposits. Where substantial variation exists in the metallurgical performance of the two zones, such as in reagent dosing, the more stringent of the two criteria was applied.

Table 17-1: Key Design Criteria

Design Parameter	Units	Value
Plant Throughput	t/d	6,460
Gold Head Grade – Maximum for Design	g/t Au	1.76
Silver Head Grade – Maximum for Design	g/t Ag	3.80
Crushing Plant Availability	%	67
Mill Availability	%	92
Bond Crusher Work Index (CWi)	kWh/t	26.7
Bond Rod Mill Work Index (BWi), 75th percentile	kWh/t	18.3
Bond Ball Mill Work Index (BWi), 75th percentile	kWh/t	16.0
SMC Axb, 25th Percentile	-	26
Bond Abrasion Index (Ai)	g	0.564
Primary Crusher	-	C130 or equivalent
Secondary Crusher	-	HP400 or equivalent
Ore Specific Gravity	t/m^3	2.74
SAG Mill Dimensions	m dia. x m EGL	7.0 x 3.4
SAG Mill Installed Power	MW	2.7
Ball Mill Dimensions	m dia. x m EGL	5.5 x 27.5
Ball Mill Installed Power	MW	4.4
Primary Grind size (P_{80})	µm	85
Gravity Gold Recovery, for ADR sizing	%	16
Gravity Gold Recovery, for intensive leaching sizing	%	25
Pre-aeration Residence Time	h	5
Leach Residence Time	h	4.4
CIL Residence Time	h	15.4
Leach Extraction	% Au	93.3
Leach-CIL Operating Density	w/w%	55
Leach Sodium Cyanide Addition, Design	kg/t	0.63
Leach Hydrated Lime Addition, Design	kg/t	1.0
Pre-aeration Tanks	-	1
Leach & CIL Tanks	#	2 + 7
Elution Column Capacity	t	4
Detoxification Residence Time	min	60
Detoxification Tanks	#	2
Detoxification SO_2 Addition, Design	SO_2:CN_{WAD} ratio	6.5
Detoxification lime Addition, Design	$Ca(OH)_2$:SO_2	0.75
Final Tails Thickener Underflow Density	w/w%	60

Note: Design basis for cyanide and lime addition was selected to represent cyanide and lime consumptions suitable for a feed consisting of 20% telluride mineralized material.

17.2.2 Crushing and Ore Stockpiling

ROM production is delivered by haul truck to the ROM feed bin where production from the Goliath, Goldlund and Miller deposits are fed to the crushing circuit. ROM stockpiles can be blended as required to stabilize plant feed grade and ore hardness when deposits are being mined simultaneously.

The crushing circuit is designed for an annual operating time of 5,869 hours or 67% availability at a capacity of 6,460 t/d.

Ore is hauled from the mine and direct tipped into to the run-of-mine (ROM) bin, where oversized ore is removed by a static grizzly screen with an aperture of 800 mm. The ROM bin is equipped with an apron feeder and vibrating grizzly feeder. The vibrating grizzly feeder oversize is fed to the jaw crusher. The jaw crusher discharge combines with the grizzly feeder undersize and is conveyed to the secondary screen. The oversize ore is removed and conveyed through a surge bin and vibrating pan feeder to be further crushed by a secondary cone crusher. The secondary screen undersize is combined with the secondary cone crusher discharge and conveyed to a covered stockpile at an 80% passing product size of 34 mm.

The covered stockpile provides approximately 3,511 t or approximately 12 hours of live storage. The stockpile disconnects crushing from the mill to allow for crusher maintenance to be carried out without interrupting feed to the mill.

The SAG mill feed of 293 t/h is regulated by two apron feeders and is conveyed by the SAG mill feed conveyor. SAG mill discharge passes through a trommel screen, from where oversize pebbles are recirculated by a pebble conveyor back to the SAG mill feed.

The material handling and crushing circuit includes the following key equipment:

- ROM feed bin
- vibrating grizzly
- primary crusher apron feeder
- primary jaw crusher
- secondary screen
- secondary cone crusher
- mill feed apron feeders
- material handling equipment.

17.2.3 Grinding Circuit

A SAG mill and a ball mill in closed circuit with hydrocyclones make up the grinding circuit. The circuit is sized based on SAG mill feed size of 80% passing 34 mm and a ball mill product of 80% passing 85 µm. The SAG mill trommel screen undersize discharges into the cyclone feed pumpbox to feed the cluster of hydrocyclones and the gravity circuit via dedicated pumps. In the event of changes in ore hardness or throughput surges, the SAG mill is powered by a variable speed drive (VSD) to allow for changes in motor speeds.

The cyclone underflow feeds the ball mill. From there, the ball mill discharges through a trommel screen. Trommel undersize discharges into the cyclone feed pumpbox at a density of 72 w/w% solids and the oversize material is discharged to the scats bunker.

Water is added to the cyclone feed pumpbox achieve the appropriate cyclone feed density of 62 w/w% solids. Cyclone overflow at 39 w/w% solids is sent to a trash screen followed by a pre-leach thickener. Slurry from the cyclone feed pumpbox is also sent to the gravity recovery circuity.

The grinding circuit includes the following key equipment:

- 23 ft diameter x 11.0 ft effective grinding length18 ft diameter x 27.5 ft EGL 4,400 kW ball mill
- cyclone feed pumpbox
- classification cyclones
- trash screen.

17.2.4 Gravity Recovery Circuit

The gravity recovery circuit comprises one centrifugal concentrator complete with a feed scalping screen. The 2 mm aperture scalping screen is fed from the cyclone feed pumpbox by a dedicated pump. The gravity scalping screen oversize is sent to the cyclone feed pumpbox.

The centrifugal gravity concentrator is fed by the scalping screen undersize. Operation of the gravity concentrator is semi-batch and the gravity concentrate is subsequently leached by the intensive cyanidation reactor circuit. The tails from the gravity concentrator also report to the cyclone feed pumpbox.

The gravity recovery circuit includes the following key equipment:

- gravity feed scalping screen
- gravity concentrator.

17.2.5 Intensive Cyanidation Reactor

The gravity concentrate reports to the intensive cyanidation reactor (ICR). The ICR leach solution (mixture of NaCN, NaOH and LeachAid® - an oxidant) is made up within the heated ICR reactor vessel feed tank. From the feed tank, the leach solution is circulated though the reaction vessel, then drained back into the feed tank. The gold extracted, or the ICR pregnant solution, is washed and pumped to the ICR pregnant solution tank to be treated for gold recovery using a dedicated electrowinning cell.

The gold sludge is combined with the sludge from the leach/adsorption elution electrowinning cells and smelted. It can also be smelted separately for metallurgical accounting purposes.

The ICR circuit includes the following key equipment:

- intensive leaching feed tank
- intensive cyanidation reactor
- ICR electrowinning cell.

17.2.6 Leach and Adsorption Circuit

Cyclone overflow at 39 w/w% solids is sent to a trash screen followed by a pre-leach thickener. Flocculant is added to the feed to the thickener to improve the settling rate of the ground solids. The thickener overflow is recycled as process water in the circuit. The thickener underflow continues to the leach and adsorption circuit. The circuit consists of one pre-aeration tank, two leach tanks and seven carbon-in-leach (CIL) tanks.

The pre-aeration tank is fed by the pre-leach thickener underflow at 55 w/w% solids, with barren solution from the electrowinning cells periodically transferred into leach tanks. The pre-aeration tank has a residence time of 5 hours, and the leach and CIL tanks have a total circuit residence time of 20 hours.

Oxygen is sparged to the pre-aeration tank, the first leach tank, and the first CIL tank to maintain adequate dissolved oxygen levels for leaching at 15 mg/L. Hydrated lime is added to maintain the operating pH at the desired set point of 11 and sodium cyanide is added to the first leach tank. Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently to the slurry flow by pumping slurry and carbon. The intertank screen in each CIL tank retains the carbon while allowing the slurry to flow by gravity to the following tank. This counter-current process is repeated until the loaded carbon reaches the first CIL tank. Recessed impeller pumps are used to transfer slurry between the CIL tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit. Slurry from the last CIL tank flows to the cyanide detoxification tanks.

The leach and carbon adsorption circuit includes the following key equipment:

- pre-leach thickener, 24 m diameter
- pre-aeration/leach/CIL tanks and agitators
- oxygen supply system
- loaded carbon screen
- intertank carbon screens
- carbon sizing screen.

17.2.7 Cyanide Destruction

CIL tailings at 55 w/w% solids flow by gravity to the two cyanide detoxification tanks in parallel, then to the carbon safety screen. The screen oversize (recovered carbon) is collected in carbon bulk bags. The screen undersize is fed to the thickener feed pump box and is pumped to the tailings thickener. The water used for acid rinse and carbon transfer is also included in the feed to detoxification circuit.

The slurry remains in the cyanide detoxification tanks for a total of 60 minutes. The circuit is designed to decrease weak acid dissociable cyanide (CN_{WAD}) concentration from 125 mg/L to less than 2.5 mg/L. The circuit includes two tanks operating in parallel.

Cyanide destruction is accomplished using the SO_2/air method. The reagents required are oxygen, lime, copper sulphate, and sodium metabisulphite (SMBS) as a sulphur source. The cyanide destruction tank is equipped with oxygen addition points and an agitator to ensure thorough mixing.

Cyanide detoxification discharges to the carbon safety screen, and the screen undersize feeds tailings thickener.

The main equipment in this area includes:

- carbon safety screen
- cyanide destruction tanka and agitators
- oxygen supply system.

17.2.8 Tailings Thickening

The carbon safety screen undersize is thickened before being discharged to the tailings management facility (TMF). The overflow of the thickener is reused as process water in the plant. Flocculant is combined with the feed to the thickener to improve the solids settling rate. The underflow is pumped to the TMF for final deposition with decant water from the TMF returned for use as process water. Excess water from the process water tank is sent to effluent treatment prior to discharge.

The main equipment in this area includes:

- high-rate thickener, 24 m diameter
- final tailings pumps.

17.2.9 Carbon Acid Wash, Elution and Regeneration Circuit

17.2.9.1 Carbon Acid Wash

Prior to gold elution, loaded carbon is treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide detoxification tanks.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

The main equipment in this area includes:

- acid wash carbon column – 4 t capacity
- hydrochloric acid feed pump
- spent solution discharge sump pump.

17.2.9.2 Carbon Elution and Electrowinning

The carbon elution circuit uses the Anglo-American Research Laboratory (AARL) process.

The column is initially filled with a strong-cyanide, caustic solution which is recirculated through a pressure-elution column heated to 120°C for a pre-soak period. At the end of the pre-soak period, the solution is discharged to electrowinning and heated water is used to complete the stripping of the precious metals from the carbon. The precious metal-rich solution from the column exchanges heat with elution water going to the column. Cooled pregnant solution then flows through electrowinning cells to deposit the gold and silver on the cathodes before the solution is discharged to the leach tanks. The carbon stripping circuit includes the following key equipment:

- carbon elution column – 4 t capacity
- elution solution heater (electric) with heat exchangers
- strip eluate, and pregnant solution tanks.

17.2.9.3 Gold Room

Gold/silver sludge is recovered from the electrowinning cells and smelted to produce doré bars.

The electrowinning circuit consist of three independent cells, two dedicated to the carbon elution pregnant solution and one dedicated to the intensive leaching reactor pregnant solution. The gold-rich sludge is washed off the stainless steel cathodes in the electrowinning cells using high-pressure spray water and transfers by gravity to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold/silver doré bullion. The electrowinning and smelting process takes place within a secure and supervised gold room equipped with access control, intruder detection, and closed-circuit television equipment.

The electrowinning circuit and gold room include the following key equipment:

- electrowinning cells with rectifiers
- sludge pressure filter
- drying oven
- flux mixer

- induction smelting furnace with bullion moulds and slag handling system
- bullion vault and safe
- dust and fume collection system
- gold room security system.

17.2.9.4 Carbon Reactivation

Carbon is reactivated in a gas-fired rotary kiln. Dewatered barren carbon from the stripping circuit is held in a 4-tonne kiln feed hopper. A screw feeder metres the carbon into the reactivation kiln, where it is heated to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln is quenched in water and screened on a carbon sizing screen located on top of the CIL tanks to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, whilst carbon screen oversize is directed to the CIL circuit.

As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit includes the following key equipment:

- carbon dewatering screen
- regeneration kiln (electric) including feed hopper and screw feeder
- carbon quench tank.

17.3 Reagent Handling and Storage

The reagent mixing and storage systems are located within the reagent building near the process plant. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Sumps and sump pumps are provided for spillage control. Appropriate ventilation, fire and safety protection, eyewash stations, and stations with material safety data sheets (MSDSs) are located throughout the facilities.

The following reagent systems are required for the process:

- hydrated lime
- sodium cyanide
- hydrochloric acid
- copper sulphate pentahydrate

- sodium metabisulphite
- sodium hydroxide
- flocculant
- activated carbon
- smelting fluxes
- liquid oxygen.

17.3.1 Hydrated Lime

Hydrated lime is delivered in bulk bags, which are lifted using a frame and hoist into the hydrated lime bag breaker on top of the mixing/storage tank to minimize plant footprint. The bag contents discharges into the tank and is mixed with process water to achieve the required slurry density. The hydrated lime slurry is pumped through a ring main with distribution points in leaching and in cyanide destruction. An extraction fan is provided over the lime bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.2 Sodium Cyanide (NaCN)

Sodium cyanide is delivered to site in secured boxes containing the 1 t bulk bags. Bags are lifted using a frame and hoist into the sodium cyanide bag breaker on top of the tank. The cyanide briquettes discharge into the tank and is dissolved in water to achieve the required dosing concentration.

After the mixing period is complete, cyanide solution is transferred to the cyanide solution storage tank using a transfer pump. Sodium cyanide is delivered to the leach circuit, intensive leach circuit and elution circuit with dedicated dosing pumps. An extraction fan is provided over the sodium cyanide bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

Sodium cyanide is delivered to the leach circuit and elution circuit with dedicated dosing pumps.

17.3.3 Copper Sulphate

Copper sulphate pentahydrate is delivered in solid crystal form in small bags and stored in the warehouse. Process water is added to the agitated copper sulphate mixing tank. A pallet of bags is lifted using a frame and hoist, and periodically a single bag is placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration.

Copper sulphate solution is transferred by gravity to the copper sulphate storage tank, which has a stacked arrangement with the mixing tank. Copper sulphate is delivered to the cyanide detoxification circuit using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.4 Sodium Metabisulphite (SMBS)

SMBS is delivered in the form of solid flakes in bulk bags and stored in the warehouse. Process water is added to the agitated SMBS mixing tank. Bags are lifted using a frame and hoist into the SMBS bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required concentration. After the mixing period is complete, SMBS solution is transferred to the SMBS storage tank using the SMBS transfer pump. SMBS is delivered to the cyanide detoxification circuit using the SMBS dosing pump. An extraction fan is provided over the SMBS mixing tank to remove SO_2 gas that may be generated during mixing. The SMBS mixing area is ventilated using the SMBS area roof fan.

17.3.5 Sodium Hydroxide (NaOH)

Sodium hydroxide (caustic soda) is delivered in intermediate bulk containers (IBCs) as a solution and stored adjacent to the elution circuit until required. During winter months, the reagent concentration may be adjusted to prevent it from freezing in the IBCs. Dosing pumps automatically deliver the reagent to the required locations—elution circuit, electrowinning and cyanide mixing—to ensure the dosing requirements are met.

17.3.6 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in IBCs as a solution and stored adjacent to the elution circuit until required. Hydrochloric acid is mixed with raw water (inline) to achieve the required 3% w/v concentration. Hydrochloric acid is delivered to the acid wash circuit using the hydrochloric acid dosing pump.

17.3.7 Flocculant

Powdered flocculant is delivered to site in bulk bags and stored in the warehouse. A self-contained mixing and dosing system is installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powdered flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water.

Flocculant solution at 0.50% w/v is agitated in the flocculant mixing tank. The flocculant is then transferred to the flocculant storage tank by gravity. Flocculant is dosed to the pre-leach and tailings high-rate thickeners using variable speed helical rotor style pumps. Flocculant is further diluted to 0.05% w/v concentration prior to the addition point using an in-line mixer.

17.3.8 Activated Carbon

Activated carbon is delivered in bulk bags. On a regular basis, fresh carbon is introduced to the carbon quench tank, and incorporated into the carbon regeneration system to be added to the final CIL tank.

17.3.9 Oxygen

Oxygen is injected into the leach tanks to achieve a dissolved oxygen level of 15 mg/L. For this purpose, bulk liquid oxygen is supplied by the vendor and stored in a vendor-supplied bulk liquid storage tank. From there, it is passed through vaporizers to feed the leach and detoxification tanks as required.

17.3.10 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and soda ash are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.4 Services & Utilities

17.4.1 Plant/ Instrument Air

High-pressure air at 690 kPag is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.4.2 Electrical Power Supply

An estimated 72,000 MWh are nominally required per year for processing Goliath ore at non-telluride conditions and 88,400 MWh are nominally required per year for processing Goldlund ore at telluride conditions. On average, it is estimated that the process plant will require approximately 78,840 MWh per year during operations.

17.5 Water Supply

17.5.1 Fresh Water Supply System

Fresh water is supplied to a raw water storage tank. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- gland water for pumps
- reagent make-up
- elution circuit make-up
- fresh water is treated and stored in the potable water storage tank for use in safety showers and other similar applications
- fire water for use in the sprinkler and hydrant system
- cooling water for mill motors and mill lubrication systems (closed loop).

Total consumption for fresh water is 11.4 m^3/h, with an additional 43.1 m^3/h which will be supplied from surface runoff/precipitation captured over time in the tailings facility.

17.5.2 Process Water Supply System

Overflow from the pre-leach thickener, final tailings thickener, and filtration plant clarifier meet the main process water requirements. Mine wastewater and mill contact water provide any additional make-up water requirements.

17.5.3 Gland Water

One dedicated gland water pump is fed from the freshwater tank to supply gland water to all slurry pumps in the main plant. At the filtration plant, one dedicated pump pulls clarifier overflow water and filters it for use as gland water for all filtration plant slurry pumps.

17.6 Reagent & Consumable Requirements

Reagent consumptions are based on testwork results and standard industry practices. A summary of the estimated reagent and consumables rates is shown in Table 17-2.

Table 17-2: Nominal Reagent and Consumable Consumption Rates

Reagent / Consumable	Form	Unit	Consumption	
			Goliath	Goldlund
Hydrated Lime	Powder, 90% minimum available CaO	t/y	2,802	2,943
Sodium Cyanide	Briquettes, 98% minimum purity	t/y	516	469
SAG Mill Media	125 mm balls	t/y	589	1,085
Ball Mill Media	50-75 mm balls	t/y	495	2,358
Jaw Crusher	Cheek and Swing Set	Qty/y	3	
Secondary Screen Decks	Deck panel	Qty/y	6	
Secondary Crusher	Mantle and Bowl Liner	Qty/y	6	
SAG Mill Liner	Complete liner set	Qty/y	1	
Ball Mill Liner	Complete liner set	Qty/y	1	
Sodium Hydroxide	Liquid, 50% w/w	t/y	3.4	
SMBS	Powder, 97.5% minimum purity	t/y	2,525	
Oxygen	Bulk liquid	t/y	948	
Flocculant	Powder, 97.5% minimum purity	t/y	83	
Copper Sulphate	Blue crystal, pentahydrate, 99.5% minimum purity	t/y	53	
Hydrochloric Acid	Liquid, 33% w/w	t/y	86	
Sulphamic Acid	Powder	t/y	4.7	
Borax	Powder	t/y	2.2	
Silica	Powder	t/y	1.1	
Sodium Nitrate	Powder	t/y	0.2	
Sodium Carbonate	Powder	t/y	0.2	
Activated Carbon	Coconut shell, grade 6 x 12 mesh	t/y	94	

Note: Goldlund reagent consumption includes a portion of Zone 1 material according to the mine plan, which requires telluride leaching conditions to treat.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The Goliath Gold Complex is located in northwestern Ontario, 20 km east of Dryden Ontario and is accessible year-round from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road.

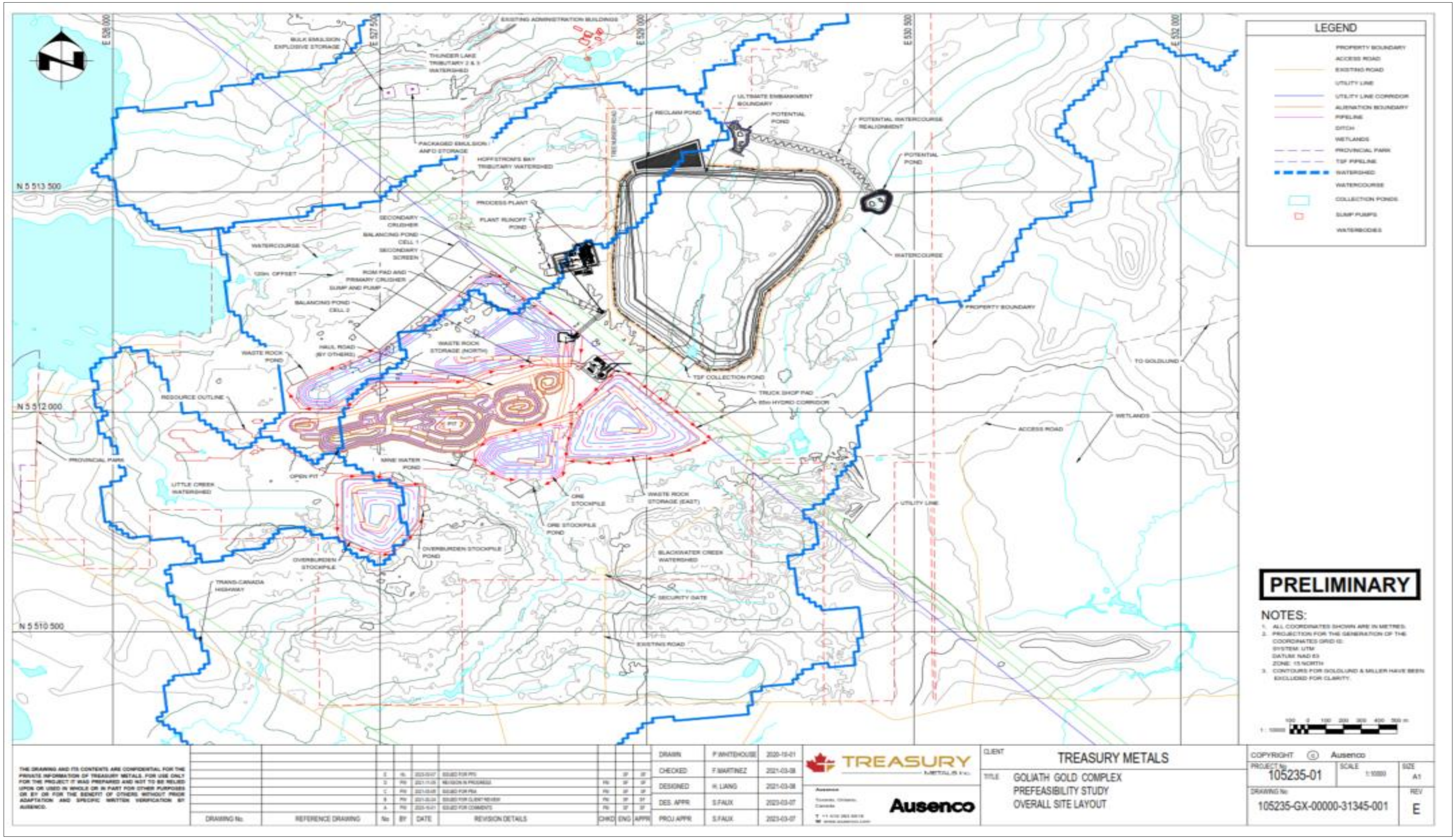
The Goliath Gold Complex is show in Figure 18-1 and Figure 18-2. Figure 18-1 shows the overall layout of the Goliath project including the Goliath open pit mine, ore stockpile storage, process plant, tailings storage facility (TSF), existing administration buildings and roads, and new access roads. Figure 18-2 shows the overall Goldlund and Miller project layout including the open pit mines, rock storage facilities, mill feed transfer pad, truck shop, and haul roads.

The processing plant and tailings storage facility will be located at the Goliath property, along with most of the ancillary project infrastructure. Table 18-1 shows the infrastructure applicable to each property.

Table 18-1: Infrastructure Applicable to Each Project

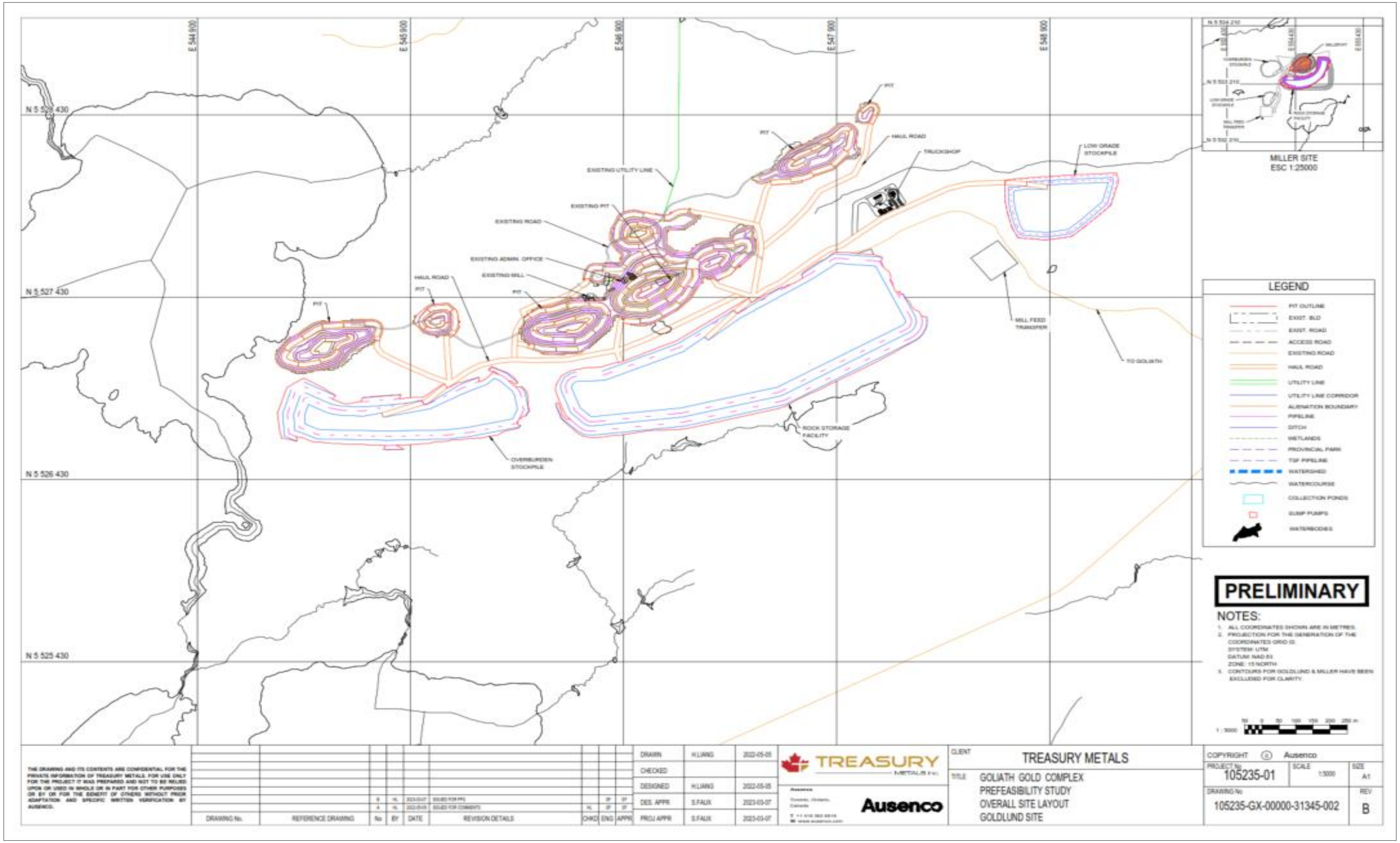
Infrastructure	Goliath	Goldlund	Miller
Plant Access Road	X		
Light-Vehicle Equipment Roads	X		
High-Grade Stockpiles	X		
Medium-Grade Stockpiles	X		
Low-Grade Stockpiles	X	X	X
Heavy-Vehicle Equipment Roads	X	X	X
Rock Storage Facilities	X	X	X
Mill Feed Transfer Pads		X	X
Overburden Stockpiles	X	X	X
Mine Dewatering Pumps and Pipelines	X	X	X
Mine Facility Platforms and Process Facility Platforms	X	X	
Water Management Ditches and Collection Ponds	X	X	X
Tailings Storage Facility (TSF)	X		
Process Plant, including Crushing, Stockpile, Mill, Gold Room & Reagent Storage Buildings	X		
Effluent Water Treatment Plant	X		
Incoming Power High-Voltage Substation and Site-Wide Electrical Distribution	X		
Mine and Process Administration Offices and Changerooms	X		
Mine Truck Shop, Truck Wash and Refuelling Station	X		
Workshop and Warehouse Facilities	X		

Figure 18-1: Goliath Gold Complex, Goliath Project Layout



Source: Ausenco, 2023.

Figure 18-2: Goliath Gold Complex, Goldlund and Miller Layout



Source: Ausenco, 2023.

The location of the site facilities was based on the following considerations:

- within the claim boundary
- suitable geotechnical conditions
- outside the battery limits of the potential future high-voltage power corridor expansion
- stockpiles are near mine pits to reduce haul distances
- process plant is in an area safe from flooding
- administration, processing plant and offices are sited to limit travel distances.

18.2 Site Preparation

Forest clearing and topsoil removal will be required for the processing plant, mining pits, stockpiling areas, and other buildings and facilities.

Existing roads connected to the project site enable access to the properties. Typical methods of clearing and topsoil removal, excavation, drains, safety bunds and aggregates will be employed to construct additional roads and upgrade existing roads as required.

18.3 Existing Roads and Logistics

18.3.1 Roads

Access to the Goliath project is from the Trans-Canada Highway 17 via Anderson Road and Tree Nursery Road. Highway 17 is located south of the project. Anderson and Tree Nursery Roads are maintained by the Wabigoon Local Services Board, with minor care and maintenance by Treasury Metals.

Access to the Goldlund project is east off Highway 72 via Goldlund Mine Road. The Miller project is accessed via forestry road east off Highway 72. Access roads for the Goldlund and Miller projects are maintained by the Sustainable Forest Licence Holder (Domtar) for the area. Travel to site from Dryden is approximately 20 minutes by road.

Load and size limits for trucking goods to site are governed by the Highway Traffic Act, R.S.O. 1990, c. H.8.

18.3.2 Road Construction

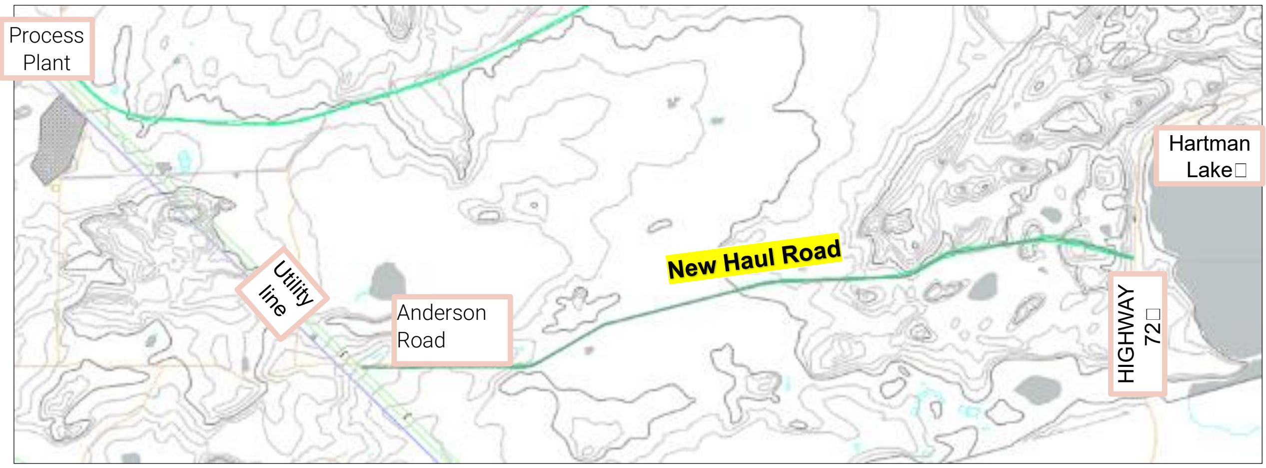
The new haul road will be built to connect the process plant to the existing Highway 72 (near to Hartman Lake) passing through the utility line. It is located on the existing Anderson Road. The length of this haul road is almost 6+935 m. It includes two lanes (2*4 m = 8 m).

After clearing and grubbing the natural ground, topsoil layer (about 500 mm) shall be removed and hauled to the organic stockpile. Consequently, bulk earthwork, excavation of the cut areas or spread and compaction of the fill areas shall be done.

For surface layers, 300 mm Granular B and 150 mm Granular A will be used. Along the way, some trenches, especially in the cut area will be excavated. Also, at the cross points with the natural streams, some culverts will be constructed.

