Lemhi Gold Project

NI 43-101 Technical Report and Preliminary Economic Assessment

Idaho, United States of America

Effective Date: October 13, 2023

Report Date: November 20, 2023

Prepared for:

Freeman Gold Corp.
1600-595 Burrard Street
Vancouver, BC, Canada, V7X 1L4

Ausenco Engineering Canada ULC.
1050 W Pender Street, Suite 1200
Vancouver, BC, Canada, V7Y 1G5

List of Qualified Persons:

Kevin Murray, P. Eng., Ausenco Engineering Canada ULC.
Scott C. Elfen, P. E, Ausenco Sustainability ULC.
Peter Mehrfert, P. Eng., Ausenco Engineering Canada ULC.
James Millard, P. Geo., Ausenco Sustainability ULC.
Jonathan Cooper, P. Eng., Ausenco Sustainability ULC.
Marc Schulte, P. Eng., Moose Mountain Technical Services.
Michael Dufresne, P. Geo, APEX Geoscience Ltd.

CERTIFICATE OF QUALIFIED PERSON

Kevin Murray, P.Eng.

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

1. I am a Professional Engineer, currently employed as Manager Process Engineering with Ausenco Engineering Canada ULC. (“Ausenco”), with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC Canada, V6E 3S7.
2. This certificate applies to the technical report titled, “*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment*,” (the “Technical Report”), prepared for Freeman Gold Corp. (the “Company”), with an effective date of October 13, 2023 (the “Effective Date”) and a report date of November 20 2023.
3. I graduated from the University of New Brunswick, Fredericton NB, in 1995 with a Bachelor of Science in Chemical Engineering.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Province of British Columbia, Registration number# 32350 and the Northwest Territories Association of Professional Engineers and Geoscientists’ Registration# L4940.
5. I have practiced my profession continuously for 22 years. I have been directly involved in all levels of engineering studies from preliminary economic analysis (PEA) to feasibility studies including being a Qualified Person for different projects including Fortune Bay’s Goldfields Gold Project PEA, SilverCrest Metals’ Las Chispas Operations technical report, and Skeena Resources Ltd’s Eskay Creek Feasibility Study. I have been directly involved with test work and flowsheet development from preliminary testing through to detailed design and construction with Teck and have direct operations support experience at Red Lake Gold Mine, Porcupine Gold Mine and Éléonore Gold mine as well has commissioning support for the Magino Gold mine. I have also developed operating cost estimate and contributed to and reviewed capital cost estimates and financial models.
6. I have read the definition of “Qualified Person” set out in National Instrument 43-101 - Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for the purposes of NI 43-101 in connection with those sections of the technical report that I am responsible for preparing.
7. I have not visited the Lemhi Gold Project site.
8. I am responsible for Sections 1.1, 1.15, 1.16.1-1.16.8, 1.17, 1.19-1.22, 2, 3.3, 17, 18.1-18.3, 19, 21.1, 21.2.1, 21.2.2, 21.2.4-21.2.9, 21.2.10.1, 21.2.10.3, 21.2.11, 21.3.1, 21.3.3, 21.3.4, 22, 23, 24 , 25.1, 25.8, 25.9, 25.11, 25.13 - 25.15, 25.16.4, 25.16.5.1, 25.16.7, 25.17.3, 26.1, 26.5, and 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no previous involvement with the Lemhi Gold Project.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023

“Signed and Sealed”

Kevin Murray, P. Eng.

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

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

1. I am employed as the Global Lead Geotechnical and Civil Services within Ausenco Engineering Canada ULC., with an office address of 1050 West Pender Street, Suite 1200, Vancouver, BC V6E 3S7, Canada.

2. This certificate applies to the technical report titled, "*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment*" (the "Technical Report"), prepared for Freeman Gold Corp. (the "Company"), with an effective date of October 13, 2023 (the "Effective Date") and a report date of November 20, 2023.

3. I graduated from the University of California, Davis, California, in 1991 with Bachelor of Science degree in Civil Engineering (Geotechnical).

4. I am a Registered Civil Engineer in the State of California (license no. C056527) by exam since 1996 and I am also a member in good standing of the American Society of Civil Engineers (ASCE), and the Society for Mining, Metallurgy & Exploration (SME).

5. I have practiced my profession continuously for 26 years with experience in the development, design, construction, and operations of mine waste storage facilities, such as waste rock storage facilities and tailings storage facilities ranging from slurry to dry stack facilities, focusing on precious and base metals, both domestic and international. In addition, I have developed geotechnical design parameters for pit slope design, plant foundation design, and other supporting infrastructure. Examples of projects I have worked on include: Skeena's Eskay Creek Project PEA, PFS and FS, O3 Mining's Marban Project PEA and PFS, First Mining Gold's Springpole PEA and PFS. SSR Mining's Puna Silver In-Pit Tailings Disposal PFS, and Detailing Engineering, and the Company's Cangrejos Project PEA.

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

7. I have visited the Lemhi Gold Project on November 3, 2022, for visit duration of 1 day.

8. I am responsible for Sections 1.16.9, 18.4, 25.10, 25.16.5.2, 26.6, and 27 of the technical report.

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

10. I have had no previous involvement with Lemhi Gold Project.

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

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

Dated: November 20, 2023

"Signed and sealed"

Scott C. Elfen, P.E.

CERTIFICATE OF QUALIFIED PERSON
Peter Mehrfert, P. Eng.

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

1. I am employed as a Process Engineer with Ausenco Engineering Canada ULC., with an office at 1050 W Pender St, Vancouver, BC V6E 3S7.
2. This certificate applies to the technical report titled *"Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment"* (the "Technical Report"), prepared for Freeman Gold Corp. (the "Company") with an effective date of October 13, 2023 (the "Effective Date") and a report date of November 20, 2023.
3. I graduated from the University of British Columbia in 1996 where I obtained a Bachelor of Applied Science in Mining and Mineral Process Engineering.
4. I am a Professional Engineer, registered with Engineers and Geosciences of British Columbia, member number 100283.
5. I have practiced my profession for 28 years and have been involved in the design, evaluation and operation of mineral processing facilities during that time. Approximately half of my professional practice has been the supervision and management of metallurgical test work related to feasibility and prefeasibility and PEA studies projects. Previous projects that I have worked on include Blackwater, 15 Mile Stream, Cochrane Hill, Eskay Creek, and Courageous Lake.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Lemhi Project site.
8. I am responsible for sections 1.12, 13, 25.5, 25.16.1, 25.17.1, 26.3, and 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no involvement with the Lemhi Project prior to this study.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023.

"Signed and sealed"

Peter Mehrfert, P. Eng.

CERTIFICATE OF QUALIFIED PERSON
James Millard, P. Geo.

I, James Millard, P. Geo., certify that:

1. I am employed as a Director, Strategic Projects with Ausenco Sustainability ULC., a wholly owned subsidiary of Ausenco Engineering Canada ("Ausenco"), with an office address of Suite 100, 2 Ralston Avenue, Dartmouth, NS, B3B 1H7, Canada.
2. This certificate applies to the technical report titled "*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment,*" (the "Technical Report"), prepared for Freeman Gold Corp. (the "Company"), with an effective date of October 13, 2023 (the "Effective Date") and a report date of November 20, 2023.
3. I graduated from Brock University in St. Catharines, Ontario in 1986 with a Bachelor of Science in Geological Sciences, and from Queen's University in Kingston, Ontario in 1995 with a Master of Science in Environmental Engineering.
4. I am a member (P. Geo.) of the Association of Professional Geoscientists of Nova Scotia, Membership No. 021.
5. I have practiced my profession for 25 years. I have worked for mid- and large-size mining companies where I have acted in senior technical and management roles, in senior environmental consulting roles, and provided advice and/or expertise in a number of key subject areas. These key areas included: feasibility-level study reviews; NI 43-101 report writing and review; due diligence review of environmental, social, and governance areas for proposed mining operations and acquisitions, and directing environmental impact assessments and permitting applications to support construction, operations, and closure of mining projects. In addition to the above, I have been responsible for conducting baseline data assessments, surface and groundwater quantity and quality studies, mine rock geochemistry and water quality predictions, mine reclamation and closure plan development, and community stakeholder and Indigenous peoples' engagement initiatives. Recently, I acted in the following project roles: Qualified Person for the environmental/sustainability aspects for "Puquios Project, Feasibility Study Report, La Higuera, Coquimbo Region, Chile", "Volcan Project, NI 43-101 Technical Report on Preliminary Economic Assessment, Tierra Amarilla, Atacama Region, Chile" and, "Colomac Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment, Northwest Territories, Canada"; and principal author for the environmental/sustainability sections for the "Kwanika-Stardust Project, NI 43-101 Technical Report and, Preliminary Economic Assessment, British Columbia, Canada".
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Lemhi Gold Project.
8. I am responsible for sections 1.18, 3.2, 4.3, 20, 25.12, 25.16.8, 25.17.4, 26.7, and 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101
10. I have had no previous involvement with the Lemhi Gold Project.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023

"Signed and sealed"

James Millard, P. Geo.

CERTIFICATE OF QUALIFIED PERSON
Jonathan Cooper, M.Sc., P.Eng.

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

1. I am employed as a Water Resources Engineer with Ausenco Sustainability ULC., with an office address of 11 King Street West, Suite 1500, Toronto, Ontario M5H 4C7.
2. This certificate applies to the technical report titled, “*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment,*” (the “Technical Report”), prepared for Freeman Gold Corp. (the “Company”), with an effective date of October 13, 2023 (the “Effective Date”) and a report date of November 20, 2023.
3. I graduated from the University of Western Ontario with a Bachelor of Engineering Science in Civil Engineering in 2008, and University of Edinburgh with a Master of Environmental Management in 2010.
4. I am a Professional Engineer registered and in good standing with Engineers and Geoscientists British Columbia (EGBC), registration number 37864.
5. I have practiced my profession for continuously for over 15 years with experience in the development, design, operation, and commissioning of surface water infrastructure. Previous projects that I have worked on that have similar features to the Lemhi Gold Project are Kwanika-Stardust for NorthWest Copper located in British Columbia, Colomac Gold Project located in the Northwest Territories, KSM for Seabridge Gold located in British Columbia and Borden Advanced Exploration for Goldcorp, located in Ontario.
6. I have read the definition of “Qualified Person” set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a “Qualified Person” for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Lemhi Gold Project site.
8. I am responsible for section 1.16.10, 18.5, 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no previous involvement with the Lemhi Gold Project.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023

“Signed and sealed”

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

CERTIFICATE OF QUALIFIED PERSON
Marc Schulte, P.Eng.

I, Marc Schulte, P.Eng., certify that:

1. I am employed as Mining Engineer with Moose Mountain Technical Services, with an office address of #210 1510 2nd Street North Cranbrook, BC V1C 3L2.
2. This certificate applies to the technical report titled "*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment,*" (the "Technical Report"), prepared for Freeman Gold Corp. (the "Company"), with an effective date of October 13, 2023 (the "Effective Date") and a report date of November 20, 2023.
3. I graduated Bachelor of Science in Mining Engineering from the University of Alberta in 2002.
4. I am a member of the self-regulating Association of Professional Engineers, Geologist and Geophysicists of Alberta Registration number #71051.
5. I have practiced my profession for 21 years since graduation. Throughout my career I have worked on numerous open pit precious metals projects, within project engineering studies and within mine operations, on Mineral Reserve estimates, mine planning, and mine cost estimates.
6. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the technical report that I am responsible for preparing.
7. I have not visited the Lemhi Gold Project.
8. I am responsible for Sections 1.14, 15, 16, 21.2.3, 21.2.10.2, 21.3.2, 25.7, 25.16.3, 26.4, and 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had no previous involvement with Lemhi Gold Project.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the effective date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023.

"Signed and sealed"

Marc Schulte, P.Eng.

CERTIFICATE OF QUALIFIED PERSON
Michael Dufresne, M.Sc., P.Geol., P.Geo.

I, Michael Dufresne, M.Sc., P.Geol., P.Geo., do hereby certify that:

1. I am a Professional Geologist, currently employed as a Principial Geologist with APEX Geoscience Ltd. ("APEX"), with an office address of 100, 11450 – 160 Street NW, Edmonton, Alberta Canada T5M 3Y7.
2. This certificate applies to the technical report titled, "*Lemhi Gold Project, NI 43-101 Technical Report and Preliminary Economic Assessment*," (the "Technical Report"), prepared for Freeman Gold Corp. (the "Company") with an effective date of October 13, 2023 (the "Effective Date") and a report date of November 20, 2023.
3. I graduated from the University of North Carolina Wilmington, NC in 1983 with a Bachelor of Science in Geology and from the University of Alberta, Edmonton, Alberta in 1987 with a Master of Science in Economic Geology.
4. I am a member in good standing of the Association of Professional Engineers and Geoscientists of the Provinces of Alberta, British Columbia and New Brunswick, Registration numbers # 48439, # 37074 and F6534, respectively and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists' Registration# L3378.
5. I have practiced my profession continuously for more than 35 years. I have been directly involved in mineral resource estimation and geology from exploration through to feasibility studies and mining including being a Qualified Person for numerous studies and reports on structurally controlled hydrothermal sediment hosted gold deposits over the last 20 years. Similar projects I have recently worked on and completed resource reviews or new resources for basement (Paleozoic to Proterozoic) sediment hosted hydrothermal precious metal deposits include Getchell Gold' Fondaway Canyon Project, Scorpio's Mineral Ridge Mine, Augusta Gold's Reward Project, Waterton's Converse and Mt Hamilton Projects, Provenance Gold's White Rock Project, Lode Metals Cracker Creek Project, and a private company's Basin Gulch Project in Montana.
6. I have read the definition of "Qualified Person" set out in National Instrument 43-101 – Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for the purposes of NI 43-101 in connection with those sections of the technical report that I am responsible for preparing.
7. I have visited the Lemhi Gold Project on a number of occasions over the last 5 years, with the most recent field visit on February 18, 2022.
8. I am responsible for Sections 1.2-1.11, 1.13, 3.1, 4.1, 4.2, 5-12, 14, 25.2-25.4, 25.6, 25.16.2, 25.17.2, 26.2, and 27 of the technical report.
9. I am independent of the Company as independence is described by Section 1.5 of NI 43-101 and in Section 1.5 of the Companion Policy to NI 43-101.
10. I have had previous involvement with the Lemhi Gold Project authoring or co-authoring technical reports in 2020 and 2021.
11. I have read NI 43-101, Form 43-101F1 and the sections of the technical report for which I am responsible have been prepared in compliance with NI 43-101, Form 43-101F1.
12. As of the Effective Date of the technical report, to the best of my knowledge, information and belief, the sections of the technical report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the technical report not misleading.

Dated: November 20, 2023

"Signed and sealed"

Michael Dufresne, M.Sc., P.Geol., P.Geo.

Important Notice

This report was prepared as National Instrument 43-101 Technical Report for Freeman Gold Corp. (Freeman Gold), by Ausenco Engineering Canada ULC. and Ausenco Sustainability ULC. (Collectively "Ausenco"), Moose Mountain Technical Services (MMTS), and APEX Geoscience Inc. (APEX), 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 Freeman Gold subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposes legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Freeman Gold Corp. (Freeman Gold) commissioned Ausenco Engineering Canada ULC. to compile a preliminary economic assessment (PEA) of the Lemhi Gold Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco Engineering Canada ULC. and Ausenco Sustainability ULC. (collectively, "Ausenco") managed and coordinated the work related to the report. Ausenco developed the PEA-level design and cost estimate for the process plant, review of the metallurgical test program, general site infrastructure, site water management infrastructure, tailings facility and environmental studies and permitting. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- Moose Mountain Technical Services (MMTS) designed the open pit, the mine production schedule, and mine capital and operating cost estimates.
- APEX Geoscience Inc. (APEX) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, Terms of Reference.

1.2 Property Description and Location

The Lemhi Gold Project (project or property), including 11 patented and 286 unpatented claims that are 100% owned, along with 46 unpatented claims that are under option by Freeman Gold, is located in Lemhi County, Idaho (ID), USA, within the Salmon River Mountains, a part of the Bitterroot Range that forms the Idaho-Montana border. The property is 40 km north of the town of Salmon and 6 km west of Gibbonsville, ID, USA. The project is comprised of ten patented mining claims (placer and lode), one patented mill site claim and 332 unpatented mining claims, totalling 2,727 hectares of mineral rights and 249 hectares of surface rights. Freeman Gold controls a 100% interest in all 11 patented claims and all 332 unpatented mining claims outright or through its wholly owned subsidiary company Lower 48 Resources (Idaho) LLC (Lower 48) subject to certain cash payments over time and royalties.

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

A total of 11 patented claims and 53 unpatented mining claims were purchased from Lemhi Gold Trust (LGT) by Freeman's subsidiary Lower 48 Resources (Idaho) LLC (Lower 48) through a closed auction bid process in November 2019. Freeman has also optioned 46 unpatented mining claims that are owned by BHLK2, LLC (BHLK). Freeman recently purchased outright the Moon #100 and Moon #101 unpatented mining claims (Moon Claims) from Vineyard Gulch Resources, LLC (Vineyard), located within the historical resource area. An additional 231 unpatented claims were staked by Freeman (Lower 48) in 2020 and 2021. Freeman closed a transaction to acquire Lower 48 and its parent Company 1132144 British Columbia (B.C.) Ltd. (1132144) on April 16, 2020, which resulted in the transfer of the 46 unpatented BHLK mining claims under option and the subsequent extinguish of 1132144. BHLK retains a 2% net smelter return (NSR) royalty on production from certain claims within the Lemhi Gold Project including the 11 patented Lower 48 claims and most of the unpatented claims within an area of interest.

The patented mining claims originated as unpatented mining claims and were converted to private ownership through the Patent and Mineral Survey process. The patented claims on the Lemhi Gold Property were patented between 1890 and 1910. Corner survey monuments are intact (with several observed by the author) and the United States Forest Service (USFS) has placed markers delineating USFS land boundaries along the claim boundaries. In order to keep the claims in good standing annual real estate taxes must be paid to Lemhi County. If the annual taxes are paid the patented claims will remain in good standing in perpetuity.

The 332 unpatented Bureau of Land Management (BLM) federal lode mining claims are administered by the USFS. The claims are ultimately owned by two entities (Freeman/Lower 48 and BHLK):

- 46 unpatented claims staked by BHLK of Missoula, Montana in 2011 and 2017
- 53 claims staked by LGT in September 2019, and purchased by Lower 48
- 223 claims staked by Lower 48 in April 2020 and 8 claims staked by Lower 48 in April 2021
- Two claims (the Moon Claims) purchased by Lower 48 from Vineyard in 2020.

Any portion of an unpatented claim overlapping a patented claim is deemed invalid; the valid portion of all unpatented claims totals 2,479 ha.

The Mining Law of 1872 states that with respect to unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine mineral mining claims. Surface rights are not included and remain property of the United States government. No payment of production royalties to the Federal government is required. To maintain existing unpatented claims in good standing an annual maintenance fee of US$165 must be paid per claim to the BLM prior to September 1 of each year or the claims will be invalidated and will expire. New lode mining claims require a US$10 recording fee payable to the Country Courthouse of the relevant jurisdiction in which the claims are located. In addition, the BLM requires a further maintenance fee of US$165, a US$20 processing fee and a US$40 claim location fee. The total fee payable to BLM for recording a new claim is US$225 per claim. All 332 unpatented mineral claims were understood to be in good standing based on the information received from Freeman. The status of the claims was checked against the BLM MLRS register database on October 17, 2023, and they are confirmed to be in good standing.

BHLK obtained a 2% NSR royalty on all 11 patented mining claims and 74 surrounding unpatented mining claims through a deed of royalty upon LGT's purchase of the project in 2011. The deed of royalty details a 2-mile area of interest and is still active today. The 74 unpatented mining claims were optioned by LGT from BHLK in 2011 and cover the area currently represented by BHLK's 46 unpatented mining claims. Subsequently, 1132144 signed an option to purchase agreement on August 31, 2019 with BHLK for the 46 unpatented mining. Freeman may earn a 100% interest in the claims with cash payments totalling US$1.0M over seven years, at which time the BHLK 2% NSR will extend over most of the unpatented claims through the active deed of royalty. The Meridian group of patented mining claims consists of three placer and two lode patents; the Ditch Creek, Hamilton, Marysville, Canola, and Copperstain patented mining claims. Meridian Gold Inc. (Meridian) purchased the five patented claims from Ashanti Goldfields lnc. (Ashanti) in 1997. Ashanti (now AngloGold Ashanti Ltd.) retained a cash royalty of US$2.0 M, payable in full within 30 days after the first commercial production pour of doré gold or silver mined from any, or all, the 11 patented mining claims. At that time, the Proksch group of patents were under lease.

LGT purchased the Meridian group of five patented mining claims from Meridian (which is now a wholly owned subsidiary company of Yamana Gold Inc. (Yamana) in 2011) for a one-time payment of US$2.5 M. The purchase was subject to Ashanti's royalty and a 'back-in' whereby Meridian can 'back-in' to a 51% ownership of the Meridian group of five patented mining claims if and when the mineral reserve reaches 2.5 M mineable ounces of gold. This 'back-in' right was purchased outright by Freeman in September of 2020 for 4,035,273 shares. These patented claims were recently purchased by Lower 48 at auction. Real estate taxes paid to Lemhi County annually for the Meridian group of patented mining claims total US$406.46.

Freeman was recently granted a Permit to Appropriate Water (No, 75-15005), which allows for water rights for both potential future mining and domestic use in four sections within the company's patented mining claims. The permit allows the use of 0.54 m^3/s of water from ground water sources for future processing in a gold operation and 24600 L/day for domestic use. The permit was obtained from the Idaho Department of Water Rights (IDWR).

Freeman has also recently received an approval of a plan of operations (POO) application to the USDA-Forest Service (USFS), Salmon and Challis National Forests, North Fork Ranger District, submitted in September 2021. The plan was approved May 23, 2022, as POO-2021-081646 and allows for an expanded drill program with additional access on the unpatented BLM mineral claims. Freeman is currently permitted to draw water from a number of wells on the patented mineral claims for drilling.

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

The Lemhi Gold Project is located within the Gibbonsville mining district in Idaho, USA. Placer gold was first discovered in 1867 at Hughes Creek west of the town of Gibbonsville, followed by discoveries in the Dahlonega Creek and Andersen Creeks and the North Fork Salmon River. In 1877, gold-bearing quartz veins were discovered on the slopes of Dahlonega Creek (5 km east of the Lemhi Gold Project) and the mining of lode gold deposits ensued.

In the Ditch Creek area, overlapping the current Lemhi Gold Project, placer gold mining commenced in the 1890s. During this time several mining claims were located and patented. In 1891, a group of six lode claims (MS 784A: Beauty Lode, Fraction Load, Atlanta Lode, Ironstone Lode, Chamaleon Lode, Copperstain Lode), was consolidated as the Bull of the Woods Mine. These six patented claims are part of the current Lemhi Gold Property. Extensive placer mining has been conducted along most of Hughes Creek and many of its tributaries, such as Ditch Creek, which drains north to south through the middle of the Lemhi Project area. The placer and surface mining have been intermittently active in the area over a period of more than 100 years with extensive placer dredge tailings piles still visible today in Hughes Creek.

The project can be accessed by paved and gravel road from Salmon, Idaho by following US Highway 93 north for 34 km to North Fork and then an additional 7.4 km to the Hughes Creek Road (USFS Road 091). The property can be reached by traveling 3.2 km west along the Hughes Creek Road and another 3.1 km north along the Ditch Creek Road to a two-track road leading northwest to the Lemhi Gold Property. The Hughes Creek and Ditch Creek roads are public graded gravel roads maintained by USFS and/or Lemhi County and provide all-weather access to the project area.

Alternatively, the property can be accessed via the Granite Mountain Road (USFS Road 092), which heads west from Highway 93, 7.5 km north of Hughes Creek Road. The Granite Mountain Road follows Votler Creek westward, then wraps around the south side of Granite Mountain and drops into the Little Ditch Creek drainage to intersect the Ditch Creek Road near the north end of the Lemhi Gold Property, 8 km from Highway 93. This road could provide a good access route for heavy equipment, supplies, and personnel in the summer months, but in its present condition would be unacceptable for winter travel due to the high altitude and lack of adequate berms.

The project is located in the Salmon River Mountains within the Rocky Mountain physiographic province. The Bitterroot Range, which forms the border between Idaho and Montana lies to the east, across the Salmon River. The claims are centered on Ditch Creek, a south-draining tributary of Hughes Creek, which in turn flows into the North Fork of the Salmon River. The area is mountainous and characterized by steep slopes (30% to 100% grade) along Hughes Creek and Ditch Creek. Total relief is 500 m, with elevations ranging from 1,500 to 2,000 masl. Pine forest, consisting of ponderosa pine and Douglas fir with minor lodgepole pine, covers most of the project area. Mammals in the area include mule deer, elk, coyote, wolf, black bear, mountain lion, beaver, rabbits, and a variety of small rodents.

The climate is typical of the central Rocky Mountains. Summers (June-September) are generally warm with average daytime highs of 15°- 20° C. Summer nights are cool. Winter temperatures are cold with overnight lows often below -10° C. Annual precipitation is largely a function of elevation with Gibbonsville at 1366 masl receiving 34 cm, and Moose Creek at 1890 masl receiving 82, mostly as snow, between November and March (Carroll, 1996). Snowstorms are frequent, but access routes to the property can be kept open with minimal snow plowing.

The town of Salmon has a population of 3,300 and its economy is based on ranching, forestry, mining, and tourism. Salmon is home to the regional offices of the USFS, BLM, and Idaho Fish and Game (IDFG) as well as other state and federal agencies. Basic supplies are available, as are food and lodging. Steele Memorial hospital and medical clinic in Salmon provides basic medical needs, but the nearest hospital is in Dillon, Montana, 90 km north of Salmon. The Lemhi County airport, located 8 km south of town, handles regularly scheduled commuter flights to/from Idaho Falls and Boise as well as charter flights. Salmon has historically provided both skilled and unskilled labor for the mining industry.

The patented mining claims at the Lemhi Gold Project provide adequate area for mine infrastructure. The placer claims of MS 1120 contain 193 ha of gently sloping private land suitable for mine offices, leach pads, a processing plant, and waste dumps. There is no power or other mining infrastructure on the Lemhi Gold Property. A 35.5 power line passes through the settlement of North Fork, 16 km by road from the property. Sufficient water for exploration is available from Ditch Creek, which has good perennial water flow. Two of the patented mining claims carry water rights totaling 99 L/s. Water wells would have to be drilled to provide sufficient water for mining and a processing plant.

The Lemhi area has a rich history of exploration and metallic mineral mining. The region has the availability and sources of power, water, and mining personnel. The project can be accessed year-round. Most exploration activities associated with fieldwork and drilling can likely be conducted year-round, although there may be periods in December to March where snow conditions may temporarily impede fieldwork. The authors do not see any significant obstacles that would prevent the potential development of a mine on the Lemhi Gold Property.

1.5 History

The Lemhi Gold Project is located within the Gibbonsville mining district in Idaho, USA. Placer gold was first discovered in 1867 at Hughes Creek west of the town of Gibbonsville, followed by discoveries in the Dahlonega Creek and Andersen Creeks and the North Fork Salmon River. In 1877, gold-bearing quartz veins were discovered on the slopes of Dahlonega Creek, which is 5 km east of the Lemhi Gold Project, and the mining of lode gold deposits ensued.

In the Ditch Creek area, overlapping the current Lemhi Gold Project, placer gold mining commenced in the 1890s. During this time several mining claims were located and patented. In 1891, a group of six lode claims (MS 784A: Beauty Lode, Fraction Load, Atlanta Lode, Ironstone Lode, Chamaleon Lode, Copperstain Lode), was consolidated as the Bull of the Woods Mine. These six patented claims are part of the current Lemhi Gold Property. Extensive placer mining has been conducted along most of Hughes Creek and many of its tributaries, such as Ditch Creek, which drains north to south through the middle of the Lemhi Project area. The placer and surface mining have been intermittently active in the area over a period of more than 100 years with extensive placer dredge tailings piles still visible today in Hughes Creek.

Since the early 1900s, the Gibbonsville district has seen little modern exploration and mining activity until 1984, when FMC Gold Company (FMC) staked claims at Ditch Creek. After conducting regional grass-roots exploration programs in the area, FMC staked additional claims surrounding the Bull of the Woods property (patent claim: MS 784A). FMC leased and purchased some of the key patented claims and accumulated a land package of over 700 unpatented claims surrounding the patented mining claims in the area of the current Lemhi Gold Project. FMC also acquired the Beartrack property, located 48 km southwest of the Lemhi Gold Project.

FMC explored the property from 1984 until 1991 (known at the time as the Ditch Creek Project, later renamed the Ponderosa Project). FMC's Ponderosa Project largely overlapped the current Lemhi Gold Project and extended up to Allan Creek west of the current project boundary. During that period FMC completed 192 reverse-circulation (RC) drill holes and four core holes. American Gold Resources (AGR) acquired the property in 1991 and held it until 1996. AGR drilled a total of 156 RC holes and nine core holes during the period they held the property. After 1996, work on the property was limited due to numerous corporate takeovers and downturns in the mining sector.

In 2011, Lemhi Gold Trust (LGT), a joint venture between Idaho State Gold Company (ISGC) and Northern Vertex, acquired the newly consolidated Lemhi Gold Project and commenced an aggressive exploration program. The historical LGT Property included the Lemhi (Humbug) Gold Deposit. In 2011 and 2012, LGT commenced and completed an aggressive pre-development program consisting of historical data compilation and review, core and RC drilling, baseline environmental studies, and geotechnical work. Drilling included 7 m of core in 40 holes and 2,672 m in 15 holes of RC drilling. LGT also completed terrestrial vegetation and wetland delineation studies, a petrographic analysis and additional metallurgical work, and re-addressed cultural resources, fisheries, wildlife resources, water rights, and right-of-way.

Based on the 2012 core drilling, LGT proposed a new geologic model for the Lemhi Gold Deposit. The new interpretation indicated that the mineralization consists of a structurally controlled, hydrothermal deposit associated with varying amounts of sulfides in a quartz-carbonate gangue hosted by late-Proterozoic metasediments within the structurally complex Trans-Challis fault system. Gold mineralization was interpreted to have been introduced during a tectonically active period (early Tertiary) that is likely temporally and spatially related to intrusive activity associated with the Idaho Batholith. Mineralization is also spatially associated with a number of intruded sills, often spatially associated with the contact zones. Gold mineralization is strongly associated with base metal (copper (Cu) and molybdenum (Mo)) mineralization and occurs as multiple hydrothermal (epithermal – mesothermal) silica replaced structures resembling multiple flat-lying veins to stockwork zones.

Historical drilling has defined a fairly large area of gold mineralization measuring 650 m in an east-west direction by 500 m in a north-south direction with a typical thickness of 10 to 70 m, today known as the Lemhi Gold Deposit and historically known as the Humbug Gold Deposit. Historically, a total of 419 RC and core holes were drilled at the project, 11 of which have no collar information and are henceforth excluded from the following hole and assay tallies. Of the historical holes with collar information (n=408), 395 are within the current claim boundaries, with geology logs and assays complete for 387 of the on-claim holes. Anomalous gold (Au) mineralization (>0.15 g/t Au) has been intersected in 356 of the 408 historical drill holes, with the complete historical database totalling more than 70,000 m of drilling and including over 43,000 gold assays.

Most of the historical drilling (pre-2000) was completed using RC drilling methods. At the time, this approach was justified, however, as it became apparent that the Lemhi Gold Property lies in a very structurally complex area the lack of geological detail from RC chips hindered the development of an accurate geological model. The 2012 core drilling program with 40 core holes facilitated the collection of more detailed geological data and resulted in the development of a new deposit model for the property, as outlined above.

Several historical resource estimates have been constructed based on the historical RC drilling with more recent estimates incorporating the 2012 core and RC drilling. A wide range of results have been presented and are summarized in Section 6 below and are discussed in detail in Dufresne (2020). The authors are not treating these historical estimates as current mineral resources or mineral reserves as per the CIM Definition Standards for Mineral Resources & Mineral Reserves (2014) and the CIM Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019). These historical resources have all been superseded by the current MRE presented in this technical report, which includes 35 core holes drilled during 2020 and 71 holes drilled during 2021-2022. Eleven of the 71 2021-2022 hole were drilled at the Beauty satellite deposit.

Prior metallurgical studies, preliminary engineering studies, and initial baseline environmental studies all suggest that the Lemhi Gold Deposit had the potential to be developed as an open pit tank-leach, or combination tank-leach and heap-leach operation. Processing and operating costs provided in various economic studies from the 1990s are out-of-date and not currently applicable. A new cost analysis using current gold prices and updated processing, mining, and permitting costs will be necessary.

A number of baseline environmental, archaeological, and geotechnical studies were conducted on the project in the 1990's, as well as between 2011-2013. Several reports document a summary timeline and overview of permitting required for the development of a potential heap-leach operation and are summarized by Brewer (2019) and Cuffney (2011). Based on the initial baseline studies and preliminary permitting completed by AGR in 1995-96, subsequent baseline studies commissioned by LGT, and on public comments received during LGT's tenure, there does not appear to be any major obstacles that would prevent the potential development of a mine on the Lemhi Gold Property. It was concluded that there were no significant impediments identified to the potential development of an open pit mine, particularly on the patented mining claims.

1.6 Geology and Mineralization

The Lemhi Gold Project is located within the Cordilleran fold and thrust belt and more locally the Trans-Challis fault system. This broad 20-30 km-wide system of en-echelon northeast-trending structures extends from Idaho City, ID northeast to the Idaho-Montana border; over 270 km in strike length. It is one of many structures within the Idaho-Montana porphyry belt, a wide northeast-trending alignment of porphyry-related deposits, which parallels the contact between the Cordilleran fold and thrust belt and the Idaho batholith and corresponds to a zone of strike-slip faults, late graben faults and northeast-trending magnetic features.

Locally, the Lemhi Gold Property is largely underlain by Mesoproterozoic quartzites and phyllites with porphyritic dacite sills, dykes and flows of the Eocene Challis volcanics preserved in down-dropped fault blocks. Numerous faults crosscut the property forming grabens and half grabens. On the property, a large low-angle fault passes through Ditch Creek and is filled with Quaternary gravels covering part of the mineralization that comprises the Lemhi Gold Deposit. The mineralization on the property is hosted in structurally controlled quartz vein swarms and quartz flooded zones and occurs in close spatial association with low-angle faulting and several intrusive bodies.