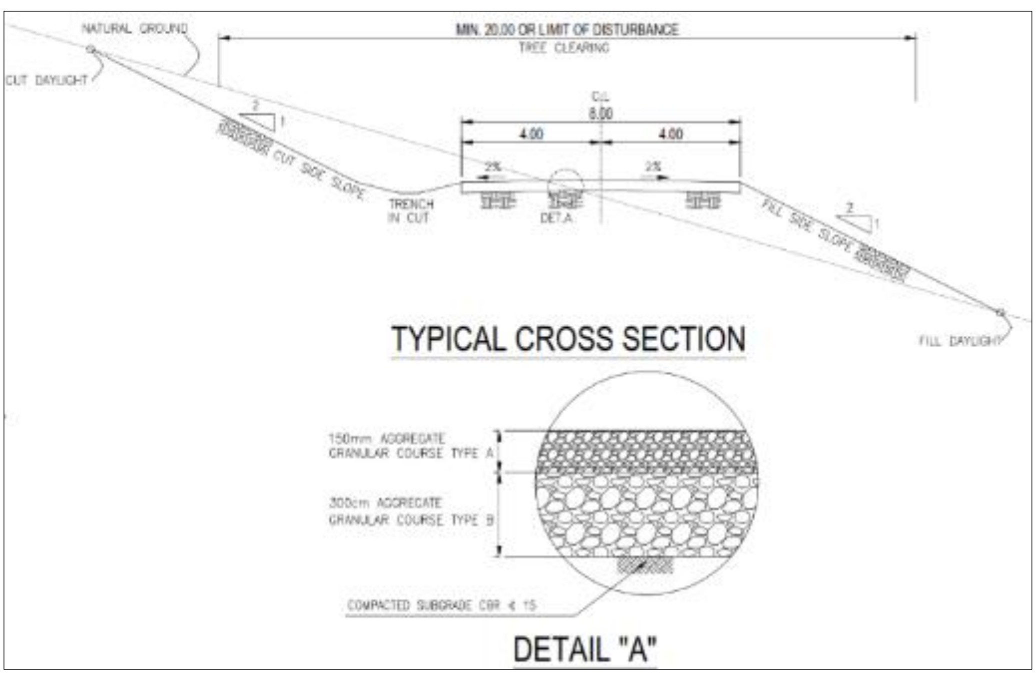
For more information, please refer to drawing number 105235-EL-00000-31345-SK123.

Figure 18-3: Plan of New Haul Road



Source: Ausenco, 2023

Figure 18-4: Details of New Haul Road



Source: Ausenco, 2023

18.3.3 Tree Nursery Road Realignment

The process plant and project have some conflict with the existing Tree Nursery Road. So, it shall be realigned near the process plant. The length of realignment is almost 1+216.5m. It includes two traffic lanes and two shoulders (2*4 m + 2*1 = 10 m).

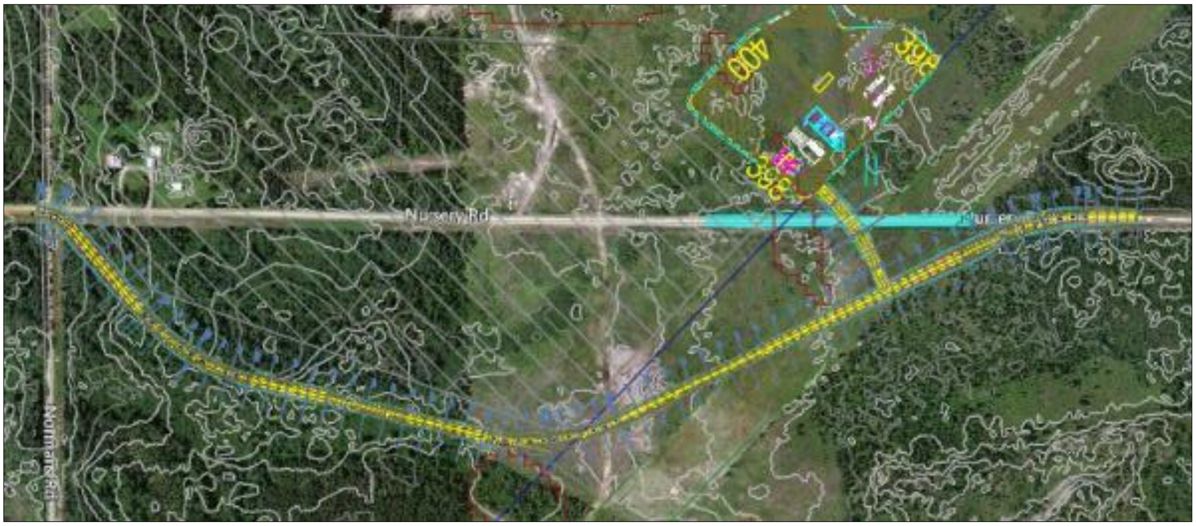
After clearing and grubbing the natural ground, topsoil layer (about 500 mm) shall be removed and hauled to the organic stockpile. Consequently, bulk earthwork, excavation of the cut areas or spread and compaction of the fill areas shall be done.

For surface layers, 300 mm Granular B and 150 mm Granular A shall be used. Along the way, some trenches, especially in the cut area shall be excavated. Also, at the cross points with the natural streams, some culverts shall be constructed.

For more information, please refer to drawing number 105235-EL-00000-31345-001.

Figure 18-5: Tree Nursery Road Realignment by Google Earth

Figure 18-6: Tree Nursery Road Realignment – modelled by Civil 3D®



18.3.4 Rail

The Canadian Pacific Railway runs parallel along the Trans-Canada Highway 17. The two stations that are closest to the property are Dryden (22 km) and Dinorwic (16 km).

18.3.5 Air

There is no airport at the project site. Nearby airport facilities are listed in Table 18-2.

Table 18-2: Nearby Airports

Airport	Distance to Site (Road Travel) (km)
Dryden Regional Airport (CYHD)	22
Vermilion Bay Airport (CKQ7)	71
Sioux Lookout Airport (CYXL)	88
Kenora Airport (YQK)	154
International Falls Airport (INL) (U.S.A)	215
Fort Frances Airport (CYAG)	219
Thunder Bay Airport (YQT) (International)	338
Winnipeg James Armstrong Richardson Airport (YWG) (International)	380
Toronto Pearson Airport (YYZ) (International)	1,693

18.3.6 Port Facilities

The Port of Thunder Bay is the closest port facility. The port is located approximately 350 km southeast of the project and is equipped with a 104t capacity mobile crane. Port facilities are listed in Table 18-3.

Table 18-3: Port Facilities

Port	Distance to Site (Road Travel)
Port of Thunder Bay	336
Port of Marathon	621
Michipicoten Harbour	810
Toronto Port	1,721

18.4 Electrical Power System

18.4.1 Facility Power Supply

Hydro One will supply power to the project site via a 115 kV transmission line which will be branched to the new Goliath Gold Complex substation.

The 115 kV outdoor substation will be constructed adjacent to the Goliath project process plant and will comprise three 115 kV / 13.8 kV, 12 / 16 MVA, forced air-cooled, liquid filled power transformers, line terminating structures, disconnect switches, and circuit breakers. The transformers will be connected to the 13.8 kV primary switchgear housed in the process plant main electrical room located next to the substation.

Two of the transformers are dedicated to powering the process plant's nominal load whereas the third transformer powers the underground mine loads. Each transformer will be sized such that any two transformers can supply the peak demand, and the system will be configured to ensure reliability in the event a single transformer is temporarily out of service.

The Goldlund and Miller projects will not require permanent electrical infrastructure.

18.4.2 Electrical System Demand

The peak demand for process plant and mine is estimated at 22.5 MVA.

18.4.3 Plant Power Distribution

Power will be distributed from the primary switchgear housed in the process plant main electrical room via 13.8 kV cable circuits and overhead powerlines to electrical rooms provided at the following locations:

- primary crusher
- stockpile / reclaim

- grinding / gold recovery
- leaching / reagents / tailings
- mine portal.

Distribution transformers will be located next to each electrical room to step down to voltage to the required utilization level.

The electrical rooms will consist of medium voltage switchgear, low voltage motor control centres (MCCs), VFDs, soft starters, plant control system cabinets, lighting and services transformers, distribution boards, and uninterrupted power supply (UPS) power distribution systems.

The crushing plant, stockpile, mine portal and offsite areas including the truck shop and warehouse and other remote site infrastructure will be fed via 13.8 kV overhead powerlines.

To reduce installation time, the electrical rooms were considered prefabricated modular buildings, installed on structural framework above ground level for bottom entry of cables. The electrical rooms will be installed with HVAC units and suitably sealed to prevent ingress of dust. They will be in the process plant area and as close as possible to the main load points to minimise costs.

18.4.4 Site Power Reticulation

An overhead 13.8 kV network will provide power to various remote facilities. Pole-mounted or pad-mounted distribution transformers will step down the voltage at each location to supply the 4160 V and 600 V loads.

18.4.5 Emergency Power

Standby diesel generators with automatic transfer systems will be provided to supply emergency power to critical loads. One low voltage generator will be dedicated for the process plant loads and will be located close to the electrical room feeding the critical loads.

Underground operations do not require emergency power provisions.

18.4.6 Underground Mine Zones

The underground mine areas will be powered from the 13.8 kV primary switchgear via an overhead powerline routed to a dedicated electrical room located at the mine portal area. Normal power to the underground mine shall be supplied by a dedicated power transformer located at the 115kV outdoor substation.

18.5 Site Buildings

The Goliath Gold Complex will have several buildings. The buildings are listed in Table 18-4. Additional details are provided in the following subsections.

Table 18-4: Site Buildings

Building Function	Location	Building Construction	Length (m)	Width (m)	Height (m)	Area (m^2)
Administrative	Goliath	Existing	19	19	3	361
Gold Room	Goliath	Pre-Engineered	16.5	11	11	181.5
Mill Building	Goliath	Pre-Engineered	36	32	25	1,152
Mill Office	Goliath	Modular	18.3	18.3	3	334.9
Mine Office & Change Room	Goliath	Modular	23	19	3	437
Plant Warehouse and Maintenance	Goliath	Fabric	38	20	6	760
Primary Crusher	Goliath	Stick-Built				
Reagent Building	Goliath	Pre-Engineered	40	20	11	800
Reagent Storage	Goliath	Fabric	21	18	6	378
Secondary Crusher	Goliath	Stick-Built				
Security Gatehouse	Goliath	Modular	12	3.6	4	43.2
Stockpile Cover	Goliath	Fabric	53	47	24	2,491
Truck Shop	Goliath	Fabric	38	17	14	646
Truck Wash	Goliath	Fabric	25	18	14	450
Truckshop Office	Goliath	Modular	24	18	2.41	432
Truckshop Warehouse	Goliath	Fabric	24	16	6	384
Truck Shop at Goldlund	Goldlund	Fabric	38	17	14	646
Truck Wash at Goldlund	Goldlund	Fabric	25	18	14	450
Truckshop Office at Goldlund	Goldlund	Modular	24	18	2.41	432
Truckshop Warehouse at Goldlund	Goldlund	Fabric	24	16	6	384

18.5.1 Crushing Area Buildings

Crushing area buildings will be of stick-built design and equipped with dust collection systems.

The primary crushing building will house the ROM hopper equipped with a static grizzly, vibrating grizzly feeder, primary jaw crusher, chutes and additional platework. The rock breaker will also be within the building. In addition, access platforms and reinforced concrete will be utilised for the pad to support the primary jaw crusher. Additional screening and crushing will also be completed prior to the mill feed stockpile. The secondary screen and crusher will be housed in a dedicated building. Conveyors and feeders will be used to control the movement of material between the buildings. A fabric building cover and concrete reclaim tunnel will be used for the mill feed stockpile.

18.5.2 Process Plant Buildings

The process plant complex is comprised of the following separate buildings:

- mill building
- reagents building

- reagents storage building
- gold room.

Large-scale buildings will be constructed from pre-engineered metal, supported on reinforced concrete footings and are complete with concrete slabs and pedestals. To account for winter conditions, buildings will be built with insulated metal panel (IMP) roof and wall cladding. Area cranes will be available for equipment servicing in the process plant.

The mill building will be 32 m (wide) x 36 m (long) x 25 m (high) and will include a ground floor, one elevated concrete floor. The various equipment will be accessed by purpose-built mezzanine platforms for maintenance, service and sampling. The grinding and gravity building will contain the ball mill, sag mill, cyclone feed hopper/pumps, cyclone cluster and trash screen, as well as dedicated areas for the gravity circuit equipment, acid wash column, the elution column and regeneration equipment.

The reagent building will be 20 m (wide) x 40 m (long) x 11 m (high) and will contain the reagent mixing tanks, and dosing tanks. The reagent profile consists of cyanide, lime, sodium hydroxide, hydrochloric acid, carbon, copper sulphate, sodium metabisulphite, flocculant, oxygen and antiscalant. Where possible totes of reagents will be used directly, to conserve space and tankage.

The Reagents Storage Building will measure 18 m (wide) x 21 m (long) x 6 m (high) and will be a fabric building providing space for longer-term storage of reagents for use in the Process Plant. The Reagent Storage Building will be located in close proximity to the main reagents building.

The gold room will house the pregnant solution tank, electrowinning cells, sludge filters, furnace, drying oven and vault. The building will be a two-storey concrete wall structure measuring 11 m (wide) x 16.5 m (long) x 11 m (high).

External parts of the processing plant include a 13 m diameter pre-aeration tank, as well as two leach and seven carbon-in-leach tanks, all of which are 10 m in diameter, and two detoxification tanks that are 6.5 m diameter. The tanks will be accessed by a purpose-built mezzanine platform and walkway to allow servicing, sampling and maintenance. An area crane will provide access to screens, tanks, pumps and agitators. The tailings will report to a pumpbox before being pumped to the tailings storage facility.

18.5.3 Administrative and Offices

The Goliath project will construct a mill office and mine office and change room of modular building design. The process plant area will have administrative and mill offices, while the mine office and change room will be located near the truckshop.

18.5.4 Security Gatehouse

The security gatehouse will be a small, modular building with a single boom gate, located on Tree Nursery Road between Normans Road and Anderson Road. Site inductions for visitors and new employees can be conducted at this point.

18.5.5 Maintenance Shop and Warehouse Building

The plant maintenance shop and warehouse building is in close to the process plant. The building is a fabric building and measures 20 m (wide) x 38 m (long) x 6 m (high).

18.5.6 Truck Shop – Goliath

The Goliath Truck Shop Building is a 17 m (wide) x 38 m (long) x 14 m (high) fabric building located approximately 750 m south of the main process plant area. The Goliath Truck Shop will be equipped with three bays for truck service.

The Goliath Truck Shop is equipped with air compressors, air filters, and pumps for gear oil, engine oil, transmission oil, grease, coolant, and windshield washer distribution.

18.5.7 Truck Wash – Goliath

The Goliath Truck Wash Building is an 18 m (wide) x 25 m (long) x 14 m (high) fabric building located next to the Goliath Truck Shop Building. The Goliath Truck Wash Building will include one wash bay.

18.5.8 Truck Shop Warehouse – Goliath

The Goliath Truck Shop Warehouse is a 16 m (wide) x 24 m (long) x 6.0 m (high) fabric building located next to the Goliath Truck Wash Building.

18.5.9 Truck Shop Office – Goliath

The Goliath truck shop offices, lunchroom and washrooms will be inside a modular building located south of the truck shop. The Goliath truck shop office building will measure 18 m (wide) x 24 m (long) x 2.4 m (high). Additional storage will be available inside shipping containers placed adjacent to the truck shop.

18.5.10 Truck Shop – Goldlund

The Goldlund Truck Shop Building is a 17 m (wide) x 38 m (long) x 14 m (high) fabric building located near the entrance to the Goldlund mine. The Goldlund Truck Shop will be equipped with three bays for truck service.

The Goldlund Truck Shop is equipped with air compressors, air filters, and pumps for gear oil, engine oil, transmission oil, grease, coolant, and windshield washer distribution.

18.5.11 Truck Wash – Goldlund

The Goldlund Truck Wash Building is an 18 m (wide) x 25 m (long) x 14 m (high) fabric building located immediately next to the Goldlund Truck Shop Building. The Goldlund Truck Wash Building will include one wash bay.

18.5.12 Truck Shop Warehouse – Goldlund

The Goldlund Truck Shop Warehouse is a 16 m (wide) x 24 m (long) x 6.0 m (high) fabric building.

18.5.13 Truck Shop Office – Goldlund

The truck shop offices, lunchroom and washrooms will be inside a modular building located northeast of the Goldlund truck shop. The truck shop office building will measure 18 m (wide) x 24 m (long) x 2.4 m (high). Additional storage will be available inside shipping containers placed adjacent to the truck shop.

18.5.14 Existing Facilities

Existing structures originally were formerly used by the Ontario Ministry of Natural Resources and Forestry and are currently used for office space and warehousing. These are approximately 1.5 km north of the future process plant and are accessed via Tree Nursery Road. These facilities will be leveraged during construction and operations as overflow office and warehousing space as required.

18.6 Site Geotechnical

Geotechnical site investigations were previously completed for the Goliath project in 2014, 2017 and 2018 (KP, 2018). The investigations were completed for preliminary characterization of subsurface conditions at the TSF, plant site (original location near waste rock storage area), mine rock stockpiles, collection ponds and overburden stockpiles. Based on the information in KP 2018, a generalized stratigraphic sequence is as follows:

- Topsoil consists of organic sand to organic clay with some roots.
- Sand and silt primarily consist of poorly graded sand and silt, trace clay, loose to compact and moist to wet. The water content varies from 14% to 40%.
- Silt and clay with trace sand, firm to stiff with depth and from low to intermediate plasticity. Plasticity index ranges from 3% to 26%, water content ranges from 21% to 51% and the peak shear strength from 26 kPa to 56 kPa.
- Grey silt generally consists of low plastic clayey silt with some clay, trace sand. Plasticity index varies from 3% to 25% and water content from 20% to 38% with peak shear strength 8 kPa to 54 kPa. Consolidation testing was completed on the grey silt.
- Sand consists of well graded silty sand with some gravel and trace clay, moist to wet with water content of 11% and loose to compact in compactness.
- Bedrock depth varies from visible surficial outcrops to 18 m below ground surface and consists of grey metasediment to gneiss with well-defined beddings dipping at 65° from surface. The bedrock is generally of good quality with rock mass rating (RMR) varying from 61 to 80, however, some zones of poor quality (RMR 21-40) were also encountered near the surface. Schmidt hammer testing on the intact rock core indicated strong rock with 50 MPa to 100 MPa UCS. The hydraulic conductivity measured in the bedrock varies from 1.0×10^{-5} cm/s to 2.0×10^{-4} cm/s based on in-situ testing conducted in the proposed TSF area to estimate the flow of water from the tailings into bedrock.

A review of KP 2018 was completed to identify gaps and requirements for additional geotechnical investigations. These are as follows:

- Surface investigation coverage is limited and not considered to be adequate to support preliminary geotechnical design input. More detailed information is required over a representative area of the site, and existing information including data at depth should be confirmed for general site characterization and preliminary design.
- Potential borrow areas for material sources should be investigated for development of the mine.
- Limited or no hydrogeological information is available. The installation of standpipes and/or vibrating wire piezometers at select locations to measure groundwater levels and allow for in-situ permeability tests is recommended.
- There is an insufficient characterization of soil strata including clay shear strength and consolidation behaviour and silt and sand permeability and shear strength. Additional in-situ testing and laboratory testing should include index testing, strength testing (soil and bedrock), corrosivity and geochemistry.

Based on a review of the available information, additional geotechnical investigations were proposed for the TSF, plant site, mine rock stockpiles, collection ponds, and overburden stockpiles for preliminary site characterization and design purposes.

Geotechnical investigations completed within the TSF in 2022 consisted of drilling and CPT investigations, as well as rock coring and packer testing at select locations (SLR 2022).

Based on the findings of the 2022 TSF investigations, the subsurface soil strata generally consist of the following primary soil and bedrock units in descending order:

- topsoil (peat, sandy silt with organics and rootlets)
- outwash sands (sands, silty sands)
- lacustrine deposit (sandy silts, clayey silt, silt)
- sandy till (sands with varying silt and gravel content)
- metamorphic bedrock.

The surficial geology within the proposed TSF area is considered relatively complex with variable thicknesses of overburden overlying bedrock with varying geologic relief. The lacustrine deposits were found to have a soft to firm consistency. The depth of the bedrock can vary from surface outcrops to significantly deeper overburden deposits generally featured between areas of moderately higher terrain relief. Measurements taken from standpipes during the 2022 investigation indicate preliminary water levels at the time of the investigation ranged from about ~0.6 m (on the west side of the TSF) to ~2.2 m (on the east side of the TSF) below ground surface.

A test pit investigation was also completed in 2022 to identify the depth to bedrock for preliminary siting purposes within the proposed alternate plant site area (southwest of the TSF). The investigation comprised 15 test pits with general grid spacing of about 75 m to 100 m over the plant site area. The test pits were terminated at the bedrock surface which ranged from ground surface (i.e., outcropping bedrock) to about 4.3 m below ground surface.

Further geotechnical investigations and assessment for the mine rock area, water management ponds, overburden stockpile, ore stockpiles, revised plant site areas and other infrastructure are recommended for the Goliath project site. Geotechnical site investigations have not been completed for the Goldlund or Miller project sites.

18.7 Tailings Storage Facility

The Goliath Gold Complex involves the storage of 18.5 Mt of tailings in an on-site TSF. The selected TSF site is similar to the 11 Mt facility assessed in the Environmental Assessment (EA) of the Goliath project but has been expanded by approximately 50% to contain the additional resource. Revisions to the TSF footprint were also made to avoid over-printing Blackwater Creek. The EA, previous PEA, and proposed TSF footprints are shown in Figure 18-7.

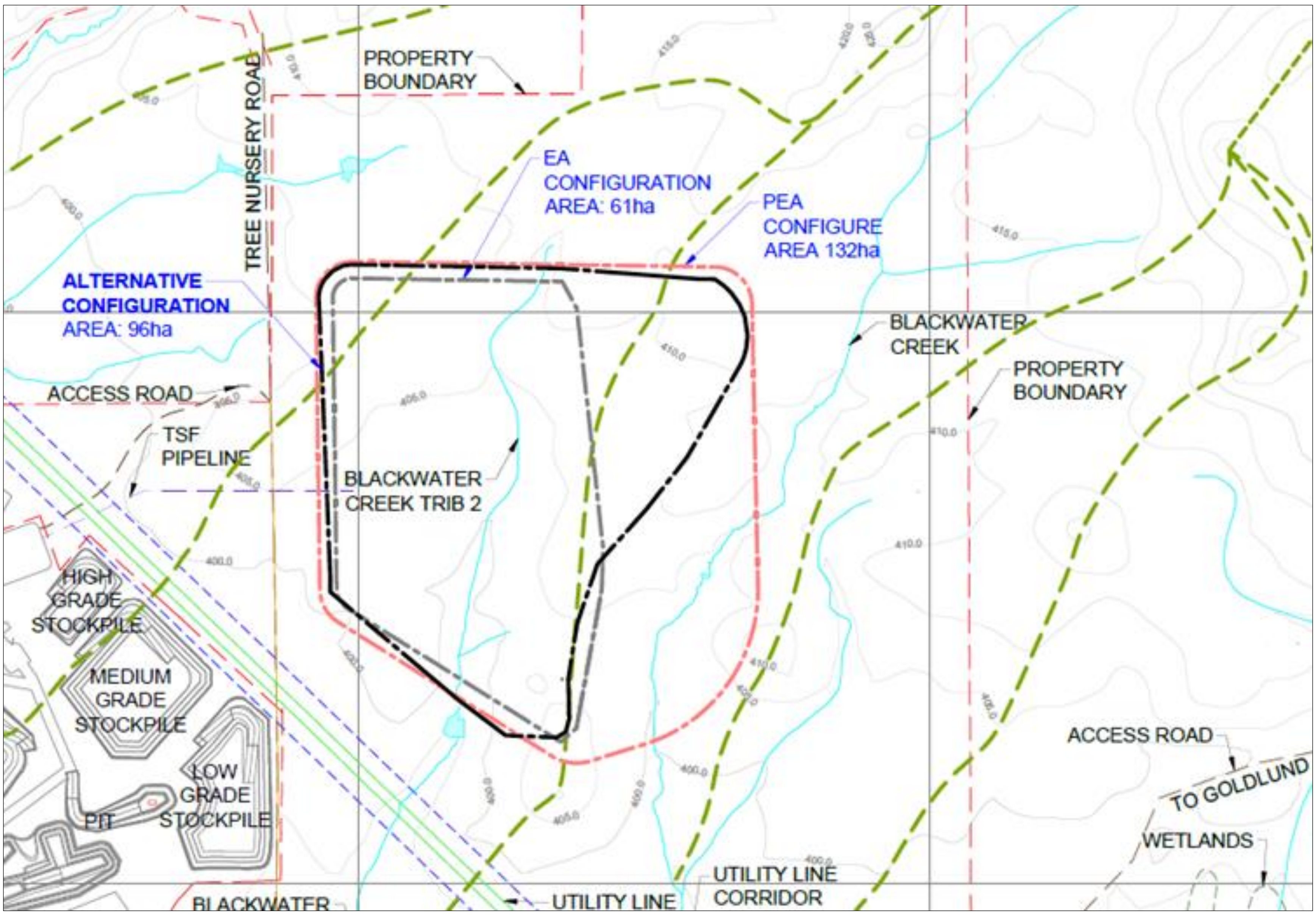
The tailings disposal concept involves the discharge of thickened, non-segregating tailings with a 3% surface slope and containment by rockfill embankment dams with HDPE geomembrane face liners keyed into natural clayey soils that have been determined to be extensive across the TSF area. A provision is included to line the area of the starter pond to allow time to establish the thickened tailings deposit that will displace the pond and blanket the ground surface, thus inhibiting seepage. Tailings bleed and consolidation water and precipitation runoff report by gravity to a lined reclaim pond located at the northwest corner of the TSF. The reclaim pond is initially a fully excavated pond below the original ground surface and is raised to approximately 5 m height above ground at the ultimate stage.

The primary advantage of thickened non-segregating tailings is the ability to maintain a positive surface slope with no need for a pond within the TSF for solid-liquid separation. In this way the deposited tailings volume is increased for any given dam height, dam safety risks are mitigated, and the seepage potential is reduced.

The general arrangement of the TSF, reclaim pond and TSF collection ditches and pond is shown in Figure 18-8.

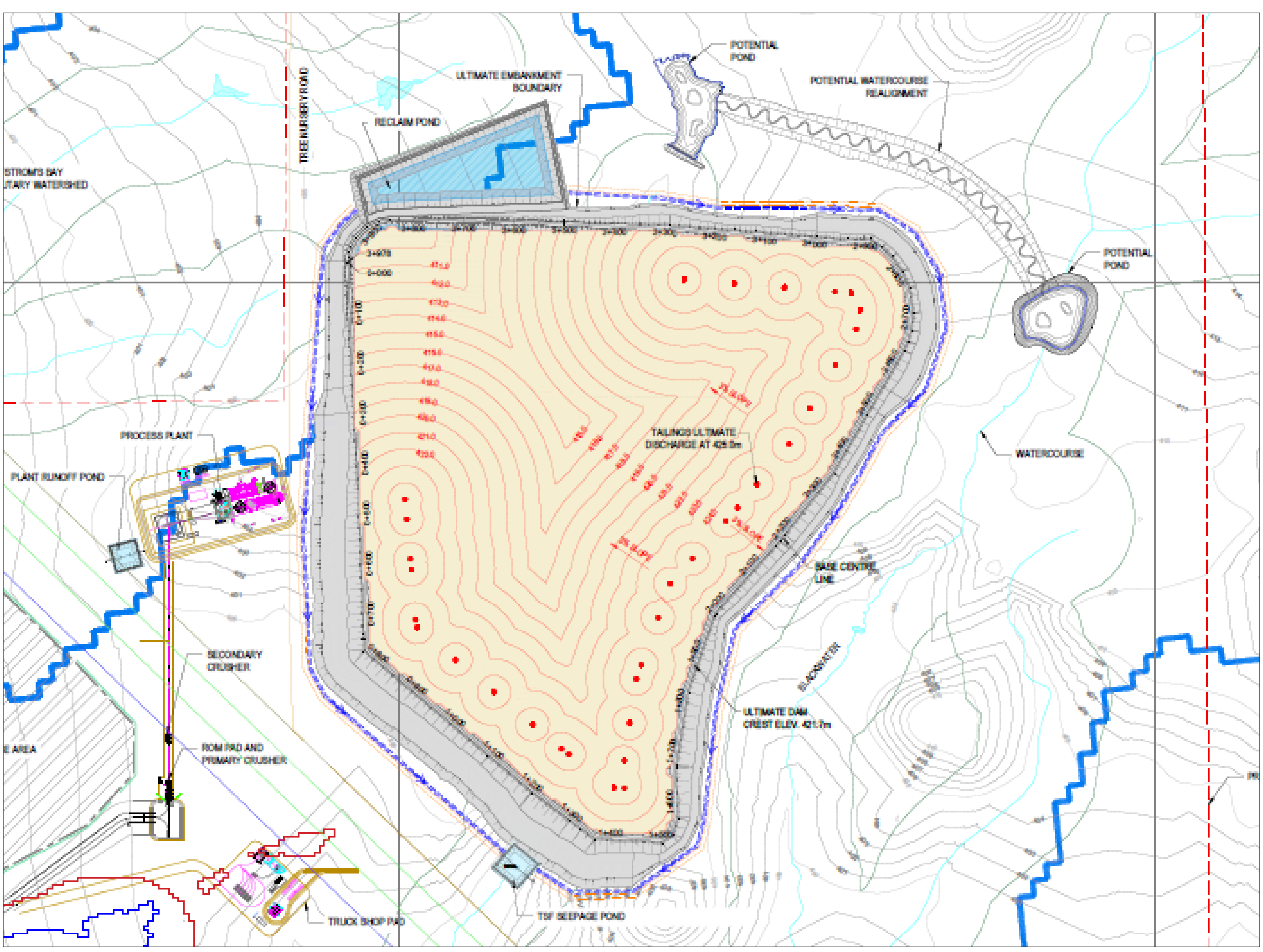
The thickened tailings disposal plan addresses the primary project driver of inhibiting oxidation of the potentially acid generating Goliath tailings by cycling the discharge location to maintain the tailings in a nearly saturated condition until they are buried beneath non potentially acid generating Goldlund tailings, or the closure cover can be placed.

Figure 18-7: TSF Site Selection and Footprint



Source: SLR, 2023

Figure 18-8: TSF General Arrangement



Source: SLR, 2023

18.7.1 TSF Design Criteria & Objectives

The key design drivers for the TSF relate to the tailings geochemistry and requirement to limit seepage due to the proximity to Blackwater Creek. Other important considerations include the intent of the EA conditions, risk management, dam safety stewardship and tailings governance.

Tailings operating data used for the design are summarized in Table 18-5.

Table 18-5: Tailings Operating Data

Parameter	Value	Units
Ore Production – Total	20.9	Mt
Mass of Tailings Used for Stockpile –Total	2.4	Mt
TSF Storage Requirement –Total	18.5	Mt
Tailings Production Rate (Nominal Daily)	6,460	t/d
Tailings / Ore Ratio	1.0	-
Discharge Slurry Density	63	% solids by mass
Specific Gravity of Tailings Solids	2.8	-
Void Ratio of Deposited Tailings (Volume of Voids / Volume of Solids)		
Volume of Deposited Tailings – Annual	1.6	Mm^3/a
Volume of Deposited Tailings – Total	12.6	Mm^3

Geochemical considerations for the Goliath Gold Complex PFS include:

- The Goliath tailings are reportedly all potentially acid generating (PAG), whereas the Goldlund tailings are reportedly non-PAG.
- Isolation of sulphidic tailings to mitigate oxygen ingress into previously deposited tailings has been demonstrated at various mine sites. Various methods include water covers, burial beneath saturated tailings, or isolation by soil covers or non-PAG tailings layers.
- Acid-generation from sulphidic tailings normally lags months or years after initial sub-aerial tailings deposition.
- Cycling the tailings discharge to maintain saturated conditions and limit oxygen ingress and acid generation, in a manner that is supported by geochemical testing and modelling, is a common technique that can be utilized.

Tailings deposition planning will need to address maintenance of the tailings at a high degree of water saturation to inhibit oxidation. Seepage to Blackwater Creek and Thunder Lake tributaries needs to be inhibited to very low values to avoid potential environmental impacts.

Based on the total impounded tailings volume, and the proximity to the power transmission line and Trans-Canada Highway corridors SLR considers that the TSF dams correspond to a hazard potential classification (HPC) of between ‘high’ and ‘very high’ according to the Canadian Dam Association (CDA, 2014). The ‘very high’ HPC assumes the possibility of very high economic losses due to transportation and/or power transmission interruptions that might result from a dam failure. In terms of environmental impacts, an HPC seems appropriate based on significant impacts to important fish and wildlife habitat, but with the possibility of restoration. A dam breach inundation study would be required to demonstrate the potential impacts to justify use of a lower HPC. Seismic and flood conveyance criteria corresponding to a 'very high' HPC have been adopted for the dam design.

The tailings containment dams are earth and rockfill embankments with an upstream geomembrane face liner. The embankment alignments have been optimized to provide the required environmental setback from waterbodies (125 m) and make use of the topographically higher ridge along the eastern side of the TSF (Figure 18-2). The dams have minimum crest width of 10 m for constructability.