Gold was discovered and mined from the area in the 1890s to mid-1900s. Modern exploration of the property area commenced in 1984. FMC conducted exploration over the current Property area between 1984 and 1991. FMC completed geologic mapping; rock, soil, and vegetation sampling; geophysical surveys; and RC and core drilling over the property. FMC defined an area of strong gold mineralization along the western slope of Ditch Creek. AGR acquired the property in 1991 and conducted exploration over the area until 1996. The FMC and AGR drilling delineated a gold deposit: the Humbug Deposit (now known as the Lemhi Gold Deposit), on the patented claims (MS 784 A and B, 2512 and 1120) which comprise the current Lemhi Gold Property. The Lemhi Gold Deposit is 1000 m east-west by 1100 m north-south. A prominent west-northwest-trending zone of higher-grade mineralization and a northeast-trending zone of strong mineralization were identified within the deposit. The mineralization is interpreted to be structurally controlled by northwest and northeast high-angle faults that intersect a low-angle (possible thrust) fault. In the footwall of an intrusion and along its western terminus the gold mineralization is thick, ranging from 30-70 m, and can occur in multiple stacked zones. In the hanging wall gold mineralization is considerably thinner and more erratic. In the core of the deposit, the low-grade envelope of mineralization is greater than 200 m thick.

1.7 Deposit Types

A mesothermal level of emplacement is evidenced by the following: 1) lack of open space filling, 2) crystalline quartz and lack of very fine-grained or chalcedonic quartz, 3) copper - molybdenum association, 4) coarse-grained sulfides, 5) associated bismuth, 6) low arsenic, antimony and mercury and 7) spatial association with the porphyritic intrusions"

The Lemhi Gold Deposit is localized within a major low-angle shear zone (possible thrust fault) and is spatially associated with a high-level porphyritic intrusion. Precious metals mineralization at Lemhi has historically been classified as shear-hosted intrusion related (porphyry-style) mineralization. Both FMC and AGR recognized this deposit type and used a porphyry-related model to guide their exploration program. Key elements of the exploration model were major structures (structural permeability); high-level intrusions (source of heat and fluids); alteration consisting of silicification and sericitization; and gold, copper, and molybdenum geochemical anomalies.

An alternate deposit model has been suggested consisting of a structurally controlled hydrothermal deposit with varying amounts of sulfides in a quartz-carbonate gangue hosted by late-Proterozoic metasediments within the structurally complex Trans-Challis fault system. It has been suggested that gold mineralization was introduced during a tectonically active period and was likely temporally related to intrusive activity associated with the Idaho Batholith. The observed gold mineralization is strongly associated with base metal (Cu and Mo) mineralization and occurs as multiple hydrothermal (epithermal – mesothermal) silica replaced structures resembling multiple flat-lying veins.

The gold deposit on the Lemhi Gold Property shares many similarities with the Beartrack mine, 35 km to the southwest, and the Musgrove deposit, 25 km further southwest. Both Beartrack and Musgrove are quartz stockworks hosted within major shear zones cutting the Apple Creek Formation.

Sericite believed to be a product of hydrothermal alteration yielded an age date of 65.5 +/-2.5 million years ago (Ma). The age date is close to that of the Beartrack deposit (68 Ma) and Napoleon Hill porphyry molybdenum deposit.

1.8 Exploration

The 2020-2022 exploration programs conducted by Freeman consisted of the following activities:

- Soil orientation survey (conventional B-horizon, partial extraction leach techniques such as ionic leach (IL) and mobile metal ion (MMI) sampling)
- Soil sampling
- Prospecting, rock, and chip sampling
- UAV (unmanned aerial vehicle)
- Geological mapping
- Ground magnetic survey
- 3D Induced polarization survey
- Core and RC drilling.

Until Freeman's 2020 program, no significant surface exploration other than drilling had been conducted on the property since the late 1980s. During Freeman's 2020 exploration program, modern soil geochemical techniques utilizing partial extraction methods, including MMI and IL, were implemented. In 2021, the results of this soil orientation program guided future exploration, utilizing IL, in under-explored areas with significant glacial or glacio-fluvial cover, such as areas west and north of the main deposit. In conjunction with the 2021 soil program, UAV studies, prospecting and geological mapping were conducted. Freeman's surface exploration is ongoing.

The Freeman 2020 exploration program also saw the entire claim group covered with a ground magnetic survey, additional staked claims in 2021 were covered with a ground magnetic survey during 2021. The core resource area was covered with a 3D Induce Polarization (IP) electromagnetic survey. The surveys and interpretations have been completed.

During the lead author's site visits, the author confirmed the locations of several historical collars on the property. Pulp re-assays of 2012 core drilling collected in 2019 (Dufresne, 2020) returned values which have close correlation with the original assays for the samples in question, confirming the validity of the 2012 assay results.

Based on the review of historical information and recent re-assay results, the authors consider the Lemhi Gold Property a property of significant merit, warranting further exploration and delineation work.

Drilling completed on the property in 2012 by LGT and in 2020-2022 by Freeman has returned encouraging results in both infill and step-out drilling. All 55 LGT holes and all 106 Freeman core and RC holes intersected gold mineralization. The new geological interpretation resulting from the data obtained from the core drilling has also identified additional potential exploration targets, including:

- Deep feeder zones
- Down-dip mineralization to the south
- Extensions of known mineralization to the west and southwest spatially associated with intrusions
- "Hidden" targets below the glacial cover immediately to the north of the known deposit.

Freeman's 2020 drilling program consisted of 7,149 m in 35 core holes of infill and step-out drilling. As part of the drilling program, Freeman commissioned a series of metallurgical studies to characterize the amenability of the mineralized material to certain recovery processes. During 2021-2022, Freeman completed another drilling program, including core and RC drilling, for a total of 15,351 m in 71 drill holes. Eleven of the 71 drill holes were exploration holes on Freeman's newly discovered "Beauty" zone, ~600 m west of the Lemhi Gold Deposit. Metallurgical studies along with the new drilling have assisted in delineation and improvement of the existing geological and mineralization model into a coherent 3D model allowing for the construction of a MRE, as presented within this technical report.

In 2023, APEX personnel validated and compiled an updated drill hole database (DHDB) to correct mistakes identified in the 2012 DHDB and include additional historical drill results discovered while verifying the 2012 database. The new 2023 Freeman DHDB was utilized in constructing the MRE in this technical report.

1.9 Drilling

During 2020-2022, Freeman completed 106 holes at the Lemhi Gold Project, on the Lemhi and Beauty prospects. From November 14, 2021, to November 21, 2022, Freeman conducted a 15,351 m drill program consisting of 58 core holes and 13 RC holes on the Lemhi Project. The aim of this program was to expand and infill the existing resource at Lemhi, and to provide increased confidence in areas with less reliable historical drilling results.

The 2021-2022 drilling expanded the mineralization at Lemhi down- and up-dip of the existing Lemhi resource. Best intersections were encountered in the north-central portion of the deposit, and along south-east extensions to the deposit. Examples of these intersections include 3.7 m at 10.2 g/t Au from 20.27 m in FG22-017C, 10.7 m at 3.0 g/t Au from 74.68 m in FG22-061R, 15 m at 2.1 g/t Au from 93 m in FG22-050C and 7.2 m at 3.8 g/t Au from 121.31 m in FG22-022C.

Drilling at the Beauty target confirmed the continuation of mineralization at depth from the veins mapped and sampled at surface. Best intersections included 5.2 m at 78.7 g/t Au from 57.8 in FG21-003C and 3.0 m at 4.4 g/t Au from 134.1 m including 1.5 m at 8.0 g/t Au from 135.6 m in FG22-056R.

1.10 Sampling Preparation and Security

A total of 7,215 drill core samples, 145 rock samples, 633 soils samples were collected during the 2020 exploration program. Of the soil samples, 291 were submitted to SGS Mineral Services – Burnaby (SGS) for MMI analysis, all remaining core, rock, and soil samples were submitted to ALS Geochemistry – Vancouver (ALS). A total of 12,144 drill core samples, 1,432 RC chip samples, 394 rock samples, and 1,006 soil samples were collected during the 2021-2022 exploration program. All core, chip, rock, and soil samples were submitted to ALS.

During the 2020 program, a total of 7,993 drilling samples, including standards, blanks, and duplicates (Certified Reference Materials or CRM) were submitted for analysis. This total includes 875 quality assurance or quality control (QA/QC) or CRM samples (10.9 %) which falls within the industry standard of at least 10% QA/QC samples for ongoing quality control and future resource work. Known standards were inserted after every 20 core or RC chip samples, duplicates after every 20 core or RC chip samples and coarse blanks were inserted after predicted high-grade intersections. Six different CRMs were selected from CDN Resource Laboratories Ltd. The selected CRMs include: CDN-BL-10, CDN-CM-40, CDN-GS-6F, CDN-GS-P4J, CDN-ME-1705 and CDN-CGS-28.

Overall, the 2020 dataset shows both high precision and accuracy with only a few analyses falling outside of three standard deviations (SD) in error and the vast majority within two SDs for error. This further demonstrates the high reliability of ALS and validity of the 2020 core sample dataset. The 2020 core sample data is considered suitable for use in the 2023 MRE presented in this technical report. The 2020 QAQC program is discussed in detail by Dufresne et al., 2021.

During the 2021-2022 program no conventional soils or MMI analysis on soils were completed. All other sample preparation, analyses, and security methods remained the same between the 2020 and 2021-2022 programs. As samples were collected, they were recorded within custom built Fulcrum Apps and relevant information such as sample location, geological information and photographs of the sample and site were recorded within the apps. Sample locations were recorded with a handheld GPS and input into the apps.

During the 2021-2022 drilling program a total of 13,062 core and 1,573 RC chip samples were submitted to ALS for analysis. This includes 1,593 QA/QC samples (12.2 %) for drill core and 188 QA/QC samples (12.0 %) for RC chip samples, which falls within the industry standard of at least 10% QA/QC samples for ongoing quality control and future resource work. Known standards were inserted after every ten samples and represent ~7% of the analyses. Duplicates were submitted after every 30 samples and represent ~3% of the analyses. Coarse blanks were inserted after every 50 samples and after predicted high-grade intersections and represent ~2% of the total analyses. Eight different CRMs were selected from CDN Resource Laboratories Ltd. The selected CRMs include: CDN-BL-10, CDN-CM-40, CDN-GS-P4J, CDN-ME-1705, CDN-CGS-22, CDN-CGS-27, CDN-CM-44 and CDN-ME-2104. Re-assays for pulps were completed when CRMs plotted outside 3 SDs for error, pulp blanks >2 times the lower detection limit (LOD) or coarse blank >3 times LOD. Five natural samples on either side of the failed standard were re-assayed in case of a failure. A total of 25 CRMs from 19 drill holes failed. The database utilized for Section 11 includes all the re-assays and is the final 2021-2022 assay database used in the MRE in this report.

1.11 Data Verification

APEX personnel compiled a drillhole database (DHDB) containing the historical and 2020 drilling data and incorporated the new 2021-2022 drilling by Freeman. The DHDB includes collar, downhole survey, assay, geology, structural, and geotechnical data.

Once the re-construction of the DHDB was complete, spot checks of ~10% of the DHDB collars and assays by APEX personnel confirmed it was in good condition and suitable for ongoing MRE studies. The DHDB contains a total of 514 holes. Of these, 501 are within the current property boundaries. In total, 506 drill holes have complete collar, assay, and drill log data.

Historical drill hole data completed by pre-Freeman operators were reviewed and it was determined that the pre-2000 drill hole data is deemed to be not as reliable as drill hole data obtained in 2012 and 2020 to 2022 with current industry best practices for sample preparation, analyses, QA/QC, and security. The discrepancies in the pre-2000 era dataset included lower accuracy in collar location due to collar coordinates often being based on rectified collar location maps, and discrepancies between check assays and umpire assay results based on a review of previous reports. Previous industry best practices for sample preparation, assay, and security standards did not include adequate QA/QC of lab assay results and therefore confidence in pre-2000 assay results is lower than current assay results.

Upon further review of the pre-2000 drill hole data, Mr. Dufresne considers the pre-2000 drill hole data to be well documented and in good condition and suitable for ongoing MRE studies. The inclusion of the American Gold Resource drilling data from the 1990s should present no risk in the MRE based upon the review. The inclusion of the 1980s FMC drill hole data does, however, increase the risk of a slightly biased estimate in areas that rely on the 1980s FMC data. To this end, the MRE in Section 14 has adjusted the classification to a lower confidence level in areas that significantly rely on the 1980s FMC data.

Mr. Dufresne considers the current Lemhi drill hole database to be in good condition and suitable for ongoing resource estimation studies. As discussed in Sections 25 and 26, recommendations for conducting modern drilling in areas of the MRE that rely on significant numbers of historical 1980s FMC drill holes have been made in order to enable higher confidence in the database and the MRE.

1.12 Metallurgical Testwork

Three metallurgical test programs were conducted on samples from the project since 1994. The test programs evaluated cyanide leaching, tank leaching, gravity concentration, and comminution properties.

Kappes Cassiday completed three phases of column and bottle roll leach testing between 1994 and 1995, however the origins of the samples are not clearly identified in the reports. The results agree with more recent testing, in that 93% of the feed gold could be extracted at a primary grind size of 150 µm P_{80}, for material with a feed grade of 1 g/t gold.

A phased program of metallurgical testing was conducted by SGS in 2022, which provides the majority of the metallurgical data on the project. Samples originated from both the 2012 and 2020 drill programs, utilizing both crushed assay rejects and half PQ core. In total, 11 composites and 31 variability samples were tested. The composites had a moderate range in gold feed grades and averaged 1.2 g/t. The composites contained low levels of sulphur and copper, averaging 0.13% and 0.05% respectively. The SGS testing included comminution testing, gravity concentration, leaching of both direct feed and gravity tails, and a small amount of flotation testing on selected samples.

The most recent metallurgical test program was completed at Base Metallurgical Laboratories using half-core samples from the 2020 drill program. The program included comminution testing on five samples, and metallurgical testing on two master composites. Metallurgical testing included gravity concentration and bottle roll leaching of the gravity tails. Additional testing to support design criteria was conducted which included CN detox and oxygen uptake rates. Solid liquid separation testing was conducted on tailings in both the SGS and Base Met programs.

The samples from both the SGS and Base Met programs consistently showed a very low resistance to breakage in a semi-autogenous grinding (SAG) mill, returning Axb values of greater than 80 and as high as 312. The average standard circuit specific energy (SCSE) value of six samples was 6.6 kWh/t. The samples also measured a moderate ball mill work index, the average BMWi value of 13 samples was 14.5 kWh/t.

The samples showed a moderate response to gravity concentration. Gold recovery to gravity concentrates averaged 40.8% to an average concentrate mass of 0.12% for the SGS and Base Met composites.

Leach extractions of gold from either the direct feed or gravity tails was consistently in the range of 95 to 96% at primary grind sizes that averaged 125 µm P_{80}. Usually, extractions were nearly complete within the first 24 hours. With a gravity concentrate intense leach extraction factor of 98% included, total circuit recoveries averaged 96% for both the SGS and Base Met test programs. The SGS test program investigated direct feed leaching and the results appear to be similar to the gravity plus leach extractions. Sodium cyanide consumptions were moderate, averaging 0.9 kg/t for the composites.

Two of the 31 variability samples within the SGS program contained feed copper levels of 0.21% and measured lower gold leach extraction values given their respective gold feed grades. While cyanide consumptions were higher for these samples, leach solution cyanide concentrations were maintained at sufficient levels for gold leaching. It is uncertain whether a portion of the gold in these samples occurred as inclusions within chalcopyrite and was not amenable to cyanide leaching. Preliminary froth flotation tests indicated that the inclusion of a flotation circuit could mitigate this recovery issue, however it is unclear how prevalent this issue might be across the resource. A composite with feed grades of 2 g/t gold and 0.18% Cu did not appear to have any leach performance issues.

A gold recovery equation was developed from composite and variability results from the two recent test programs that had feed grades within the range of the mine plan and primary grinds within the range of 90 to 130µm P_{80}. The equation relates gold recovery to gold feed grade and was used to estimate gold recoveries in the annual mine plan.

1.13 Mineral Resource Estimate (MRE)

The Lemhi Project database contains a total of 506 drill holes with collar information, and assays covering 91,747 m of drilling with 64,299 drill hole sample intervals. The sample database contains a total of 62,670 samples assayed for gold. The 2023 Lemhi MRE utilized 442 drill holes that intersected the estimation domains of which 284 drill holes were completed between 1983 and 1995, and 158 drill holes were completed between 2012 and 2022. Inside the mineralized domains, there is a total of 16,234 samples analyzed for gold. Standard statistical treatments were conducted on the raw and composite samples resulting in a capping limit of 17.3 g/t gold (Au) applied to the composites for the Main Zone and 50 g/t Au for the Beauty Zone. The current DHDB was validated by APEX personnel and is deemed to be in good condition and suitable for use in ongoing MRE studies Mr.Michael Dufresne, M.Sc., P.Geol., P.Geo., President of APEX, is an independent qualified person (QP) and is responsible for the database validation and MRE.

Mineral resource modelling was conducted in the Universal Transverse Mercator (UTM) coordinate system relative to the North American Datum (NAD) 1983, and Idaho State Plane Central FIPS 1102 (EPSG:6448) The mineral resource block model utilized a selective mining unit (SMU) block size of 2.5 m (X) by 2.5 m (Y) by 2.5 m (Z) to honour the mineralization wireframes. The percentage of the volume of each block below the top of bedrock surface and within each mineralization domain was calculated using the 3-D geological models and a 3-D topographic surface model. The Au grades were estimated for each block using ordinary kriging with locally varying anisotropy (LVA) to ensure grade continuity in various directions is reproduced in the block model. The MRE is reported as undiluted within a series of optimized pit shells. Details regarding the methodology used to calculate the MRE are documented in this section.

Gold mineralization at the Lemhi Gold Project is primarily of two dominant styles. The primary mineralization occurs as a halo around the granodiorite intrusion, concentrated on the bottom side of the intrusive bodies, with secondary mineralization along faults and shallow dipping foliation. It appears that both styles of mineralization generally occur in zones of stacked parallel sub-horizontal sheets. The Beauty zone is ~700 m west from the nearest modelled intrusion and is primarily controlled by a structurally complex fault zone.