Table 18-6: Dam Design Criteria

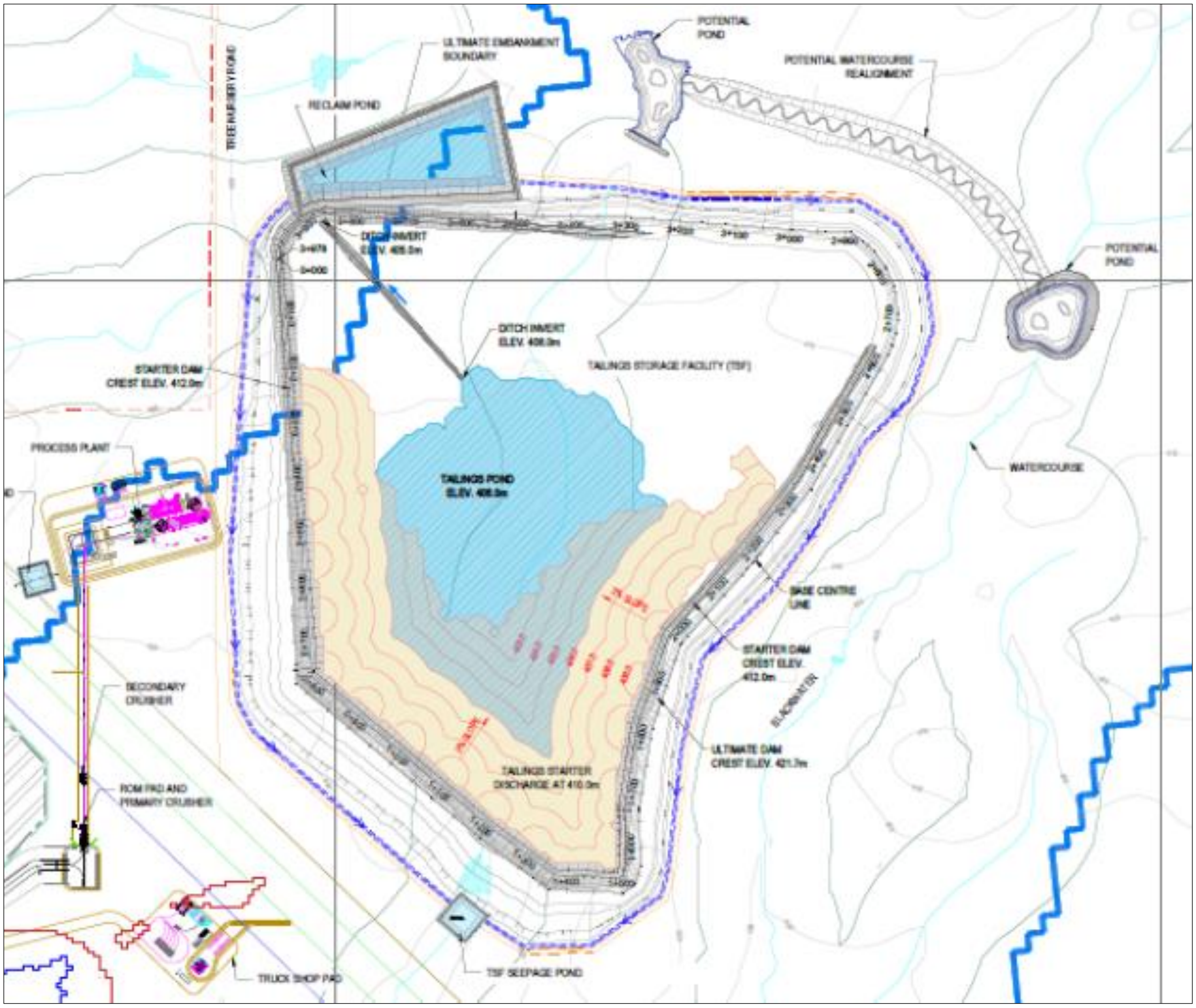
Design Criteria	Value	Source or Reference
Dam Design Criteria		
Dam Hazard Classification Category	Very High	SLR
Inflow Design Flood (IDF) for Spillway Design	Inflow Design Flood (IDF) for Spillway Design	CDA (2014)
Maximum Design Earthquake (MDE)	Maximum Design Earthquake (MDE)	CDA (2014)
Minimum Crest Width	Minimum Crest Width	SLR
Flood & Seismic Design Criteria		
Inflow Design Flood (for Spillway Sizing)	2/3 between 1:1,000 Year and Probable Maximum Flood (360 mm)	CDA (2014)
Maximum Design Earthquake (MDE)	Maximum Credible Earthquake (pga Estimate 0.1 g)	CDA (2014)
Minimum Slope Stability Factors of Safety		
Short Term (end of Construction, before Filling)	1.3	CDA (2014)
Long Term (during Operation and Post-closure)	1.5	CDA (2014)
Pseudo-Static for MCE	1.1	CDA (2014)

18.7.2 Tailings Deposition Plan

Tailings are deposited with a thickened, non-segregating (high-density) slurry consistency estimated at 63% solids by mass based on dynamic thickening tests and a corresponding yield stress of 70 Pa inferred through vane shear tests. The TSF geometry is well-suited to accommodate flexibility in terms of the deposited slope realized between about 1.5% and 3%, provided the main objective that the tailings are non-segregating is met. This avoids the need for a large pond within the TSF to facilitate solid liquid separation. Further rheological characterization of the thickened tailings will be required to advance the design to feasibility level.

The TSF starter arrangement is shown on Figure 18-9 with starter dam crest elevation 412.0 m, and internal drainage channel leading to the initial Reclaim Pond which has an operating water level below the original ground surface.

Figure 18-9: TSF Starter Arrangement



Source: SLR, 2023

Tailings deposition planning was carried out assuming a deposited slope of 3% based on previous experience with similar tailings. Deposition of thickener, non-segregating tailings is advantageous in that a relatively uniform surface gradient can be maintained and there is little or no segregation of tailings particles after discharge. Unlike conventional slurry, where coarser particles fall out of suspension near the discharge points, progressively finer particles settle on the beach and a pond is required to force solid-liquid separation. A thickened tailings beach is relatively uniform and a pond is only required to collect bleed water and surface runoff.

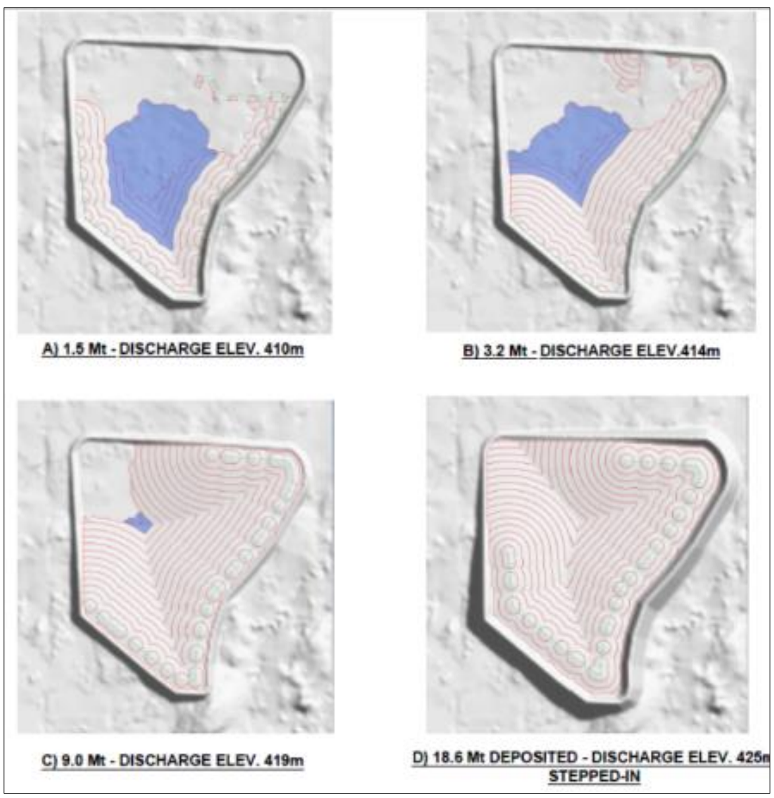
The tailings are deposited from spigots located at or near the crest of the staged-raised perimeter dams. To suit the overall sloping geometry of the deposited tailings the dam crest elevation rises from the low section at the northwest to higher elevations as required around the east, south and west sides to maintain discharge availability.

In general, the deposition set out on Figure 18-5 comprises four stages:

- Start-up – the starter dam crest elevation 412.0 m supports tailings discharge at elevation 410.0 m around the southern portion of the TSF (closest to the mill)
- Stage 1 – the dam crest raised to elevation 415.0 m facilitates discharge from elevation 414.0 m
- Stage 2 – further dam crest raising to elevation 420.0 m facilitates discharge from elevation 419.0 m
- Stage 3 – involves raising to the ultimate dam crest to support discharging tailings in a mounded configuration upstream of the dam by stepping-in the discharge with a moderate slope, reaching a peak elevation several meters above the dam crest elevation.

Figure 18-10 illustrates the deposition plan including the concept of stepping-in the discharge spigots to deposit additional tailings for a given dam crest elevation. This concept is presented to highlight the flexibility with thickened tailings in the event that the dam raising schedule is delayed, for example. Figure 18-9 and Figure 18-10 highlight the positive surface drainage and flow to the reclaim pond at all stages.

Figure 18-10: Tailings Deposition Plan Overview



Source: SLR, 2023

18.7.3 TSF Dam Design

Stability of the rockfill embankment dams slopes will be governed by the foundation soil shear strength of the lightly over-consolidated lacustrine soils as highlighted on Figure 18-11. With toe stabilization berms required to meet stability requirements.

Based on a review of previous geotechnical investigations and field vane tests, Standard Penetration Tests (SPTs) and Cone Penetration Test with pore pressure measurement (CPTu) probes, the undrained shear strength of the lacustrine soils was found to vary from 23 to 57 kPa with an average of 42 kPa across the TSF. The minimum value of 23 kPa was measured in the northeast corner of the TSF dam footprint by field vane test.

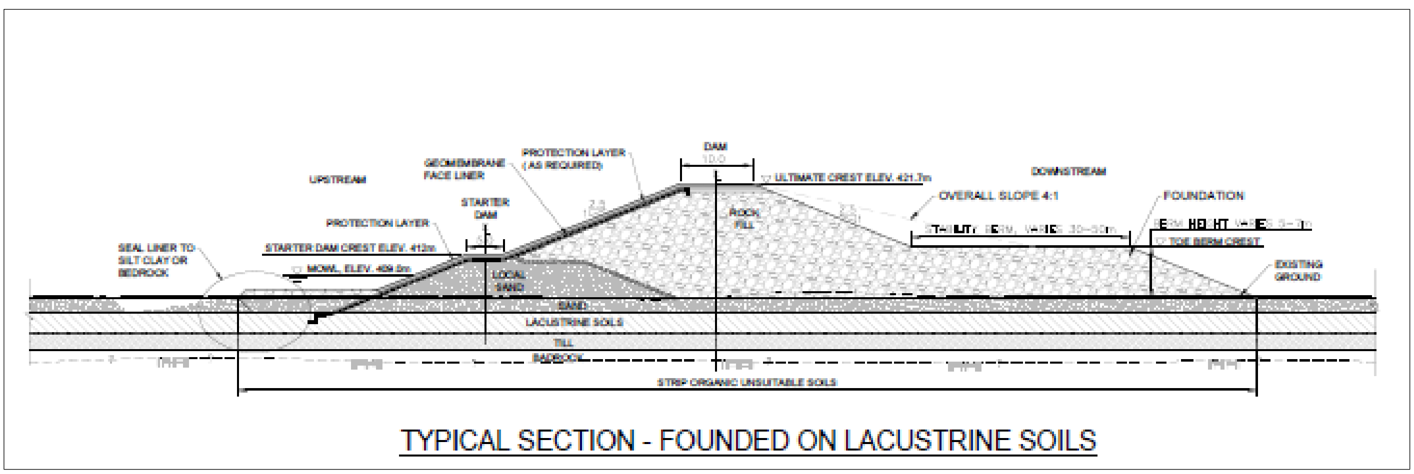
Slope stability modelling has been carried out using an undrained strength ratio of 0.2 with a minimum value of 25 kPa.

To satisfy stability requirements the design includes toe stabilization berms as shown on Figure 18-12 for the starter dams and Figure 18-13 for the ultimate dam. A low berm is also required on the upstream side of the starter dam (Figure 18-11).

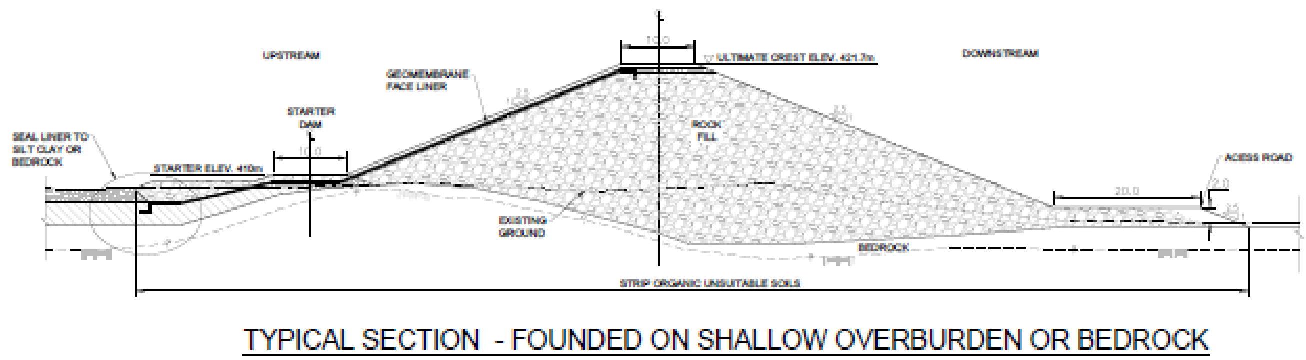
Future geotechnical investigations should focus on delineating the extent of shallow bedrock at the east, south and north sides of the TSF which may allow for optimization of the alignment of the higher segments of the embankment to reduce the size of the toe berms required.

Detailed sequencing of the toe berms construction in stages will be optimized during subsequent design efforts to maximize the consolidated shear strength gain and therefore minimize berms sizes.

Figure 18-11: Typical Sections – TSF Containment Dams



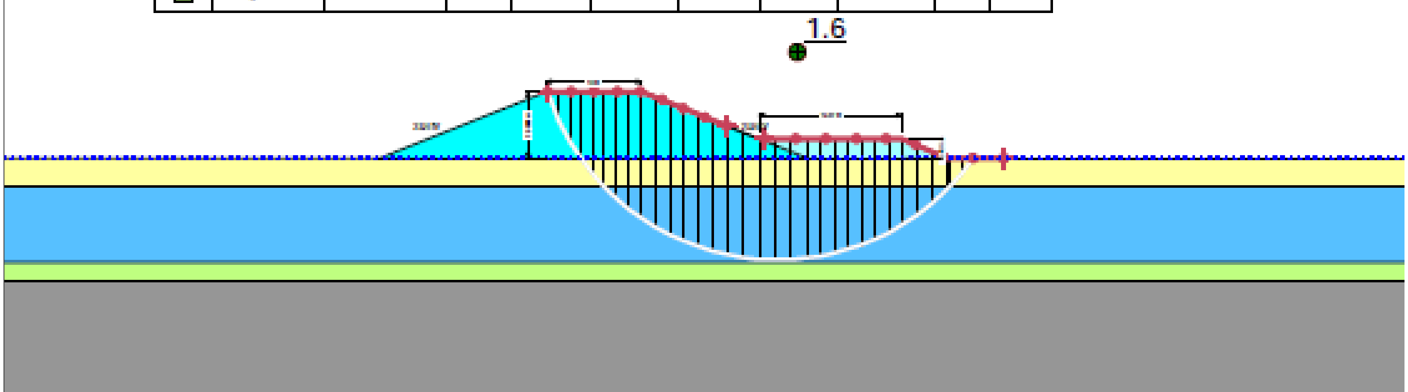
Source: SLR, 2023



Source: SLR, 2023

Figure 18-12: Slope Stability Model – TSF Starter Dam West Dam on Lacustrine Soil Foundation

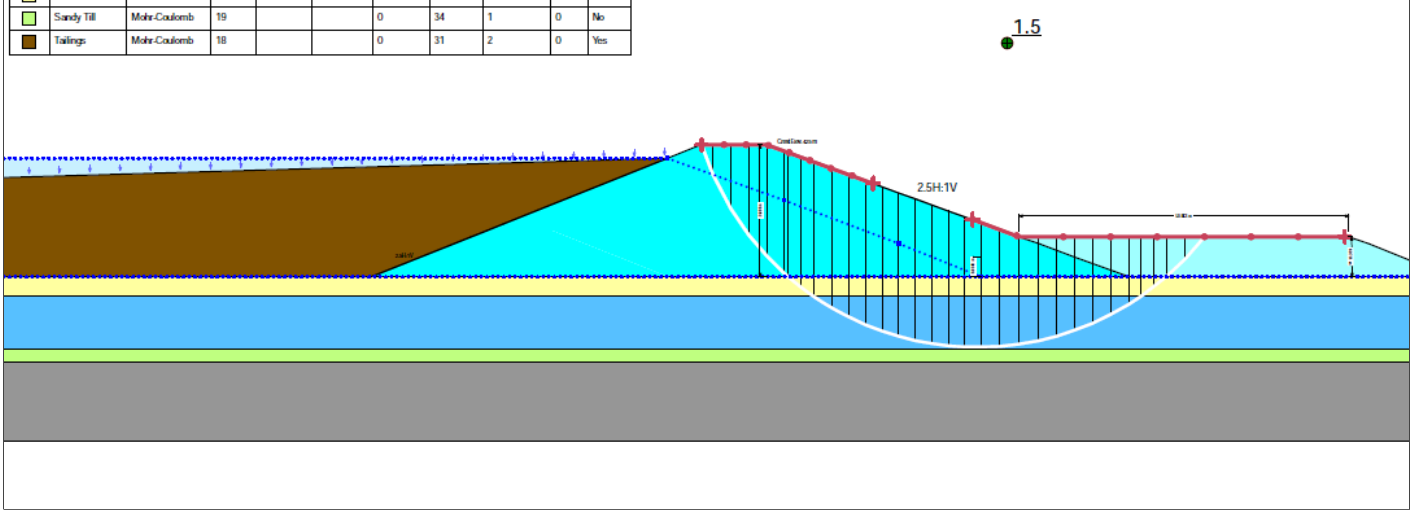
Color	Name	Slope Stability Material Model	Unit Weight (kN/m³)	Minimum Strength (kPa)	Tau/Sigma Ratio	Effective Cohesion (kPa)	Effective Friction Angle (°)	Piezometric Line	B-bar	Add Weight
	Bedrock	Bedrock (Impenetrable)						1	0	No
	Buttress	Mohr-Coulomb	18			0	35	1	0	Yes
	Dam Shell	Mohr-Coulomb	18			0	35		0	Yes
	Lacustrine	SHANSEP	18.5	25	0.2			1	0.3	No
	Outwash Sand	Mohr-Coulomb	19			0	30	1	0	No
	Sandy Till	Mohr-Coulomb	19			0	34	1	0	No



Source: SLR, 2023

Figure 18-13: Typical Section – Slope Stability Model – TSF Ultimate Dam on Lacustrine Soil Foundation

Color	Name	Slope Stability Material Model	Unit Weight (kN/m³)	Minimum Strength (kPa)	Tau/Sigma Ratio	Effective Cohesion (kPa)	Effective Friction Angle (°)	Piezometric Line	B-bar	Add Weight
	Bedrock	Bedrock (Impenetrable)						1	0	No
	Buttress	Mohr-Coulomb	18			0	35	2	0	Yes
	Dam Shell	Mohr-Coulomb	18			0	35	2	0	Yes
	Lacustrine	SHANSEP	18.5	25	0.2			1	0.3	No
	Outwash Sand	Mohr-Coulomb	19			0	30	1	0	No
	Sandy Till	Mohr-Coulomb	19			0	34	1	0	No
	Tailings	Mohr-Coulomb	18			0	31	2	0	Yes



Source: SLR, 2023

18.7.4 Operational and Closure Considerations

To inhibit oxidation of potentially acid-generating (PAG) tailings from the Goliath ore body it will be necessary to cycle the tailings discharge locations to repeatedly re-wet the surface during operations. As a preliminary estimate, the spigots might be cycled monthly or more frequently to maintain a balance between depositing thin lifts (~0.3 to 0.5 m) that drain and partially desiccate to achieve a target strength and density and maintaining a continually re-wetted surface.

After Year 3, the tailings will largely be generated by processed Goldlund ore, which is predicted to be non-PAG, meaning the underlying Goliath tailings will be isolated from interacting directly with atmospheric oxygen. Later, during Years 7 to 9, Goliath tailings account for only 10% of the total tailings and may be geochemically benign. If this blend is determined to be PAG, however, it may need to be isolated so a more complex oxygen-barrier or water-shedding cover can be placed.

The conceptual closure plan for the tailings facility involves constructing a vegetated cover on the deposited tailings. During the active closure phase, batch pumping from the reclaim pond to the pit will take place.

The complexity of the tailings closure cover will depend on the nature of the final tailings deposited near the surface and the depth to the water table within the tailings. Potentially acid-generating tailings at surface will require at least a low permeability, water-shedding cover or perhaps an oxygen-barrier cover including a capillary break(s). Laboratory testing and instrumentation of the tailings during operations will be an important consideration to understand the hydraulic conditions within the tailings including soil-water characteristics (matric suction), water table depth and permeability.

18.7.5 Tailings Geochemistry

Geochemical test programs have been conducted to assess the metal leaching and acid rock drainage (ML/ARD) potential of the Goliath and Goldlund tailings. Tailings samples included in the test program were produced as part of bench scale metallurgical testwork conducted for the project from 2020 to 2022. The test program included a suite of static and kinetic tests to evaluate short-term static conditions and the long-term potential for acid rock drainage and metal leaching.

Synthetic tailings samples included in the tailings geochemistry program included two samples of Goliath tailings, two samples of Goldlund tailings, and four samples of blended tailings. Specific blend ratios were guided by mine planning information available at the time the testing was conducted. Static testing (acid-base accounting, net acid generation testing, elemental content analysis, shake flask extraction testing, and mineralogical testwork) and kinetic testing (humidity cell tests) were conducted on all of the Goliath (n=2), Goldlund (n=2), and blended tailings samples (n=4). Water quality analysis was conducted for synthetic tailings decant water (supernatant) where available. Tailings geochemical testwork is proposed for Miller once metallurgical testwork tailings are available.

Results indicated that the Goliath tailings are potentially acid generating (PAG; NPR<1) and metal leaching, with a short lag time to acid onset (e.g., approximately one year). Metal leaching risks were observed prior to acid onset, under neutral pH conditions, and following acidification of the tailings. Available data suggest that the Goldlund tailings are non-potentially acid generating (NPAG; NPR>2), with an apparently low potential for metal leaching.

Geochemical programs and lab testing indicated that co-processing Goliath and Goldlund ores may generate an NPAG blended tailing with a low potential for metal leaching; however, the ML/ARD potential of the blended tailing is highly dependent on the ore feed characteristics and the proportion of each ore feed used to generate the blend. Geochemical programs are proposed to further evaluate the risk of PAG and/or metal leaching tailings of being produced under planned ore feed profiles.

If NPAG / non-metal leaching tailings are generated, they can be deposited subaerially. However, if PAG and/or metal leaching tailings are generated they will need to be physically isolated to limit their contact with oxygen and surface runoff. PAG tailings will need to be physically isolated prior to their acidification (i.e., within less than one year). However, testwork indicates that such tailings may also release metals prior to their acidification; water quality will need to be monitored and the materials managed accordingly. The use of NPAG / non-metal leaching tailings has been proposed to isolate PAG/metal leaching tailings during mine operations. The efficacy of this approach will be influenced by the depth of the overlying layer of NPAG/non-metal leaching tailings and the tailings saturation level. Geochemical modelling is proposed to further evaluate these factors and support deposition planning.

Closure approaches will need to support the management of PAG and/or metal leaching tailings in a physically isolated condition with a high degree of water saturation to inhibit oxidation and acidification. Such approaches need to ensure that the PAG and/or metal leaching tailings remain in a permanently saturated condition to support the long-term geochemical stability of these materials.

Geochemical programs (e.g., modelling and field evaluations) are proposed to further evaluate tailings deposition approaches and the proposed closure concept to maintain long-term geochemical stability of the tailings.

18.8 Goliath Water Systems

18.8.1 Site Water Balance

A site-wide water balance assessment was conducted to estimate the Goliath water availability from the proposed Goliath Gold Complex footprint on a monthly and annual basis as well as the resulting water surplus/deficit under dry, average, and wet annual climatic conditions.

18.8.1.1 Modelling Approach

A deterministic flow model in linked electronic spreadsheets was developed to simulate the site-wide water balance for the project site during operations. The flow model was set to simulate the transfer of water between the various mine facilities of the project site on a monthly basis over a one-year period. Figure 18-14 presents the flow logic diagram for the last year of Goliath pit mining. Figure 18-15 presents the flow logic diagram for the last year of in-pit tailings disposal.

Three climate scenarios have been considered for water balance simulations: average, 1:20-year dry and 1:20-year wet annual climatic conditions.

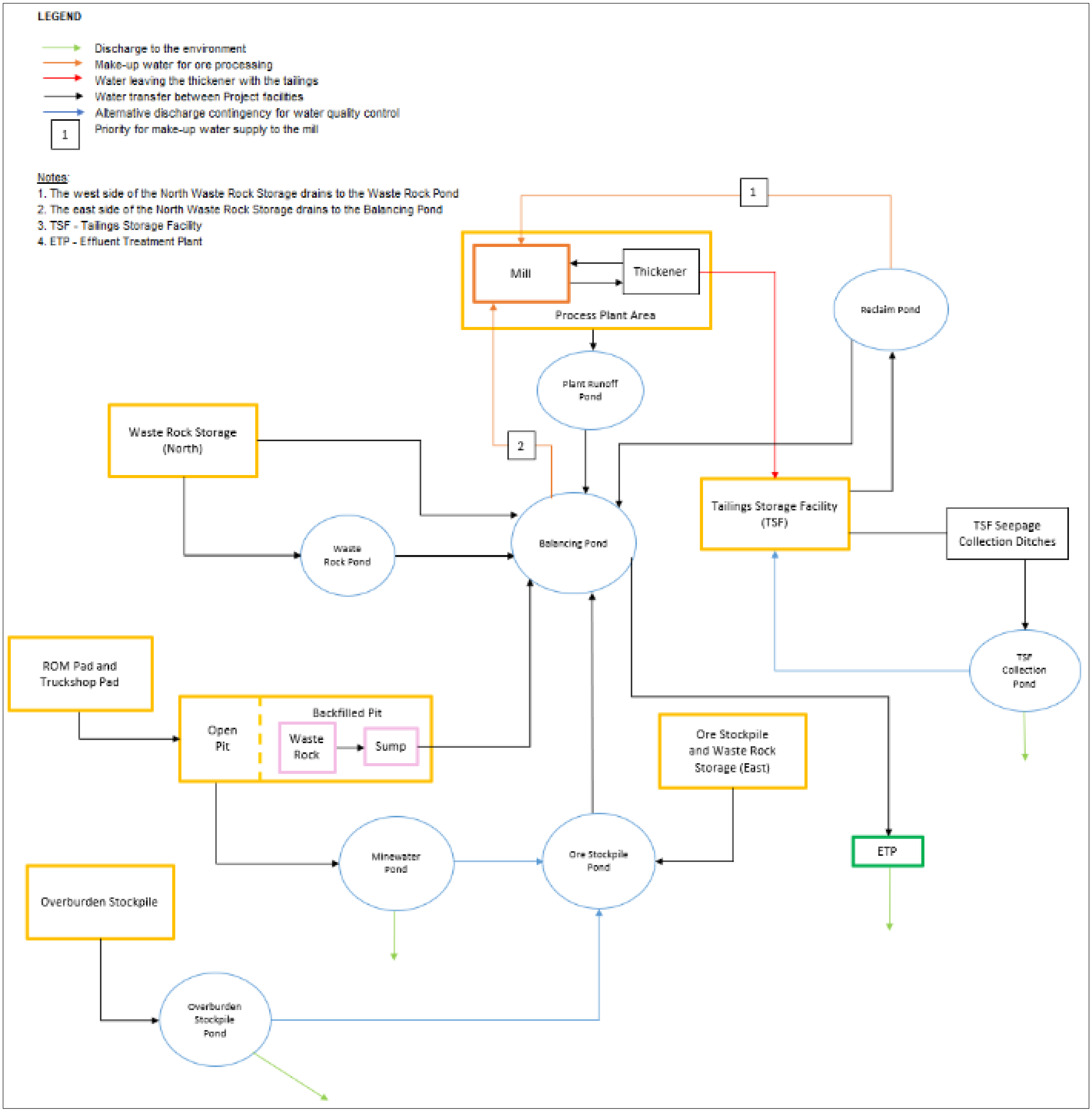
Three snapshots of the project mine life have been simulated for PFS water balance modelling purposes:

1. Production Start-up – Approximately one year after ore processing has started, when the Goliath pit and waste rock storage footprints are not yet fully developed. The mill is processing 5,000 t/d of ore and contact water treated in the effluent treatment plant (ETP) is conveyed from a smaller balancing pond.
2. Last Year of Goliath Pit Mining – The Goliath pit and waste rock storage footprints are fully developed, and tailings are being deposited in the TSF. The mill is processing 6,460 t/d of ore and the balancing pond has been expanded to its ultimate design volume/footprint. Approximately 500,000 m^3 of waste rock have been deposited in the Goliath pit. The north and east waste rock storage areas remain operational (i.e., no closure cover has been constructed).
3. Last Year of In-Pit Tailings Disposal – Tailings are being deposited in the Goliath pit and ore extraction is taking place in the underground mine. The mill is processing 6,460 t/d. Water from the pit is being pumped to the balancing pond to support deposition of tailings sub-aerially, maximize the density and minimize pit lake turbidity. No further disposal of waste rock is taking place in the pit, only tailings. The north and east waste rock storage areas and the ore stockpile are inactive and closed (i.e., a closure cover has been constructed). All the stored overburden has been used in progressive reclamation activities and the overburden stockpile area has been rehabilitated.

In all three snapshots, the primary source of mill make-up water is the reclaim pond followed by the balancing pond. Excess water collected in the reclaim pond is discharged to the balancing pond, not to the environment.

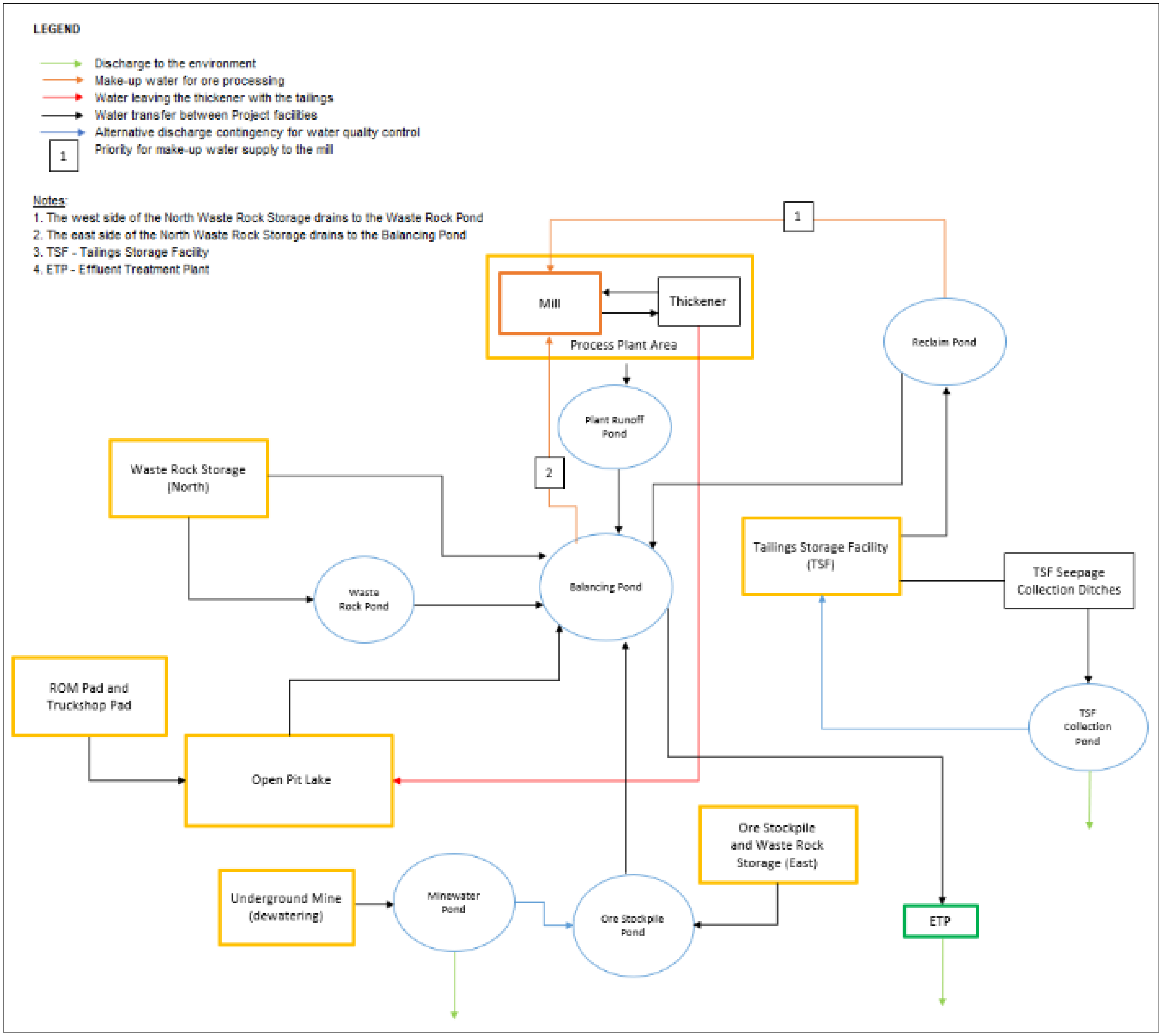
The inflows to the system are the runoff from precipitation, flows associated with processing the ore (i.e., make-up water requirements), seepage (i.e., groundwater inflow) into the Goliath pit, and underground mine dewatering. Evaporation from pond surfaces and the wet tailings beach is accounted for as a loss to the system. The water balance model also accounts for toe seepage from the waste rock storage facilities and the ore stockpile.