In total, 14,208 bulk density samples are available from the Lemhi Property drillhole database. APEX personnel performed exploratory data analysis (EDA) of the bulk density sample data available. Three main geologic units showed significant variation in density. The median specific gravity (SG) value for each geological unit was used for assigning density for material in the MRE. The EDA resulted in a change in the SG used in the 2021 MRE from 2.62 g/cm^3 (Dufresne 2021) for mineralized material and unmineralized material to 2.64 g/cm^3 for metasedimentary rock material, 2.58 g/cm^3 for intrusion material, and 2 g/cm^3 for silty breccia material.

All reported mineral resources occur either within a pit shell optimized using values of US$1,750 per ounce of gold or in shapes outside of the pit shell that display potential for underground stopes. The measured, indicated, and inferred mineral resources are undiluted and constrained within an optimized pit shell at a 0.35 g/t Au lower cut-off. Out-of-pit potential underground mineral resources utilized a 1.5 g/t Au lower cut-off and are constrained with continuous shapes that yield a minimum of 1,400 m^3. The MRE comprises a combined measured and indicated mineral resource of 30.022 Mt at 1.00 g/t Au for 988,100 oz of gold, and an inferred mineral resource of 7.634 Mt at 1.04 g/t Au for 256,000 oz of gold and shown in Table 1-1. The MRE for the Main Zone covers a surface area of 1,320 m by 740 m and extends down to a depth of 240 m, and remains open on strike to the north, south, and west, as well as at depth. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 1-1: **2023 Lemhi Gold Project Mineral Resource Estimate [1-8].**

Au Cut-off (g/t)	Zone	RPEEE Scenario	Classification	Tonnes	Au (oz)	Au Grade (g/t)	Au Grade (oz/st)
0.35	Main and Beauty	Open Pit	Measured	4,469,000	168,800	1.15	0.033
0.35	Main and Beauty	Open Pit	Indicated	25,553,000	819,300	0.98	0.029
0.35	**Main and Beauty**	**Open Pit**	**M and I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35	**Main and Beauty**	**Open Pit**	**Inferred**	**7,338,000**	**234,700**	**1.01**	**0.029**
1.5	Main and Beauty	Underground	Inferred	296,000	21,300	2.27	0.066
0.35/1.5	Main and Beauty	Combined	Measured	4,469,000	168,800	1.15	0.033
0.35/1.5	Main and Beauty	Combined	Indicated	25,553,000	819,300	0.98	0.029
0.35/1.5	**Main and Beauty**	**Combined**	**M and I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35/1.5	**Main and Beauty**	**Combined**	**Inferred**	**7,634,000**	**256,000**	**1.04**	**0.030**

Notes:
1. Contained tonnes and ounces may not add due to rounding.
2. Mr.Michael Dufresne, PGeol., P. Geo. of APEX Geoscience Ltd., who is deemed a qualified person as defined by NI 43-101 is responsible for the completion of the updated mineral resource estimation.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
5. The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could potentially be upgraded to an indicated mineral resource with continued exploration.
6. The mineral resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
7. The constraining pit optimization parameters assumed US$1,750/oz Au sale price, NSR Royalty of 1%, US$2.10/t mineralized and US$2.00/t waste material mining cost, 50° pit slopes, a VAT process cost of US$8.00/t, HL process cost of US$2.40/t and a general and administration (G&A) cost of US$2.00/t.
8. The effective date of the mineral resources estimate is March 15, 2023.

The 2023 Lemhi MRE is classified according to the CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (November 29, 2019) and CIM "Definition Standards for Mineral Resources and Mineral Reserves" (May 10, 2014).

1.14 Mining Methods

The deposit is amenable to open pit mining practices. Open pit mine designs, mine production schedules, and mine capital and operating cost estimates have been developed for the Lemhi deposit at a scoping level of engineering. Mineral resources form the basis of the mine planning.

Mine planning is based on conventional drill/blast/load/haul open pit mining methods suited for the project location and local site requirements. The open pit activities are designed for two years of construction followed by twelve years of operations. The subset of mineral resources contained within the designed open pits are summarized in Table 1-2, with a 0.25 g/t gold cut-off, and form the basis of the mine plan and production schedule.

Table 1-2: PEA Mine Plan Production Summary

Parameter	Value
PEA mill feed (LOM)	31,128 kt
Mill feed gold grade	0.88 g/t
Waste overburden and rock	121,903 kt
Waste to resource ratio	3.9

Notes:
1. The PEA Mine Plan and Mill Feed estimates are a subset of the March 15, 2023, Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a Scoping level for the Lemhi deposit.
2. PEA Mine Plan and Mill Feed estimates are mined tonnes and grade; the reference point is the primary crusher.
3. Mill Feed tonnages and grades include open pit mining method modifying factors, such as dilution and recovery.
4. Cut-off grade of 0.25 g/t assumes $1,700/oz. Au; 99.95% payable gold; $4/oz off-site costs (refining, transport, and insurance); a 1.0% NSR royalty; and a 92% metallurgical recovery for gold.
5. The cut-off grade covers processing costs of $9.20/t, administrative (G&A) costs of $1.10/t, and low-grade stockpile Rehandle costs of $1.00/t.
6. Estimates have been rounded and may result in summation differences.

The economic pit limits are determined using the Pseudoflow implementation of the Lerchs-Grossman algorithm. Ultimate pit limits are split up into six phases or pushbacks to target higher economic margin material earlier in the mine life. Upper benches will be accessed via internal cut ramps on topography, or via ramps left behind on phased pit walls. In-pit ramps will access material below the pit rim.

Pit designs are configured on 5 m bench heights, with minimum 8 m wide berms placed every four benches, or quadruple benching. Slopes of 25° are applied in the thin overburden layer above the deposit bedrock. Since there has been no geotechnical test work or analysis completed on the bedrock, the applied bench face and inter-ramp angles, 70-75°and 50-55°, respectively, are scoping level assumptions based only on the rock type and overall depth of the open pit.

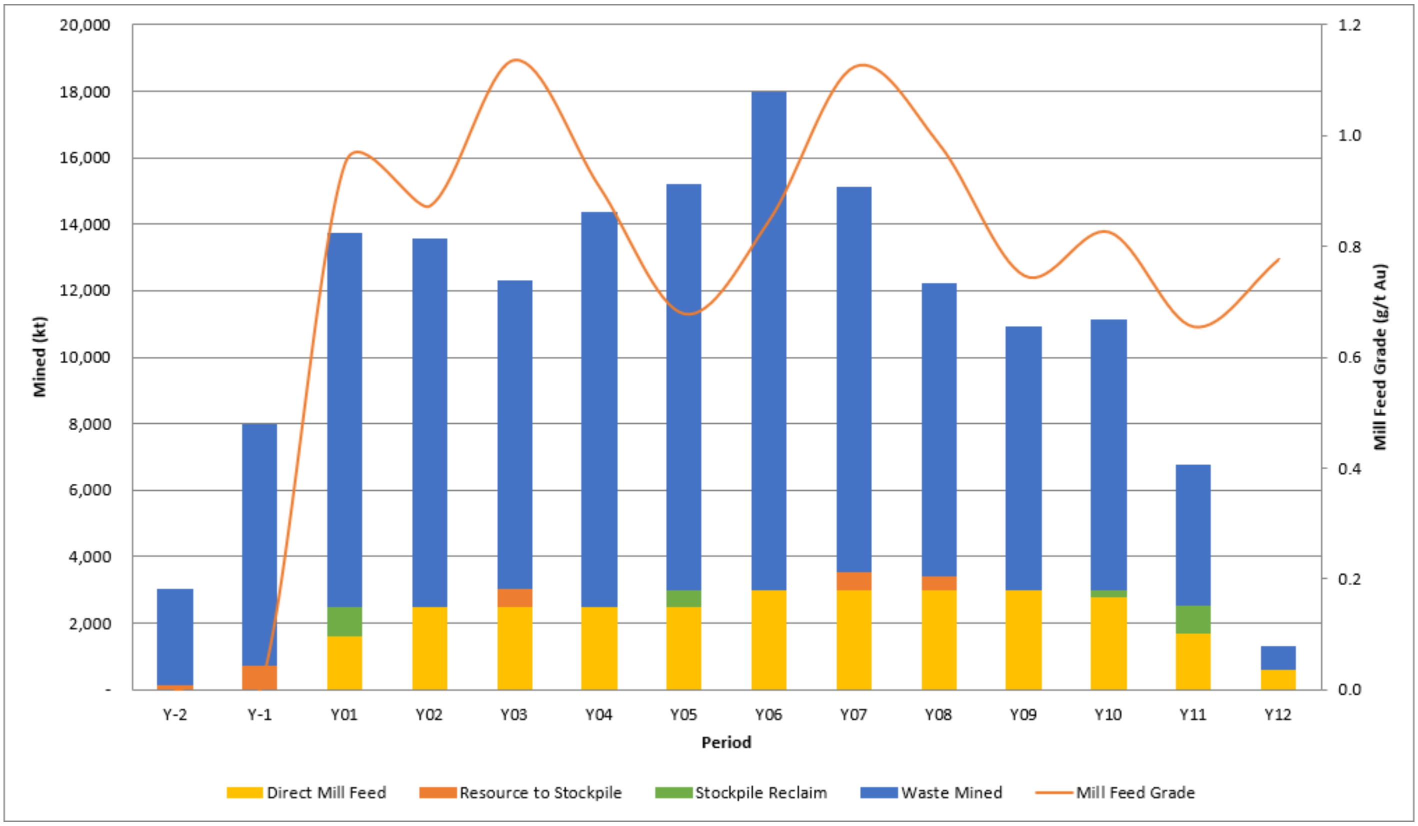
Resource from the open pit will report to a run-of-mine (ROM) pad and primary crusher directly northeast of the pit rim. The mill will be fed with material from the pits at an average rate of 2.5 Mt/a (6.8 kt/d), increasing to 3.0 Mt/a (8.2 kt/d) after four years of operation. Resources mined in excess of mill feed targets will be stored in a low-grade stockpile directly south of the ROM pad, and east of the open pit. This stockpile is planned to be completely reclaimed to the mill at the end of the mine life.

Waste rock will be placed in one of two facilities, each planned as a comingled facility with processed tailings. The north facility sits directly adjacent and uphill from the open pit, with its most northern point lying 1.2 km from the pit rim. The south facility sits 0.6 km southeast and downhill of the open pit, with its most southern point lying 2.0 km from the pit rim. The waste rock from the open pit has not been tested or analyzed for potential acid generation (PAG).

Topsoil and overburden encountered at the top of the pits will be placed in a dedicated stockpiles directly south of the open pit and kept salvageable for closure at the end of the mine life.

The mine production schedule is summarized in Figure 1-1.

Figure 1-1: Mine Production Schedule Summary



Source: Moose Mountain, 2023.

Mining operations will be based on 365 operating days per year with two 12-hour shifts per day. Owner managed operations are planned, utilizing a diesel-powered mining fleet.

Cost estimates for mining are based on benchmarking to other similar sized operations in western United States, mining 12-16 Mt/a. These operations typically include RC drills for bench-scale grade control drilling, down the hole (DTH) drills with 140 mm bit size for production drilling, emulsion based on blasting agents targeting 0.3 kg/t powder factors, 12 m^3 bucket size diesel hydraulic excavators and 14 m^3 bucket sized wheel loaders for production loading, and 91 t payload rigid-frame haul trucks for production hauling, plus ancillary and service equipment to support the mining operations.

In-pit dewatering systems will be established for the pit. All surface water and precipitation in the pits will be gravity drained, or directed via submersible pumps, to ex-pit settling ponds directly outside the pit limits.

The mine equipment fleet is planned to be purchased via lease financing arrangement, with down payments occurring when the equipment is commissioned, and lease payments deferred for 1 year after the equipment is operational. Maintenance on mine equipment will be performed in the field with major repairs and planned interval maintenance in the shops located near the process facilities.

1.15 Recovery Methods

The plant is designed for a throughput of 2.5 Mt/a in the initial phase (Phase 1) and ramp-up to 3.0 Mt/a (Phase 2) with availability of 92%. The crusher plant circuit design is set at 75% availability and the gold room availability is set at 52 weeks per year. The plant will operate two shifts per day, 365 days per year, and will produce doré bars. The project has an estimated life of 11.2 years.

The process plant features the following:

- primary crushing of ROM material
- SAG mill followed by ball mill with cyclone classification
- leach and carbon-in-leach (CIL) adsorption, a pre-leach thickener will be added for the expansion
- acid washing and elution of loaded carbon
- electrowinning and smelting to produce doré
- carbon regeneration
- cyanide destruction and wet tailings disposal.

The simplified process flow diagram for the Lemhi Project is shown in Figure 1-2.

1.16 Project Infrastructure

1.16.1 Overview

- Infrastructure at the Lemhi Project includes on-site infrastructure such as earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, power supply, piping, camp, and tailings storage facility. The site infrastructure will include:
- Mine facilities include administration offices, truck shop and wash bay, and mine workshop.
- Common facilities, including an entrance/exit gatehouse, a security/medical office. Overall site administration building, potable water and fire water distribution systems, compressed air, power distribution facilities, diesel reception and communications area.
- Process facilities housed in the process plant, including grinding and classification, leach and CIL adsorption, acid washing and electrowinning and smelting, carbon regeneration, cyanide destruction, assay laboratory and process plant workshop and warehouse.
- Other infrastructure includes on-site camp and waste management facilities and two co-placement storage facilities (CPSFs).

The mine and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication utilities, and sanitary systems.

Site selection and location for project infrastructure was guided by the following considerations:

- Locating the facilities described above on the Lemhi patent land to the greatest extent possible.
- Locating two CPSFs close to the open pit to reduce haul distance.
- Locating primary crushing close to the Lemhi deposits to reduce haul distance.
- Utilizing the natural high ground for the ROM pad as much as possible.
- Separating heavy mine vehicle traffic from non-mining, light vehicle traffic.
- Locating the process plant near an existing primary access road.
- Locating the process plant in an area safe from flooding.
- Placing mining, administration, and process plant staff offices close together to limit walking distances between them.

The Lemhi Project site layout is shown in Figure 1-3.

1.16.2 Site Access

The Lemhi Project site is accessible via multiple routes. The primary access is through Salmon, Idaho, via paved and gravel roads. To access the Lemhi Project site and process plant, routing will be upgraded as the access is through mountainous terrain that features some switchbacks and sharp turns. As part of upgrading activities, some of these switchbacks and turns will be improved to meet the transportation needs of the site. The proposed access route avoids both residential areas in the region and the project's 300 m blast radius for the project's open pit mine design.

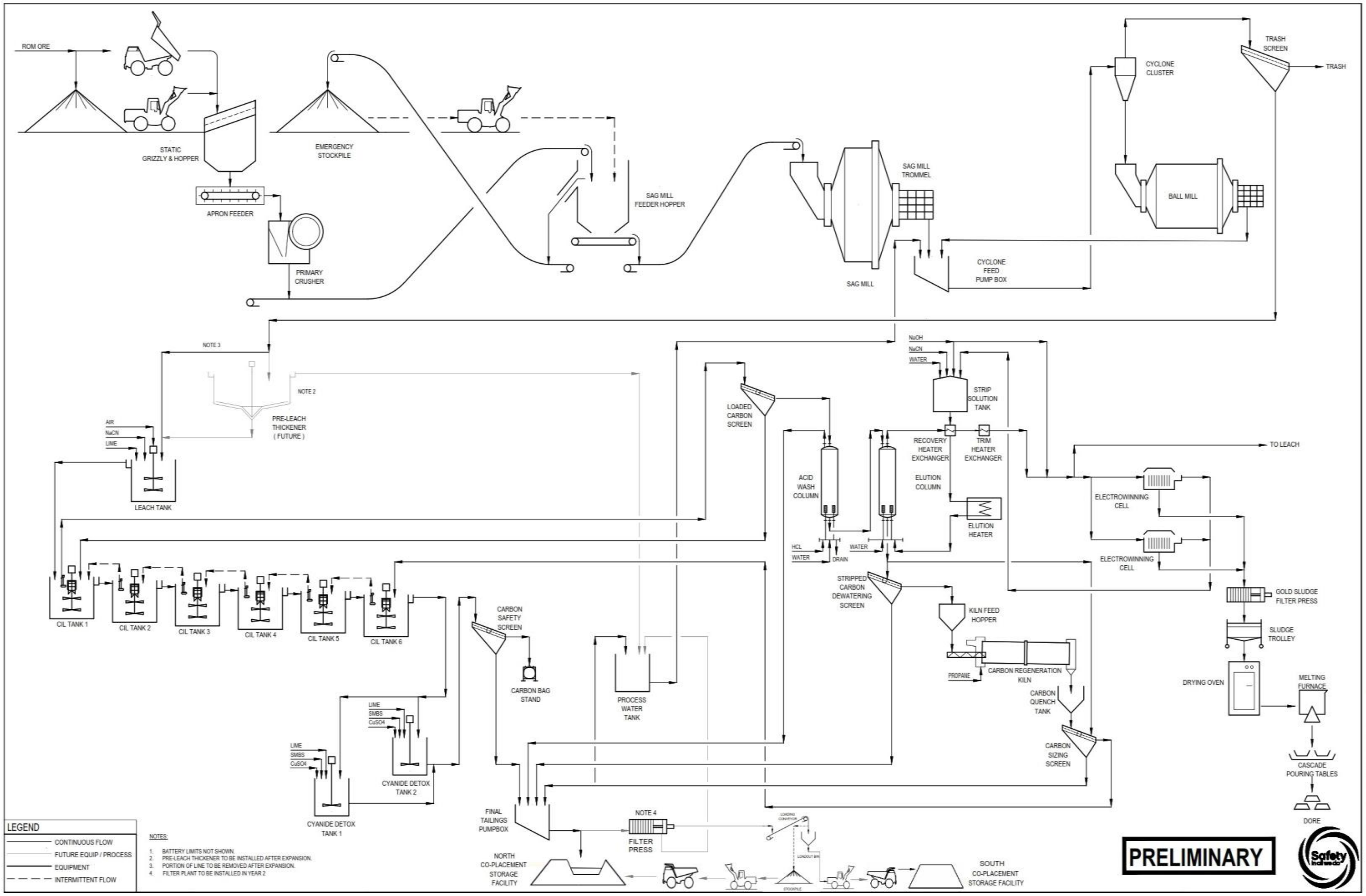
1.16.3 On-Site Roads

The typical method of clearing, topsoil removal, and excavation will be employed, incorporating drains, safety bunds, and backfilling with granular material and aggregates for road structure. Clearing forest and removing topsoil is expected to allow construction of the processing plant and other buildings and facilities.