Figure 18-14: Flow Logic Diagram for the Last Year of Goliath Pit Mining



Source: SLR, 2023

Figure 18-15: Flow Logic Diagram for the Last Year of In-pit Tailings Disposal



Source: SLR, 2023

18.8.1.2 Flow Model Inputs

Inputs to the flow model included the following:

- pit groundwater inflow rate
- underground mine dewatering rate
- operating data
- process flows associated with ore processing
- precipitation
- evaporation
- catchment areas
- runoff factors.

Steady state groundwater modelling conducted as part of the Federal Environmental Assessment predicted a seepage rate into the proposed fully dewatered mine ranging from 1,000 m^3/d to 1,900 m^3/d, considering open pit and underground workings (Amec 2018a). Under the Base Case scenario developed for the EA, the stabilized seepage rates into the proposed fully dewatered mine (i.e., open pit and underground mine workings) were estimated to be about 1,320 m^3/d. This is the pit groundwater inflow rate used in the flow model. A preliminary underground pit dewatering rate of 620 m^3/d provided by SRK Consulting was used in the flow model. The value is considered a gross estimate at this time.

It is noted that no groundwater numerical model has been developed in support of the PFS. The water balance assessment should be revisited at the next stage of project engineering once results of groundwater numerical modelling become available to inform pit groundwater inflow rates through the operating and closure phases based on modelling predictions. Likewise, the underground mine dewatering rate should be better defined at the next stage of project engineering.

18.8.1.3 Water Balance Modelling Assumptions

The following main assumptions were made in the water balance simulations:

- Water collected in the Balancing Pond can be treated in the ETP all year round.
- Water discharge to the environment takes place from April through December. There is no discharge from January through March. This is a conservative scenario for the PFS in terms of water storage capacity requirements given the shorter time window for discharge of treated effluent to the environment. The ability to discharge year-round via a piped outlet to Wabigoon Lake will be evaluated at the next stage of project engineering.
- In the last year of in-pit tailings disposal the north and east waste rock storage areas and the Ore Stockpile are not operational and have been closed. A closure cover has been constructed, which reduces infiltration from precipitation.

- There is no Overburden Stockpile in the last year of in-pit tailings disposal. All the stored overburden has been previously used in progressive reclamation activities and the overburden stockpile area has been rehabilitated (including its pond).
- Water mounding inside the mine rock area facilities and the ore stockpile will reach steady state by Year 5 of operations. At this time, all the precipitation that infiltrates uncapped waste rock storage facilities and the ore stockpile becomes toe seepage that gets conveyed to the water management ponds (i.e., balancing pond, waste rock pond and ore stockpile pond). This is a conservative scenario for the PFS regarding volume of toe seepage. It is anticipated that the project will undertake progressive reclamation of the north mine rock area, which would reduce the amount of surface runoff and seepage to be treated at the ETP.
- The calculation of ETP nominal flow capacity assumes that all water collected within the project footprint is either used for mine operation activities or conveyed to the ETP for treatment, except for water collected in the overburden stockpile pond (settling pond) which is discharged directly to the environment.

18.8.1.4 Modelling Results

The main observations made from the water balance modelling results for the PFS water management concept are:

- The project exhibits a net positive water balance. Surplus collected water must be discharged to the environment under the 1 in 20-year dry, average and 1 in 20-year wet annual climatic conditions to avoid net accumulation of water in the ponds on an annual basis. Under the dry conditions, water should be stored seasonally to assure availability of water as required to meet the water demand of the mine operation.
- Enough water would be collected from the project footprint to meet the make-up water demand on an annual basis for process plant operation. There is no water deficit on an annual basis, except during dry and average annual climatic conditions at start-up (initial months of mill production). Water harvesting in the TSF would be required prior to mill commissioning.
- During the period of Goliath Pit mining and the period of in-pit tailings disposal water must be withdrawn from both the Reclaim Pond and the Balancing Pond to meet the mill make-up water demand for ore processing.
- Depending on the annual climatic conditions, the total excess water volume to be discharged to the environment in the last year of Goliath Pit mining when tailings are being deposited in the TSF is approximately 0.9 Mm^3/a, 1.7 Mm^3/a and 2.4 Mm^3/a for the 1 in 20-year dry, average and 1 in 20-year wet annual climatic conditions, respectively. These volumes account for water discharged from the ETP and the settling ponds such us the Overburden Stockpile Pond.
- Depending on the annual climatic conditions, the total excess water volume to be discharged to the environment in the last year of Goliath Pit mining in the last year of in-pit tailings disposal is approximately 0.43 Mm^3/a, 1.2 Mm^3/a and 1.9 Mm^3/a for the 1 in 20-year dry, average and 1 in 20-year wet annual climatic conditions, respectively. These volumes account for water discharged from the ETP and the settling ponds such us the Overburden Stockpile Pond.
- The ETP nominal flow capacity required under the average and the 1 in 20-year wet annual climatic conditions is 1,896 m^3/d and 2,443 m^3/d, respectively, for production start-up (Year 1).
- The ETP nominal flow capacity required under the average annual climatic conditions is 4,336 m^3/d and 3,179 m^3/d for the last year of Goliath Pit mining and the last year of in-pit tailings disposal, respectively.

- The ETP nominal flow capacity required under the 1 in 20-year wet annual climatic conditions is 6,318 m^3/d and 5,168 m^3/d for the last year of Goliath Pit mining and the last year of in-pit tailings disposal, respectively.

18.8.2 Surface Water Management

18.8.2.1 Water Management Objectives

The main objectives of the water management plan for the project are as follows:

- develop a flexible water management system that provides Treasury with alternatives to store, convey and treat the contact water across the project site
- ensure a continuous supply of make-up water to the Process Plant
- minimize the use of freshwater by maximizing recirculation of process water and contact water runoff
- minimize the volume of contact water generated within the project footprint where possible
- intercept and store the contact water generated on site (open pit dewatering, underground mine dewatering, surface runoff from disturbed surfaces and toe seepage), by channeling it or pumping it to water management ponds
- store excess contact water that does not meet water quality standards and use it for mine operation activities (if suitable) or provide water treatment prior to its release to the environment
- routing of contact water that needs treatment to the Effluent Treatment Plant
- provide sediment control for surface water runoff from mine facilities that meets water quality standards and can be discharged directly to the environment
- allow safe operation of the mine waste management facilities (i.e., tailings and waste rock) and the associated water management components for a wide range of climatic and operating conditions in facilities that will be continuously growing and expanding.

18.8.2.2 Description of the Water Management System

The water within the basin where the mine facilities will be located is classified into two categories, "contact" and "non-contact" water. Contact water is surface water that has been exposed to excavated materials (e.g., ore, tailings and waste rock) or mining process facilities (e.g., water within the Process Plant). Non-contact water is surface runoff that has not been in contact with any disturbed surface within the project area (i.e., freshwater) and is diverted around the mine facilities, including the proposed realignment channel to divert Blackwater Creek Tributary #2 to Blackwater Creek to create the space required for the TSF. Any non-contact water that mixes with contact water becomes contact water and will be managed as appropriate.

Contact water will be collected in ditches and ponds. Water transfer between ponds and/or facilities such as the mill and the ETP will be achieved mainly with pumps and pipelines although gravity drainage will be prioritized when the topography allows. Non-contact water diversion berms and/or ditches will be implemented, if required.

The surface runoff from the catchment areas within the project footprint will be conveyed to contact water management ponds where the majority of the total suspended solids could settle out prior to sending the water to a user within the mine operation (make-up water for ore processing, water for dust suppression, etc.) or releasing it to the environment. Water collected in the Reclaim Pond will be the primary source of mill make up water. The Balancing Pond will be the secondary source.

Surplus collected water will be discharged to the environment following water treatment, as required, in order to meet applicable water quality standards. EA commitments include meeting Provincial Water Quality Objectives (PWQO) at final discharge, and treating contact water from the project footprint. It is noted that for the purposes of the Federal EA and determination of mitigation/management measures, it was assumed both waste rock and tailings from the Goliath ore deposit were potentially acid generating and that the time to acid onset will be quite rapid (< 2 years).

The water management system includes the following ponds:

- Overburden Stockpile Pond – It will collect surface runoff from the Overburden Stockpile to promote settlement of solid particles before releasing the water to Tributary 1 of the Blackwater Creek.
- TSF Collection Pond – It will collect mainly the surface runoff from the dam crest, the downstream face of the dam and the TSF perimeter ditches. The pond is in the lowest topographic spot around the TSF (where the existing Tributary 2 of the Blackwater Creek runs through), to make possible the collection of seepage from the TSF that could potentially reach the pond. Potential seepage collected in the pond is anticipated to be a small fraction of the total volume of water collected in the pond. If the water quality does not meet the criteria for direct release to the environment (PWQO), the pond provides Treasury with a control point to pump the water back to the TSF or pump it to the Balancing Pond for subsequent treatment in the ETP.
- Minewater Pond – It will collect the Goliath Pit dewatering and the underground mine dewatering to promote settling of solid particles. The Minewater Pond will be engineered to provide Treasury with the flexibility to either convey water to the ETP via the Ore Stockpile Pond and the Balancing Pond under normal operation, or release water directly to the environment if PWQO at final discharge are met. Such flexibility could result in noticeable reduction of flow rates to be treated in the ETP as the operation phase advances. Of note, all water collected in the Minewater Pond will be conveyed to the ETP at the beginning of operations.
- Reclaim Pond – It will collect water from the TSF by gravity. The water will be pumped to the mill to offset mill make-up water requirements. Excess water will be pumped to the Balancing Pond.
- Waste Rock Pond – It will collect surface runoff and toe seepage from the west side of the Waste Rock Storage (North) for pumping to the Balancing Pond.
- Ore Stockpile Pond – It will collect surface runoff and toe seepage from the Ore Stockpile and the Waste Rock Storage (East) for pumping to the Balancing Pond.

- Balancing Pond – It will collect surface runoff and toe seepage from the east side of the Waste Rock Storage (North). It will also receive the water from the Waste Rock Pond, Ore Stockpile Pond, Plant Runoff Pond and excess water collected in the Reclaim Pond. Once mining of the Goliath Pit has ceased, the Balancing Pond will receive water pumped from the Goliath Pit while in-pit tailings disposal is taking place, and underground mine dewatering via the Minewater Pond. Surplus water collected in the Balancing Pond (i.e., in excess of water requirements to support mine operation activities) will be conveyed from the Balancing Pond to the ETP for release to the environment, ultimately reporting to Wabigoon Lake directly via Keplyn's Bay or indirectly via the Blackwater Creek. An important function of the Balancing Pond will be detaining runoff from the project site during high flow conditions allowing a more uniform inflow to the ETP.

- Plant Runoff Pond – It will collect surface runoff from the Process Plant area for discharge to the Balancing Pond.

Construction of the Balancing Pond will be staged. An initial cell will be built prior to commissioning of the mill with capacity to collect flows from the project site at start-up. A second larger cell will be constructed later as the footprint of the project facilities increases, the Goliath Pit deepens, and more contact water is collected and sent to the ETP via the Balancing Pond.

The treatment concept involves the implementation of a modular ETP to allow expansion of the flow capacity by implementing additional modules as needed. At the time of commissioning the mill (i.e., beginning of operations) the ETP should have capacity to treat all flows collected within the project site when production starts and the disturbed footprint is smaller. Expansion of the flow capacity may be implemented in the future for a larger disturbed footprint, as required by operations to meet PWQO at final discharge.

18.8.2.3 Design Criteria for Water Management Facilities

The sizing of the water management ponds that have active discharge to other facilities accounted for the following:

- A provision for dead storage (i.e., minimum pond volume) for pump operation and to provide hydraulic residence time (i.e., sediment control). The minimum pond volume was calculated considering a retention time of 2 days for the maximum monthly inflow in the summer under average annual precipitation conditions. The minimum volume for the Reclaim Pond was defined as a 2 m depth for progressive accumulation of sediment.

- A provision for live storage (i.e., operating pond volume) equal to 3 days of maximum monthly inflow in the spring under the 1 in 20-year wet annual precipitation conditions (obtained from water balance modelling) accounting for spring freshet.

- A provision to store the Environmental Design Flood defined as the flood volume resulting from the 1:100 year, 24-hour rainfall event, which generates 110 mm of rain in 24 hours.

The ponds that have passive discharge to the environment were sized considering a retention time of 5 days for the maximum monthly inflow in the summer under average annual precipitation conditions.

The Reclaim Pond, the Waste Rock Pond, the Ore Stockpile Pond and the Balancing Pond will be lined with geomembrane.

The management of storm flows in the water management ponds, resulting from discrete storm events, is based on two criteria for dam safety: an Environmental Design Flood (EDF) and an Inflow Design Flood (IDF).

The EDF is defined as the runoff resulting from the largest storm event that can be stored without discharge to the environment through the emergency spillway. Retention of water during the EDF requires storage capacity above the normal operating water level (CDA 2014). Runoff events exceeding the EDF will be discharged through the emergency spillway. The 1 in 100-year, 24-hour storm event was used to determine the EDF. Only the ponds discharging actively to other facilities (pumping), which will collect contact water that requires treatment in the ETP, have been sized with storage capacity to contain the EDF. The intent is to equip such ponds with adequate storage capacity to prevent uncontrolled release of water with poor quality to the environment during extreme precipitation events.

The water management ponds will be formed by excavation and feature perimeter berms at locations where it is not efficient to rely solely on excavation to provide containment of targeted pond size. The water management ponds will be equipped with emergency overflow spillways to prevent overtopping of the containment berms. The pond emergency spillways are designed to safely convey the IDF while maintaining a minimum freeboard (i.e., minimum vertical distance between the maximum water elevation and the berm crest elevation).

As indicated in CDA (2013), the maximum flood for which a dam is to be designed or evaluated is termed the IDF. The IDF is the most severe inflow flood (peak, volume, shape, duration, timing) for which a dam and its associated facilities are designed (CDA 2014).

A 'Very High' HPC was selected for the TSF and the Reclaim Pond (Section 18.6.1). A 'Significant' HPC was selected for the remaining water management ponds based on potential adverse environmental impacts associated with uncontrolled release of water with poor quality. According to the CDA guidelines and the selected dam consequence classification, the IDF used for the sizing of the TSF and Reclaim Pond emergency spillway is two thirds the interval between the 1 in 1,000-year, 24-hour storm runoff event and the 24-hour Probable Maximum Flood (PMF). The PMF is determined based on the 24-hour Probable Maximum Precipitation (PMP) rainfall event. The IDF used for the design of the emergency spillways for the remaining water management ponds is the 1 in 1,000-year, 24-hour storm runoff event.

The contact water collection ditches were sized for the peak flow resulting from the 1 in 100-year, 24-hour rainfall storm event.

18.8.3 Process Water Supply & Distribution

Process water from the pre-leach thickener overflow tailings thickener overflow and decant TSF water will report to the process water tank. From there it will be distributed to required users in the plant, such as grinding mills, reagent mixing, and vibrating screen spray bars.

18.8.4 Fresh Water Supply & Distribution

Fresh water will be directed to the fresh/fire water tank, where it will be distributed to required points in the plant, and feed the potable water treatment system, elution circuit, and reagent systems. The bottom section of the fresh/fire water tank will be dedicated for the fire water system.

18.8.1 Potable Water System

The quality requirement for the potable water treatment plant will match the local drinking water guidelines. Fresh water will be sourced from the freshwater intake pump and processed through the potable water treatment skid before being stored in the potable water tank.

Prior to further use, the potable water will be heated by the tepid water heating skid before being distributed to safety showers and other points in the Process Plant facilities. The distribution piping will be heat traced and insulated wherever it is not inside a heated building. Where necessary, manual drain points will be included.

18.8.2 Fire Water System

All facilities will have a fire suppression system in accordance with the structure's function. Fire water will be distributed with an underground ring main network around the facilities. All buildings will have hose cabinets and handheld fire extinguishers. Electrical and control rooms will be equipped with dry-type fire extinguishers. Ancillary buildings will be provided with automatic sprinkler systems. For the reagents, appropriate fire suppression systems will be included according to their material safety datasheets.

18.8.3 Sewage Collection

A domestic effluent and sanitary system package will be supplied at the Process plant area to treat all domestic waste collected within the site. The collection network will be underground. Office and domestic waste will be collected and disposed of off site in accordance with applicable regulations.

18.8.4 Effluent Water Treatment

The effluent treatment plant (ETP) will be located on the Goliath Property to intake, treat and discharge contact water from the Goliath project. The plant will be engineered by a specialist vendor and customized for the application considering influent water quality data, effluent discharge requirements from the completed Federal EA process (20.2.4), and the Goliath site-wide water balance (18.7.1).

The proposed plant will include a two stages of metal precipitation, the first for iron and second for other metals. The precipitated metals will be collected in a flocculated settling stage. After settling the effluent will be treated for sequential destruction of cyanide species, ammonia and nitrate in biological reactors. A recirculating load of effluent is provided for periods of low flow to maintain a sufficient base of bacterial loading. The plant will be expanded from the initial installation in the first year of project operations to accommodate increased treatment flow requirements.

Discharge from the ETP will be released to the environment, ultimately reporting directly or indirectly to Wabigoon Lake via Keplyn Bay or Blackwater Creek, respectively.

18.9 Goldlund Water Systems

18.9.1 Goldlund Surface Water Management

Water management for the Goldlund site will consist of decentralized water treatment and management consisting of water management ponds, diversion berms, drainage ditches and pumps to collect and contain surface water runoff from the waste rock stockpiles/rock storage facilities, overburden stockpile and pits (including groundwater seepage). The water management infrastructure is designed with a decentralized water treatment approach with gravity drainage for non-pit features to reduce pumping requirements. Pumps will be required to dewater the five open pits.

This considers only water management infrastructure based on a Q25 return period. Water will be managed via in-pit pump and sedimentation ponds around infrastructure and discharge into the receiving environment. Post-Pit #1 exhaustion, the depleted open pit will be used to manage water for Open Pit #2, and so on. It is assumed that water will only require treatment for suspended solids, and therefore only retention is required for settling via gravity (vs. treatment) prior to discharge.

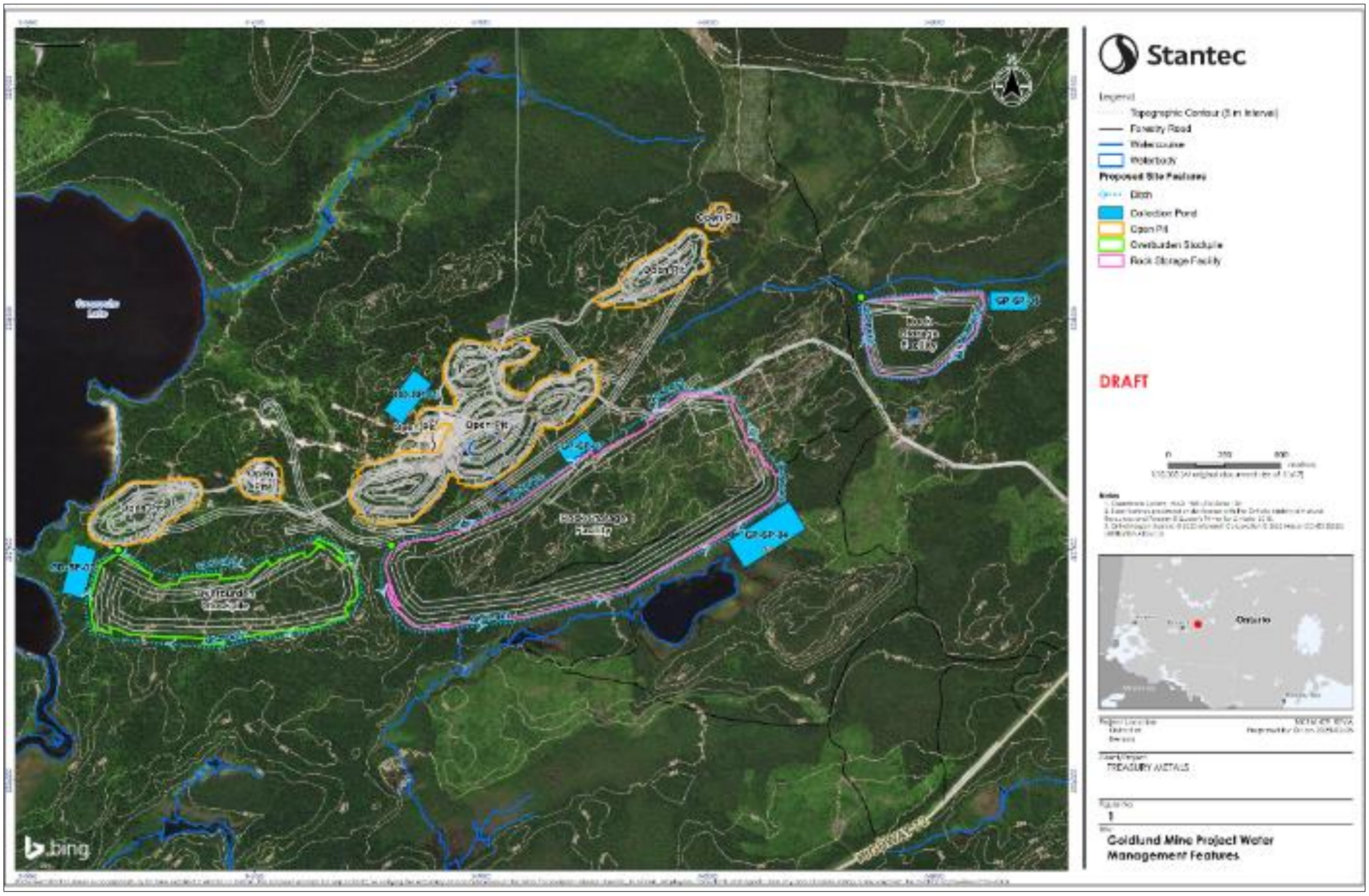
Perimeter ditches will be constructed to convey surface runoff, toe drainage and groundwater seepage from the stockpiles to water management ponds. A standard trapezoidal design will be used for the ditches with 2H:1V side slopes tied to existing grade. Ditch excavation materials will be side cast and used to construct a shallow berm (0.5 m in height). The berms will be constructed on the outside bank of the ditches (opposite side of the stockpiles). Ditches will be seeded with vegetation in lower slopes zones (<5%) and lined with non-acid generating (NAG) waste rock in steeper zones for erosion protection (>5%).

Water management ponds will be designed to provide storage with a permanent pool providing inactive storage at the pond's low operating water level to promote solids settling and store sediments. The ponds are designed with 3H:1V side slopes, 0.5 m permanent pool depth, 2.5 m active pond depth, 1.0 m dam height constructed above grade and a 4.0 m width on the dam crest for vehicle access and a 2H:1V outer slope. Inlets to the water management ponds will include vegetated and rock lined ditches and ditch outlet splash pad/aprons for energy dissipation to reduce the velocity of the flow into the pond thus reducing velocity currents in the ponds. The ponds will have free-flowing continuous discharge via multi-stage outlets controls to provide slow, controlled releases after appropriate sedimentation settling for different design precipitation events. Further treatment requirements for surface water runoff and pit dewatering, in addition to sedimentation of suspended solids, will be assessed as part of additional studies developed as design is progressed from prefeasibility to feasibility study level design.

Figure 18-16 on the following page presents locations of the water management components.

Runoff from the overburden stockpile will drain into a series of perimeter ditches and to a water management pond that will discharge into Crossecho Lake within the Finlayson Creek watershed. The central rock storage facility will be served by a series of perimeter ditches and water management ponds that will discharge into either Crossecho Lake or an unnamed tributary to Tom Chief Lake within the Finlayson Creek watershed. The east rock storage facility will be served by a series of perimeter ditches and a sedimentation pond that will discharge into Franciscan Lake within the Franciscan Creek watershed. Runoff and seepage from the central open pit will be pumped to a water management pond that will discharge to Crossecho Lake within the Finlayson Creek watershed. Following full excavation of the central open pit, dewatering flows from other open pits will be pumped to the central open pit to accelerate pit filling.

Figure 18-16: Locations of Water Management Components



18.9.2 Goldlund Site Water Balance

Ore will be direct shipped off site to the Goliath site, with no tailings produced at the Goldlund site. The mine water demand for the Goldlund site processes will be relatively very low (e.g., dust suppression). Based on project component surface runoff, seepage collection and pit dewatering, it is estimated that less than 10% of all site-generated flows will be reused in Goldlund site operations with most flows discharging from the respective water management ponds to the environment after regulatory discharge criteria are met. Surface water will be managed via a series of sedimentation ponds, gravity fed through ditches around major infrastructure. Water reused on site for uses such as drilling and dust control will be drawn from water management pond outlets and be compliant with discharge criteria.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

Treasury Metals has not completed any formal marketing studies with regards to gold production that will result from the mining and processing of ore from the Project into gold/silver doré bars. Gold and silver production is expected to be sold on the spot market, with the terms and conditions of sales contracts expected to be typical of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold and silver are bought and sold, and it is not difficult to obtain a market price at any particular time. The gold and silver markets are very liquid with a large number of buyers and sellers active at any given time.

19.2 Commodity Price Projections

The economic analysis for the Project was performed assuming a base case gold price of US$1,750/oz, and silver price of US$21/oz. These prices are in alignment with consensus forecasts from numerous financial institutions. As of February 21, 2023, the trailing two-year gold and silver prices were US$1,804 and US$23.22 respectively, and the trailing three-year gold and silver prices were US$1,806 and US$22.72 respectively. The exchange rate used in the economic analysis is C$1.00:US$0.75.

19.3 Contracts

Treasury Metals plans to contract out the transportation, security, insurance, and refining of gold/silver doré bars. Treasury metals may enter into contracts for forward sales of gold and silver or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in Canada and in countries throughout the world. For the PFS, costs of C$5.0/oz Au and C$0.26/oz Ag were assumed for transportation and refining.

19.4 Comments on Market Studies and Contracts

The QP is of the opinion that the marketing and commodity price information is suitable to be used in cashflow analysis to support the 2023 Pre-feasibility Study.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

The Goliath, Goldlund and Miller projects are three distinct project properties and will go through their own permitting processes as each site is developed. Accordingly, Section 20 presents a summary of environmental studies and site characterizations, ongoing environmental monitoring, anticipated project permitting efforts required for the proposed life of mine, and other relevant environmental considerations for each of the three projects. Treasury Metals' overall social and community programs are also discussed.

Requirements and plans for mine rock storage, tailings disposal, and related site water management facilities for all three project sites are detailed in Section 18 that pertains to site infrastructure.

20.1 Regulatory Framework Overview and Status

An overview of the federal regulatory framework associated with the three project sites is presented below to provide context on the status of environmental baseline studies, monitoring programs, and permitting for each project site to date.

20.1.1 Goliath Project

Having established that the proposed Goliath project would likely be subject to the *Canadian Environmental Assessment Act, 2012* (CEAA 2012), Treasury Metals initiated the federal environmental assessment (EA) process with submission of a Project Description to the Canadian Environmental Assessment Agency (CEA Agency) on November 26, 2012. The Goliath project was defined in the Project Description as an open pit and underground gold mine and associated infrastructure, with an ore input capacity of 2,500 tonnes per day (t/d) and an anticipated mine and mill life of up to 12 years. Accordingly, the proposed Goliath project involved designated physical activities as prescribed under the associated CEAA 2012 regulation, as follows:

- Regulations Designating Physical Activities, Item 15(c) [since revised to 16 (c)]: The construction, operation, decommissioning and abandonment of a [new rare earth element mine or] gold mine, other than a placer mine, with a production of 600 t/d or more.

The CEA Agency subsequently determined on January 17, 2013 that the Goliath project did require an EA and issued final guidelines by February 21, 2013 for the preparation of an Environmental Impact Statement (EIS) to conduct an environmental assessment pursuant to CEAA 2012. On April 25, 2015, Treasury Metals submitted the Goliath Project EIS for the construction, operation, decommissioning, and abandonment of an open pit and underground gold mine and associated infrastructure, having an ore production capacity of 5,424 t/d and an ore input capacity of 3,240 t/d, with an anticipated mine and mill life of 12 years, which was subsequently accepted by the CEA Agency. Following the CEA Agency's review and stakeholder comment period, and Treasury Metals' provision of responses to requested information, Treasury Metals submitted a final revised EIS on April 20, 2018. Following additional technical review and stakeholder comments, Treasury Metals provided a second round of responses and information in March 2019, whereupon the CEA Agency released their Draft Environmental Assessment Report on June 14, 2019.

On August 19, 2019, the Minister of the Environment issued their Decision Statement. The Decision Statement concluded, with required mitigative measures and conditions, that the Goliath project is not likely to cause significant adverse environmental effects, and the Goliath project was thereby permitted to proceed under CEAA 2012. The conditions outlined in the Decision Statement are legally binding and have been considered as part of the ongoing environmental monitoring programs and development of the Goliath project. Furthermore, Treasury Metals is obligated to notify the federal and relevant authorities and consult with specified Indigenous groups regarding any changes to the Goliath project that may result in potential adverse environmental effects, including proposed mitigation measures, and follow-up requirements to be implemented pertaining to the proposed changes.

The Goliath project, as presented in this technical report, has undergone optimizations since the 2018 EA/EIS, including optimized designs of the tailings facility, mine rock stockpiles and water management systems. The milling of ore from the Goldlund and Miller project sites at the Goliath project site is also being proposed, with an increased milling rate of up to 6,480 t/d. Additional environmental data continue to be collected to support the proposed changes, Indigenous consultation is ongoing, and preparation of the required notification, as stipulated in the Decision Statement, is currently in progress.

The currently proposed design changes at the Goliath project site are not anticipated to significantly affect the overall description of the Goliath project as assessed under the federal CEAA process, nor are they expected to trigger a review of the environmental impact assessment under the current *Canadian Impact Assessment Act* (IAA) and the associated Regulations Designating Physical Activities (IAA Regulations), which came into force in August 2019. More specifically, the activities listed in the updated IAA Regulations that could potentially apply to the optimized Goliath project site, and which are not expected to be triggered, are:

- Item 19 (c): The expansion of an existing mine, mill, quarry or sand or gravel pit in the case of an existing metal mine, other than a rare earth element mine, placer mine or uranium mine, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore production capacity would be 5,000 t/d or more after the expansion; and

- Item 19 (d) The expansion of an existing mine, mill, quarry or sand or gravel pit in the case of an existing metal mill, other than a uranium mill, if the expansion would result in an increase in the area of mining operations of 50% or more and the total ore input capacity would be 5,000 t/d or more after the expansion.

Nevertheless, while the IAA Regulations are not expected to be triggered, Treasury Metals must still submit a Project Change Notice (PCN) for the Goliath project to notify the Impact Assessment Agency of Canada (IAAC) of changes to the project in accordance with the project's EA Decision Statement. In parallel to the PCN, Treasury Metals must also undertake the required Indigenous consultation on the respective changes.

Based on the above, Treasury Metals is preparing to commence with applicable provincial and federal permitting processes to align with the PCN and Indigenous consultation; details of the anticipated permits are discussed further in Section 20.2.4.

20.1.2 Goldlund and Miller Projects

Based on current proposed designs for the Goldlund and Miller projects, neither project is expected to trigger any of the applicable designated physical activities stipulated under the IAA Regulations listed above (i.e., 19(c), or 19(d)) or the following item listed for a new mine:

- Item 18 (c): The construction, operation, decommissioning and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 t/d or more.

Neither project is therefore expected to require completion of a federal impact assessment. Accordingly, these two sites can proceed with provincial and federal environmental permitting, including potential provincial EAs, which will need to be supported by appropriate baseline data. Further details of anticipated permits and approvals for these two project sites are discussed in Sections 20.3.4 and 20.4.3, respectively.

20.2 Goliath Project Site

20.2.1 Baseline / Environmental Studies

20.2.1.1 Overview

Infrastructure for the Goliath project is located primarily within the Blackwater Creek sub-watershed (Figure 18-1). Two large lakes are proximal to the project area; Wabigoon Lake, and Thunder Lake which drains into Wabigoon Lake via Thunder Creek. Small tributaries drain from the project site into the Blackwater Creek, which flows southwest to Keplyn's Bay in Wabigoon Lake. Other nearby tributaries (Hoffstrom's Bay Tributary and Little [formerly Unnamed] Creek) are located north and west of the project site, respectively, and drain into Thunder Lake. The watersheds of these watercourses, and the bays of Thunder Lake and Wabigoon Lake that receive these watercourses, were defined during the EA/EIS studies as the Local Study Area (LSA), while the Regional Study Area (RSA) was delineated to include Thunder Lake and Wabigoon Lake.

Environmental data collection was initiated in 2008, with more rigorous baseline studies being undertaken in 2010. Additional environmental studies were subsequently undertaken to support the federal environmental assessment process. Since the completion of the Goliath project federal EA/EIS in August 2019, Treasury Metals has continued to collect supplemental environmental baseline monitoring data while optimization and engineering designs for the Goliath project are progressed. The following subsections provide a summary of the environmental baseline studies and site characterizations undertaken at the Goliath project site to date. The majority of the available data have been extracted from the various studies that were conducted for the EA/EIS, with additional or updated information provided where available.

20.2.1.2 Water Quality and Aquatic Studies

As ongoing baseline data collection has advanced, surface water sampling locations and frequencies have continued to be refined to develop a robust understanding of the project footprint.

Monthly surface water quality sampling began in November 2010 with samples being collected from Thunder and Wabigoon Lakes and four streams in the vicinity of the Goliath project. In 2012 and 2013, DST Consulting Engineers Inc. (DST) conducted surface water quality monitoring at additional creek and lake locations adjacent to the project site, with results indicating that total iron concentrations were naturally above the Provincial Water Quality Objective (PWQO) criterion at 14 of the 15 sampling locations. The results also indicated that pH was lower and total metal concentrations of cobalt, copper, lead, selenium, silver, vanadium, and zinc were naturally elevated above the PWQO on several occasions at a number of sample locations throughout the study area (DST 2014). Many of the elevated metals concentrations were often correlated with higher suspended solids concentrations. Results from ongoing water quality monitoring indicate consistent results with earlier sampling programs. These results are similar to other surface waters throughout northwestern Ontario and are indicative of oligotrophic lakes in general.