1.16.4 Mining Infrastructure

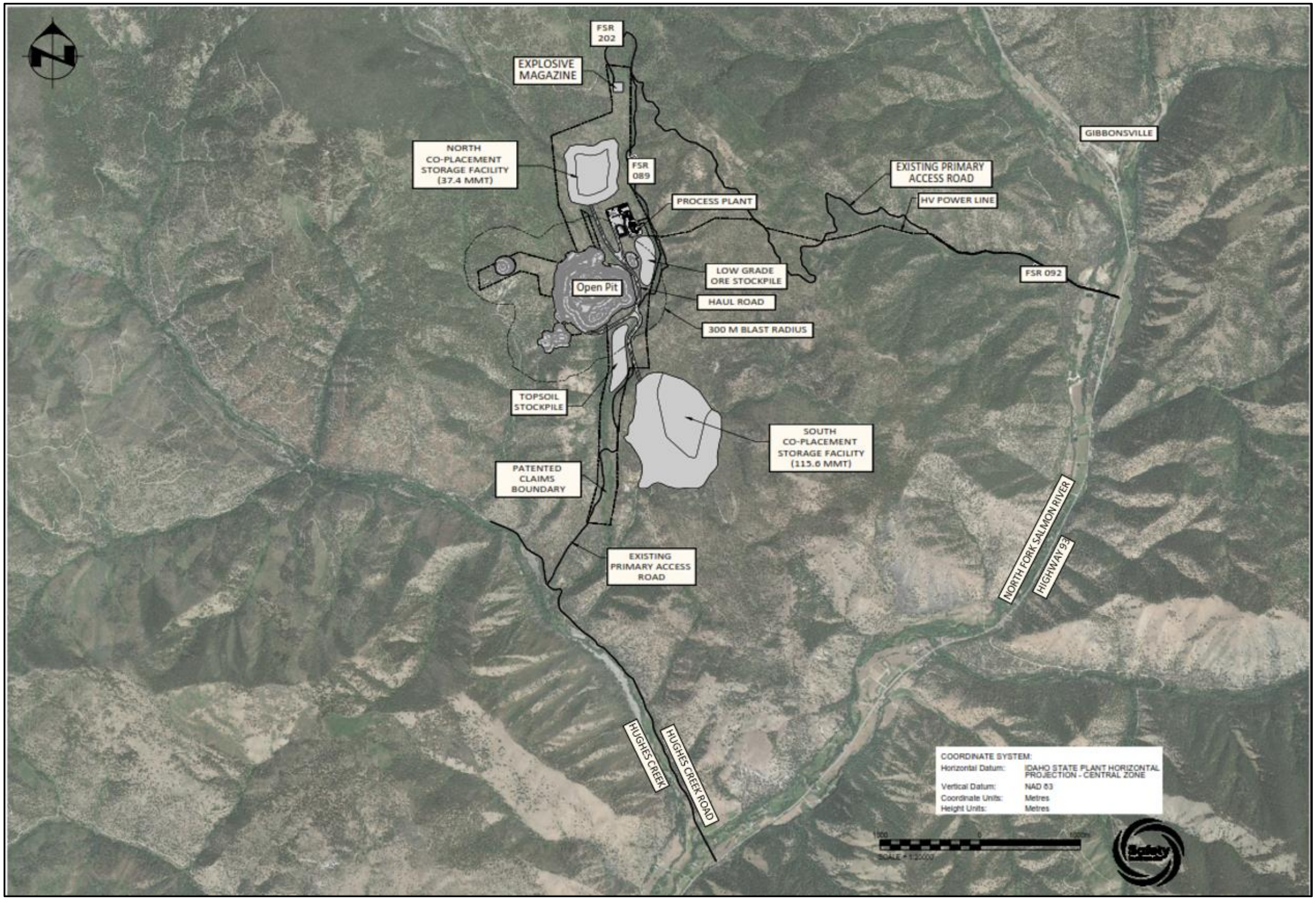
The mining infrastructure includes haul roads from the pit to the different areas on site, explosive facility, truck shop and truck wash bay, mine warehouse, office, and workshop. Other on-site roads will be constructed to allow access between the administration building, warehouses, mill building, crushing buildings, mining truck shop. These roads will be constructed to allow two-way, light vehicle and, in some areas, mine truck traffic. All internal mine roads will be all-season, gravel-paved roads.

Figure 1-2: Process Flowsheet



Source: Ausenco, 2023.

Figure 1-3: Infrastructure Layout Plant



Source: Ausenco, 2023

1.16.5 Buildings

The plant site will consist of infrastructure necessary to support the processing operations with all buildings and structures constructed to comply with all applicable codes and regulations. The project site will include an administration building, plant maintenance shop, warehouse, and other buildings.

1.16.6 Water Supply

The fresh water will be supplied from the wells located on site. This water will be the source of potable water on site, used for the building facilities and the process plant.

1.16.7 Power Supply

Electrical power will be supplied from the local grid via a 5 km power line to be constructed for the project. The power line will be connected to a high voltage line that passes nearby the project site and distributed to different power requirements across the project site.

1.16.8 Fuel Storage

Fuel will be delivered to the mine site via tanker trucks. The fuel storage tanks are insulated and heated to prevent fuel gelling. The tanks will be contained in a lined containment berm to assure no fuel can leak into the environment.

1.16.9 Co-placement Storage Facility (CPSF)

Two waste materials are generated during the mining process: waste rock and tailings. Tailings and coarse waste rock material will be transported independently, but not mixed to form a single discharge stream, into co-placement storage facility (CPSF). For the Lemhi Project, two co-placement storage facilities will be constructed over the life of mine (LOM), the north CPSF and the south CPSF.

The north CPSF will be constructed first since it is within their patented claims boundary while Freeman Gold obtains the permit to store waste materials on federal National Forest lands. The North CPSF has a storage capacity of 37.4 Mt of tailings and waste rock. This facility will be a slurry tailings facility with upstream raises since there is sufficient waste rock to develop a starter embankment. The facility has storage capacity for over two years of tailings and waste rock.

The south CPSF will be commissioned in Year 3 after obtaining the permit to place waste materials on federal lands. There is insufficient time to construct an embankment for slurry tailings; therefore, the north CPSF will be a paddock style construction utilizing waste rock to create cells and the tailings will be filtered to a cake and placed in the cells. The south CPSF has a storage capacity of 115.6 Mt of tailings and waste rock.

For the north CPSF tailings will be conveyed to the site in a pipeline and decant water will be reclaimed from the back of the facility. For the south CPSF both tailings and waste rock will be transported by haul trucks.

1.16.10 Water Management Structures

Non-contact runoff will be diverted away from mining facilities via excavated channels. Runoff from the waste rock dump (WRD) and processing plant area will be collected in channels and conveyed to storage ponds for treatment or release to the environment. A diversion berm adjacent to the Little Ditch Creek will be constructed outside the watercourse to prevent non-contact water from migrating into the WRD.

1.17 Market Studies and Contracts

It is assumed in this PEA that the Lemhi Gold Project produces gold in the form of doré bars. The market for doré is well-established and accessible to new producers. The doré bars is refined in a certified North American refinery—there are many in the eastern United States and Canada—and the gold to be sold on the spot market.

Freeman Gold has not completed any formal marketing studies with regard to gold production resulting from the mining and processing of mineralized material in the form of gold doré bars from the project. Gold 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 is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with large numbers of buyers and sellers active at any given time.

1.18 Environmental, Permitting and Social Considerations

The Lemhi Project involves the development of the Lemhi Gold Deposit. The project is in Lemhi County in east-central Idaho, within the Salmon River Mountains, a part of the Bitterroot Range which forms the Idaho-Montana border. The project area is located within the Hughes Creek Basin. Hughes Creek is a tributary to the North Fork Salmon River, which subsequently flows into the Salmon River 15 km south of the project area. Most of the proposed mine facilities are within the Ditch Creek sub-basin.

1.18.1 Environmental Considerations

Several limited field and screening environmental baseline studies and reports were completed between 1995 and 2012. The programs involved the collection of baseline data within the proposed project footprint area and commenced the process of identifying potential environmental constraints and opportunities related to the proposed development of the project.

The environmental baseline studies included:

- Fish population and habitat (1995)
- Acid rock drainage (ARD) testing for mineralized and waste material (1995)
- Surface water hydrology and water quality (1995-1996)
- Hydrogeology and groundwater quality (1995-1996)
- Wetland delineation (1995, 2012)

- Vegetation (2012).

Notably, much of the data collected for baseline studies is not recent; therefore, new baseline studies documenting existing or recent conditions will be required to support baseline development and impact assessments. In assessing the utilization of older baseline data, direct discussions with state and federal regulators will be required.

Additionally, there have been no baseline studies completed to date on air quality, meteorology, noise, greenhouse gases and climate change, wildlife and wildlife habitat, or cultural resources. Ongoing and expanded baseline studies will be required to support the project through pre-feasibility, feasibility, and environmental impact statement/permitting stages. The results of baseline studies will be used to minimize impact of the project on valued ecosystem components and optimize the location and operation of project infrastructure.

As the project progresses through the pre-feasibility, feasibility, and environmental impact statement/permitting stages, environmental management and monitoring plans will be required for the purpose of guiding the development and operation of the project and mitigating and limiting environmental impacts. These plans will be complementary to the engineered designs that will be required for the storage of tailings, waste rock, mineralized material, and conveyance/storage/treatment of mine contact water (refer to Section 18 of this technical report).

Ditch Creek is the primary watercourse through the project area. This stream is designated as critical habitat for Bull trout and is protected under the Endangered Species Act. To mitigate project impacts to this stream, alternative designs for site infrastructure, including potential realignment of Ditch Creek and its associated riparian area, may be required by federal regulators.

There are environmentally sensitive areas located downstream of the project, including: the main stem of the Salmon River, located 15 km south of the project, and the Frank Church River of No Return Wilderness Area, located 40 km southwest of the project.

1.18.2 Closure and Reclamation Considerations

All surface mines must submit and obtain approval of a comprehensive reclamation and closure plan for mining activities on patented land as administered by the Idaho Department of Lands. This includes detailed operating plans showing pits, mineral stockpiles, overburden piles, tailings facilities, haul roads, and all related facilities. A reclamation and closure plan must also align with appropriate best management practices and provide for financial assurance in the amount necessary to reclaim those mining activities.

The Mine Plan of Operations (MPO) submitted to United States Forest Service (USFS) under the National Environmental Policy Act (NEPA) process must include a reclamation and closure plan. In addition, a reclamation report with a Reclamation Cost Estimate (RCE) for the closure of the project is required.

A key closure objective for the mine will be for effluent to meet applicable water quality objectives without ongoing treatment. The current conceptual closure and reclamation plan for the project includes the following measures:

- Partial backfilling of open pits with non-acid generating waste rock, and flooding of the remaining open pit
- The mineralized material stockpile will be reclaimed, once depleted.

- The surface infrastructure on the site will be decommissioned and removed from the site upon completion of mining.
- Explosives, explosives magazines, fuel, and storage facilities will be removed from the site.
- Concrete slabs and footings will be broken and placed appropriately to meet project closure and reclamation objectives.
- Process buildings, pipelines, conveyor systems, and equipment will be removed from site or appropriately landfilled in an approved facility.
- Comingled waste and tailings facilities (CWTF) will be re-contoured for geotechnical stability, capped with a graded earthfill/rockfill cover to facilitate runoff and minimize infiltration, and revegetated.
- Compacted surfaces including laydowns, civil pads, and roads will be decompacted, re-contoured, capped with a graded earthfill/rockfill cover to facilitate runoff and minimize infiltration, and revegetated.
- Water treatment will be continued until water quality meets discharge criteria. Once water quality meets discharge criteria, water treatment will be stopped, diversions will be decommissioned, and the site will be allowed to discharge naturally.
- For mine roads, Freeman Gold will remove all culverts and install cross-ditches for drainage. The mine site access road will not be deactivated as it will be required for access for continued reclamation activities and monitoring.

Closure planning will include dialogue with appropriate stakeholders to determine post-mining land use objectives and necessary investigations required to achieve and monitor those objectives.

1.18.3 Permitting Considerations

Some project infrastructure is located on federal National Forest lands administered by the U. S. Forest Service (USFS). As such, permitting and approval for the mine will be subject to the National Environmental Policy Act (NEPA) review process and the requirements stipulated in a Record of Decision (ROD) for an Environmental Impact Statement (EIS) prepared by the USFS as the lead agency.

Modification and alteration of Ditch Creek, a federally designated critical habitat for Bull trout, is a permitting risk for the project and may raise concerns from stakeholders.

The major federal authorizations anticipated for the project beyond the NEPA Record of Decision include Biological Opinions under the Endangered Species Act for listed species or their critical habitats, and a Dredge and Fill permit under the Clean Water Act to place fill materials into Waters of the United States.

The major state permits anticipated for the project include the following: wastewater discharge permits under the Idaho Pollutant Discharge Eliminate System, air quality permit to construct and operate, groundwater point of compliance permit, cyanidation permit, and a stream channel alteration permit.

1.18.4 Social Considerations

Baseline socio-economic and cultural baseline studies have not yet been completed for the Lemhi Project. These assessments will be required at the appropriate time as the project advances into the feasibility and permitting phases and the full extent of the disturbed footprint of the project has been identified.

Based on the available information, there are no indications to date of community or tribal consultation completed by Freeman Gold. Environmental review of the project Plan under NEPA will include public scoping to obtain input from the local community and tribal members and to develop alternatives to the proposed action. The NEPA review will likely include Government-to-Government consultation between USFS and the Nez Perce and Shoshone-Bannock Tribes. During this consultation, a determination will be made if traditional cultural properties, cultural landscapes, sacred sites, or tribal resource collection areas would be adversely impacted in project areas.

1.19 Capital and Operating Cost

1.19.1 Capital Cost Estimate

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with -30% /+50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2023 United States dollars, based on based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and advanced studies including experience from similar operations.

The estimate includes open pit mining, processing, on-site infrastructure, tailings and waste rock facilities, off-site infrastructure, project indirect costs, project delivery, owner's costs, and contingency. The capital cost summary is presented in Table 1-3. The total initial capital cost for the Lemhi Project is US$190.2 M; and LOM sustaining costs are US$101.2 M. The cost of expansion in the fifth year is estimated at US$7.6 M. Closure costs are estimated at US$29.9 M, with salvage credits of US$12.0 M applied at the end of the LOM.

1.19.2 Operating Cost Estimate

The estimate includes mining, processing, maintenance, power, and general and administration (G&A) costs. Table 1-4 provides a summary of the project operating costs.

The overall LOM operating cost is US$670.3 M over 11.2 years, or an average of US$21.53/t of material milled in a typical year.

Table 1-3: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	LOM Sustaining Capital Cost (US$M)	Expansion Cost (US$M)	LOM Total Capital Cost (US$M)
1000	Mine	41.3	60.4	2.1	103.8
3000	Process plant	67.0	1.7	3.5	72.2
4000	Tailings	10.2	37.9	-	48.1
5000	On-site infrastructure	18.5	0.2	-	18.7
6000	Off-site infrastructure	2.3	-	-	2.3
	Total Directs	**139.2**	**100.2**	**5.6**	**245.1**
7100	Field indirects	6.4	-	0.3	6.6
7200	Project delivery	11.8	-	0.4	12.2
7500	Spares and first fills	2.9	1.0	0.2	4.1
8000	Owner's cost	3.7	-	-	3.7
	Total Indirects	**24.7**	**1.0**	**0.9**	**26.6**
9000	Contingency	26.2	-	1.1	27.3
	Project Total	**190.2**	**101.2**	**7.6**	**298.9**

Note: Totals may not sum due to rounding.

Table 1-4: Operating Cost Summary

Cost Area	LOM Cost (US$M)	LOM Annual Cost (US$M)	LOM Unit Cost (US$/t milled)
Mining	355.8	31.7	11.43
Process	281.2	25.0	9.03
G&A	33.2	3.0	1.07
Total	**670.3**	**59.7**	**21.53**

Note: Totals may not sum due to rounding.