On a regional level, mercury concentrations in the English-Wabigoon River system (downstream of the Goliath project) have been shown to be elevated, and the production of methylmercury is reported to have caused adverse effects on human health to members of Asubpeeschoseewagong Netum Anishinabek (Grassy Narrows First Nation). Treasury Metals is mindful of the sensitivities regarding mercury in the regional area and has made firm commitments regarding effluent discharge as part of the federal EA review for the Goliath project to ensure that the environment and human health are protected.

Various aquatic studies, including sediment, benthic invertebrate, fish and fish habitat studies have also been undertaken since 2010, as summarized in Table 20-1.

Table 20-1: Goliath Project Baseline Sediment Quality and Aquatic Life Data Collection (2011 to 2021)

Report	Year	Components	Areas
KCB 2012	2010, 2011	sediment quality and benthic invertebrate community (Blackwater Creek only), fish community, fish habitat	Blackwater Creek, Thunder Lake Tributaries (including Hoffstrom's Bay Tributary and Little Creek), east bays of Thunder Lake, and Thunder Creek
DST 2014 (EIS App. P)	2012, 2013	sediment quality, benthic invertebrate community, fish community, fish habitat	Blackwater Creek, Thunder Lake Tributaries (including Hoffstrom's Bay Tributary and Little Creek), east bays of Thunder Lake, Keplyn's Bay of Wabigoon Lake, and a Reference Bay of Wabigoon Lake
KBM 2017 (EIS App. Q)	2010 to 2016	benthic invertebrate community, fish habitat, fish community, fish spawning, fish tissue quality	Blackwater Creek, Thunder Lake Tributaries (including Hoffstrom's Bay Tributary and Little Creek), east bays of Thunder Lake, Keplyn's Bay of Wabigoon Lake, and a Reference Bay of Wabigoon Lake
Amec Foster Wheeler 2018b (EIS App. Q)	2010 to 2017 (summary)	benthic invertebrate community, fish habitat, fish community, fish spawning, fish tissue quality	Blackwater Creek, Thunder Lake Tributaries (including Hoffstrom's Bay Tributary and Little Creek), east bays of Thunder Lake, Keplyn's Bay of Wabigoon Lake, and a Reference Bay of Wabigoon Lake
Portt 2019	2018	benthic invertebrate community, fish habitat, fish community, fish spawning	Blackwater Creek, Little Creek, Hoffstrom's Bay Tributary, and Thunder Lake Tributaries 2 and 3
Portt 2021 (raw data)	2019	benthic invertebrate community, fish habitat, fish community, fish spawning	Blackwater Creek, Little Creek, Hoffstrom's Bay Tributary, and Thunder Lake Tributaries 2 and 3
Minnow 2022 (draft)	2020, 2021, 2022	benthic invertebrate community, fish habitat, fish community, fish spawning, fish tissue quality	Blackwater Creek, Little Creek, Hoffstrom's Bay Tributary, Thunder Lake Tributaries 2 and 3, and Keplyn's Bay of Wabigoon Lake

Sediment samples were collected by Klohn Crippen Berger (KCB; 2012) in October 2011 at four stations in Blackwater Creek and at the outlet of Thunder Creek, concurrently with benthic invertebrate samples (Table 20-1). Results indicated that total organic carbon (TOC), chromium, copper, iron, manganese, and nickel concentrations were above the Provincial Sediment Quality Guidelines (PSQG) lowest effect levels (LELs) at most of the Blackwater Creek stations, and below the severe effect levels (SELs). Copper and zinc concentrations were above their LEL (and below their SEL) in the sediment sample at the Thunder Lake outlet. TOC and manganese concentrations were above their SEL at one Blackwater Creek location and at the Thunder Creek outlet, respectively. Sediment samples collected by DST in October 2012 from creek and lake locations were analysed for nutrients, mercury, and zirconium only. All results were below PSQG LELs, with the exception of phosphorus in one sample from Wabigoon Lake, one from Unnamed Creek, and both samples from Wabigoon Lake

Reference area. Such exceedances are not unusual due to the metal-rich nature of the bedrock of the Canadian Shield region.

Aquatic studies to date include characterization of fish habitat and community structure of the local water bodies, including benthic communities and small and large fish populations (Table 20-1). Baseline investigations of fish and fish habitat were conducted by KCB in 2010 and 2011, and by DST in 2012 and 2013. Additional fish sampling was conducted by Treasury Metals in 2014. C. Portt and Associates (C. Portt) subsequently conducted reconnaissance level investigations at a number of locations and side-scan sonar investigations of Keplyn's Bay on Wabigoon Lake and an unnamed bay of Thunder Lake in 2016. Results of these studies were consolidated and reported by C. Portt (2016; and presented in Amec 2018b). Aquatic environmental baseline data collection continued post-publication of the Goliath Project EIS, including reporting of 2018 and 2019 data (Portt 2019, 2021) and completion of additional monitoring by Minnow Environmental Inc. (Minnow) in 2020, 2021 and 2022.

In general, benthic invertebrate communities were reflective of general conditions found throughout northern Ontario. Lake samples were characterized by invertebrates which were resistant to poor water quality (Chironomidae) and fine-grained substrates. Creek samples had more Ephemeroptera, Plecoptera, Tricoptera (EPT) members which are commonly known to be good indicators of clean, well oxygenated water.

The fish community within Blackwater Creek is primarily comprised of small-bodied forage fish, although within the lower reaches near Wabigoon Lake, large bodied species, such as northern pike, could ascend from the lake. Seven fish species were identified in Blackwater Creek during baseline sampling conducted between 2010 and 2017. Fish communities in tributaries of Thunder Lake were also primarily composed of small-bodied fish, which included up to thirteen different species. Twenty-three fish species were identified within Wabigoon Lake, while fourteen species were identified within Thunder Lake.

Thunder Lake is a cold-water lake that supports a large-bodied fish community, including lake trout, lake whitefish, walleye, northern pike and smallmouth bass. The lake contains several areas of spawning habitat for lake whitefish and lake trout. Thunder Lake supports both recreational and commercial fishing. Wabigoon Lake is a cool-water lake. In particular, there are two fish sanctuaries on Wabigoon Lake created to protect spawning walleye and sauger. Wabigoon Lake also supports an active sport fishery focused on walleye and muskellunge angling. There are eight active tourist outfitters operating on Wabigoon Lake, which receives enhanced management and supports an active sport fishery focused on walleye and muskellunge angling.

The sampling programs did not indicate evidence of any aquatic species at risk (such as lake sturgeon), either under the federal *Species at Risk Act* (SARA) or Ontario's *Endangered Species Act* (ESA).

20.2.1.3 Hydrological Studies

Regionally, the Goliath project site is in the English River watershed, which is a tributary of the Winnipeg River and within the Nelson River primary watershed, which drains to Hudson Bay. Locally, the Goliath project site is located east of Thunder Lake and northeast of Wabigoon Lake, and sits within sub-watersheds that drain to either Thunder Lake or Wabigoon Lake. Thunder Lake ultimately discharges to Wabigoon Lake via Thunder Creek. The sub-watersheds surrounding the project site include Thunder Lake Tributaries 2 and 3, Hoffstrom's Bay Tributary, and Little Creek in the Thunder Lake watershed, and Blackwater Creek in the Wabigoon Lake watershed. Blackwater Creek flows to Keplyn's Bay of Wabigoon Lake; a portion of this bay has been cut off by a railway and flows from this isolated portion into Keplyn's Bay via culverts under the railway.

Blackwater Creek and its tributaries provide low-gradient stream habitat, with predominantly clay and silt substrates, and punctuated by extensive active and inactive beaver dams and ponds. The measurement of hydraulic flow at baseline monitoring stations along Blackwater Creek were indicative of the challenges associated with accurately measuring continuous streamflow in small, low-gradient runoff-dominated stream systems that experience frequent beaver impoundments.

20.2.1.4 Hydrogeological Studies

The regional hydrogeology of the project area consists of relatively shallow (less than 10 m), localized overburden aquifers, as well as fractured metamorphic bedrock aquifer conditions. The Goliath project is located in the west-central portion of a hydrological basin containing low to moderate relief topographic features, including low lying marsh type lands and exposed bedrock ridges. This basin has been defined by inferred groundwater divides associated with topographic watersheds, and is bordered by upland areas to the east, in the vicinity of Hartman Lake, and to the north, part of which is occupied by a significant wetland area; the Thunder Lake Tributary drainage basin to the west; and Wabigoon Lake to the south. This basin contains the Thunder Lake drainage area to the west, Blackwater Creek drainage area through the central region, and the Hughes and Nugget Creek drainage areas in the east.

Groundwater levels at the Goliath project site are relatively close to surface and approximately follow the topography and surface drainage pathways, with greatest flows occurring along the contact between upper weathered and fractured bedrock and/or basal sand. Local groundwater flow is generally in a southerly direction from an elevated wetland area to the northeast (Lola Lake Provincial Nature Reserve), with flow splitting off in the general vicinity of the project site area both to the west, towards Thunder Lake, and to the south, towards Wabigoon Lake. Groundwater provides minimal baseflow to creeks in the immediate vicinity of the project site and for much of the project area. The surface watercourses in the local area of the Goliath project are runoff dominated, with groundwater baseflow representing a small proportion of the total flow. Rates of groundwater flow are expected to be much lower in the deeper bedrock.

Supplementary work is currently proposed to further develop and refine the understanding of the local hydrogeological conditions, including the installation of additional groundwater monitoring wells in the basal sand unit and the shallow bedrock at key locations within the project site area in 2023. Groundwater elevation monitoring and quarterly groundwater quality sampling continue to be monitored at existing groundwater stations.

20.2.1.5 Terrestrial Studies

Regionally, the Goliath project is located within the Ontario Shield Ecozone and the Lake Nipigon Ecoregion. The vegetation in the area consists of boreal forest types, generally characterized as a black spruce forest, with dominant woody vegetation that includes white spruce, balsam fir, trembling aspen, white birch, tamarack and jack pine. The ecoregion is also characterized by abundant wetlands, ponds, lakes, and rivers.

The Goliath project is also located within the Canadian Shield in the west-central portion of a hydrological basin containing low to moderate relief topographic features, including low-lying wetlands and marsh type lands, and exposed bedrock ridges.

Terrestrial baseline studies were conducted as part of the EA process by Klohn Crippen Berger (KCB), DST, and KBM Resources Group (KBM) in 2011, 2012, and 2015 to 2016, respectively, involving surveys for breeding birds, eastern whip-poor-wills (EWPW), waterfowl, marsh birds, amphibians, reptiles, and small mammals. An additional survey was conducted in 2022 by NorthWinds Environmental Services (NWES), focussing on EWPW, bats and barn swallow.

In general, bird densities and species richness observed during the field surveys were typical of the boreal forest. Avian species at risk detected during the 2011 to 2016 surveys conducted within the local study area (that encompassed the Goliath project site and the adjacent southwest area of Lola Lake Provincial Nature Reserve) included barn swallow, Canada warbler, common nighthawk, and olive-sided flycatcher, although no evidence of active nesting within the project site area was observed. No EWPW were detected during these surveys. No EWPW were observed on the project site during the 2022 survey, and there were no signs of barn swallow within the project site except for considerable nesting activity associated with the Treasury Metals current office location.

Ultrasonic recorders were set up throughout the local study area in 2011, 2012, and 2016, with bats being recorded at most of the locations. Although exact population numbers were not determinable based on recorder information, there was a clear indication that bats were present within the local study area. In January 2013, three species of bats were officially added to the Ontario species at risk list (tri-coloured, little brown myotis, and northern myotis). Two of the three species detected at the project site are provincially listed. The detected species included little brown myotis, northern myotis and big brown bats. No bats were observed on the project site during the 2022 survey.

Several large mammals and furbearers characteristic of the area include moose, white-tailed deer, black bear, gray wolf, American beaver, and snowshoe hare.

20.2.1.6 Geochemical Studies

Baseline geochemical programs have been conducted for the Goliath project since 2011 to evaluate the metal leaching and acid rock drainage (ML/ARD) potential of Goliath project materials in support of the federal EA and ongoing engineering studies. Geochemical programs to date have included a suite of static and kinetic tests to evaluate short-term static conditions and the long-term potential for acid rock drainage and metal leaching. The static testing assessment includes on the order of 200 drill core samples and has involved a number of laboratory analytical methods, including acid-base accounting, elemental content analysis, shake flask extraction testing, and mineralogical testwork. Kinetic testing has also been conducted and has included humidity cell tests and field cell tests. A number of humidity cell tests continue to operate as part of ongoing geochemical programs.

Results indicate that almost all project mine rock (i.e., waste rock and ore) is potentially acid generating (PAG; at a neutralization potential ratio of less than 1 [NPR<1]) with a very low neutralization potential content (on the order of 5 to 10 kg $CaCO_3$/t). Humidity cell testing indicated that these materials may become net-acid generating after several months to one year of exposure.

A preliminary static testing assessment was conducted to characterize overburden samples (n=20) from two proposed borrow areas, including acid base accounting analysis, elemental content analysis, and shake flask extraction testing. Results indicated that overburden materials from these borrow areas are expected to be non-potentially acid generating (NPAG) with a low potential for metal leaching. Geochemical assessment of overburden from other locations to be used as borrow material or developed for site infrastructure is proposed.

A tailings geochemical assessment, including testing of the Goliath tailings, is currently on-going, and is discussed in Section 18.7.5.

20.2.1.7 Air Quality and Noise Studies

The Goliath project is located in a mostly forested area between the communities of Dryden and Wabigoon, and north of Highway 17. The site is at least 10 km from any existing sources of significant air emissions. There are several aggregate operations on the east side of Airport Road in Dryden. The town of Dryden, located approximately 15 km to the west, is home to a kraft pulp mill currently operated by Domtar, which would contribute to the background air quality in the area, primarily due to emissions from the natural gas and wood-waste fired boilers, recovery boiler, and lime kiln. Due to the distance of the sources at the Dryden pulp mill and the aggregate operations from the Goliath project site, interaction between these sources and the expected emissions from the project site are expected to be minimal.

Baseline ambient air quality indicator levels at the Goliath project site were estimated by RWDI (2014a) based on data from two (2) monitoring stations in the Thunder Bay area (MECP Stations No. 63203 and 63064). The MECP monitoring station results indicated that the existing baseline ambient air quality levels do not exceed the relevant assessment criteria. The baseline ambient air quality levels at the Goliath project site are expected to be typical of other forested areas of northern Ontario; however, since the only available data near the project site were collected from a more urbanized area, the data are considered to be a conservative (higher) estimate of current local project site conditions.

Ambient long-term background sound levels measured by RWDI (2014b) in 2011 and 2013 at the Goliath project site were similar to background ambient sound levels characteristic of remote areas (25 to 45 A-weighted decibels; dBA). The sound from these levels would be described as faint. Noise observed during the study consisted mostly of wind, small animals, birds, and vehicle noise from the TransCanada Highway. The difference between daytime and nighttime sound levels was generally small and was attributed mainly to a very low level of noise from human activity which could not be screened out. The noise measurement results indicated that the baseline sound levels did not exceed the guideline sound level limits (MOE 1995). In accordance with the project's EA Decision Statement, ambient air monitoring efforts and updated emissions modelling are being advanced in 2023.

20.2.2 Environmental Monitoring

Based on the federal EA process, the issued Decision Statement stipulates a number of conditions that require Treasury Metals to undertake various environmental monitoring programs at the Goliath project site, including surface and groundwater quality sampling, surface and groundwater flow monitoring, dust monitoring, and wildlife monitoring, in consultation with interested stakeholders and Indigenous communities as applicable. The existing environmental baseline monitoring programs conducted to date provide the basis for the monitoring frameworks and may be modified to meet regulatory and reporting requirements as the project moves through the permitting phase. The required monitoring programs apply to the construction, operation, closure, and post-closure phases of the project, as appropriate, and will confirm compliance of activities with anticipated provincial and federal environmental approvals and permits criteria, while providing information to determine the effectiveness of proposed mitigation measures.

As the Goliath project site is developed, ongoing monitoring is expected to provide for an adaptive management approach, should environmental effects vary from those predicted; if mitigation measures prove less effective than anticipated; or as new information becomes available. Mitigation strategies may be modified accordingly, and monitoring parameters, locations and/or frequencies will be adapted as appropriate.

20.2.3 Water Management

Infrastructure for on-site water management during the life of mine is discussed in detail in Section 18.9.1.

A watercourse realignment channel has also been designed to redirect water around the proposed tailings facility to the main stem of Blackwater Creek. This realignment channel will be designed and utilized to compensate for the overprinting of the Blackwater Creek Tributaries by site infrastructure, including the tailings facility.

20.2.4 Permitting Considerations

Having completed the federal EA process and accumulating substantial baseline information to support the EA, the Goliath project site is prepared to move into the next permitting phase, which will involve acquiring provincial environmental permits and approvals primarily from the Ministry of the Environment, Conservation and Parks (MECP), the Ministry of Natural Resources and Forestry (MNRF), and the Ministry of Mines (MINES). Additional agencies that may be involved in permitting include the Ontario Energy Board (OEB), the Ministry of Transportation, and the Ministry of Tourism, Culture and Sport (MTCS).

Provincial environmental approvals expected for construction and operation of the Goliath project include (but are not limited to) those shown in Table 20-2.

Two tributaries of Blackwater Creek will be partially overprinted by project infrastructure, resulting in the unavoidable harm to fish and fish habitat, infilling of waters frequented by fish, and/or reduction of flow. The loss of fish habitat will require a Fisheries Act Authorization (FAA), including development and implementation of an offsetting plan for compensation of the lost fish habitat, pursuant to the federal *Fisheries Act.* A regulatory amendment to Schedule 2 of the Metal and Diamond Mining Effluent Regulations (MDMER), will also be required for mine waste facilities that overprint fish habitat. As part of the FAA, Treasury Metals will need to provide financial assurance to the department of Fisheries and Oceans Canada (DFO) until it can be demonstrated that the fish habitat offsets are constructed and functioning as intended.

The proposed Blackwater Creek realignment channel is expected to be utilized as part of the compensation measures; however, additional measures may be required (the delineation of habitat loss and compensation requirements are currently in progress).

Additional federal environmental approvals are also expected to be required to construct and operate the project and include those shown in Table 20-3.

In addition, engineering approvals related to explosives manufacturing and/or storage will be required.

Table 20-2: Expected Key Provincial Environmental Approvals and Permits

Agency	Permit/ Approval/ EA	Act	Relevant Components
MECP	Environmental Compliance Approval – Industrial Sewage Works	Ontario Water Resources Act	For establishment and operation of an industrial sewage treatment facility (e.g., tailings storage facility, mine water ponds, effluent treatment plant), including a domestic sewage treatment plant, for collection, treatment, and discharge of mine site waters.
	Permits to Take Water	Ontario Water Resources Act	For taking of ground or surface water (in excess of 50 m^3/d), such as for domestic/potable needs and pit dewatering. During construction, permits will be required for excavation dewatering (e.g., for dam and mill construction).
	Environmental Compliance Approval – Air and Noise	Environmental Protection Act	For discharge of air emissions and noise, such as from mill processes, on-site laboratory, and haul trucks (road dust).

Agency	Permit/ Approval/ EA	Act	Relevant Components
	Environmental Compliance Approval – Waste Disposal Site	Environmental Protection Act	For operation of a landfill and/or waste transfer site.
MNRF	Various Work Permits / Land Use Permits for Construction	Lakes and Rivers Improvement Act/Public Lands Act	For work/construction on Crown land. May be required for access roads, water crossings, buildings or possibly transmission lines.
	Lakes and Rivers Improvement Act (LRIA) Permit	Lakes and Rivers Improvement Act	Construction of online (in-stream) dams or berms, including approval for location of the dam(s) and its plans and specifications. Also for water crossings, work on/near shorelines, and watercourse realignments.
	Forest Resource License	Crown Forest Sustainability Act	For clearing of Crown merchantable timber.
	Aggregate Permit	Aggregate Resources Act	For extraction of aggregate (e.g., sand/gravel/ rock for tailings dam or other site construction).
	Endangered Species Permit	Endangered Species Act	For any activity that could adversely affect species or their habitat identified as 'Endangered' or 'Threatened' in the various schedules of the Act (none known at this time).
Mines	Closure Plan	Mining Act	For mine construction/production and closure, including financial assurance, inclusive of offline dams (e.g., for a tailings storage facility or mine water ponds).
MTCS	Clearance Letter	Heritage Act	For confirmation that appropriate archaeological studies and mitigation, if required, have been completed.

Table 20-3: Expected Additional Federal Environmental Approvals

Agency	Permit/ Approval	Act	Relevant Components
DFO	Sections 35 and 36. Authorization for serious harm to fish that are part of a commercial, recreational, or Aboriginal fishery, or to fish that support such a fishery	Fisheries Act	For construction of the tailings facility, mine rock stockpiles, access road creek crossings, water works for water intake structures, and/or groundwater dewatering effects, that would cause disruption to creeks and/or ponds supporting fish that are part of, or support a fishery.
ECCC	Schedule 2 Listing	Fisheries Act (Metal and Diamond Mining Effluent Regulations; MDMER)	For overprinting of waters frequented by fish, by a deleterious mineral waste (tailings management facility, mine water ponds).
NRCan	Licence for an explosives factory	Explosives Act	For operation of an on-site facility to supply explosives for use in the open pit operations.

20.2.5 Closure and Reclamation Planning

Submission of a Closure Plan for the proposed Goliath project will be required for approval (filing) by MINES in accordance with the *Ontario Mining Act*, the related "Advanced Exploration, Mine Development and Closure" regulation (Ontario Regulation [O.Reg.] 240/00), and the accompanying "Mine Rehabilitation Code". The Closure Plan is also to be accompanied by a detailed rehabilitation budget and submission of financial assurance to ensure that sufficient funds are in place to carry out the Closure Plan measures. The objective of closure is to rehabilitate and reclaim the project site which, while different from the pre-existing environment, will be capable of supporting plant, wildlife and fish communities, and applicable land uses.

Although previous mining activities at the Goliath project site ceased prior to the enactment of the closure regulations made under the *Mining* Act in 2000, the portal constructed during earlier exploration activities is reported to have been sealed and the area contoured, reseeded, and remediated in late 1999. The Goliath project Closure Plan with therefore only need to cover measures for the proposed future project infrastructure.

Conventional methods of closure are expected to be employed at the Goliath project site. The closure measures for the tailings management facility are to be designed to ensure chemical stability of the tailings solids, and physically stabilize the tailings surface to prevent erosion and dust generation and to limit the potential for oxygen penetration. The pit will be allowed to flood through active and passive measures. Monitoring at appropriate sampling locations, including those established during baseline studies and operations, will continue after closure until chemical stabilization is confirmed and to ensure performance of the rehabilitation measures.

The key rehabilitation measures to be included in the Closure Plan for the Goliath project are expected to include:

- Progressive rehabilitation of mine rock stockpiles during operations, to the extent practical;
- Removal of buildings, machinery, and infrastructure (including remaining petroleum products, chemicals, and explosives);
- Breaking of above grade concrete structures to near grade and infilling with clean mine rock if needed;
- Recontouring the tailings management area, if needed, covering with non-acid generating / non-metal leaching material (e.g., crushed rock, tailings, or mineral soils) and organic soil, and vegetating;
- Breaching of mine water pond dams and stabilizing;
- Capping/backfilling of any mine openings (portals and/or vent raises);
- Construction of a safety berm and/or boulder fence around the pit perimeter;
- Flooding of the pit via seepage and surface runoff inputs from the local area;
- Undertaking of geotechnical stability assessments of the pit walls and any crown pillars;
- Covering of remaining potentially acid-generating / metal leaching mine rock stockpiles with low permeability cover and seeding, with drainage directed to the open pit;

- Scarifying access roads to promote revegetation;
- Covering of the remaining general site area with a growth material and revegetating; and
- Post closure physical / chemical stability and biological monitoring.

Preparation of the Goliath Closure Plan is currently in progress. The financial assurance required to rehabilitate the Goliath project is currently based on conceptual closure measures; and a preliminary estimate is presented in Section 21.

20.3 Goldlund Project Site

20.3.1 Baseline and Supporting Studies

20.3.1.1 Overview

The Goldlund project is located within the former Goldlund mine site, which last operated in the mid 1980s. The site primarily sits within the Finlayson Creek watershed, which includes Kathlyn Lake, Kathlyn Lake South, Philcot Lake, and Crossecho Lake, and drains to Big Sandy Lake and thence to the Minnitaki Lake Chain that forms part of the English River system. A historic tailings impoundment area related to the former Goldlund mine (located southeast of the project site), drains south via an unnamed creek to Tom Chief Lake, which then drains to Finlayson Creek immediately upstream of Tablerock Lake (Figure 18-16). Franciscan Lake (east of the project site) also drains into Minnitaki Lake, downstream of the Big Sandy Lake outlet via the Sandy River.

Environment baseline data collection for the Goldlund site was initiated in 2020, which built upon basic scoping level aquatic information for three lakes and associated creeks within the proposed Goldlund project footprint by Story Environmental (Story 2017).

The following subsections provide a summary of the environmental baseline studies and site characterizations undertaken by Minnow and Stantec at the Goldlund project site to date.

20.3.1.2 Water Quality and Aquatic Studies

Water chemistry samples were collected at six lakes and six creek/tributary monitoring locations, including the historic tailings impoundment, between 2021 and 2023 during the ice-free period. The pH results across the project site area were circumneutral and had generally low hardness in the soft water range. The metal concentrations were below the PWQO at all stations, with the exception of occasional silver, arsenic, chromium, cobalt, copper and zinc concentrations that were naturally elevated above the PWQO on at least one occasion at one or more sample locations. Concentrations of total phosphorus and iron were naturally elevated above their PWQO values at 11 sampling locations. Water quality results at the outlet of the historic tailings impoundment were similar to those observed at the 11 water body and watercourse monitoring locations.

Various aquatic studies, including sediment, benthic invertebrate, fish, and fish habitat studies have also been undertaken since 2017, as summarized in Table 20-4.

Sediment quality samples were collected concurrent with benthic invertebrate samples in six lakes in the summer of 2021. Sediment results indicated some nutrients and metals had concentrations above the PSQG LEL, including TOC, chromium, copper, iron, manganese, nickel, and zinc (Amec Foster Wheeler, 2018b). Such elevated concentrations are not unusual due to the metal-rich nature of the bedrock of the Canadian Shield region and natural weathering.

Fish habitat characteristics were assessed at each waterbody considering the various life history stages of fish species present within the waterbody based on published literature accounts of preferred habitat. Habitat suitability was based on the quality of spawning, rearing (juvenile), and foraging (juvenile/adult) habitat available for the fish species captured within each respective waterbody.

Table 20-4: Goldlund Project Baseline Sediment Quality and Aquatic Life Data Collection – 2017 to 2021

Report	Year	Components	Areas
Story Environmental Inc. 2017	2017	Fish Habitat, Fish Community	Crossecho Lake, Historic Tailings Impoundment (L4), Tablerock Lake, Unnamed Stream S-2, Crossecho Lake Tributary, Tablerock Tributary, and Finlayson Creek
Minnow 2022	2020, 2021, 2022	Sediment Quality, Benthic Invertebrate Community, Fish Habitat, Fish Community, Fish Spawning, Fish Tissue Quality	Kathlyn Lake (incl. Kathlyn South), Philcot Lake, Crossecho Lake, Tom Chief Lake, Tablerock Lake, Historic Tailings Impoundment (L4), Franciscan Lake, Big Sandy Lake, Crossecho Lake Tributary, Tablerock Tributary, Finlayson Creek, Crossecho Lake Outlet, and Franciscan Lake Outlet

Fish communities in the Franciscan watershed were composed of both small-bodied and large-bodied species. Common small-bodied species found in lakes within the Franciscan watershed included Iowa darter, fathead minnow, spottail shiner, common shiner and Johnny darter. Common large-bodied species included white sucker, northern pike, yellow perch and black crappie. The community mostly reflects fish species preferring a cool-water thermal regime. Twelve species were reported from Philcot Lake. The community reflected species preferring both cool- and cold-water thermal regimes, suggesting water temperature within profundal depths remains cold through the summer months. The community reflects fish species preferring a cool-water thermal regime. The sampling programs did not indicate evidence of any aquatic species at risk (such as lake sturgeon), either under the federal SARA or Ontario's ESA.

Fish tissue chemistry sampling, using both large-bodied and small-bodied fish, was undertaken by Minnow in September 2021 to document baseline concentrations/accumulation of metals, total mercury, and methylmercury in lakes associated with the Goldlund project.

For Big Sandy Lake, Tablerock Lake, and Crossecho Lake, total mercury concentrations in northern pike generally increased with fish size, as mercury naturally bioaccumulates within food webs and larger fish tend to occupy higher trophic positions. Total mercury concentrations in most individual northern pike from Crossecho Lake naturally exceeded the MECP (2015) consumption advisory level for sensitive populations and half also exceeded the consumption advisory level for the general population. All fish samples, however, had total mercury concentrations below consumption restriction levels and the guideline for the protection of aquatic life and fish-consuming birds.

20.3.1.3 Hydrology Studies

The Goldlund project is primarily situated within the Finlayson Creek watershed, which drains to Big Sandy Lake and thence to the Minnitaki Lake Chain that forms part of the English River system. From the headwaters of Finlayson Creek, Kathlyn Lake, Kathlyn Lake South, and Philcot Lake are located upstream of potential direct influences from the Goldlund project site area, and discharge into Crossecho Lake immediate west of proposed future project infrastructure. The historic tailings impoundment drains south via an unnamed creek to Tom Chief Lake, which then drains to Finlayson Creek immediately upstream of Tablerock Lake. The topography of the Finlayson Creek watershed includes areas of both low and moderate local relief. A small portion of the Goldlund project site and a key access road are located within the Franciscan Lake watershed. Franciscan Lake also drains into Minnitaki Lake, downstream of the outlet from Sandy River. The entire area is part of the Hudson Bay drainage, with flow from the English River entering the Winnipeg River and then Lake Winnipeg, which subsequently drains into Hudson Bay via the Nelson River.

20.3.1.4 Hydrogeological Studies

Regionally, the Goldlund project is located within the Wabigoon Subprovince geological formation, which is an area underlain by sedimentary and volcanic rocks, numerous intermediate to mafic subvolcanic intrusive sheets, and intrusions of granitoid stocks that extend from Wabigoon Lake to Sioux Lookout (Wood 2022).

The local project area is primarily comprised of glaciolacustrine and glaciofluvial outwash deposits with some organic and glaciofluvial ice contact deposits. A significant portion of the project site is bedrock, as it is located in the Canadian Shield.

Based on provincially available surficial geology mapping, the surface overburden is fine grained (silt or clay) glaciolacustrine sediments with glaciofluvial sediments to the north, generally in an area of higher ground (Wood 2022). Other sand bodies are indicated by local gravel pits. According to exploration drillhole data, the local overburden geology in the vicinity of the main open pit and stockpile area consists of relatively thin overburden (generally <2 m thick) over crystalline bedrock, with thicker overburden present at the smaller pits to the east and west of the main pit.

A baseline hydrogeological investigation program was conducted in 2021 by Wood (2022), consisting of advancing 6 new boreholes into bedrock and conducting packer testing and installing nested vibrating wire piezometers in 3 of these boreholes to assess hydraulic gradients near the proposed open pit areas. Based on limited groundwater monitoring data available to date, groundwater levels within the upper bedrock appear to generally follow the topography (i.e., at the project site, groundwater flow follows the surface drainage pattern, with flow westward towards Crossecho Lake). Two of the boreholes indicated artesian conditions that may be attributed to potential influence of the flooded underground mine workings (200 level), which are located very close to these boreholes.

Under the current site conditions, most of the local groundwater flow is assumed to be driven by recharge into the surficial sand and gravels found at the higher elevation areas to the east and northeast of the proposed open pits, with preferential flow through basal sand and shallow fractured bedrock towards surface water features such as Crossecho Lake. Discharge to surface water features, however, is anticipated to be limited by surficial silty clay which is found across much of the project area. Groundwater flow in the deeper bedrock is anticipated to be minimal. Based on the available data, a preliminary three-dimensional numerical groundwater flow model has been developed by Wood (2022) to estimate groundwater inflows to the open pits, their zone of influence, and baseflow reduction in nearby surface water features.

Groundwater quality data is available from samples collected in May and November of 2017 (Northrock 2018) from six boreholes at the site, including the two artesian holes at the west end of the project site. The available groundwater quality information indicated the groundwater is hard, with occasional metal concentrations above the PWQO for arsenic, iron, manganese, and molybdenum.

Supplementary work is currently proposed to further develop and refine the understanding of the local hydrogeological conditions, including the installation of additional boreholes in the overburden at key locations within the project site area in 2023.

20.3.1.5 Terrestrial Studies

Terrestrial baseline studies were conducted by Northwest Environmental Services (NWES; 2022) in 2020 and 2021, involving surveys for vegetation communities, wetlands, breeding birds, Whip-poor-wills, waterfowl, marsh birds, owls, bats, amphibians, reptiles, and large and small mammals.

The local area of interest (LAI) delineated for avian and wildlife surveys was relatively large, covering more than a 5 km buffer around the Goldlund project site. Avian SAR detected in the LIA included bald eagle, barn swallow, Canada warbler, common nighthawk, eastern whip-poor-will, eastern wood pewee, evening grosbeak, and short-eared owl, with only two barn swallows observed within the proposed project footprint at the former mine site.

Bat species recorded within the LAI included eastern red bat, hoary bat, silver-haired bat, little brown myotis and evidence of tri-colored bat; with little brown myotis (recorded at 75% of the 2020 bat survey locations) and tri-colored bat being identified as provincially listed.

Several large mammals reside within the LAI (in order of abundance, include moose, gray wolf, white-tailed deer, snowshoe hare, and black bear).

Six amphibian species were observed during the surveys; no SAR were observed during these surveys.