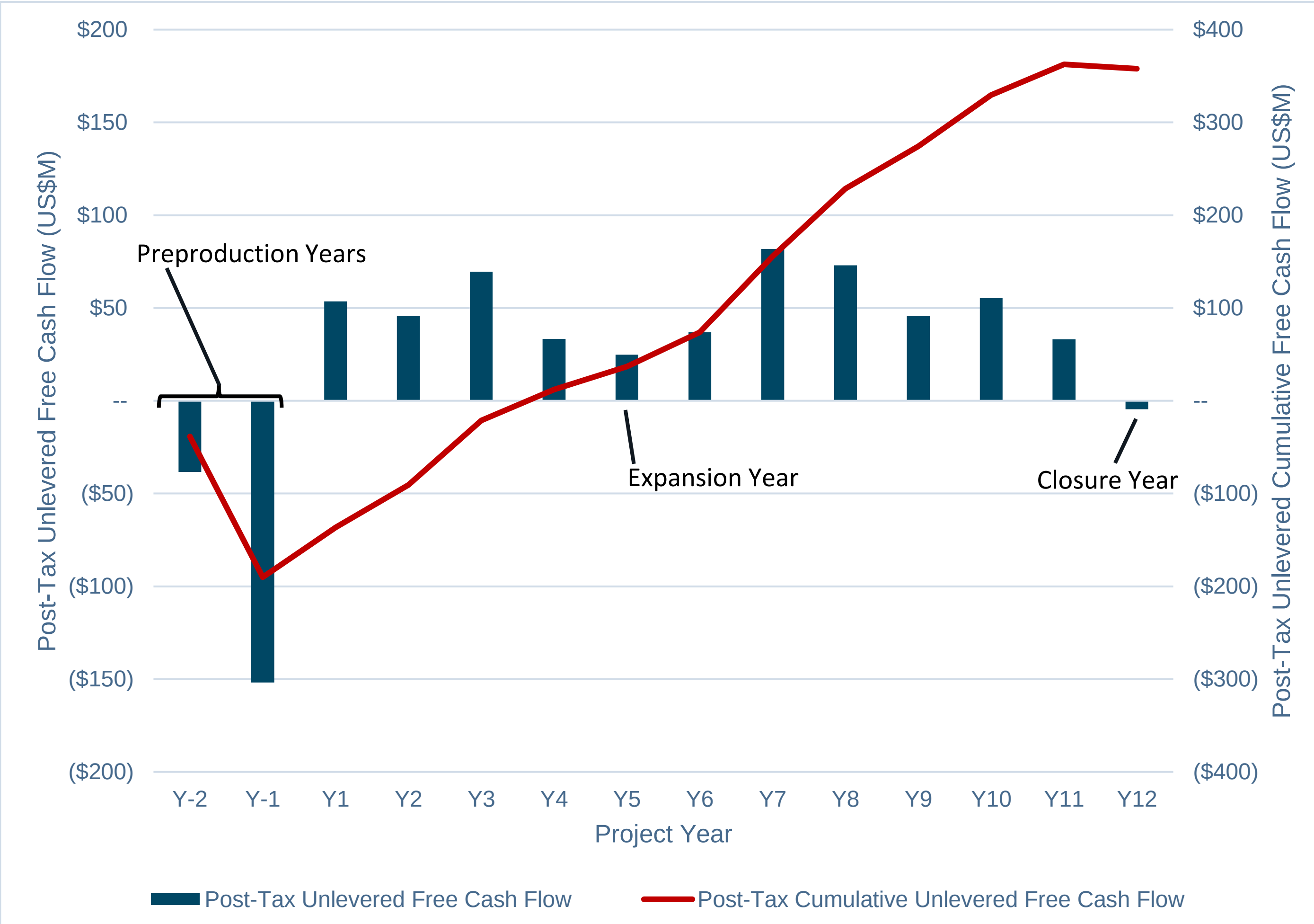
1.20 Economic Analysis

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

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 US$297 M; the IRR is 26.9%; and payback period is 3.3 years. On a post-tax basis, the NPV discounted at 5% is US$212.4 M; the IRR is 22.8%; and payback period is 3.6 years. A summary of the post-tax project economics is shown graphically in Figure 1-4 and listed in Table 1-5.

Figure 1-4: Projected LOM Post-Tax Unlevered Free Cash Flow



Source: Ausenco, 2023.

Table 1-5: **Economic Analysis Summary**

General	Unit	LOM Total/Avg.
Gold price	US$/oz	1,750
Mine life	years	11.2
Total waste tonnes mined	kt	121,903
Total mill feed tonnes	kt	31,128
Strip ratio	waste:mineralized rock	3.9
Production	**Unit**	**LOM Total/Avg.**
Mill head grade	g/t	0.88
Mill recovery rate	%	96.7
Total payable mill ounces recovered	koz	851.9
Total average annual payable production	koz	75.9
Operating Costs	**Unit**	**LOM Total/Avg.**
Mining cost (incl.rehandle)	US$/t mined	2.51
Mining cost (incl.rehandle)	US$/t milled	11.43
Processing cost	US$/t milled	9.03
General and administrative cost	US$/t milled	1.07
Total operating costs	US$/t milled	21.53
Treatment and refining cost	US$/oz	4.30
Net smelter royalty	%	1.0
Cash costs1	US$/oz Au	809
All-in sustaining costs2	US$/oz Au	957
Capital Costs	**Unit**	**LOM Total/Avg.**
Initial capital	US$M	190
Expansion capital	US$M	8
Sustaining capital	US$M	101
Closure costs	US$M	30
Salvage value	US$M	12
Financials – Pre-Tax	**Unit**	**LOM Total/Avg.**
Net present value (5%)	US$M	297
Internal rate of return	%	26.9
Payback	years	3.3
Financials – Post-Tax	**Unit**	**LOM Total/Avg.**
Net present value (5%)	US$M	212
Internal rate of return	%	22.8
Payback	years	3.6

Notes:
1. Cash costs consist of mining costs, processing costs, mine-level G&A, and treatment and refining charges, and royalties.
2. All-in sustaining costs include cash costs plus expansion capital, sustaining capital, closure costs, and salvage value.

1.20.1 Sensitivity Analysis

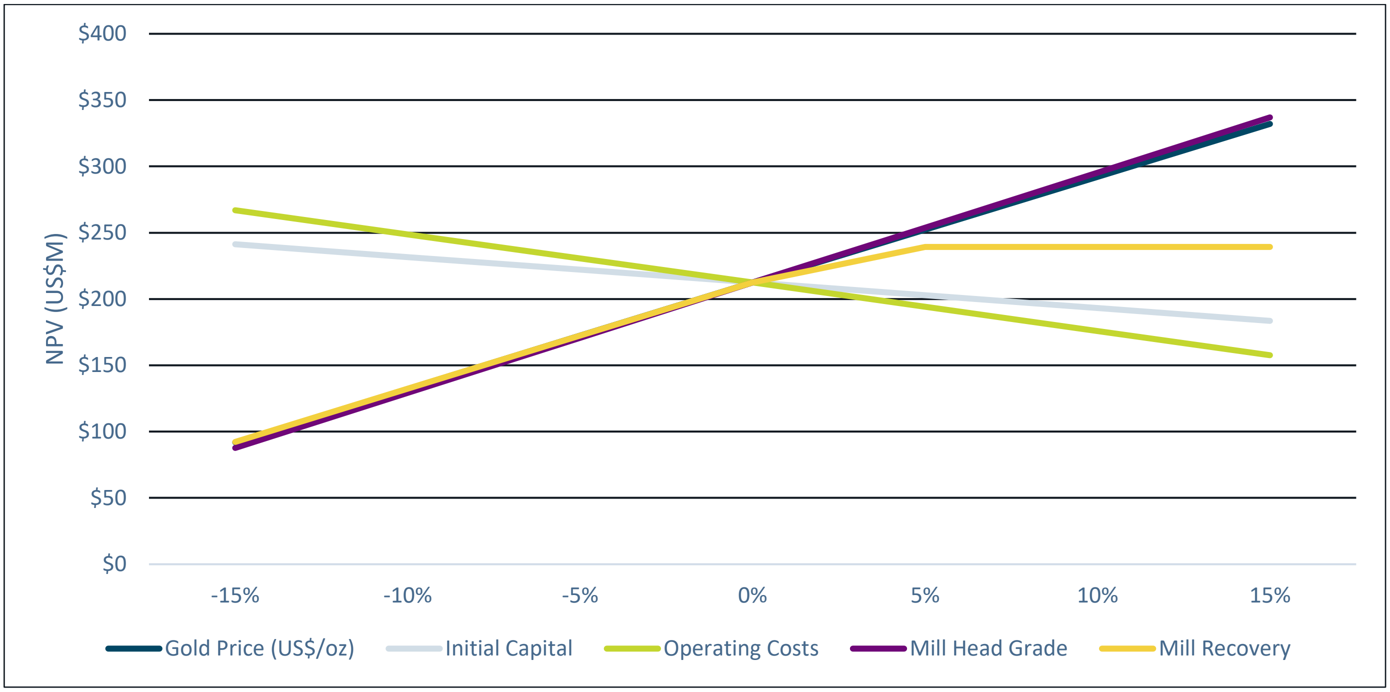
A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using the following variables: gold price, discount rate, operating costs, initial capital costs, mill recoveries, and mill head grades. Table 1-6 summarizes the post-tax sensitivity analysis results.

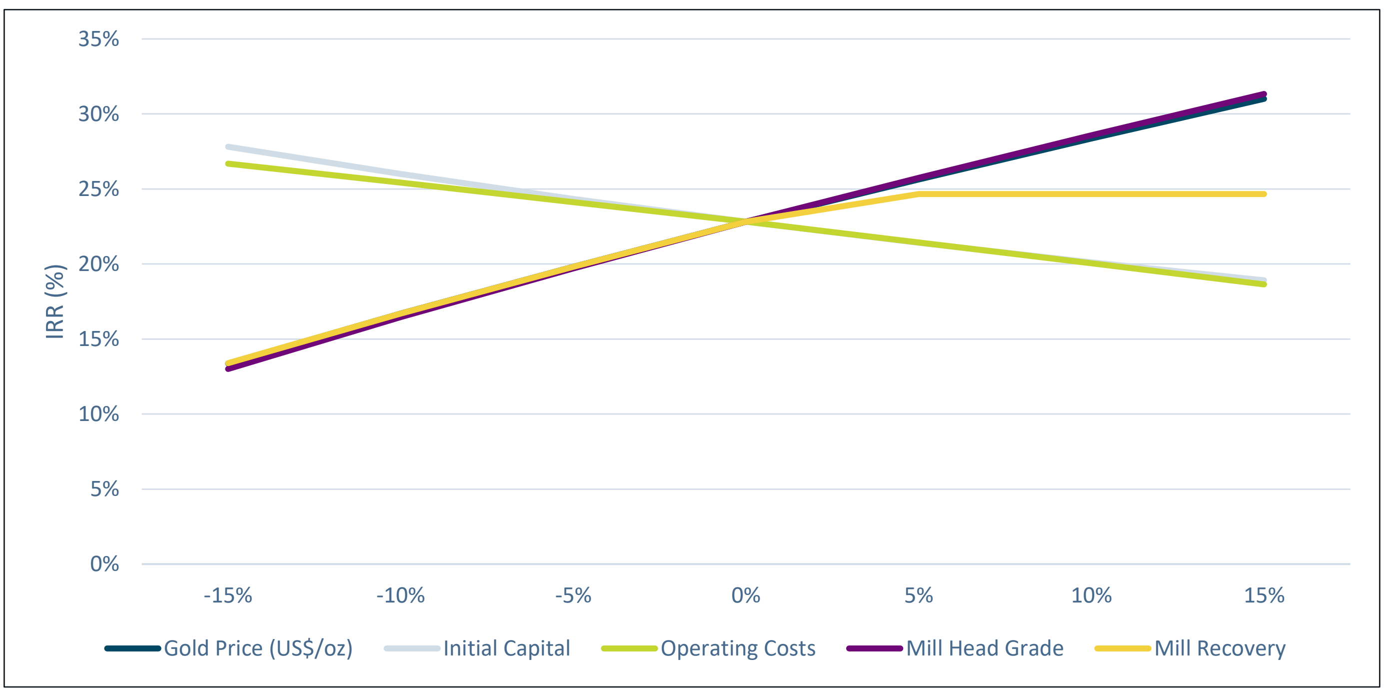
Table 1-6: Post-Tax Sensitivity Analysis

	Post-Tax NPV (US$M) Sensitivity to Discount Rate					
	Gold Price (US$/oz)					
Discount Rate		$1,450	$1,600	$1,750	$1,900	$2,050
	1.0%	145	234	323	412	501
	3.0%	106	185	263	340	418
	5.0%	74	144	212	281	349
	8.0%	36	95	152	209	266
	10.0%	16	68	120	170	221
	Post-Tax NPV (US$M) Sensitivity to Operating Costs					
	Gold Price (US$/oz)					
Operating Costs		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	148	217	285	353	422
	(10.0%)	111	180	249	317	385
	--	74	144	212	281	349
	10.0%	37	107	176	244	313
	20.0%	(1)	70	139	208	276
	Post-Tax NPV (US$M) Sensitivity to Initial Capital					
	Gold Price (US$/oz)					
Initial Capital		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	113	182	251	319	388
	(10.0%)	94	163	232	300	368
	--	74	144	212	281	349
	10.0%	55	124	193	262	330
	20.0%	36	105	174	242	311
	Post-Tax NPV (US$M) Sensitivity to Recovery Mill					
	Gold Price (US$/oz)					
Recovery Mill		$1,450	$1,600	$1,750	$1,900	$2,050
	(2. 0%)	61	129	196	263	330
	(1. 0%)	68	136	204	272	340
	--	74	144	212	281	349
	1. 0%	81	151	220	289	358
	2. 0%	88	158	228	298	368
	Post-Tax NPV (US$M) Sensitivity to Head Grade					
	Gold Price (US$/oz)					
Head Grade		$1,450	$1,600	$1,750	$1,900	$2,050
	(2. 0%)	60	128	196	263	330
	(1. 0%)	67	136	204	272	339
	--	74	144	212	281	349
	1. 0%	82	151	221	290	359
	2. 0%	89	159	229	299	369

	Post-Tax IRR (%) Sensitivity to Discount Rate					
	Gold Price (US$/oz)					
Discount Rate		$1,450	$1,600	$1,750	$1,900	$2,050
	1.0%	11.9	17.6	22.8	27.6	32.1
	3.0%	11.9	17.6	22.8	27.6	32.1
	5.0%	11.9	17.6	22.8	27.6	32.1
	8.0%	11.9	17.6	22.8	27.6	32.1
	10.0%	11.9	17.6	22.8	27.6	32.1
	Post-Tax IRR (%) Sensitivity to Operating Costs					
	Gold Price (US$/oz)					
Operating Costs		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	18.0	23.2	27.9	32.5	36.8
	(10.0%)	15.0	20.4	25.4	30.1	34.5
	--	11.9	17.6	22.8	27.6	32.1
	10.0%	8.5	14.6	20.1	25.1	29.7
	20.0%	4.9	11.4	17.2	22.4	27.2
	Post-Tax IRR (%) Sensitivity to Initial Capital					
	Gold Price (US$/oz)					
Initial Capital		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	17.1	23.8	29.8	35.4	40.7
	(10.0%)	14. 3	20.4	26. 0	31. 1	36. 0
	--	11. 9	17. 6	22. 8	27. 6	32. 1
	10.0%	9. 8	15. 2	20.1	24. 6	28. 9
	20.0%	7. 9	13. 1	17. 8	22. 1	26. 1
	Post-Tax IRR (%) Sensitivity to Recovery Mill					
	Gold Price (US$/oz)					
Recovery Mill		$1,450	$1,600	$1,750	$1,900	$2,050
	(2. 0%)	10.7	16. 4	21. 6	26. 4	30.9
	(1. 0%)	11. 3	17. 0	22. 2	27. 0	31. 5
	--	11. 9	17. 6	22. 8	27. 6	32. 1
	1. 0%	12. 4	18. 2	23. 4	28. 2	32. 7
	2. 0%	13. 0	18. 7	24. 0	28. 8	33. 3
	Post-Tax IRR (%) Sensitivity to Head Grade					
	Gold Price (US$/oz)					
Head Grade		$1,450	$1,600	$1,750	$1,900	$2,050
	(2. 0%)	10. 6	16. 4	21. 6	26. 4	30.9
	(1. 0%)	11. 2	17. 0	22. 2	27. 0	31. 5
	--	11. 9	17. 6	22. 8	27. 6	32. 1
	1. 0%	12. 5	18. 2	23. 4	28. 2	32. 8
	2. 0%	13. 1	18. 8	24. 0	28. 8	33. 4

Figure 1-5: Post-Tax NPV, IRR Sensitivity Results





Source: Ausenco, 2023.

1.21 Conclusions and Interpretations

The mineral resource estimate (MRE) is comprised of combined measured and indicated mineral resource of 30.022 Mt at 1.00 g/t Au for 988,100 oz of gold, and an inferred mineral resource of 7.634 Mt at 1.04 g/t Au for 256,000 oz of gold (Table 1-1). The MRE covers a surface area of 1320 m by 740 m and extends down to a depth of 240 m.

The Lemhi Property is amenable to conventional truck and shovel open pit mining. Mining operations are planned to feed mineralized material (averaging 0.88 g/t Au) for processing over an 11.2-year project life. Initial operations will process 2.5 Mt/a, and in Year 5, will expand throughput to 3.0 Mt/a is targeted. Based on the assumptions and parameters in this report, the PEA shows positive economics (i.e., US$212.2 M post-tax NPV (5%) and 22.8%, post-tax IRR). The PEA supports a decision to carry out additional studies to further progress the project.

1.22 Recommendations

The Lemhi Project demonstrates positive economics, as shown by the results presented in this technical report. Continuing to develop the project through to pre-feasibility study is recommended. Table 1-7 summarizes the proposed budget to advance the project through the pre-feasibility stage.