The Goldlund project is noted to fall within an area of active forest management (NWES 2022). Black Ash, a recently assigned Ontario SAR species, was identified in twenty-two (22) ecosite stands within the LIA during field surveys, and were primarily located southwest of the proposed project site in the area of Tablerock Lake. The MECP has paused the protection of Black Ash until 2024 while protection and recovery plans are developed. No other SAR or provincially tracked vegetative species were found in the survey area.

20.3.1.6 Geochemistry Studies

A baseline geochemical assessment was undertaken for the Goldlund project in 2021 to evaluate the metal leaching and acid rock drainage potential of Goldlund project materials. The assessment included static and kinetic testing methods. Static testing included acid-base accounting, net acid generation testing, elemental content analysis, shake flask extraction testing, and mineralogical testwork. This testing was carried out on approximately 200 drill core samples. Kinetic testing (humidity cell tests) was also conducted for 11 drill core samples as part of the ongoing baseline program.

Drill core static testing results indicated that mine rock is expected to be sulphide-bearing but have a generally low sulphur content (often on the order of 0.1%), although some samples had a sulphur content as high as approximately 2%. Most of the tested samples (i.e., approximately 98%) were classified as non-potentially acid generating (NPAG, NPR>2).

Although the majority of the samples were NPAG, kinetic testwork identified that selenium, cobalt, and molybdenum may be of interest for metal leaching for some of the mine rock, as indicated by increasing release rates in some of the humidity cell tests. The tests are ongoing and will continue to be monitored. Results of the ongoing kinetic tests will be used to inform mine rock and water quality management needs at the project site. The preparation of water quality estimates is proposed.

A static testing program was also conducted in 2021 for historical wastes present around the site, including the collection of surface grab samples from several historical mine rock piles (n=8 samples) and the historic tailings area (n=5 samples). Static testing was carried out on the samples including acid-base accounting, net acid generation testing, and elemental content analysis. Nine water samples representing drainage from various mine waste features were also collected. The assessment was intended to be a preliminary, screening-level assessment of the ML/ARD potential of historical site materials.

Results indicated that most of the samples of historical mine rock had a generally low sulphur content (on the order of 0.5%) and were classified as NPAG (NPR>2). Two samples were classified as PAG (NPR<2) and had sulphur contents on the order of 0.8 to 1%. The historical tailings samples (n=5) had low sulphur contents (< 0.03%) and were classified as NPAG (NPR>2).

The tested samples generally contained low metal concentrations with most metals below qualitative screening values. Site water quality data collected to represent drainage from historical mine waste features (n=9 samples) showed a neutral pH and low metal concentrations, suggesting a low potential for metal leaching from the historical wastes. Iron concentrations were identified to be elevated relative to screening values in some samples, attributed to local redox (low oxygen) conditions at the sampling locations (comprising groundwater, pore water, and ponded water in swampy areas). Additional characterization and delineation of historical wastes is proposed.

A tailings geochemical assessment, including testing of the Goldlund tailings, is currently on-going, and is discussed in Section 18.7.5.

20.3.2 Environmental Monitoring

Environmental monitoring programs, including routine surface water quality monitoring, hydrological monitoring, and groundwater quality monitoring, will continue to be undertaken for the Goldlund project, commensurate with the scale of the project and potential environmental effects, and in consultation with interested stakeholders and Indigenous communities. In 2022, Treasury Metals initiated a review of its historical monitoring programs to support anticipated federal and provincial regulatory processes.

20.3.3 Water Management

Infrastructure for on-site water management for the Goldlund project is discussed in detail in Section 18.4.5.

20.3.4 Permitting Considerations

Similar to the Goliath project, there will be a number of provincial and federal permitting approvals and authorizations that will be required for the development of the Goldlund project. The majority of these permits and approvals will be required from the MECP, the MNRF, and MINES, as listed in Table 20-2. Additional agencies that may be involved in permitting of the Goldlund project include the Ministry of Transportation for any highway upgrades and/or site entrance requirements, and Ministry of Tourism, Culture and Sport (MTCS).

The Goldlund project may also require completion of one or more provincial class environmental assessment processes, pursuant to the *Ontario Environmental Assessment Act*, depending on the final project designs. Based on preliminary designs, it is anticipated that there would only be a requirement for a Class Environmental Assessment(s) for Resource Stewardship and Facility Development Projects, subject to regulatory confirmation.

Various infrastructure (e.g., overburden, mine rock, and ore stockpiles) for the Goldlund site is being designed and located to avoid any overprinting of watercourses. As such, the requirement for an FAA or Schedule 2 listing under requirements of the MDMER is currently not envisioned.

20.3.5 Closure and Reclamation Planning

Previous mining and milling activities at the Goldlund project site ceased in 1985, prior to the enactment of the closure regulations (O.Reg. 240/00) under the *Ontario Mining Act* in 2000. As such, some physical hazards currently remain that will require rehabilitation. Accordingly, a Closure Plan and financial assurance will be required for rehabilitation of the proposed Goldlund project, as presented herein, as well as any remaining physical and/or chemical hazards from earlier mining and milling activities.

The key rehabilitation measures to be included in the Closure Plan for the Goldlund project are expected to include:

- Progressive rehabilitation of mine rock stockpiles during operations, to the extent practical;
- Removal of buildings (including the former mill), machinery, and infrastructure (including remaining petroleum products, chemicals and explosives);
- Breaking of any above grade concrete structures to near grade and infilling with clean mine rock if needed;
- Capping/backfilling of any mine openings (portals and/or vent raises);
- Construction of a safety berm and/or boulder fence around the pit perimeters;
- Flooding of the pits via seepage and surface runoff inputs from the local area;
- Undertaking geotechnical stability assessments of the pit walls and any crown pillars;
- Breaching of mine water pond dams and stabilizing;
- Scarifying access roads to promote revegetation;
- Covering of the remaining general site area with a growth material and revegetating; and
- Post closure physical / chemical stability and biological monitoring.

Preparation of a Closure Plan for the Goldlund project is expected to commence in late 2023. The financial assurance required to rehabilitate the Goldlund project is currently based on conceptual closure designs, and a preliminary estimate is presented in Section 21.

20.4 Miller Project Site

20.4.1 Baseline and Supporting Studies

To date, baseline and supporting studies at the Miller project site have been limited.

A preliminary aquatic baseline monitoring program was conducted by Minnow at seven locations in the Miller project site area in 2021 and included collection of information related to habitat characteristics, sediment chemistry, benthic invertebrate community, and fish habitat, community, and tissue chemistry. Reporting is still in progress.

A preliminary geochemical assessment was conducted for the Miller project including static testing of approximately 50 drill core samples representing mine rock. Static testing included acid-base accounting, net acid generation testing, elemental content analysis, and shake flask extraction testing.

Most (i.e., approximately 98%) of the tested samples were classified as non-potentially acid generating (NPAG, NPR>2) based on the preliminary assessment. A screening level assessment of metal leaching potential indicated that some samples had selenium and arsenic contents that were greater than qualitative screening values, but concentrations of these elements were low in leachates from short term leaching tests. Kinetic testing is proposed to further assess the metal leaching potential of Miller samples.

20.4.2 Environmental Monitoring

Environmental monitoring programs will be established for the Miller project, commensurate with the scale of the project and potential environmental effects, and in consultation with interested stakeholders and Indigenous communities.

20.4.3 Permitting Considerations

Similar to the Goliath and Goldlund projects, there will be a number of provincial and federal permitting approvals and authorizations that will be required for the development of the Miller project. The majority of these permits and approvals will be required from the MECP, the MNRF, and MINES, as listed in Table 20-2. Additional agencies that may be involved in permitting include the Ontario Energy Board (OEB) if the site requires construction of a transmission line, Ministry of Transportation for any highway upgrades and/or site entrance requirements, and Ministry of Tourism, Culture and Sport (MTCS).

Similar to the Goldlund project, the Miller project may require completion of one or more provincial environmental assessment processes pursuant to the *Ontario Environmental Assessment Act*, depending on the final project designs. Based on current preliminary designs, it is anticipated that there would only be a requirement for a Class Environmental Assessment(s) for Resource Stewardship and Facility Development Projects, subject to regulatory confirmation.

At this time, designs and layout for the Miller project are still preliminary. Similar tor the Goldlund project, various infrastructure (e.g., overburden, mine rock, and ore stockpiles) for the Miller project site will need to be designed and located to avoid any overprinting of watercourses and thereby avoid the requirement for an FAA or Schedule 2 listing under requirements of the MDMER.

20.4.4 Closure and Reclamation Planning

A Closure Plan and financial assurance will be required for rehabilitation of the proposed development of the Miller project.

The key rehabilitation measures to be included in the Closure Plan for the Miller project are expected to include:

- Progressive rehabilitation of mine rock stockpile(s) during operations, to the extent practical;
- Removal of any buildings, machinery, and infrastructure (including remaining petroleum products, chemicals and explosives);
- Breaking of any above grade concrete structures to near grade and infilling with clean mine rock if needed;

- Capping/backfilling of any mine openings (portals and/or vent raises);
- Construction of a safety berm and/or boulder fence around the pit perimeters;
- Flooding of the pits via seepage and surface runoff inputs from the local area;
- Undertaking geotechnical stability assessments of the pit walls and any crown pillars;
- Breaching of mine water pond dams and stabilizing;
- Scarifying access roads to promote revegetation;
- Covering of the remaining general site area with a growth material and revegetating; and
- Post closure physical / chemical stability and biological monitoring.

Preparation of a Closure Plan for the Miller project is expected to commence once more details of the proposed site layout and required infrastructure are better known. The financial assurance required to rehabilitate the Miller project is currently based on conceptual closure designs, and a preliminary estimate is presented in Section 21.

20.5 Social Considerations

20.5.1 Human Environment

The Goliath project is located 20 km east of the City of Dryden, Ontario, which has a population of approximately 8,000 people. The three proposed projects are located in areas used by the public for recreational fishing, hunting, boating, and commercial activities, including tourism, fishing, trapping, and wild rice and bait harvesting. For example, Thunder Lake is popular for fishing and hiking trails, and snowmobile trails exist in the area.

20.5.2 Traditional Land and Resource Use

The three project sites are located within the Treaty 3 (1873) area of Ontario, which affords hunting, trapping and fishing rights and protections, and it has been shared with Treasury Metals that there are areas within the Goliath Gold Complex property boundaries that are used by Indigenous communities for traditional land and resource use. The Indigenous communities nearest to the project are Eagle Lake First Nation, Wabigoon Lake Ojibway Nation and Lac Seul First Nation.

Information regarding traditional land and resource use that was shared with Treasury Metals throughout the EA process was included wherever possible in the federal EA for the project. Formal traditional knowledge and traditional land and resource use studies for the project were provided to Treasury Metals by Eagle Lake First Nation the Métis Nation of Ontario. While the specific details of these studies are confidential, it can be confirmed that there is overlap of the impacts and effects of the Goliath project with areas currently used by members of Indigenous communities for hunting large game, non-commercial fishing and gathering of plant material.

20.5.3 Non-Indigenous Stakeholders

Non-Indigenous public interest groups have been identified as part of past, present and future consultation and engagement efforts. This includes the Village of Wabigoon, City of Dryden, the Town of Sioux Lookout, and other regional industrial partners and stakeholders.

20.5.3.1 Local Citizens (Proximal to the Goliath Project)

The residents of Anderson Road, Tree Nursery Road, East Thunder Lake Road, Thunder Lake Road, Highway 11/17, those proximal to Wabigoon Lake, and those proximal to Thunder are the parties in closest proximity to the Goliath project. Residents from these locations have interests in the potential effects and impacts to their environment, health, lifestyle, and economic conditions due to the development of the project. A number of meetings specifically for these residents have been held to provide a forum for these community members to ensure their concerns are expressed and for Treasury Metals to ensure appropriate mitigation measures are implemented and communicated.

20.5.3.2 Village of Wabigoon

The Village of Wabigoon has a long history associated with gold mining. With the discovery of gold on Upper Manitou Lake at the town of Gold Rock, and development of mines there during the 1890s, Wabigoon with its location on the Canadian Pacific Railway, became the transportation hub, supply depot and jumping off location for personnel and supplies destined for the Gold Rock mining area. Personnel and supplies arriving by rail at the Village of Wabigoon followed the freight route across Wabigoon, Dinorwic and Minehaha Lakes and then portaged overland to the Gold Rock mines. Many Wabigoon area families have historical ties to the Gold Rock mining activities. Since the closure of the Gold Rock mines, Wabigoon's employment and economic base has been tied primarily to forestry and tourism. The Village of Wabigoon has significant interest in the project due to the potential effects and impacts to their environment, health, lifestyle, and economic stimulus to the community. A number of meetings have been held to provide a forum for these community members to ensure their concerns are expressed and for the company to ensure proper mitigation measures are implemented and communicated. A number of meetings specifically for the residents of Wabigoon have been held to provide a forum for these community members to ensure their concerns are expressed and for the company to ensure proper mitigation measures are implemented and communicated.

20.5.3.3 City of Dryden

The City of Dryden also has some early ties to gold mining with mines operating just south of the City of Dryden and Wabigoon Lake in the Larson Bay/Contact Bay area during the early part of the 20th Century. Dryden also has some ongoing links to the mining industry as an industrial supply area for northwestern Ontario including sales and maintenance of mining equipment. However, the mainstay of Dryden's economy has been the forest industry. Until recently, the mill complex in Dryden included pulp and paper operations, paper converting and a sawmill; along with the associated woodlands operations. Recent closures of the sawmill, followed by closures of the paper machines and converting facility have left the complex with a pulp mill only and significantly reduced employment in the Dryden area. Reduced employment opportunity has resulted in numerous people having to relocate away from the Wabigoon/Dryden area. This in turn, has adversely affected the retail sector as well as real estate values in the area. Therefore, the City of Dryden is seen as a key partner and is seen as having significant interest in the project due to the potential effects and impacts to residents' environment, health, lifestyle, and economic prosperity of the community. A number of meetings specifically for the residents of Dryden have been held to provide a forum for these community members to ensure their concerns are expressed and for the company to ensure proper mitigation measures are implemented and communicated.

20.5.4 Community and Indigenous Consultation

20.5.4.1 Indigenous Communities/Partners

The Goliath project is located within the Treaty 3 (1873) area of Ontario, which affords hunting, trapping and fishing rights and protections to its signatories throughout the Treaty territory. The Indigenous communities nearest to the project are Eagle Lake First Nation, Wabigoon Lake Ojibway Nation, and Lac Seul First Nation.

Information regarding traditional land and resource use that was shared with Treasury Metals throughout the EA process was included wherever possible in the federal EA for the project. Formal traditional knowledge and traditional land and resource use studies for the project were provided to Treasury Metals by Eagle Lake First Nation the Métis Nation of Ontario. While the specific details of these studies are confidential, it can be confirmed that there is overlap of the impacts and effects of the Goliath project with areas currently used by members of Indigenous communities for hunting large game, non-commercial fishing and gathering of plant material.

Treasury Metals is committed to working collaboratively with Indigenous and regional communities to ensure informed and engaged dialogue throughout the life of the project. To date, Treasury Metals has participated in meaningful consultation and engagement activities with the following Indigenous communities.

Table 20-5: Summary of Agreements between Treasury Metals and Indigenous Groups

Year	Date	Indigenous Group	Summary
2017	December 11	Eagle Lake First Nation	Treasury Metals and Eagle Lake First Nation execute Memorandum of Understanding fostering trust between the Parties, potential support, and meaningful participation with the permitting and development of the Goliath project.
2017	December 11	Metis Nation of Ontario	Treasury Metals and the Metis Nation execute Memorandum of Understanding fostering trust between the Parties, potential support, and participation with the permitting and development of the Goliath project.
2019	January 14	Eagle Lake First Nation	Treasury Metals and Eagle Lake First Nation execute Memorandum of Understanding enabling continued communication, consultation, and meaningful participation with the permitting and development of the Goliath project.
2019	February 14	Wabauskang First Nation	Treasury Metals and Wabauskang First Nation execute Engagement Agreement to enable continued meaningful participation with the permitting and development of the Goliath project.

Treasury Metals will endeavour to maximize participation with its Indigenous partners wherever possible. Treasury Metals is focused on building and strengthening relationships, integrating traditional knowledge into its decision-making frameworks, and actively communicating and sharing information in a transparent manner via phone calls, meetings, letters, delivery of reports and presentations. As part of the federal EA Approval on the Goliath project, Treasury Metals made several firm commitments to its Indigenous partners regarding consultation and engagement, which may also be extended to the Goldlund project and Miller project.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PFS for the Goliath Gold Complex are based on open pit and underground mining of the Goliath deposit, open pit mining of the Goldlund deposit, open pit mining of the Miller deposit, and the construction of a process plant, tailings storage facility, and requisite supporting infrastructure. The purpose of the capital estimate is to provide substantiated costs which can be used to assess the economics of the project.

21.2 Capital Costs

21.2.1 Overview

The capital cost estimate was developed in Q3 2022 from budgetary quotations, Ausenco's in-house database of projects and studies, and experience from similar operations to a level of accuracy of ±25% (Class 4). The level of accuracy is in accordance with the Association for the Advancement of Cost Engineering International (AACE International) for prefeasibility studies.

The capital cost summary is presented in Table 21-1. The total initial capital cost for the Goliath Gold Complex is $335.0 million and LOM sustaining costs are $197.6 million. Closure costs are additional and are estimated at $28.9 million.

Table 21-1: Summary of Capital Costs

WBS2	WBS Description	Initial Capital (C$M)	Sustaining Capital (C$M)	Total Capital (C$M)
11000 12000 21000	Goldlund Mine Miller Mine Goliath Open Pit Mine	63.4	41.7	105.1
22000	Goliath Underground Mine	3.7	91.3	95.0
Mining Total		67.1	133.0	200.1
13000	Goldlund-Miller On-Site Infrastructure	0.0	11.5	11.5
Goldlund-Miller Property Total		0.0	11.5	11.5
23000	Process Plant	98.6	0.0	98.6
24000	On-Site Infrastructure	75.3	36.6	112.0
Goliath Property Total		173.9	36.6	210.6
31000	Main Access Road Diversion	0.0	1.1	1.1
32000	HV Line Tie-In	0.1	0.0	0.1
33000	Goldlund-Goliath Transport Connection	0.0	5.8	5.8
34000	Watercourse Realignment	2.0	0.0	2.0
35000	Water Management Pipeline	1.8	0.0	1.8
Off-Site Infrastructure Total		3.9	6.9	10.8
41000	Temporary Construction Facilities and Services	20.6	9.7	30.3
42000	Commissioning Reps and Assistance	0.3	0.0	0.3

WBS2	WBS Description	Initial Capital (C$M)	Sustaining Capital (C$M)	Total Capital (C$M)
43000	Spares (Commissioning, Initial and Insurance)	2.1	0.0	2.1
44000	First Fills & Initial Charges	1.0	0.0	1.0
Project Indirects Total		24.0	9.7	33.7
50000	Project Delivery	14.3	0.0	14.3
60000	Owner's Costs	16.6	0.0	16.6
Project Delivery and Owner's Costs Total		30.9	0.0	30.9
71000	Contingency	35.1	0.0	35.1
Contingency Total		35.1	0.0	35.1
	Closure Cost	0.0	28.9	28.9
	Salvage Value	0.0	(9.9)	(9.9)
Closure Costs and Salvage Value Total		0.0	19.0	19.0
Grand Total		**335.0**	**216.6**	**551.7**

Source: Ausenco, 2023

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 Estimate Responsibilities by WBS

WBS	Description	Engineering	Cost Estimating
11000	Goldlund Mine	SRK	SRK
12000	Miller Mine	SRK	SRK
13300	Goldlund Site Water Management	Stantec	Ausenco
21000	Goliath Open Pit Mine	SRK	SRK
22000	Goliath Underground Mine	SRK	SRK
23000	Process Plant	Ausenco	Ausenco
24000	Goliath On-Site Infrastructure	Ausenco, except: 24800 (SITE WATER MANAGEMENT), and 24900 (Tailings Storage Facility) by SLR	Ausenco
30000	Off Site Infrastructure	Ausenco	Ausenco
40000	Project Indirects	Ausenco	Ausenco
51000	EPCM	Ausenco	Ausenco
50000	Project Delivery	Treasury	Treasury
70000	Provisions (Contingency)	SRK, Ausenco	SRK, Ausenco

Source: Ausenco, 2023

21.2.3 Basis of Estimate

Data input for the estimates has been obtained from numerous sources, including the following:

- mining schedule
- PFS level engineering design
- mechanical equipment requirements determined from first principles and Ausenco's database of historical projects
- budgetary equipment quotes from Canadian and International suppliers
- budgetary unit costs from local contractors for civil, concrete, steel, electrical, piping and mechanical works
- geotechnical investigations
- topographical information.

The following parameters and qualifications were considered:

- there is no escalation added to the estimate
- a growth allowance was included
- percentage of contingency was allocated to major cost categories on a line-item basis based on the accuracy of the data.

21.2.4 Exchange Rates

Vendors and contractors were requested to price in native currency. The estimate is prepared in the base currency of Canadian dollars (C$). Pricing has been converted to Canadian dollars using the exchange rates in Table 21-3. The US dollar contributions to the capital estimate are 4.3% of initial capital costs, and the Australian Dollar 1.9% of initial capital costs. All sustaining capital contributions are in Canadian Dollars.

Table 21-3: Estimate Exchange Rates

Symbol	Abbreviation	Initial Currency	Exchange Rate
C$	CAD	Canadian	1.00
AU$	AUD	Australian Dollar	0.89
€	EUR	Euro	1.32
US$	USD	United States Dollar	1.34

Source: Ausenco, 2023

21.2.5 Direct Costs – Mining (WBS 11000, 12000, 21000, 22000)

The mining capital cost estimate is grouped into two main categories: open pit capital costs and underground capital costs. These costs are summarized in Table 21-4.

Table 21-4: Mining Capital Cost Estimate

Mining Capital Category	Initial Cost (C$M)	Sustaining Cost (C$M)	Total Capital Cost (C$M)
Open Pit Capital	63.4	41.7	105.1
Underground Mining Capital	3.7	91.3	95.0
Total	67.1	132.9	200.1

Source: SRK, 2023

21.2.5.1 Open Pit Mining

A mining cost model has been developed to estimate the open pit mining capital and operating expenditures based on owner-operation, except for the haulage between Goldlund and Goliath which will be undertaken by a contractor. The capital cost estimate has been completed to a PFS level. The cost estimate is in C$ and has been developed by SRK based on quotes obtained from local manufacturers and suppliers and SRK's internal cost database. The open pit mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM).

The open pit mining capital cost estimate is shown Table 21-5.

Table 21-5: Open Pit Mining Capital Costs

Parameter (C$M)	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12
Capital Costs	**105.1**	**63.4**	**11.4**	**7.8**	**7.8**	**2.6**	**1.5**	**4.8**	**1.4**	**2.0**	**1.4**	**0.7**	**0.0**	**0.2**
Capitalized Mining Operating Cost	46.9	46.9	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0	0.0
Financed Equipment	42.0	7.7	11.3	7.8	7.8	2.1	1.4	0.2	1.1	1.6	0.0	0.7	0.0	0.2
Non-Financed Equipment	16.2	8.8	0.1	0.0	0.0	0.5	0.1	4.5	0.3	0.4	1.4	0.0	0.0	0.0

Source: SRK, 2023

21.2.5.2 Underground Mining

The mining cost model for Goliath underground assumes that the majority of activities will be undertaken by a contractor, who will furnish their own personnel and equipment. As discussed in Section 16.5.9, there are some capital requirements associated with the contracted backfilling process, and as such have been included here.

The capital cost estimate has been completed to a PFS level. The cost estimate is in C$M and has been developed by SRK based on quotes obtained from original equipment manufacturers and suppliers, and SRK's internal cost database. The underground mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM).

The underground mining capital cost estimate is shown Table 21-6.

Table 21-6: Underground Mining Capital Costs

Parameter (C$M)	Total	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Capital Costs	95.0	3.7	8.8	16.9	15.2	15.9	12.9	11.9	3.4	1.4	0.8	3.2	0.5	0.3	0.1	-
Infrastructure (Project)	3.7	3.7	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Infrastructure (Sustaining)	34.3	-	4.7	8.5	4.0	4.3	3.5	3.1	2.1	1.4	0.8	1.2	0.4	0.3	0.0	-
Capital Development	56.9	-	4.1	8.4	11.1	11.6	9.4	8.9	1.3	-	-	2.0	0.1	-	0.1	-

Source: SRK, 2023

21.2.6 Process Plant (WBS 23000) and On-Site Infrastructure (WBS 24000)

The definition of process equipment requirements is 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 sent for budgetary pricing to equipment suppliers. For mechanical equipment costs, 85% of the equipment value was sourced from budgetary quotes; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies. Installation costs were provided as budgetary quotations from local contractors.

Major electrical equipment was sized based on the project's equipment list. Scopes of work were developed in order to receive budgetary pricing from equipment suppliers. For major electrical equipment, 70% of the equipment value was sourced from budgetary quotations, inclusive of the incoming electrical substation and emergency power generators discussed in (Section 18.4.1). The remainder was sourced by benchmarking against other recent Canadian gold projects and studies. Installation costs were provided as budgetary quotations from local contractors.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design were completed to a PFS level of definition, allowing for the bulk material quantities (platework, structural steel, concrete, earthworks) to be derived. Budgetary quotations from contractors were used to complete construction pricing for these commodities. Earthworks contractor budgetary quotations provided the construction costs for the tailings storage facility (section 18.8) and water management structures (Section 18.9)

Equipment supply and installation costs for small-bore piping, electrical bulks and instrumentation were generated by benchmarked against recent similar projects within Ausenco's in-house database.

Building costs for the building list described in Section 18.6 were provided by modular, pre-engineered and fabric building suppliers as turn-key supply and installation packages.

The effluent treatment plant capital cost was received by vendor budgetary quotations, considering the nominal flow rates described in section 18.9.1, influent criteria, and effluent discharge requirements in section 20.2.4.

21.2.7 Direct Costs – Off Site Infrastructure (WBS 30000)

21.2.7.1 High-Voltage Overhead Line

The cost for the high-voltage overhead power line connecting the incoming power substation to the existing 115 kV powerline crossing the Goliath site (section 18.5.1) was developed considering Ausenco's in-house database of costs.

21.2.7.2 Watercourse Realignment

The watercourse realignment (Section 20.2.3) cost is an allowance based on the actual cost incurred by a project of similar scope in the Northern Ontario, provided by Treasury Metals.

21.2.8 Indirect Costs (WBS 40000)

21.2.8.1 Temporary Construction Facilities and Services

Contractor indirect costs are related to the contractor's direct costs, and can include:

- mobilization and demobilization
- site offices and utilities
- construction equipment including mobile equipment, scaffolding, safety supplies, etc.
- head office costs/contribution
- financing charges
- insurances
- profit.

Contractors provided indirect costs as part of their pricing schedules.

21.2.8.2 Commissioning Reps and Assistance

Vendor representative costs during commissioning and construction include vendor representative support during the installation of the purchased equipment.

Vendor representative costs have been based on the engineer's evaluation of recommendations and prices provided by equipment vendors during the pricing enquiry process.

21.2.8.3 Spares

Capital and commissioning spares cost for mechanical equipment is determined by a factored allowance based on the supply price and benchmarked against Ausenco's in-house database of projects. Allowance factors were based on a 6-month period of capital spares.

21.2.8.4 First Fills and Initial Charges

Process first fill quantities (e.g., mill media and reagents) and first fill lubricants (e.g., greases, oils, and hydraulic fluids) were calculated based on the engineering design and priced using quotes that were provided by reagent and media suppliers.

21.2.9 Project Delivery (WBS 50000)

The project delivery cost estimate was build considering a hybrid consultant and Owner's team. Project delivery costs have been estimated on first principals inclusive of the following:

- engineering
- field engineering, inspection and expediting
- expenses incurred by consultant parties:
  - project office facilities
  - corporate overhead and fees
  - travel expenses
  - home office expenses
  - site office expense.

The following items are considered in Owner's Costs:

- procurement
- construction management
- commissioning team

21.2.10 Owner's Cost (WBS 60000)

Owner's costs are inclusive of the following:

- owner's project team and expenses, including roles associated with project delivery:
  - procurement
  - construction management
  - commissioning team
- pre-production labour
- administration, finance, insurance, and legal fees

- environmental consultation and management
- human resources, recruiting and training
- community relations
- site security; and
- mobile equipment and vehicle leases.

21.2.11 Contingency (WBS 70000)

Contingency accounts for the difference in costs between the estimated and actual costs of materials and equipment. The level of contingency varies depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital cost estimate was developed (in terms of the level of engineering definition, basis of the estimate, schedule development, etc.), the estimate includes provision to cover the risk from these uncertainties.

The contingency cost is from total installed costs based on the level of uncertainty for each area, using a deterministic approach. Ausenco calculated a contingency of C$35.1 M following the percentage allotments by commodity according to Table 21-7.

Table 21-7: Contingency Applied

Commodity Code	Commodity Description	Contingency Applied
A	Architectural	15%
B	Earthworks	15%
C	Concrete	15%
D	Mining	0%
E	Electrical	15%
F	Platework and Mechanical Bulks	15%
I	Instrumentation	10%
M	Mechanical Equipment	15%
N	Plant & Miscellaneous Equipment	5%
P	Pipework	15%
Q	Electrical Bulks	15%
S	Structural Steel	15%
U	Field Indirects	15%
V	Third-Party Packages/Other	15%
W	EPCM, EPC & EP	15%
X	Provisions	0%
Y	Owner's Costs	0%

Source: Ausenco, 2023.

21.2.12 Growth Allowance

Each line item of the estimate was developed initially at base cost only. A growth allowance has then been allocated to each element of those line item costs to reflect the level of definition of design and pricing strategy. Estimate growth is:

- intended to account for items that cannot be quantified based on current engineering status but empirically known to appear
- accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at feasibility study level
- pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalization of commercial terms and conditions to be used on the project.

Considering the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate, a 10% growth factor was applied to all items except in the following cases where no growth factor was applied:

- tailings storage facility and site-wide water management ponds and ditches
- mobile equipment
- mining activities
- project delivery
- fully priced turnkey contractor packages, as with pre-engineering, fabric and modular buildings and fire systems
- contractor field indirects.

21.2.13 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- land acquisitions
- wetlands compensation or relocations
- senior finance charges
- permitting
- royalty buyouts
- further testwork and drilling programs
- taxes not listed in the financial analysis
- environmental approvals
- this study or any future project studies, including environmental impact studies

- first nations impact benefit agreement costs
- sales taxes
- operating costs
- operational readiness costs
- working capital
- scope changes and project schedule changes and the associated costs
- any facilities/structures not mentioned in the project summary description
- geotechnical unknowns/risks
- financing charges and interest during the construction period
- third party costs.

21.3 Sustaining Capital

Please refer to Section 21.2.5 for a summary of sustaining capital items for mining (WBS 11000, 12000, 21000, 22000).

21.3.1 On-site Infrastructure (WBS 24000)

21.3.1.1 Equipment Installations

Electrical equipment required to support underground mining is considered as sustaining capital, valued at $2.6 million, inclusive of an expansion to the incoming substation and the installation of an electrical room positioned at the underground portal.

An expansion to the effluent treatment plant flow capacity valued at $2.1 million is considered in the sustaining capital.

21.3.1.2 Tailings Storage Facility

Progressive expansion of the tailing storage facility over the life of mine is considered in sustaining capital, totalling $31.8 million. Earthworks contractor budgetary quotations provided the construction costs.

Construction material quantities were determined using a combination of Civil 3D AutoCad modelling of the dam volume above the original ground surface and footprint area with an assumed average stripping depth of 1 m through the footprint of the embankment. The starter dam volume (Figure 18 9) and ultimate dam volume (Figure 18-8) were calculated using AutoCAD.

Volumes of the filter and transition zones within the embankment dam section were calculated manually.

The dam face liner area was also calculated by Civil 3D as were the collection ditch volumes.

21.3.2 Off-site Infrastructure (WBS 30000)

21.3.2.1 Road Works

Road works volumes were developed based on the site layout, planned road alignments, existing conditions, and site topographical information. Unit rates received from budgetary quotations were used to estimate the cost of the road works. Road works considered as off-site infrastructure are the re-alignment of Tree Nursery Road, required to accommodate the South waste rock area, and the planned haul road connecting the Goliath site to Highway 72. These works are described in (Section 18.3.1).

21.3.3 Indirect Costs (WBS 40000)

Contractor indirect costs required to execute works associated as sustaining capital. Contractor indirect inclusions are described in (Section 21.2.8).

21.4 Closure Costs

Minnow estimated the closure requirements inclusive of all necessary demolition, rehabilitation, revegetation, earth grading/contouring, scrap metal disposal/tipping fees, as well as post-closure monitoring. The total closure cost was calculated to be C$28.9 million for all three projects.

21.4.1 Process Plant

Site closure for the process plant area captures the cost associated with the demolition of equipment, process plant, and mining building infrastructure and remediation works of the site.

21.4.2 Site Rehabilitation

Site closure costs for the non-process plant footprint include works to soil cover, revegetate/hydroseed the stockpiles and TSF, and construct a closure spillway.

21.4.3 Monitoring and Inspection

Post operation maintenance, monitoring and inspection costs are included for the requisite periods applicable to the respective projects.

21.5 Salvage Value

Salvaging costs have been projected by assuming that the process plant will carry a 10% resale value at the end of the mine life. Total salvaging value was estimated at C$9.9 million.

21.6 Operating Costs

The operating cost estimate is presented in Q3 2022 C$. The estimate was developed to have an accuracy of ±25%. The estimate includes mining, processing, and general and administration (G&A) costs.

The overall life-of-mine operating cost is C$1,447 million over 15 years, or an average of C$47.71/t of ore milled in a typical year. Of this total, processing and G&A account for C$451 million and mining accounts for C$995 million. Table 21-8 provides a summary of the project operating costs.

Table 21-8: Operating Cost Summary

Cost Centre	C$/t Milled (Over LOM)	LOM Operating Cost (C$M)
Mining		
Open Pit Mining	15.4	467.2
Underground Mining	13.5	409.2
Transportation	3.9	119.0
Mining Subtotal	**32.8**	**995.3**
Process Plant		
Reagents	3.2	97.7
Consumables	2.7	81.4
Plant Maintenance	0.6	17.9
Power	2.4	72.7
Laboratory	0.1	2.1
Labour – Process Plant	2.4	72.1
Process Plant Subtotal	**11.3**	**343.9**
G&A		
G&A Expenses	1.5	45.1
Mobile Equipment	0.2	6.7
Effluent Treatment Plant	1.8	55.6
G&A Subtotal	**3.5**	**107.4**
Total Project Operating Costs	47.7	1,446.6

21.6.1 Overview

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2022 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.34 Canadian dollar to 1.00 US dollar was assumed.
- Estimated costs for diesel and gasoline are C$1.00/L and C$1.045/L, respectively.
- The annual power costs were calculated using a unit price of C$0.07/kWh. This cost was derived considering a C$0.02/kWh discount associated with the Northern Energy Advantage Program, a program supported by the Government of Ontario to support Northern Ontario's largest industrial electricity consumers.

21.6.2 Open Pit Mining

A mining cost model has been developed to estimate the open pit mining capital and operating expenditures based on owner-operation, except for the haulage between Goldlund and Goliath which will be undertaken by a contractor. The operating cost estimate has been completed to a PFS level. The cost estimate is in C$ and has been developed by SRK based on quotes obtained from local manufacturers and suppliers and SRK's internal cost database. The open pit mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM).

The open pit mining operating cost estimate is shown in Table 21-9.

Table 21-9: Open Pit Mining Operating Costs

Operating Costs	LOM Total (C$M)	Y 1 to 13 Total (C$M)	Y 1 to 13 Unit Cost (C$/t)
Category			
Labour	228.9	209.4	1.89
Maintenance	50.9	45.8	0.41
Fuel	101.0	91.8	0.83
Lubricants	23.5	21.4	0.19
Tires	13.1	12.1	0.11
Wear Parts	17.6	15.8	0.14
Explosives	67.7	60.2	0.54
Sampling	2.2	1.9	0.02
Contract	8.1	7.8	0.07
Miscellaneous	1.2	1.1	0.01
Total	514.1	467.2	4.22
Activity			
Management	33.1	30.8	0.28
Technical Services	25.7	23.8	0.22
Loading	88.3	80.0	0.72
Hauling	115.8	105.5	0.95
Ancillary	93.2	85.6	0.77
Drilling	63.9	56.8	0.51
Blasting	78.2	70.0	0.63
Water Management	4.3	4.0	0.04
Grade Control	5.6	5.4	0.05
Contract	4.7	4.3	0.04
Miscellaneous	1.2	1.1	0.01
Total	514.1	467.2	4.22
LOM: Life of Mine			

Source: SRK, 2023

21.6.2.1 Transportation

The ROM will be transported by truck from Goldlund to Goliath by contractor. The contractor will provide a loader at Goldlund to load the material onto trucks and haul the material to a stockpile at Goliath. The contractor will also maintain the road to be used for transportation. The cost estimate has been based on budget quotes received from contractors for this task. The transportation cost has been estimated at C$7.00/t.

21.6.3 Underground Mining

The mining cost model for Goliath underground assumes that the majority of activities will be undertaken by a contractor, who will furnish their own personnel and equipment.

The operating cost estimate has been completed to a PFS level. The cost estimate is in C$ and has been developed by SRK based on quotes obtained from original equipment manufacturers and suppliers, and SRK's internal cost database. The underground mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM). The underground mining operating cost estimate is shown Table 21-10.

Table 21-10: Underground Mining Operating Costs

Category	LOM Total Cost (C$M)	LOM Unit Cost (C$/t Underground Ore)
Direct Mining		
Lateral Development (less labour & equipment)	30.4	8.06
Paste Fill	109.5	28.99
Production Drilling	15.2	4.03
LHD Mucking	27.7	7.33
Production Blasting	5.7	1.51
UG Truck Haulage	18.0	4.76
Direct Mining Subtotal	206.4	54.68
Indirect Mining		
Power - Ventilation	11.3	3.00
Power – Mining & Dewatering	4.8	1.28
Underground Services & Upkeep	3.8	1.00
Supplies Handling	8.2	2.16
Direct Mining Subtotal	28.1	7.44
Labour & Salary		
Salary	52.4	13.87
Operational & Maintenance Labour	114.9	30.43
Labour & Salary Subtotal	167.3	44.31
Ventilation Heating	7.6	2.02
Total Mining Operating Cost	**409.2**	**108.44**

Source: SRK, 2023

21.7 Process Operating Costs

Unless stated otherwise, all costs presented in this chapter are in C$. This estimate aligns with the principles of a Class 4 pre-feasibility study level estimate with a ±25% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The processing operating cost estimate includes costs relating to reagent and consumable consumption, plant maintenance, power use, the laboratory, labour, and processing mobile equipment.

As the process plant is expected to run under two operating conditions, at telluride and non-telluride conditions, shared and separate process plant operating costs were derived based on mill power requirements and process plant reagent and consumable requirements. Average annual LOM operating costs for the process plant were derived from the expected contributions from each deposit per year, and weighted. A breakdown of the expected average annual plant operating costs is presented in Table 21-11. An annualized breakdown of the operating costs relating to the process plant in its entirety is provided in Table 21-12. The average yearly process operating costs amount to C$34.2 million, or $14.88 of ore milled.

Table 21-11: Average LOM Process Plant Operating Cost Summary

Cost Centre	C$/t	Average over LOM (C$M/a)
Plant Maintenance	0.59	1.38
Laboratory	0.07	0.16
Labour – Process Plant	2.38	5.54
Reagents	3.22	7.52
Consumables	2.69	6.26
Power	2.40	5.59
Process Plant Subtotal	11.34	26.45
G&A Expenses	1.49	3.47
Mobile Equipment	0.22	0.51
Effluent Treatment Plant	1.83	4.27
G&A Subtotal	3.54	8.26
Total Processing Operating Costs	14.88	34.17

Table 21-12: Annualized Process Plant Operating Costs

Year	Unit	1	2	3	4	5	6	7	8	9	10	11	12	13
Throughput														
Ore Processed at Non-Telluride Conditions	kt	2,004	537	534	533	533	533	745	1,035	1,075	1,297	1,308	1,345	1,343
Ore Processed at Telluride Conditions	kt	0	1,821	1,830	1,825	1,825	1,825	1,619	1,323	1,283	1,061	1,056	1,013	1,015
Operating Costs														
Power Costs	C$M	4.3	5.9	5.9	5.9	5.9	5.9	5.8	5.7	5.7	5.6	5.4	5.3	5.3
Reagent Costs	C$M	6.5	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.6	7.7	7.6	7.6
Consumable Costs	C$M	3.1	7.5	7,5	7.5	7.5	7.5	7.1	6.5	6.4	5.9	5.0	5.0	5.0
Labour Costs	C$M	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5	5.5
Maintenance Costs	C$M	1.2	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4
Laboratory Costs	C$M	0.1	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2	0.2
General and Administrative Costs	C$M	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5	3.5
Mobile Equipment Costs	C$M	0.4	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5	0.5
Effluent Treatment Costs	C$M	2.5	5.3	4.0	4.0	4.0	4.0	5.3	5.3	5.3	4.0	4.0	4.0	4.0
Total Operating Costs	C$M	27.2	37.4	36.1	36.1	36.1	36.1	36.9	36.1	36.0	34.1	33.1	33.0	33.0

21.7.1 Basis of Estimate

The following was used to determine the project's LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a facility capable of processing 6,460 t/d of ore.

The following was used to determine the project's LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a facility capable of processing 6,460 t/d of ore.Assumptions made in developing the process operating cost estimate are listed below:

- mill production is set at an average of 2.4 Mt/y
- process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables
- off-site gold refining, insurance, and transportation costs are excluded, as they are included elsewhere
- labour rates were provided by Treasury Metals Inc
- general and administration (G&A) costs were baselined against previous project experience, defined along with specific inputs from Treasury Metals Inc
- no factor for spare parts has been applied to adjust for consumption of fewer spare parts in early years of operation
- grinding media consumption rates have been estimated based on the ore characteristics
- reagent consumption rates have been estimated based on the metallurgical testwork results at a nominal basis
- mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease.

21.7.2 Reagents and Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Major reagent costs were obtained from vendor quotations to Dryden, including SAG and ball mill media, sodium hydroxide, hydrated lime, flocculant, activated carbon, and sodium metabisulphite (SMBS). Other reagent cost was obtained through benchmarking for similar projects performed by Ausenco.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using:

- metallurgical testing results (Bond abrasion testing)
- Ausenco's in-house calculation methods, including simulations
- forecast nominal power consumption.

Reagents and consumables represent approximately 52% of the average process operating costs at $5.91/t.

21.7.3 Maintenance

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using weighted average factors ranging between 2% and 6%. The factor was applied to mechanical equipment. The total maintenance consumables operating cost is C$0.59/t milled, or approximately 4% of the direct mechanical capital cost.

21.7.4 Power

The processing power draw was based on the average power utilization of each motor on the electrical load list for the process plant and services.

An estimated 72,000 MWh are nominally required per year for processing Goliath ore at non-telluride conditions and 88,400 MWh are nominally required per year for processing Goldlund ore at telluride conditions, resulting in an average annual power cost of $5.59 million, or $2.40/t milled. This represents 21% of the processing operating costs.

21.7.5 Laboratory

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year, estimated by Ausenco. Assay costs include plant solid samples taken from various samplers throughout the plant, solution samples, tests on the loaded, barren, and regenerated carbon, bullion bar testing, cyanide detoxification sampling, and environmental sampling and assaying. The laboratory and assays comprise approximately 0.6% of the total process operating cost. Approximately 16,000 internal assays are required per year.

21.7.6 Labour

The personnel requirement was estimated by benchmarking against similar projects. The labour costs incorporate personnel requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total process plant labour averages 55 employees.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. Salaries were provided by Treasury Metals Inc. Treasury metals Inc. also confirmed the specific benefits and bonuses to be allocated. The rates were estimated as overall rates, including all burden costs.

An organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-13. Process plant labour represents approximately 21% of the process operating costs.

Table 21-13: Operations and Maintenance Staffing Plan

Labour / Contractor Summary	#/Shift	# Shifts	Quantity
Process Upper Management			
Plant Superintendent	1	1	1
Senior Metallurgist	1	1	1
Mill Trainer	1	1	1
Chief Assayer	1	1	1
Maintenance Superintendent	1	1	1
Mill Operations			
Shift Foreman	1	4	4
Crusher Operator	1	4	4
Grinding/Gravity Operator	1	4	4
Leach/Elution Operator	1	4	4
Reagents Labourer	1	4	4
Control Room Operator	1	4	4
Gold Room Operator	1	4	4
Surface Crew/Tailings	1	4	4
Technical Services			
Lab Manager	1	1	1
Lead Heads	1	1	1
Assay Lab Technicians	1	1	1
Mill Maintenance			
Maintenance Foreman	1	1	1
Maintenance Planner	1	1	1
Electrician	1	2	2
Electrical Foreman	1	1	1
Welder	1	2	2
Instrument Technician	1	2	2
Millwright/Fitter	2	2	4
Apprentice	1	2	2
Total Process Plant	25	53	55

21.7.7 General and Administrative Operating Costs

21.7.7.1 G&A Expenses

General and administrative (G&A) costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed with input from Treasury Metals Inc., as well as Ausenco's in-house data on existing Canadian operations.

- The G&A costs were determined for a 12-year mine life with an average cost of C$3.54/t milled. These costs were assembled according to the following departmental cost reporting structure:
- G&A maintenance (includes vehicle maintenance and road maintenance)
- G&A personnel
- human resources (includes recruiting, training, and community relations)
- site administration, maintenance, and security (includes professional memberships and dues, office supplies and equipment, in-town office rental, sewage and garbage disposal, and bank fees)
- health and safety (includes personal protective equipment and first aid supplies)
- asset operation (includes non-operation-related vehicles)
- environmental (includes sampling)
- IT and telecommunications (includes software and microwave link)
- contract services (includes insurance, sanitation, licenses, and legal fees)
- cyanide code fees.

The G&A labour costs were estimated by developing a headcount profile for each department. Labour rates provided by Treasury Metals Inc. were applied to develop the total G&A labour cost.

G&A labour resources include 14 employees. An organizational staffing plan outlining the G&A labour requirement is shown in Table 21-14.

Table 21-14: General and Administrative Staffing Plan

Labour / Contractor Summary	#/Shift	# Shifts	Quantity
General Manager	1	1	1
H&S and Operations Training Superintendent	1	1	1
Safety Technicians	1	1	1
Manager HR	1	1	1
Manager – Procurement / Contracts	1	1	1
Manager Environment	1	1	1
Environmental Technician	1	1	1
Security EMT	1	2	2
Security Personnel	1	2	2
Warehouse Operator	1	2	2
Warehouse Attendant	1	1	1
Total G&A	11	14	14

21.7.7.2 Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment (including fuel, maintenance, spares and tires, and annual registration and insurance fees). The cost of operating and maintaining the processing mobile vehicles is estimated as $0.22/t milled.

21.7.7.3 Effluent Treatment Plant

Water treatment costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were informed by effluent treatment plant vendors regarding required power and treatment reagents for operation and treatment costs.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

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

- mineral resource estimates
- assumed commodity prices and exchange rates
- the proposed mine production plan
- projected mining and process recovery rates
- assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged the timing and amount of estimated 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:

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 being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- ability to maintain the social licence to operate
- changes to interest rates
- changes to tax rates.

22.2 Methodologies Used

The project has been evaluated using a discounted cash flow (DCF) analysis based on a 5% discount rate. Cash inflows consist of annual revenue projections. Cash outflows consist of capital expenditures, including pre-production costs, operating costs, taxes, and royalties. These are subtracted from the inflows to arrive at the annual cash flow projections. Cash flows are taken to occur at the mid-point of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations, and as such, the actual post-tax results may differ from those estimated. A sensitivity analysis was performed to assess the impact of variations in gold price, discount rate, foreign exchange, total operating cost, and initial capital cost. The capital and operating cost estimates developed specifically for this project are presented in Section 21 of this report in Q3 2022 Canadian dollars. The economic analysis has been run on a constant dollar basis with no inflation.

22.3 Financial Model Parameters

The economic analysis was performed assuming the base case gold price of US$1,750/oz, and silver price of US$21/oz. The forecasts used are meant to reflect the average metals price expectation over the life of the project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- The construction period will be 1.5 years
- the mine life is 13 years
- cost estimates are in constant Q3 2022 Canadian dollars with no inflation or escalation factors considered
- capital costs are funded with 100% equity (no financing assumed)
- all cash flows are discounted to the start of the construction period using a mid-period discounting convention
- working capital based on accounts receivable of 0 days, accounts payable of 30 days, and inventory of 30 days
- all metal products will be sold in the same year they are produced
- no contractual arrangements for refining currently exist.

22.4 Royalties

The economic analysis assumes the Company exercises its right to repurchase 50% of the 2.2% Net Smelter Returns Royalty that the Company sold to Sprott Resources Streaming and Royalty Corp for US$20 million in April 2022 and 0.5% of the 1.5% Net Smelter Returns Royalty that the Company sold to First Mining Gold Corp. in August 2020 as part of the purchase of Tamaka Gold Corporation. In addition, several other smaller royalties across the property package are assumed to be repurchased. The cost of the repurchase of these royalties are excluded from project level economic analysis.

22.5 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics. The tax model was compiled with assistance from third-party taxation professionals. The calculations are based on the tax regime as of the date of the feasibility study update. At the effective date of the cashflow, the project was assumed to be subject to the following tax regime:

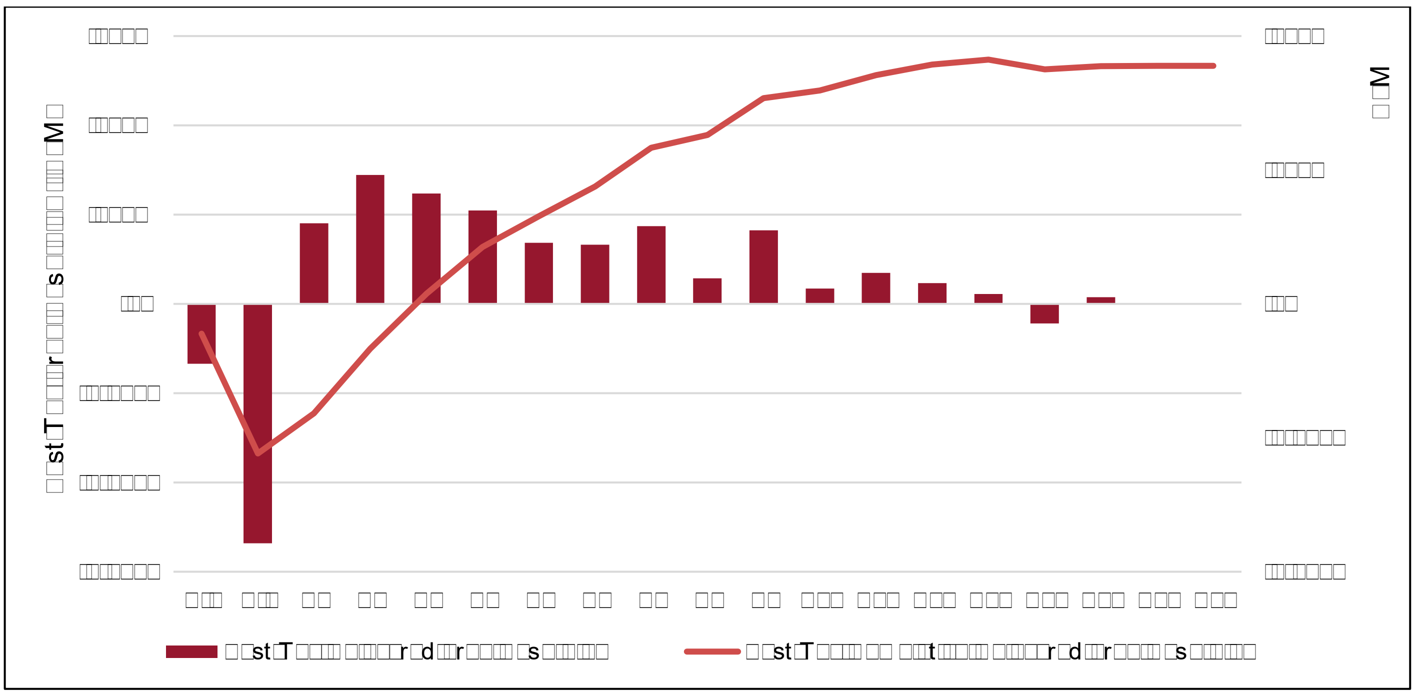
- The Canadian corporate income tax system consists of 15% federal income tax and 10% provincial income tax
- the mining tax rate in Ontario is 10%.

At the base case gold and silver price assumptions, total tax payments are estimated to be $201 million over the life of mine.

22.6 Economic Analysis

The economic analysis was performed assuming an 5% discount rate. The pre-tax NPV discounted at 5% is $469 million; the IRR is 29.3%, and payback period is 2.8 years. On a post-tax basis, the NPV discounted at 5% is $336 million, the IRR is 25.4%, and the payback period is 2.8 years. A summary of project economics is shown graphically in Figure 22-1 and listed in Table 22-1. The analysis was done on an annual cashflow basis; the cashflow output is shown Table 22-2.

Figure 22-1: Post-Tax Project Economics



Source: Ausenco, 2023.

Table 22-1: Economic Analysis Summary

Description	Unit	Life-of-Mine Total / Average
General Assumptions		
Gold Price	US$/oz	1,750
Silver Price	US$/oz	21
Discount Rate	%	5.0%
Exchange Rate	USD:CAD	0.75
Production		
Mill Head Grade Au	g/t	1.30
Mill Head Grade Ag	g/t	1.77
Mill Recovery Rate Au	%	92.8%
Mill Recovery Rate Ag	%	60.0%
Total Mill Ounces Recovered Au	koz	1,175
Total Mill Ounces Recovered Ag	koz	1,034
Total Average Annual Production Au	koz	90
Total Average Annual Production Ag	koz	80
Operating Costs		
Open Pit Mining Cost	C$/t mined	4.22
Underground Mining Cost	C$/t mined	61.23
Mining Cost (Open Pit + Underground)	C$/t milled	32.83
Goldlund Ore Haulage to Mill	C$/t milled	7.00
Processing Cost	C$/t milled	11.34
G&A Cost	C$/t milled	3.54
Refining and Transport Au	C$/oz Au	5.00
Refining and Transport Ag	C$/oz Ag	0.26
Total Operating Costs	C$/t milled	47.71
Cash Costs and All-in Sustaining Costs (Byproduct Basis)		
Operating Cash Costs*	US$/oz Au	935
All-in Sustaining Cost **	US$/oz Au	1,072
Capital Expenditures		
Initial Capital Cost	C$M	335
Sustaining Capital Cost	C$M	198
Closure Capital Cost	C$M	29
Salvage Value	C$M	10
Economics		
Pre-tax NPV @ 5%	C$M	469
Pre-tax IRR	%	29.3%
Pre-tax Payback	years	2.8
Post-tax NPV @ 5%	C$M	336
Post-tax IRR	%	25.4%
Post-tax Payback	years	2.8

Note: * Cash costs consist of mining costs, processing costs, G&A and refining charges and royalties. ** All-in Sustaining Costs (AISC) includes cash costs plus sustaining capital, closure costs and salvage value. Both cash costs and AISC are calculated on a by-product basis. Source: Ausenco, 2023.

Table 22-2: Cashflow Statement on an Annualized Basis (Real 2022 C$M Unless Otherwise Noted)

Macro Assumptions	Units	Total/ Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12	Y13	Y14	Y15
Gold Price - Flat	US$/oz	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750	1,750
Silver Price - Flat	US$/oz	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21	21
FX	US$:C$	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75	0.75
Free Cash Flow Valuation																			
Revenue	C$M	2,790	--	--	225	303	290	291	270	266	275	196	217	139	128	109	81	--	--
Operating Cost	C$M	(1,447)	--	--	(86)	(120)	(131)	(139)	(146)	(147)	(147)	(125)	(108)	(87)	(80)	(72)	(59)	--	--
Refining Charges	C$M	(6)	--	--	(1)	(1)	(1)	(1)	(1)	(1)	(1)	(0)	(0)	(0)	(0)	(0)	(0)	--	--
Royalties	C$M	(51)	--	--	(3)	(7)	(7)	(6)	(5)	(5)	(5)	(3)	(4)	(2)	(2)	(1)	(1)	--	--
EBITDA	C$M	1,286	–	–	136	176	152	144	119	113	122	67	104	49	47	36	20	–	–
Initial Capex	C$M	(335)	(67)	(268)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	C$M	(198)	--	--	(46)	(31)	(28)	(23)	(19)	(21)	(9)	(8)	(7)	(4)	(0)	(0)	(0)	--	--
Closure Capex	C$M	(29)	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	(29)	--
Salvage Value	C$M	10	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	10	--
Change in Working Capital	C$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	C$M	734	(67)	(268)	90	145	124	121	100	92	112	59	98	45	46	35	20	(19)	–
Pre-Tax Cumulative Unlevered Free Cash Flow	C$M		*(67)*	*(335)*	*(245)*	*(100)*	*24*	*145*	*245*	*337*	*450*	*509*	*606*	*651*	*698*	*733*	*753*	*734*	*734*
Unlevered Cash Taxes	C$M	(201)	--	--	--	(0)	(0)	(17)	(31)	(26)	(25)	(30)	(15)	(28)	(12)	(12)	(9)	(3)	7
Post-Tax Unlevered Free Cash Flow	C$M	533	(67)	(268)	90	144	124	105	69	66	87	29	82	17	35	23	11	(22)	7
Post-Tax Cumulative Unlevered Free Cash Flow	C$M		*(67)*	*(335)*	*(245)*	*(101)*	*23*	*128*	*196*	*263*	*350*	*379*	*461*	*478*	*513*	*536*	*547*	*525*	*533*
Production																			
Total Resource Mined - UG	kt	3,776	--	--	18	72	219	324	413	407	494	390	368	374	358	245	93	--	--
Total Resource Mined - OP	kt	30,370	--	2,196	3,902	4,086	3,468	3,489	3,577	4,222	3,158	1,142	1,130	--	--	--	--	--	--
Total Waste – UG	kt	2,905	--	--	215	340	514	539	506	426	212	23	0	111	10	2	6	--	--
Total Waste - OP	kt	94,302	--	11,804	9,783	10,008	10,367	10,144	10,403	10,270	10,742	8,008	2,773	--	--	--	--	--	--
Total Material Mined	kt	131,353	--	14,000	13,918	14,506	14,569	14,497	14,898	15,324	14,606	9,563	4,272	486	368	247	99	--	--
Strip Ratio - OP	w:o	3.11	--	5.38	2.51	2.45	2.99	2.91	2.91	2.43	3.40	7.01	2.45	--	--	--	--	--	--
Mill Feed	kt	30,318	--	--	2,004	2,358	2,364	2,358	2,358	2,358	2,364	2,358	2,358	2,358	2,364	2,358	2,358	--	--
Mill Head Grade (Au)	g/t	1.30	--	--	1.53	1.84	1.75	1.76	1.62	1.57	1.63	1.18	1.30	0.84	0.76	0.65	0.49	--	--
Mill Head Grade (Ag)	g/t	1.77	--	--	3.80	1.36	1.33	1.28	1.34	1.46	1.70	1.80	2.05	1.71	2.01	1.86	1.58	--	--
Contained (Au)	koz	1,267	--	--	99	139	133	134	123	119	124	90	99	64	58	50	37	--	--
Contained (Ag)	koz	1,723	--	--	245	103	101	97	101	111	130	137	155	129	153	141	120	--	--
Mill Recovery (Au)	%	92.8%	--	--	95.0%	92.2%	92.1%	92.1%	93.3%	94.7%	93.9%	91.8%	92.4%	90.9%	92.5%	91.8%	90.1%	--	--
Mill Recovery (Ag)	%	60.0%	--	--	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	60.0%	--	--
Gold Production	koz	1,175	--	--	94	128	123	123	114	113	116	82	91	58	54	46	34	--	--
Silver Production	koz	1,034	--	--	147	62	61	58	61	66	78	82	93	78	92	85	72	--	--
Gold % Payable	%	100%	--	--	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	100%	--	--
Payable Gold	koz	1,175	–	–	94	128	123	123	114	113	116	82	91	58	54	46	34	–	–
Payable Silver	koz	1,034	–	–	147	62	61	58	61	66	78	82	93	78	92	85	72	–	–
Gold Revenue	C$M	2,760	–	–	220	302	288	289	269	264	273	193	214	137	126	107	79	–	–
Silver Revenue	C$M	29	–	–	4	2	2	2	2	2	2	2	3	2	3	2	2	–	–
Total Revenue	C$M	2,790	–	–	225	303	290	291	270	266	275	196	217	139	128	109	81	–	–
Operating Costs																			
Total Operating Costs	C$M	1,447	–	–	86	120	131	139	146	147	147	125	108	87	80	72	59	–	–
Mine Operating Costs	C$M	995	--	--	58	82	95	103	110	111	111	89	72	53	47	39	26	--	--
Mill Processing	C$M	344	--	--	21	28	28	28	28	28	28	27	27	26	25	25	25	--	--
G&A Costs	C$M	107	--	--	6	9	8	8	8	8	9	9	9	8	8	8	8	--	--
Operating Costs per Tonne Processed	*C$/t Milled*	*47.71*	--	--	*42.67*	*50.80*	*55.32*	*59.01*	*61.80*	*62.40*	*62.37*	*52.88*	*45.96*	*37.02*	*33.66*	*30.57*	*25.03*	--	--
Refining & Royalties																			
Refining Charges	C$M	6	–	–	1	1	1	1	1	1	1	0	0	0	0	0	0	–	–
Au Refining Charges	C$M	6	--	--	0	1	1	1	1	1	1	0	0	0	0	0	0	--	--
Ag Refining Charges	C$M	0	--	--	0	0	0	0	0	0	0	0	0	0	0	0	0	--	--

Royalties	C$M	51	–	–	3	7	7	6	5	5	5	3	4	2	2	1	1	–	–
Cash Costs																			
Cash Cost *	US$/oz Au	935	--	--	671	727	828	875	977	999	968	1,141	897	1,122	1,098	1,166	1,295	--	--
All-in Sustaining Cost (AISC) **	US$/oz Au	1,072	--	--	1,034	910	995	1,015	1,101	1,140	1,029	1,214	953	1,172	1,105	1,174	1,298	--	--
Capital Expenditures																			
Initial Capital	C$M	335	67	268	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–
Capitalized Mining Opex	C$M	47	--	47	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Financed Mining Equipment	C$M	8	--	8	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Non-Financed Mining Equipment	C$M	9	--	9	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Underground Mining	C$M	4	--	4	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Process Plant	C$M	99	25	74	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
On-site Infrastructure	C$M	75	19	57	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Off-site Infrastructure	C$M	4	1	3	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Project Indirects	C$M	24	6	18	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Project Delivery	C$M	14	4	11	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Owner's Costs	C$M	17	4	12	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Provisions	C$M	35	9	26	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	C$M	198	–	–	46	31	28	23	19	21	9	8	7	4	0	0	0	–	–
Mining Equipment	C$M	42	--	--	11	8	8	3	2	5	1	2	1	1	--	0	--	--	--
Underground Mine Development	C$M	91	--	--	9	17	15	16	13	12	3	1	1	3	0	0	0	--	
Mining Infrastructure	C$M	23	--	--	21	2	--	--	--	--	--	--	--	--	--	--	--	--	
TSF	C$M	42	--	--	5	5	5	5	5	5	5	5	5	--	--	--	--	--	
Closure Cost	C$M	29	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	29	–
Salvage Value	C$M	10	–	–	–	–	–	–	–	–	–	–	–	–	–	–	–	10	–
Total Capital Expenditures Including Salvage Value	C$M	552	67	268	46	31	28	23	19	21	9	8	7	4	0	0	0	19	–

Note: * Cash costs consist of mining costs, processing costs, G&A and refining charges and royalties. ** All-in Sustaining Costs (AISC) includes cash costs plus sustaining capital, closure costs and salvage value. Both cash costs and AISC are calculated on a by-product basis. Source: Ausenco, 2023.

22.7 Sensitivity Analysis

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

Tables 22-3 and 22-4 show the post-tax sensitivity analysis results.

As shown in Figures 22-3 and 22-4, the sensitivity analysis revealed that the project is most sensitive to changes in gold price and foreign exchange, and less sensitive to discount rate, total operating cost, and initial capital cost.

Table 22-3: Post-Tax Sensitivity Summary

Gold Price	Post-Tax NPV(5%)	Initial Capital Cost		Total Operating Cost		Foreign Exchange	
US$/oz	Base Case	(-20%)	(+20%)	(-20%)	(+20%)	(-20%)	(+20%)
$1,550	$178	$242	$114	$321	$30	$486	($52)
$1,600	$218	$282	$153	$361	$73	$535	($4)
$1,750	$336	$400	$271	$479	$192	$682	$103
$1,900	$453	$518	$389	$596	$310	$829	$202
$2,000	$532	$596	$467	$675	$389	$928	$268
Gold Price	Post-Tax IRR	Initial Capital Cost		Total Operating Cost		Foreign Exchange	
US$/oz	Base Case	(-20%)	(+20%)	(-20%)	(+20%)	(-20%)	(+20%)
$1,550	16.6%	23.8%	11.4%	24.0%	7.2%	33.2%	1.0%
$1,600	18.9%	26.5%	13.5%	26.1%	10.3%	35.6%	4.8%
$1,750	25.4%	34.2%	19.3%	32.1%	17.9%	42.5%	11.9%
$1,900	31.6%	41.4%	24.6%	37.7%	24.7%	49.3%	18.0%
$2,000	35.4%	45.9%	28.0%	41.4%	29.0%	53.7%	21.7%

Source: Ausenco, 2023.