Table 1-7: Cost Summary for the Recommended Future Work

Item	Budget (US$M)
Exploration and drilling	4.00
Metallurgical testwork	0.15
Mining methods	2.20
Process and infrastructure engineering	0.80
Site-wide assessment and CPSF geotechnical studies	0.96
Environmental, permitting, social and community recommendations	0.99
Total	**9.10**

Note: Totals may not sum due to rounding

2 INTRODUCTION

2.1 Introduction

Freeman Gold Corp. (Freeman Gold) commissioned Ausenco Engineering Canada ULC. to compile a preliminary economic assessment (PEA) of the Lemhi Gold Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco Engineering Canada ULC. and Ausenco Sustainability ULC. (collectively, “Ausenco”) and managed and coordinated the work related to the report. Ausenco developed the PEA-level design and cost estimate for the process plant, general site infrastructure, site water management infrastructure, tailings facility and environmental studies and permitting. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- Moose Mountain Technical Services (MMTS) prepared the mine plan, including open pit designs, the mine productions schedule, and mine capital and operating cost estimates.
- APEX Geoscience Ltd. (APEX) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, and mineral resource estimate.

2.2 Terms of Reference

The purpose of this report is to present the results of the PEA and to support the disclosures by Freeman Gold in a news release dated October 16, 2023 and titled “Freeman announces robust maiden preliminary economic assessment for Lehmi: After tax NPV of US$212 million”.

All measurement units used in this technical report are metric unless otherwise noted. Currency is expressed in United States dollars (US$). This technical report uses English.

Mineral resources are estimated in accordance with the 2019 edition of the Canadian Institute of Mining, Metallurgy and Exploration (CIM) Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (2019 CIM Best Practice Guidelines) and are reported using the 2014 CIM Definition Standards for Mineral Resources and Mineral Reserves (2014 CIM Definition Standards).

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

2.3 Qualified Persons

The individuals presented in Table 2-1 serve as the qualified persons for this technical report as defined in National Instrument 43-101, Standards of Disclosure for Mineral Projects, and in compliance with Form 43-101.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Freeman Gold Corp.	Report Section
Kevin Murray	P. Eng.	Manager – Process Engineering	Ausenco Engineering Canda ULC.	Yes	1.1, 1.15, 1.16.1-1.16.8, 1.17, 1.19-1.22, 2, 3.3, 17, 18.1-18.3, 19, 21.1, 21.2.1, 21.2.2, 21.2.4-21.2.9, 21.2.10.1, 21.2.10.3, 21.2.11, 21.3.1, 21.3.3, 21.3.4, 22, 23, 24, 25.1, 25.8, 25.9, 25.11, 25.13 - 25.15, 25.16.4, 25.16.5.1, 25.16.7, 25.17.3, 26.1, 26.5, and 27
Scott C. Elfen	P. E.	Global Lead Geotechnical Services	Ausenco Engineering Canda ULC.	Yes	1.16.9, 18.4, 25.10, 25.16.5.2, 25.16.6, 26.6, and 27
Peter Mehrfert	P. Eng.	Principal Process Engineer	Ausenco Engineering Canada ULC.	Yes	1.12, 13, 25.5, 25.16.1, 25.17.1, 26.3, and 27
James Millard	P. Geo.	Director, Strategic Projects	Ausenco Sustainability ULC	Yes	1.18, 3.2, 4.3, 20, 25.12, 25.16.8, 25.17.4, 26.7, and 27
Jonathan Cooper	P. Eng	Water Resources Engineer	Ausenco Sustainability ULC.	Yes	1.16.10, 18.5, 27
Marc Schulte	P. Eng.	Vice President – Mine Engineering	Moose Mountain Technical Services	Yes	1.14, 15, 16, 21.2.3, 21.2.10.2, 21.3.2, 25.7, 25.16.3, 26.4, and 27
Michael Dufresne	P. Geol., P. Geo.	President	APEX Geoscience Ltd.	Yes	1.2-1.11, 1.13, 3.1, 4.1, 4.2, 5-12, 14, 25.2-25.4, 25.6, 25.16.2, 25.17.2, 26.2, and 27

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Site Inspection by Scott C. Elfen, P. E.

Scott C. Elfen visited the property on November 3, 2022 and was able to review the general topography of the project site and site access options.

2.4.2 Site Inspection by Michael Dufresne, P. Geol., P. Geo.

Mr. Dufresne visited the property on November 8 and 9, 2019 and from September 10 to 17, 2020. During the 2020 visit, Mr. Dufresne confirmed the locations of several historical collars on the property, assisted in planning of the 2020 program and reviewed core from the first couple of drill holes. The author also conducted a site visit to Freeman's core facility on February 26, 2021, and observed and reviewed a number of the gold-bearing core intersections from the 2020 drilling program. The most recent site visit was conducted on February 18, 2022, where the author visited two active drill pads, viewed core being quick-logged by the on-site geologist and visited the Beauty Zone mineral occurrence showing and several Beauty drill pads.

2.5 Effective Dates

- The effective date of the mineral resource estimate is March 15, 2023.
- The effective date of the overall report is October 13, 2023.

2.6 Information Sources and References

Reports and documents listed in Section 3 and Section 27 of hthis technical report were used to support preparation of the technical report. The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. The sources of information include historical data and reports compiled by previous consultants and researchers of the project and supplied by Freeman Gold personnel, as well as other documents cited throughout the report and referenced in Section 27 previously completed reports filed on System for Electronic Document Analysis and Retrieval (SEDAR) by previous owners. The QP's opinions contained herein are based on information provided to the QPs by Freeman Gold throughout the course of the investigations.

The QPs have relied on Freeman Gold's internal experts and legal counsel for details on project history, regional geology, geological interpretations, and information related to ownership and environmental permitting status.

This report has been prepared using the documents noted in Section 27 References. The QPs used their experience to determine if the information from previous reports was suitable for inclusion in this technical report and adjusted information that required amending. This report includes technical information that required subsequent calculations to derive subtotals, totals, and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, the QPs do not consider them to be material.

2.6.1 Previous Technical Reports

The Lemhi Gold Project has been the subject of a previous technical report, as follows:

- NI 43-101 Maiden Resource Technical Report for the Lemhi Gold Project, Lemhi County, Idaho, USA. APEX Geoscience Ltd, Effective Date: June 1, 2021.

2.6.2 Definitions

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	Atomic absorption
AACE	Association for the Advancement of Cost Engineering International
AAS	Atomic absorption spectroscopy
AB	Alberta, Canada
AD&M	The American Development, Mining and Reduction Company
AGR	American Gold Resources
Ai	Abrasion value
ALS	ALS Geochemistry Vancouver
APEX	APEX Geoscience Ltd
ARD	Acid rock drainage
Au	gold
B.C.	British Columbia, Canada
BF	Block factor
BHLK	BHLK2, LLC
BLM	Bureau of Land Management
BM	Ball mill
BO	Biological Opinion
CAPEX	Capital expenditure
CGP	Construction General Permit
CDN	CDN Resource Laboratories Ltd
CIL	Carbon-in-leach
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CN	Cyanide
CPSF	Co-placement storage facility
CRM	Certified reference materials
CSAMT	Controlled Source Audio-Frequency Magnetotelluric
CWTF	Comingled waste and tailings facilities
DHDB	Drill hole database
DTH	Down the hole
DWT	Drop weight
E	East
EDA	Exploratory data analysis
EDGM	Earthquake design ground motion

Abbreviation	Description
EIS	Environmental impact statement
EPA	Environmental Protection Agency
ESA	Endangered Species Act
FMC	FMC Gold Company
FOS	Factors of safety
G&A	General and administration
GME	General mine expense
GPS	Global positioning system
GNSS	Global navigation satellite system
GRG	Gravity recoverable gold
HDPE	High density polyethylene
HDR	HDR Engineering
IBC	Intermediate bulk containers
ICP-AES	Inductively coupled plasma atomic emission spectroscopy
IDAPA	Idaho Administrative Procedures Act
IDEQ	Idaho Department of Environmental Quality
IDF	Inflow design flood
IDL	Idaho Department of Lands
IDWR	Idaho Department of Water Resources
IL	Ionic leach
IMC	Independent mining consultants
IP	Induced polarization
IPaC	Information for planning and consultation
IPDES	Idaho Pollutant Discharge Elimination System
IR	Insufficient recovery
IRR	Internal rate of return
ISGC	Idaho State Gold Company
KCA	Kappes, Cassiday and Associates
LGT	Lemhi Gold Trust
LiDAR	Light detection and ranging
LLDPE	Liner low density polyethylene
LOM	Life of mine
LVA	Locally varying anisotropy
MAG	Ground magnetic survey
MCC	Motor control centres
MDL	Method detection limit
MLRS	Mineral and land records system

Abbreviation	Description
MMI	Mobile metal ion
MMTS	Moose Mountain Technical Services
MPEL	Mineral Processing and Environmental Laboratories Inc.
MPO	Mine plan of operations
MRE	Mineral resource estimate
MSGP	Multi-Sector General Permit
MSHA	Mine Safety and Health Administration
MTO	Material take-off
MVI	Magnetic vector inversion
N	north
NaCN	Sodium Cyanide
NEPA	National Environmental Policy Act
NPDES	National Pollutant Discharge Elimination System
NPV	Net present value
NOAA	National Oceanic and Atmospheric Administration
NSP	Net smelter price
NSR	Net smelter return royalty
NW	northwest
NWI	National Wetland Inventory
OPEX	Operating expenditure
PAG	Potential acid generation
PAH	Pincock, Allen and Holt
PAX	Potassium amyl xanthate
PEA	Preliminary economic assessment
PF	Price factor
PFS	Pre-feasibility study
POC	Point of compliance
QA/QC	Quality assurance / quality control
QEMSCAN	Quantitative evaluation of materials by scanning electron microscopy
QP	Qualified person
QQ	Quantile to quantile
RC	Reverse-circulation
RCE	Reclamation cost estimate
RCP	Reclamation and closure plan
RM	Rod mill
RMSP	Resource Modelling Solutions Platform
ROD	Record of Decision

Abbreviation	Description
ROM	Run-of-mine
S	south
SAG	Semi-autogenous grinding
SCSE	SAG circuit specific rnergy
SDS	Safety data sheet
SEDAR	System for electronic document analysis and retrieval
SG	Specific gravity
SGS	SGS Mineral Services Inc. Burnaby
SMC	SAG mill comminution
SMU	Selective mining unit
SPCC	Spill prevention, control, and countermeasure
TDIP	Time domain induced polarization
TSF	Tailings storage facility
UCF	undiscounted cashflow
UPS	Uninterrupted power supply
USFS	United States Forest Service
USFWS	United States Fish and Wildlife Service
USGS	United States Geological Survey
UTM	Universal Transverse Mercator
VFD	Variable frequency drives
VWP	Vibrating wire piezometers
W	West
WAD	Weak acid dissociable
Wi	Bond work index
WRCC	Western Regional Climate Centers
WRD	Waste rock dump
WRSF	Waste rock storage facilities

Table 2-3: Units of Measurement

Abbreviation	Description
3D	Three-dimensional
°C	degrees Celsius
C$	Canadian dollars
US$	United States dollars
BV/h	Bed volume per hour

Abbreviation	Description
cm	centimetre
%	Percent
%w/w	Dry weight concentration of a solution
µ	micron
µm	micrometre
ft	feet
ft^3/s	cubic feet per second
g	gram
gal	gallon
g/cm^3	grams per centimetre cubed
g/t	grams per tonne
h	hour
ha	hectare
HP	horsepower
kg	kilogram
km	kilometre
koz	thousand ounces
kt/d	thousand tonnes per day
kV	kilovolt
kWh	kilowatt hour
kWh/m^3	kilowatt hour per metre cubed
kWh/t	kilowatt hour per metric tonne
L/s	litre per second
M	million
m	metre
m^2	square metre
m^3	cubic metre
M^3/s	cubic metres per second
masl	metres above sea level
mamsl	metres above mean sea level
mg/L	milligrams per liter
mm	millimetres
Mt	million tonnes
Mt/a	million tonnes per annum
mV/V	millivolts per volt
MW	Megawatt
MWh	Megawatt hour

Abbreviation	Description
oz	ounce
oz/t	ounces per metric tonne
P_{80}	passing grind size
ppm	parts per million
ppb	parts per billion
s	second
t	metric tonne
t/d	tonnes per day
t/m²/h	tonnes per metre squared per hour
X	times

3 RELIANCE ON OTHER EXPERTS

3.1 Property Agreements, Mineral Tenure, Surface Rights and Royalties

The qualified persons (QPs) are not qualified to provide an opinion or comment on issues related to legal agreements, royalties, permitting, or environmental matters. Accordingly, the authors of this technical report disclaim portions of the report particularly in Section 4, Property Description and Location. The QPs have relied upon the following reports by other experts, which provided information regarding mineral rights, surface rights, property agreements, royalties, environmental, permitting, social licence, closure, taxation, and marketing for sections of this report:

- The QPs relied entirely on background information and details regarding the nature and extent of Lower 48's Land Titles (in Section 4.1) provided by Freeman. The legal and survey validation of the claims is not in the author's expertise and the QP's have relied on Freeman's land-persons and legal team to provide the relevant information.
- A title opinion was provided by Freeman from Christopher Healy of Healy Law PLLC and is dated July 28, 2020. Bureau of Land Management (BLM) Customer Information Reports were provided by Freeman. In addition, the QPs have confirmed the unpatented mineral claims are in good standing as of October 17th, 2023, using the Mineral and Land Records System (MLRS) register and have no reason to question the validity or good standing of the claims.
- The QPs relied entirely on information regarding the agreements of acquired Vineyard Gulch Resources claims provided by Freeman. See company news release dated September 15, 2020 and titled, "Freeman Gold further consolidates land package within historical resource area of the Lehmi Gold Project."
- The QPs relied entirely on information regarding royalties and back-in agreements provided by LGT, Lower 48 and Freeman, including a title opinion by Healy Law PLLC dated Jul 28, 2020, an Option Agreement between BHLK-2 LLC and 1132144 BC Ltd. dated August 31, 2019, a Deed of Royalty, Humbug Mine dated September 22, 2011.
- The QPs relied entirely on information regarding permitting and environmental status of the project provided by LGT, Lower 48 and Freeman. See company news releases dated May 31, 2022 and titled, "Freeman Gold receives approval of plan of operations and provides Lemhi Gold Project update," and June 7, 2022 and titled "Freeman Gold awarded mining water rights for Lemhi."
- This information was relied upon in Sections 1.2, 1.3, 4 and 25.2.

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

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Freeman Gold and experts retained by Freeman Gold for information related to environment, permitting, closure planning and related cost estimation, and social and community impacts as follows:

- Hart Crowser. 1995. ARD Potential of Humbug Project Rocks. Prepared for American Gold Resources Corporation.

- Hart Crowser. 1996. 1995 Baseline Monitoring Report, Surface Water and Groundwater. Prepared for American Gold Resources Corporation.
- Hart Crowser. 1996. 1996 Baseline Monitoring Data Technical Memorandum. Prepared for American Gold Resources Corporation.
- HDR Engineering. 2012. Final Terrestrial Vegetation Report, Lemhi Gold Trust Exploration Project. Prepared for Lemhi Gold Trust, LLC.
- HDR Engineering. 2012. Draft Wetland Delineation Report, Lemhi Gold Trust Exploration Project. Prepared for Lemhi Gold Trust, LLC.
- Karen Kuzis Consulting. 1995. Ditch Creek Baseline Fish Population and Habitat Surveys. Prepared for American Gold Resources Corporation.
- Selkirk Environmental. 1996. Jurisdictional Wetland Determination for Humbug Gold Project. Prepared for American Gold Resources Corporation.

This information was relied upon in Sections 1.18, 20, and 25.12.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by Freeman Gold relating to the tax model used in the economic analysis, according to the file “107121-01 Lehmi Financial Model_2023-08-28.xlsx” received via email on September 6, 2023. The tax model was compiled by Freeman Gold, assuming a blended corporate tax rate of 25% to reflect federal and Idaho state taxes.

This information was relied upon in Sections 1.20, 22, and 25.15.