Figure 22-2: Sensitivity Analysis

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
1.0%	287	336	485	634	733
3.0%	228	272	404	536	624
5.0%	178	218	336	453	532
8.0%	116	150	251	351	418
10.0%	82	113	204	295	355

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
1.0%	16.5%	18.9%	25.4%	31.6%	35.4%
3.0%	16.6%	18.9%	25.4%	31.6%	35.4%
5.0%	16.6%	18.9%	25.4%	31.6%	35.4%
8.0%	16.6%	18.9%	25.4%	31.6%	35.4%
10.0%	16.6%	18.9%	25.4%	31.6%	35.4%

Post-Tax NPV Sensitivity To Foreign Exchange

Foreign Exchange	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	486	535	682	829	928
(10.0%)	315	359	490	620	708
--	178	218	336	453	532
10.0%	64	101	209	316	388
20.0%	(52)	(4)	103	202	268

Post-Tax IRR Sensitivity To Foreign Exchange

Foreign Exchange	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	33.2%	35.6%	42.5%	49.3%	53.7%
(10.0%)	24.3%	26.7%	33.4%	39.6%	43.7%
--	16.6%	18.9%	25.4%	31.6%	35.4%
10.0%	9.4%	11.8%	18.4%	24.4%	28.2%
20.0%	1.0%	4.8%	11.9%	18.0%	21.7%

Post-Tax NPV Sensitivity To Total Operating Cost

Total Operating Cost	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	321	361	479	596	675
(10.0%)	250	289	407	525	603
--	178	218	336	453	532
10.0%	106	146	264	382	460
20.0%	30	73	192	310	389

Post-Tax IRR Sensitivity To Total Operating Cost

Total Operating Cost	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	24.0%	26.1%	32.1%	37.7%	41.4%
(10.0%)	20.5%	22.6%	28.8%	34.7%	38.4%
--	16.6%	18.9%	25.4%	31.6%	35.4%
10.0%	12.3%	14.8%	21.8%	28.2%	32.3%
20.0%	7.2%	10.3%	17.9%	24.7%	29.0%

Post-Tax NPV Sensitivity To Initial Capital Cost

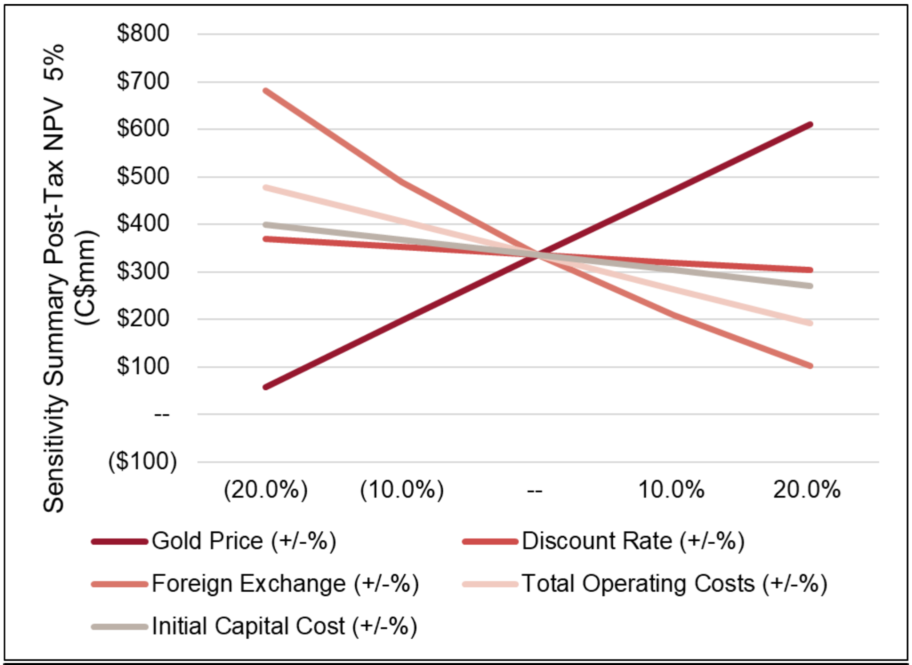
Initial Capital Cost	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	242	282	400	518	596
(10.0%)	210	250	368	486	564
--	178	218	336	453	532
10.0%	146	185	303	421	499
20.0%	114	153	271	389	467

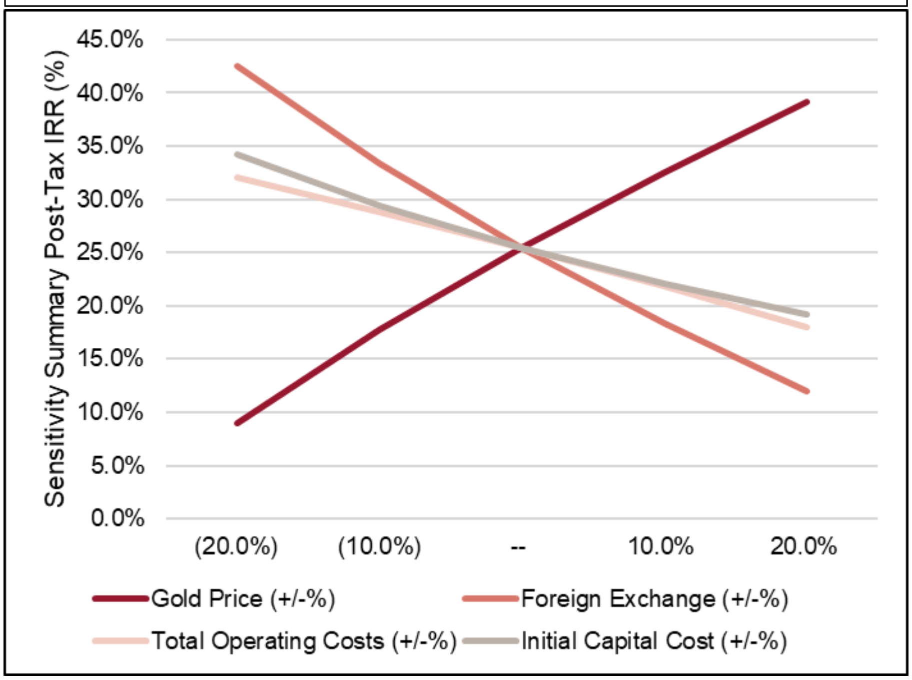
Post-Tax IRR Sensitivity To Initial Capital Cost

Initial Capital Cost	Gold Price (US$/oz) $1,550	$1,600	$1,750	$1,900	$2,000
(20.0%)	23.8%	26.5%	34.2%	41.4%	45.9%
(10.0%)	19.8%	22.4%	29.4%	36.0%	40.1%
--	16.6%	18.9%	25.4%	31.6%	35.4%
10.0%	13.8%	16.0%	22.1%	27.8%	31.4%
20.0%	11.4%	13.5%	19.3%	24.6%	28.0%

Source: Ausenco, 2023.

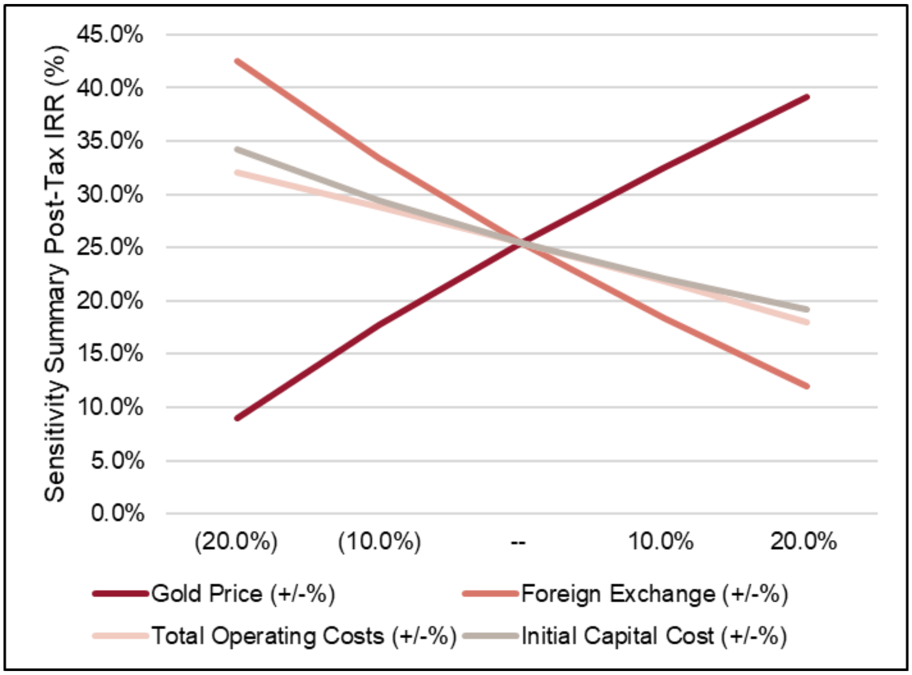
Figure 22-3: Post-Tax NPV Sensitivity Results





Source: Ausenco, 2023.

Figure 22-4: Post-Tax IRR Sensitivity Results



Source: Ausenco, 2023.

23 ADJACENT PROPERTIES

This section is not relevant to this report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this report.

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 Goliath property covers approximately 7,601 ha and consists of 284 mining claims totalling approximately 6,254 ha; four mining leases totalling 359.25 ha; and 28 land parcels (includes patented claims) totalling 1,347.189 ha. Of the 1,347.18 ha of the patents and leases, 90.2 ha are surface rights only from seven land parcels. Of the 284 mining claims, 267 are single-cell mining claims, eight are boundary cell mining claims, and nine are multi-cell mining claims. The mineral rights are 100% held by Treasury Metals and all mineral rights are in good standing.

The Goliath property is held 100% by Treasury Metals, subject to certain underlying royalties and payment obligations on 13 of the 21 land parcels, totalling approximately $103,500 per year.

The Goldlund-Miller property consists of 1,349 mining claims totalling approximately 26,634 ha, 26 patented claims totalling 390.97 ha, one mining lease of 48.56 ha, and one licence of occupation of 74.84 ha. The patented claims and mining lease allow for both mineral rights and surface rights, while the Licence of Occupation allows for mineral rights only.

The Goldlund Property is subject to the Goldlund Mines Limited Royalty Agreement, covering six patented claims as well as the three patented claims covered by the Mining Lease. Goldlund Mines will receive a 1% NSR on any ore mined above 50 m below the existing shaft collar as of the date of the agreement. The Goldlund is also subject to the Rio Algom Limited Option Agreement whereby the Property owner will pay a 2.5% NSR and will have the right but not the obligation to purchase the NSR in its entirety for a one-time payment of $2.5 million with a 10-day notification of intent to exercise the purchase right.

25.3 Geology and Mineralization

The Goliath project is located in the Archean Eagle-Wabigoon-Manitou greenstone belt in the Wabigoon Subprovince of the Superior Province. In the immediate area of the deposit, a 100 to 150 m thick unit of intensely deformed and variably altered, fine- to medium-grained, quartz-feldspar-sericite schist and biotite-quartz-feldspar-sericite schist with minor metasedimentary rocks hosts the most significant gold concentrations of gold in the Main and C Zones of the deposit.

Native gold and silver are associated with finely disseminated sulphides, coarse-grained pyrite and very narrow light grey translucent “ribbon” quartz veining. The main sulphide phases are pyrite, sphalerite, galena, pyrrhotite, minor chalcopyrite and arsenopyrite and dark grey needles of stibnite. The alteration consists of primarily sericitization and silicification in association with the gold mineralization.

At Goliath, the gold-bearing zones strike from 090° to 072° with dips that are consistently between 70° and 80° south or southeast. The mineralized zones are tabular composite units defined on the basis of moderate to strongly altered rock units, anomalous to strongly elevated gold concentrations, and increased sulphide content and are concordant to the local stratigraphic units. In the Goliath deposit, higher grade gold mineralization occurs in shoots with relatively short strike-lengths (up to 50 m) that plunge steeply to the west. The main area of gold, silver and sulphide mineralization and alteration occurs up to a maximum drill-tested vertical depth of ~805 m over a drill-tested strike-length in excess of 2,500 m. Gold mineralized zones remain open at depth.

At Goldlund, Gold mineralization is hosted by zones of northeast-trending and gently to moderately northwest-dipping quartz stockworks, comprised of numerous quartz veinlets less than 1 to 20 cm thick. The stockwork zones are hosted in albite-trondhjemite to diorite (granodiorite) strata-parallel sills, which dip from vertical to -80° southward and range in thickness from 14 m to 60 m. The stockwork zones form bands within the granodiorite sills that intrude the east-northeast-trending mafic metavolcanic rocks. The quartz veins and veinlets contain occasional fine-grained to coarse-grained pyrite. The intervening areas between the quartz veinlets exhibit strong to moderate feldspathic alteration associated with common fine to medium-grained pyrite and magnetite.

The mineralized sills strike generally northeast (065°) and dip steeply to the southeast. The quartz stockwork veins at Goldlund consist of two synchronous sets of veins, referred to as the 20 set and the 70 set (Pettigrew, 2012). The gold-bearing veins display a remarkable consistency in form across the project.

The gold mineralization has been interpreted as a series of nine northeast-trending sub-parallel zone wireframes, considering a nominal 0.1 g/t Au threshold. Wireframes of Zones 1, 7, and 5 consist principally of gold mineralization associated with the stockwork veins in the large granodiorite sills, while wireframes of Zones 2, 3, 4, 6, 8, and 9 consist of gold mineralization associated with stockwork veins that are hosted in several lithologies including andesite, and felsic to intermediate porphyries, with only a minor contribution from the granodiorite sills. While the Qualified Person for this section of the report believes that the interpretation of the mineralized zone wireframes is suitable for the estimation of mineral resources, the development of a 3D model of lithology, structure, and alteration would help to improve the interpretation of the mineralized zones and the understanding of the controls on gold mineralization.

The Miller deposit is analogous to the Goldlund deposit in that the gold mineralization is hosted within stockworks of veins and veinlets that occur within a granodiorite and feldspar porphyry lithology. The granodiorite is hosted within a sequence of regional andesite and gabbro lithologies. The deposit has been outlined along a 500 m long strike length, with a width up to 50 m, and appears to be open at depth.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The mineralisation was sampled over the years with multiple campaigns of core drilling by Teck-Corona and Treasury Metals since the 1990s. The drill database is now a mix of historical data and more recent data collected by Treasury Metals from 2008 through to 2021. Both data types were used in the resource estimate. The mineral resource estimate for Goliath is supported by 904 surface drill holes with an aggregated length of 290,6856 m.

For the Goldlund deposit, the dataset consists of 1,934 core holes representing 250,861 m of core (1,454 surface holes and 480 underground drill holes). In addition, the Goldlund data also includes 246 underground channel samples representing 3,637 m and 188 trenches and one pit for 1,444 m of sampling. Of these, 1,375 core holes contributed to the estimation of mineral resources for Goldlund. The underground channel and trench samples were not considered for grade estimate but were included in the modelling of mineralized zones.

There are 61 drill holes in the Miller database totalling 10,370 m of drilling, of these, 49 drill holes, 7,964 m contributed to the Miller resource estimate.

25.5 Metallurgical Testwork

Metallurgical testwork and associated analytical procedures were appropriate to the mineralisation type, appropriate to investigate the optimal processing routes, and were performed using samples that are typical of the mineralisation styles found within the various mineralized zones.

Samples selected for testing were representative of the various types and styles of mineralisation. Samples and were selected from a range of depths within the deposits. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Recovery factors estimated are based on appropriate metallurgical testwork and are appropriate to the mineralisation types and the selected process route. Based on the 2021 preliminary pre-feasibility testwork results on composite samples representative of the Goliath, Goldlund, and Miller deposits, doré can be produced at a high recovery of gold.

No deleterious elements that impair doré bullion quality were identified in the testing programs. Process conditions selected minimize the impact of problematic minerals which impact reagent consumptions and recoveries.

25.6 Mineral Resource Estimates

The mineral resources for the Goliath and Goldlund deposits were prepared by Dr. Gilles Arseneau and Ms. Sheila Ulansky of SRK (Canada) Inc. The mineral resources for the Miller deposit were prepared by Dr. Arseneau who is the qualified person for all three mineral resource statements presented in this technical report.

The mineral resources are prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). The estimated mineral resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. Mineral resources that are not mineral reserves do not have demonstrated economic viability. The effective date of the mineral resource statement is January 17, 2022 (Table 25-1).

Table 25-1: Goliath Gold Complex Mineral Resource Estimate

Type	Classification	Cut-off	Tonnes	Au (g/t)	Au (Oz)	Ag (g/t)	Ag (Oz)
Open Pit	Measured	0.25 / 0.3	6,223,000	1.20	239,500	4.70	940,600
	Indicated	0.25 / 0.3	58,546,000	0.82	1,545,000	2.53	1,878,500
	Measured + Indicated	0.25 / 0.3	64,769,000	0.86	1,784,500	2.99	2,819,100
	Inferred	0.25 / 0.3	32,301,000	0.73	754,900	0.80	85,200
Underground	Measured	2.20	170,000	6.24	34,100	22.34	122,100
	Indicated	2.20	2,772,000	3.59	320,000	7.08	580,800
	Measured + Indicated	2.20	2,942,000	3.74	354,100	8.04	702,900
	Inferred	2.20	270,000	3.21	27,900	4.06	6,300
Total	Measured		6,393,000	1.33	273,600	5.17	1,062,700
	Indicated		61,318,000	0.95	1,865,000	2.98	2,459,300
	Measured + Indicated		67,711,000	0.98	2,138,600	3.42	3,522,000
	Inferred		32,571,000	0.75	782,800	0.84	91,500

Notes: 1. Mineral Resources were estimated by ordinary kriging by Dr. Gilles Arseneau, associate consultant of SRK Consulting (Canada) Inc., Mineral Resources were prepared in accordance with NI 43-101 and the CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (2019). This estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, sociopolitical, marketing, or other relevant issues. Mineral Resources that are not mineral reserves do not have demonstrated economic viability. 2. Mineral Resource effective date January 17, 2022. 3. Goliath Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.25g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60% respectively. 4. Goldlund Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.3g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 5. Miller Open Pit Mineral Resources are reported within an optimized constraining shell at a cut-off grade of 0.3 g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 93.873*Au(g/t)^0.021. 6. Goliath Underground Mineral Resources are reported inside shapes generated from Deswick Mining Stope Optimiser (DSO) at a cut-off grade of 2.2g/t gold that is based on a gold price of US$1,700/oz, a silver price of US$23/oz, and a gold and silver processing recovery of 93.873*Au(g/t)^0.021 and 60% respectively. 7. Goldlund Underground Mineral Resources are reported inside DSO shapes at a cut-off grade of 2.2g/t gold that is based on a gold price of US$1,700/oz and a gold processing recovery of 90.344xAu(g/t)^0.0527. 8. Gold and Silver assays were capped prior to compositing based on probability plot analysis for each individual zones. Assays were composited to 1.5 m for Goliath, 2.0 m for Goldlund and 1.0 m for Miller. 9. Excludes unclassified mineralization located within mined out areas. 10. Silver grade and ounces are derived from the Goliath tonnage only. 11. Goliath Open Pit and Goldlund/Miller cut-off grades are 0.25g/t and 0.30g/t, respectively. 12. All figures are rounded to reflect the estimates' relative accuracy, and totals may not add correctly.

25.7 Mining Methods

The application of both open pit and underground extraction methods is seen as appropriate for the three deposits comprising the Goliath Project given individual deposit geometry, depth from surface, and economic considerations. The mining methods are modern and utilize conventional, widely accepted processes and technology. The cut-off values selected were used to identify the areas that can be economically extracted. Mining will occur for a 13-year duration, with extraction shifting from the three deposits in order to provide the most amenable mill feed.

The mineral reserve tonnages and grades presented include estimates for both external dilution and mining recovery. Process rates and mining productivities are based on first principles calculations and experience with deposits in similar settings. The production schedules were generated via Deswik software, with the individual mines' schedules blended and aggregated into a single schedule to provide an optimal feed to the mill.

25.8 Recovery Methods

The process plant is designed to process ore at a rate of 6,460 t/d to produce gold/silver doré bullion.

The process plant flowsheet designs were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

25.9 Infrastructure

There are several existing buildings at within the property ate Goliath site at different conditions. It is recommended to conduct structural assessment of these buildings and identify any requirement for repair and upgrade for utilization as part of mining operation.

Several potential rock quarries have been identified at south Goliath mine site. It is recommended to conduct further investigations to quantify the type and available volume of material for mine development use.

25.10 Environmental, Permitting and Social Considerations

The environmental baseline programs are well advanced and combine the studies completed for the EA process and ongoing program to ensure data is complete and up to date. The site-based information will be used to support federal and provincial permitting processes and provide a basis of knowledge from which to develop environmental management plans for all phases of the project(s).

The EA Decision Statement effluent criteria are quite stringent, and the Goliath project must meet PWQO or background criteria protective of aquatic life. The design and selected treatment circuits for the ETP will be critical in achieving specific and non-specific criteria. Given the stringent water quality criteria, there is both technical and non-technical risk associated with the Goliath project's selection of discharge location and the capability of the ETP system. Discharge to Blackwater Creek may present challenges related to flow and timing of discharge while a direct pipeline to Wabigoon Lake via Keplyn Bay could present routing challenges and social acceptance risk as noted during the federal EA process. Additional modelling and a geomorphological survey of Blackwater Creek are required to understand the receiving capacity of Blackwater Creek, along with modelling to assess the assimilative capacity of Wabigoon Lake.

Notification to IAAC and the applicable Indigenous communities of the proposed design updates to the Goliath project will need to be communicated during the feasibility study design stage to allow sufficient time for appropriate consultation and incorporation into permit application submissions.

25.11 Markets and Contracts

Gold Production is expected to be sold on the spot market. Terms and conditions included as part of sales contracts are expected to be typical of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold is bought and sold, and it is difficult to obtain a market price at any particular time. The gold market is very liquid with a large number of buyers and sellers willing and active at any time.

25.12 Capital Cost Estimates

25.12.1 Open Pit and Underground Mining Processes

SRK completed the estimation of capital costs as relates to the open pit and underground mining processes only. The open pit costs are based on the assumption of owner-operation, while the underground costs assume that mining will primarily be conducted by a contractor(s). The cost estimate is in C$ and has been developed by SRK based on quotes obtained from local manufacturers and suppliers and SRK's internal cost database. The mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM).

25.12.2 Process Plant and Infrastructure

Ausenco developed the capital cost estimate relating to the direct costs of the process plant, Goliath on-site infrastructure, off site infrastructure. The project indirect costs and contingency were also estimated by Ausenco. The capital cost estimate is in C$, sourced from vendor and contractor quotes as well as Ausenco's previous project experience.

The total capital cost for the Goliath Complex Project as identified by Ausenco is C$551.7 million. Initial capital costs are C$335 million, and life-of-mine sustaining costs are C$216.6 million, which includes $28.9 million of closure costs and $9.9 million of salvage value. As a total, 80% of total capital costs were derived from the first principles, with equipment quotation or contract supply/installation.

25.13 Operating Cost Estimates

25.13.1 Open Pit and Underground Mining

SRK completed the estimation of operating costs as relates to the open pit and underground mining processes only. The open pit costs are based on the assumption of owner-operation, while the underground costs assume that mining will primarily be conducted by a contractor(s). The cost estimate is in C$ and has been developed by SRK based on quotes obtained from local manufacturers and suppliers and SRK's internal cost database. The mining costs do not include any contingencies and are exclusive of engineering, procurement, and contract management (EPCM).

25.13.2 Process Plant and Infrastructure

Ausenco derived the process plant and general and administrative (G&A) operating costs. These costs include those relating to process plant power usage, reagent and consumables consumption, laboratory assays, mobile vehicles, process plant labour, equipment maintenance, and the effluent treatment plant.

The overall LOM operating cost for the Goliath Complex Project as identified by Ausenco is C$1,447M over 13 years, or an average of C$47.71M/t ore milled in a typical year. Of this total, processing and G&A account for C$451M and mining accounts for C$995M.

25.14 Economic Analysis

An engineering economic model was developed to estimate the project's annual pre-tax and post-tax flows and sensitivities based on an 5% discount rate.

The economic analysis also used the following assumptions:

- the mine life is 13 years
- cost estimates are in constant Q3 2022 Canadian dollars with no inflation or escalation factors considered
- capital costs are funded with 100% equity (no financing assumed)
- all cash flows are discounted to the start of the construction period using a mid-period discounting convention

The pre-tax NPV discounted at 5% is $469 million; the IRR is 29.3%, and payback period is 2.8 years. On a post-tax basis, the NPV discounted at 5% is $336 million; the IRR is 25.4%; and the payback period is 2.8 years.

A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using the following variables: gold price, foreign exchange, total operation cost, initial capital cost. The sensitivity analysis revealed that the project is most sensitive to changes in gold price and foreign exchange, and less sensitive to discount rate, total operating cost, and initial capital cost.

25.15 Risks and Opportunities

Risk identification and mitigation was ongoing throughout the prefeasibility study, and will continue through value/detailed engineering, construction, operations and closure. Risks were identified and qualitatively ranked in the Project Risk Register. As the project moves from prefeasibility into the feasibility, it will be necessary to update the project risk register.

25.15.1 Process Plant

The following list provides some of the main risks associated with the process plant design:

1. Multiple deposits – The Goliath Gold Complex is comprised of two major deposits (Goliath and Goldlund) and one minor deposit (Miller). As the ore characteristics and recovery behaviour of the two deposits are different, assumptions were made while designing the process plant provide allowance for possible changes in ore hardness and leach conditions. The characteristics of the blended feed could perform differently than expected, putting the comminution circuit design at risk.
2. Testwork – The metallurgical testwork program associated with the design of the leach-adsorption circuit was conducted primarily at grind sizes of k_{80}=100 µm for the Goliath samples and k_{80}=90 µm for the Goldlund samples. Goldlund samples from the West Pit demonstrated strong correlations between the leach grind size and the cyanidation residue grade. As the optimal grind size selected for the Goldlund samples was k_{80}=85 µm, there is a risk associated with the leach circuit reagent addition rates selected and the leach recovery models.

25.15.2 Mineral Resource Estimate

Mineral resources are estimated based on drillhole sampling points and grades are interpolated between sampled points. Any interpolation is subject to basic assumptions of continuity of grade and geology between sampled points. As such, the mineral resources are subject to some risks. Conversely, opportunities to expand the mineral resources in area of assumed geological continuity where grade has not been confirmed because of insufficient sampling or drilling also exist.

25.15.2.1 Goliath

For the Goliath deposits, the QP has identified the following risks:

- The high-grade zones are very narrow, they typically pinch and swell along strike and dip.
- Zones C, D, E and H zones could be less continuous than modelled.
- The high-grade zones may not be as linear as modelled which could increase dilution in the underground mining phase.

The Goliath deposit also offers the following opportunities:

- The Main and C zones remain open at depth and to the East (Goliath East).
- The narrow hanging and footwall zones (D, E and H) could provide additional resource in the pit. These ones were generally not sampled in the historical (Teck) drilling and are therefore underrepresented in the assay database. The zones are probably larger then currently modelled.
- Additional drilling at Goliath East will help better define the mineralized zones and convert the inferred mineral resource to indicated category.

25.15.2.2 Goldlund

For the Goldlund deposit, the QP has identified the following risks:

- The mineralization is hosted in stockwork that can't be easily modelled. This may result in over-estimation of grade continuity.
- The model includes historical drillholes (1940 to 1970) that have not been fully validated. The historical drill data are not easily validated. The QP limited their influence to minimize their impact on the global resource estimate.
- It is unclear if the underground workings fully represent the extend of historical mining. Extensive underground working seems to support only a small volume of stoping, there is a risk that the historical mining was more extensive than is currently modelled. However, the modelled stopes generate tonnages that are close to the historical production records suggesting that this risk may be minimal.
- Because of the narrow nature of stockwork mineralization, underground mining may encounter high dilution.

The Goldlund deposit does offer some opportunities for expansion:

- The mineralization remains open at depth.
- The high-grade core present in Zone 1 could be expanded with additional drilling, both along strike and down dip.

25.15.2.3 Miller

For the Miller deposit, the QP has identified the following risks:

- The mineralization at Miller is hosted in stockwork that cannot be easily modelled. This could result in an over-estimation of grade continuity between drillholes.
- The model is supported with wider spaced drilling than Goliath and Goldlund which increases the risk of over-estimation of the grade continuity.
- Dilution could be greater than expected.

The following opportunities have been identified for the Miller deposit:

- The deposit remains open for expansion at depth and to the southwest.
- Additional drilling could expand and find similar parallel zones to the north of the current deposit.

25.15.3 Mineral Reserves & Mine Planning

SRK has identified the following risks associated with the Mineral Reserves and mine planning:

- ROM transportation from Goldlund to the Goliath processing facility was changed after the pit optimizations were completed. The ROM transportation costs increased, however the pit optimizations and cut-off grades were not re-run or re-calculated. There is a risk that updating the transportation cost could result in a smaller inventory for Goldlund. This will be evaluated in the next phase of the study.
- Further geotechnical field investigation and analysis will better inform the assumptions made during this study. This may result in additional constraints regarding underground stope size, extraction sequence, and size of the crown pillar under the Goliath Pit.

- The pastefill design will require further testing prior to the next stage of study. The data acquired during this analysis may result in additional costs and/or mine scheduling implications.
- Both capital and operating costs have been affected by inflationary trends worldwide over the past several years. While this pressure appears to be lessening, the costs estimated in this study may be further impacted over the next few years.

SRK has identified the following opportunities associated with the Mineral Reserves and mine planning:

- The Goliath pit selection was based on limited waste storage capacity on surface. The removal of this constraint could increase the size of the pit and inventory.
- The pit optimization slope angles for all deposits were based on larger trucks than were ultimately selected for the study. A re-optimization of the deposits with steeper slopes could increase pit inventory.
- While the underground mine setting does not require the use of electric mining vehicles (either BEVs or tethered) to mitigate temperature or airflow volume concerns, further study is suggested to at the Value Engineering stage to determine if carbon taxes can be offset through the incorporation of more electric vehicles than are currently planned. While more difficult to quantify, there may be additional benefits from an ESG perspective be removing a greater amount of diesel equipment from the underground environment (e.g., long-term employee health, employee recruitment and retention, etc.).
- Consider eliminating longer auxiliary ventilated headings by developing drop raises with ramp development.
- Determine the final locations of fixed facilities (magazines, etc.) and determine best ventilation strategies for them.

25.16 Conclusions

Proven and probable mineral reserves for the Goliath Gold Complex project are estimated at 30.3 million tonnes at an average grade of 1.3 g/t Au for 1.3 million ounces of contained gold combined open pit and underground. The previous sections were provided with collaboration of qualified and approved consulting firms as indicated in this report which provided mineral resource and mineral reserve estimates, design parameters and cost estimates for mine operations, process facilities, waste and tailings storage, permitting, reclamation, equipment selection and operating and capital expenditures.

26 RECOMMENDATIONS

26.1 Overall

The following sections detail the recommended future work for the project. Estimated costs for the work are summarized in Table 26-1

Table 26-1: Estimated Cost of Work Program

Area	Estimated Cost (C$K)
Metallurgical Testwork (Exclusive of Drilling) and Recovery Methods	275
Water Management	620
Tailings Management	400
Geochemical Assessment	1,200

26.2 Metallurgy and Processing

26.2.1 Metallurgical Testwork

Additional comminution tests (e.g., SMC, Bond ball work index, and abrasion index) are recommended on samples from both the Goliath and Goldlund deposits over a range of lithologies or zones to minimize risk in the crushing and grinding circuit design.

Further leaching testwork should be conducted at a grind size of 105 um for Goliath and 85 um for Goldlund to confirm the findings of this report which were interpolated over a range of grind sizes. Tests for the Goldlund deposit should be run at both telluride and conventional cyanide leaching conditions and include measurement of the deportment of gold within telluride bearing minerals. Oxygen uptake testing should also be completed these samples to confirm the oxygen consumption requirements in leaching across the deposit.

The feasibility study metallurgical testwork program should also include additional point samples representing a variety of mineralogies in the two primary deposits to understand the recovery behaviour. It is recommended that these samples undergo gravity-leach testwork to investigate reagent addition and recovery behaviour, and that the cyanidation tailings should complete vendor thickener tests to ensure accurate equipment sizing. Specific attention should be given to understanding the metallurgical responses within both Zones 1 and 4.

26.2.2 Recovery Methods

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

- Material handling testing to support process design. The results and recommendations should be incorporated into the crushing and stockpile circuit detailed design.
- conduct additional comminution testing on additional variability samples from Goliath and Goldlund to better understand hardness variability and minimize throughput risk in the crushing and grinding circuit designs.
- Conduct validation testing at the selected grind size to confirm the interpretation used for process design and recovery estimates.
- Additional downstream testwork on telluride-bearing zones and lithologies to understand recovery behaviour at various operating conditions.
- The cost of these items is covered under metallurgical testwork, as a result zero estimation is considered for this section.

During the next phase of study, additional process design work should be performed to produce capital and operating cost estimates with an accuracy of ±15% (AACE International Class 4).

26.3 Water Management

The following work related to water balance and water management is recommended for completion during the feasibility-level study:

- Identify specific effluent water treatment requirements and solutions based on the ongoing geochemical characterization and the results of the water quality model to be updated with the new geochemical data.
- Conduct groundwater numerical modelling to inform pit groundwater inflow rates through the operating and closure phases based on modelling predictions. The model will determine the transient inflow rate during both the initial stages of dewatering and as the pit depth is increased.
- Conduct a better definition of underground mine dewatering rates.
- Develop a stochastic flow (water balance) model to simulate the water management strategy with greater detail incorporating decision making functions and considering probabilistic climatic scenarios.
- Carry out a geomorphology assessment for Blackwater Creek to define the allowable effluent discharge flow and/or period to release treated water to the creek.
- Identify potential alternative discharge locations via a piped outlet to Wabigoon Lake and carry out assimilative capacity assessment downstream of the potential discharge location.
- Prepare water quality estimates for the mine closure phase including the open pit and other key mine features.

26.4 Tailings Management

Work that will be required to advance the design includes the following:

- Hydrogeological investigations and dam seepage modelling to ensure compatibility between the dam seepage control measures and the assimilative capacity of the receivers.
- Tailings rheology and geotechnical characterization.
- Geotechnical and geophysical investigations and analyses to confirm the natural clay continuity, constructability of the geomembrane anchor trench, and potential for seismic liquefaction of the outwash sand dam foundations.

26.5 Geochemistry

The following recommendations are provided to advance the Project ML/ARD assessment

- Continue to advance ongoing geochemical studies for the Goliath, Goldlund and Miller projects to refine currently available data regarding the ML/ARD potential of mine materials and associated mine waste / water management needs for operations and closure. This should include laboratory testwork, field testwork, and geochemical modelling.
- Undertake further assessment of the potential risk of producing a PAG and/or metal leaching tailings with proposed ore feed profiles.
- Conduct additional tailings geochemical assessment to evaluate the proposed deposition strategy to mitigate ML/ARD risks during mine operations and the potential performance of closure cover concepts to maintain long term geochemical stability of the tailings.
- Evaluate geochemically suitable (i.e., non-potentially acid generating and non-metal leaching) sources of rockfill and borrow materials for use in TSF construction.
- Advance water quality estimates including updates to the Goliath water quality estimates when additional geochemical data are available and prepare water quality estimates for Goldlund and Miller projects.

A summary of estimated indicative costs associated with the above recommendations is provided in Table 26-1 as a sum for all properties. Additional geochemical assessment, beyond what is outlined here, may be required pending the outcome of these recommendations.

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