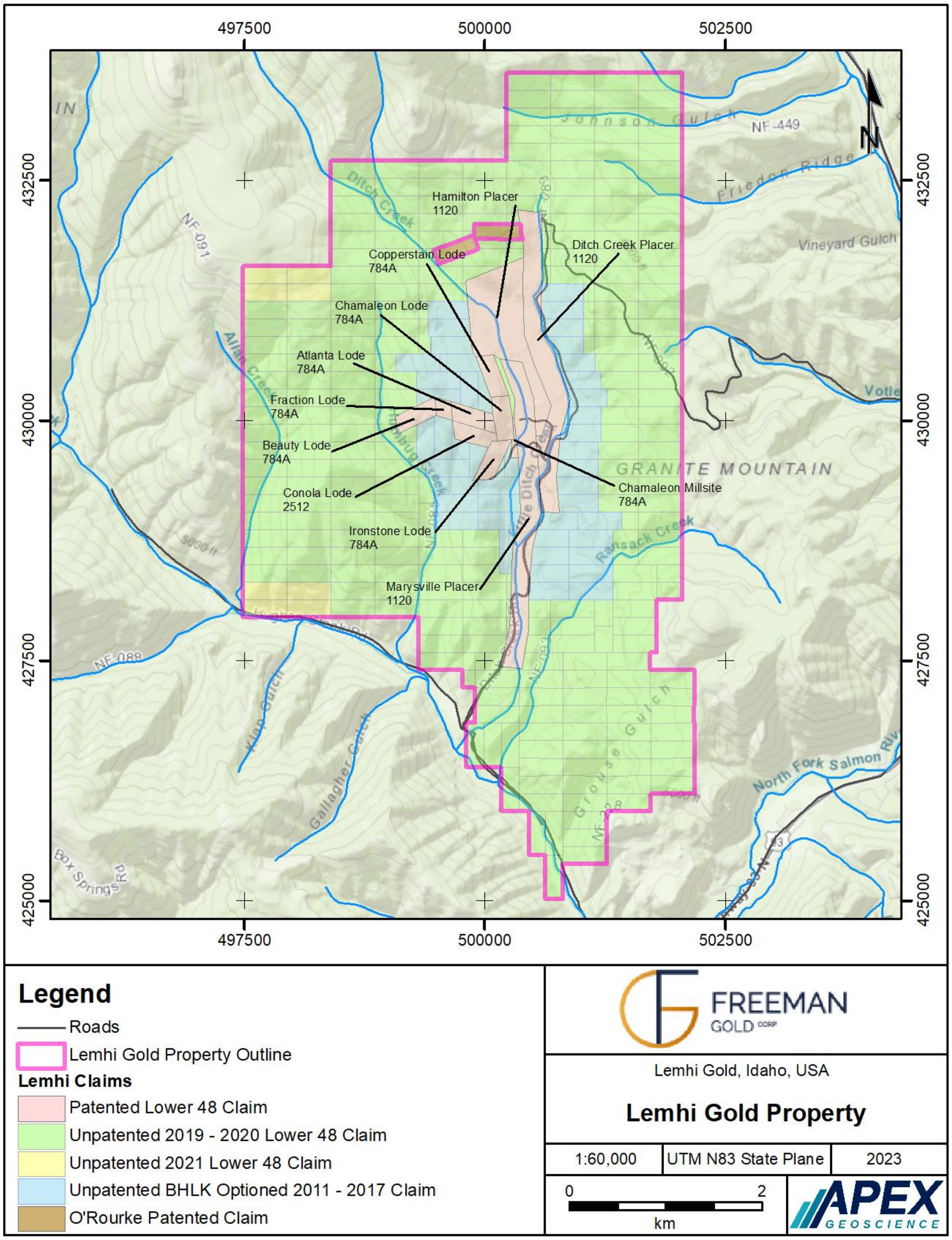
4 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

The project is located in Lemhi County in east-central Idaho, within the Salmon River Mountains, a part of the Bitterroot Range which forms the Idaho-Montana border (Figure 4-1).The property is 40 km north of the town of Salmon, ID and 6 km west of Gibbonsville, ID. The approximate center of the property in Universal Transverse Mercator (UTM Zone 11T) NAD83 Idaho State Plane coordinates is Easting 500,275, Northing 429,900.

The project consists of 10 patented mining claims (placer and lode), one patented mill site claim and 332 unpatented mining claims, totalling 2,727 ha of mineral rights and 249 ha of surface rights (Figure 4-1 and Appendix 1). The 11 patented mining claims and 53 nearby unpatented mining claims were previously owned by Lemhi Gold Trust, LLC (LGT), and were acquired by Lower 48, in a sealed bid auction in November 2019. Lower 48's parent company, 1132144 had previously signed an option with BHLK to purchase a 100% interest in 46 unpatented claims immediately adjacent to the patented claims on August 31, 2019. The option has since been transferred to Freeman and 1132144 has been extinguished. The remaining 231 unpatented mining claims were either staked or purchased by Lower 48 in 2019 to 2021. Freeman has since acquired 100% of the issued shares of Lower 48 for a total of 33,740,000 common shares of Freeman. In addition, in order to complete the transaction a finder's fee of 3,500,000 common shares was issued by Freeman to Sub C Holdings Ltd. Freeman Gold controls a 100% interest in all 11 patented claims and all 332 unpatented mining claims subject to certain cash payments over time for the 46 BHLK unpatented claims and royalties outright or through its wholly owned subsidiary company Lower 48 Resources (Idaho) LLC (Lower 48).

Figure 4-1: Lemhi Gold Project Claims



Source: APEX, 2023.

A total of 46 unpatented mining claims immediately adjacent to the patented mining claims are owned by BHLK and are under option (since August 31, 2019) to Freeman through a transfer from 1132144. Freeman has the right to purchase a 100% interest in the BHLK claims by making an aggregate of US$1 M in payments to BHLK over a 7-year period. Freeman has recently completed the required 5th year payment as of the effective date of this report. Freeman recently purchased outright the Moon #100 and Moon #101 unpatented mining claims (Moon Claims) from Vineyard Gulch Resources, LLC (Vineyard), located within the resource area. An additional 231 unpatented claims were staked by Freeman (Lower 48) in 2020 and 2021.

The 11 patented mining claims were recently purchased from LGT at auction by Lower 48 and have been transferred to Lower 48. Patented mining claims originated as unpatented mining claims and were converted to private ownership through the Patent and Mineral Survey process. The patented claims on the Lemhi Gold Property were patented between 1890 and 1910. Corner survey monuments are intact–of which several were observed by the author– and the USFS has placed markers delineating USFS land boundaries along the claim boundaries. In order to keep the claims in good standing, annual real estate taxes must be paid to Lemhi County. If the annual taxes are paid the patented claims will remain in good standing in perpetuity.

The 332 unpatented BLM federal lode claims are administered by the USFS. The following claims are ultimately owned by two entities (Freeman/Lower 48 and BHLK):

- 46 unpatented claims staked and owned by BHLK of Missoula, Montana in 2011 and 2017 and under option to purchase by Freeman through a transfer of the option from 1132144
- 53 claims staked by LGT in September 2019, purchased by Lower 48 at Auction in November 2019
- 223 claims staked by Lower 48 in April 2020 and eight claims staked by Lower 48 in April 2021
- Two claims, the Moon Claims, purchased by Lower 48 from Vineyard in 2020.

Lower 48 purchased at auction in November 2019, the 11 LGT patented claims and the 53 LGT unpatented mining claims. Additionally, 1132144 signed an option to purchase agreement on August 31, 2019 with BHLK for 46 unpatented mining claims. This option has subsequently been transferred to Freeman. The 53 LGT claims have been transferred to Lower 48. The 46 BHLK claims will be transferred to Lower 48 upon completion of the option. Any portion of an unpatented claims which overlaps a patented claim is deemed invalid. The valid portion of all unpatented claims totals 2,479 ha.

In October 2010, Vineyard of Salmon, Idaho staked two fractional claims on a narrow strip of USFS ground between the Proksch and Meridian Patented Claim groups. The two claims (Moon #100 and Moon #101) cover 3.4 ha of ground and are located toward the northern portion of the historical resource. In September 2020, Freeman purchased 100% ownership of the Moon claims via a purchase and sale agreement between Freeman (Lower 48) and Vineyard (the seller) of the Moon Claims. Freeman paid the seller cash consideration of US$150,000 and issued 375,000 common shares of Freeman to Vineyard. The transaction was not subject to a finder's fee or brokerage commission.

An additional two patented claims lie within the boundaries of the unpatented claims and are owned by John G O'Rourke of San Bruno California. These claims consist of 14.5 ha and are not part of Freeman's land package.

All information pertaining to the ownership and option agreements for ownership of the patented and unpatented mining claims was provided by Lower 48 and Freeman. The various agreements have been reviewed but have not been verified by the author.

The Mining Law of 1872 states that with respect to unpatented mining claims on federal lands, the locator has the right to explore, develop, and mine mineral mining claims. Surface rights are not included and remain the property of the United States government. No payment of production royalties to the United States Federal government is required. To maintain existing unpatented claims in good standing an annual maintenance fee of US$165 must be paid per claim to the BLM prior to September 1 of each year or the claims will be invalidated and will expire. New lode mining claims require a US$10 recording fee payable to the Country Courthouse of the relevant jurisdiction in which the claims are located. In addition, the BLM requires a further maintenance fee of US$165, a US$20 processing fee and a US$40 claim location fee. The total fee payable to BLM for recording a new claim is US$225 per claim. All 332 mineral claims were understood to be in good standing based on the information received from LGT, BHLK, Vineyard and Freeman. The status of the claims was checked against the BLM MLRS register database on October 17, 2023, and they are confirmed to be in good standing.

4.2 Royalties and Agreements

Lower 48 purchased at auction in November 2019, the 11 patented LGT mining claims, and the 53 unpatented LGT mining claims. The patented mining claims came with a couple of historical and active encumbrances in the form of royalties and a buy back clause. The 11 patented and 53 unpatented claims have been transferred to Lower 48. The royalties are still active; however, the buy-back clause has been purchased and extinguished.

BHLK obtained a 2% NSR royalty on all 11 patented mining claims and 74 surrounding unpatented mining claims through a deed of royalty upon LGT's purchase of the project in 2011. The deed of royalty details a 2-mile area of interest and is still active today. The 74 unpatented mining claims were optioned by LGT from BHLK in 2011 and cover the area currently represented by BHLK's 46 unpatented mining claims. Subsequently, 1132144 signed an option to purchase agreement on August 31, 2019 with BHLK for the 46 unpatented mining, which has subsequently been transferred to Freeman. Freeman may earn a 100% interest in the claims with cash payments totalling US$1.0M over seven years, at which time the BHLK 2% NSR will extend over most of the unpatented claims through the active deed of royalty. The Meridian group of patented mining claims consists of three placer and two lode patents; the Ditch Creek, Hamilton, Marysville, Canola, and Copperstain patented mining claims. Meridian Gold Inc. (Meridian) purchased the five patented claims from Ashanti Goldfields lnc. (Ashanti)in 1997. Ashanti (now AngloGold Ashanti Ltd.) retained a cash royalty of US$2.0 M, payable in full within 30 days after the first commercial production pour of doré gold or silver mined from any, or all, the 11 patented mining claims. At that time, the Proksch group of patents were under lease.

LGT purchased the Meridian group of five patented mining claims from Meridian (now a wholly owned subsidiary company of Yamana Gold Inc. (Yamana) in 2011 for a one-time payment of US$2.5 M. The purchase was subject to Ashanti's royalty and a 'back-in' whereby Meridian can 'back-in' to a 51% ownership of the Meridian group of five patented mining claims if and when the mineral reserve reaches 2.5 million mineable ounces of gold. This 'back-in'right was purchased outright by Freeman in September 2020 for 4,035,273 shares. These patented claims were recently purchased by Lower 48 at auction. Real estate taxes paid to Lemhi County annually for the Meridian group of patented mining claims total US$406.46.

The Proksch group of patented mining claims consists of five mining lode patents and one mill site patent and includes the Atlanta, Fraction, Ironstone and Chamaleon lode patents, along with the Chamaleon Millsite patent, shown in Table 4-1: The Proksch group of patented claims is subject to the Ashanti Cash Royalty. LGT purchased the Proksch group of patents from Joe and Hallie Proksch for US$2.5 M cash and did not include a royalty. These patented claims were recently purchased by Lower 48 at auction. The annual real estate taxes paid to Lemhi County total US$77.24. BHLK maintains a 2% royalty over the 11 patented claims.

Table 4-1: Patented Mining Claims Summary

Claim Group	Claim Names	Mineral Survey Number	Acres
Meridian (Yamana)	Ditch Creek Placer	MS 1120	477.75
	Hamilton Placer		
	Marysville Placer		
	Conola Lode	MS 2512	19.79
	Copperstain Lode	MS 784 A and B	20.66
Proksch	Beauty Consolidated Lode	MS 784 A and B	97.75
	Atlanta Lode		
	Fraction Lode		
	Chamaleon Lode		
	Chamaleon Millsite		
	Ironstone Lode		

Lower 48’s parent company, Freeman, has signed an option agreement with BHLK, in which Freeman has the option to acquire a 100% interest in the BHLK unpatented mining claims that surround the patented claims in consideration of payment of US$1.0 M paid over a seven-year period. The first 5 years of payments totalling $350,000 have been completed by Freeman. BHLK retains a 2% NSR on production from the BHLK unpatented claims and an area of interest of one mile of the outer boundary of the unpatented mining claims (excluding the patented mining claims). In addition, through the deed of royalty from the purchase of the LGT patented and unpatented claims, there is a two mile area of interest of the outer boundary of the original patented and unpatented claims purchased from LGT, which results in the 2% BHLK NSR covering most of the Lower 48 unpatented claims. Freeman has an option to buy down half the NSR for US$1.0 M at any time.

4.3 Environmental Liabilities, Permitting and Other Significant Factors

Enviroscientists Inc. conducted an environmental assessment on the project in 2008 and concluded there were no known environmental liabilities on the project (Cuffney, 2011; Brewer, 2019). Site inspections conducted by Cuffney during his 2011 property visit noted the presence of a small historical shaft located on the Conola claim which should be fenced or filled in to prevent inadvertent entry.

Section 20 of this report provides a summary of environmental conditions reported on the property and Section 25.12 provides a summary of key environmental considerations. Section 25.16.5 presents key environmental risks in relation to permitting schedule. The timely implementation of the recommendations presented in Section 26.7 will help to quantify, qualify, and potentially mitigate risks to the PFS stage of the project as well as future permitting and schedule.

Exploration and mining activities on private land, including patented mining claims. are regulated by the Idaho Department of Lands (IDL) and are subject to The Mined Land Reclamation Act of 1971. Exploration activity, including use of motorized earth moving equipment, requires that a notice of exploration be filed with the department within seven days of commencing operations. Holes and trenches must be closed with one year and reseeded. If exploration exceeds five contiguous or ten non-contiguous acres further approvals are required including a reclamation plan and bonding.

Permits to drill on federal land BLM mineral claims are administered by the BLM and USFS. For drilling on USFS land, a Notice of Intent or Plan of Operations must be submitted and accepted prior to disturbance. If the surface area disturbance is expected to be <5 acres, drilling and/or trenching can be conducted with a Notice of Intent (which can typically obtain within 60 to 180 days). For disturbances of >5 acres a Plan of Operations is required at which point reclamation bonds, archeological surveys and other requirements may be requested by USFS.

Freeman was recently granted a Permit to Appropriate Water (No, 75-15005), which allows for water rights for both potential future mining and domestic use in four sections within the company's patented mining claims. The permit allows for the use of 0.54 m^3/s of water from ground water sources for future processing in a gold operation and 24605 L/day for domestic use. The permit was obtained from the Idaho Department of Water Rights (IDWR). The usage rates are subject to change and the company can apply to amend (increase) the authorization if and when required as the Lemhi Project engineering and economic studies (Freeman News Release dated June 7, 2022).

The permit allows for ground water use in Township 26N, Range 21E sections 28, 29, 32, 33 of 15 L/s and a maximum of 24, 600 L/day 24605 L/day for domestic use. The permit is a preliminary order issued pursuant to Rule 730 of the IDWR's Rules of Procedure.

Freeman has also recently received an approval of a Plan of Operations (POO) application to the USDA-Forest Service (USFS), Salmon and Challis National Forests, North Fork Ranger District, submitted in September 2021. The plan was approved May 23, 2022, as POO-2021-081646 and allows for an expanded drill program with additional access on the unpatented BLM mineral claims (Freeman News Release May 31, 2022).

The authors are not aware of any environmental liabilities, or any other known material risks related to the Lemhi Property that may affect access, title, or the right or ability to perform work on the Lemhi Property. If the Company were to advance the Lemhi Property to pre-feasibility study, or feasibility study, the company may have to consider preparing a comprehensive EIS to ensure the project is considered in a careful and precautionary manner such that the project does not cause significant adverse environmental effects. With regard to potential environmental, permitting, and community/social risks, the company should consider the implementation of the recommendations presented in Section 26.7.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Accessibility

The project can be accessed by paved and gravel road from Salmon, Idaho by following US Highway 93 north for 34 km to North Fork and then an additional 7.4 km to the Hughes Creek Road (USFS Road 091). The property can be reached by traveling 3.2 km west along the Hughes Creek Road then another 3.1 km north along the Ditch Creek Road to a two-track road leading northwest to the Lemhi Gold Property. The Hughes Creek and Ditch Creek roads are public graded gravel roads maintained by USFS and/or Lemhi County and provide all-weather access to the project area.

Alternatively, the property can be accessed via the Granite Mountain Road (USFS Road 092), which heads west from Highway 93, 7.5 km north of the Hughes Creek Road. The Granite Mountain Road follows Votler Creek westward, then wraps around the south side of Granite Mountain and drops into the Little Ditch Creek drainage to intersect the Ditch Creek Road near the north end of the Lemhi Gold Property, 8 km from Highway 93. This road could provide a good access route for heavy equipment, supplies, and personnel in the summer months, but in its present condition would be unacceptable for winter travel due to the high altitude and lack of adequate berms.

5.2 Site Topography, Elevation and Vegetation

The project is located in the Salmon River Mountains within the Rocky Mountain physiographic province. The Bitterroot Range, which forms the border between Idaho and Montana lies to the east, across the Salmon River. The claims are centered on Ditch Creek, a south-draining tributary of Hughes Creek, which in turn flows into the North Fork of the Salmon River. The area is mountainous and characterized by steep slopes (30% to 100% grade) along Hughes Creek and Ditch Creek. Total relief is 500 m, with elevations ranging from 1,500 m to 2,000 m.

Pine forest, consisting of ponderosa pine, Douglas fir, and minorly of lodgepole pine, covers most of the project area. A fire-wise timber thinning program was conducted in 2013-2014. Riparian areas within the Ditch Creek drainage contain aspen and a few cottonwood trees. Mammals in the area include mule deer, elk, coyote, wolf, black bear, mountain lion, beaver, rabbits, and a variety of small rodents.

5.3 Climate

The climate is typical of the central Rocky Mountains. Summers (June-September) are generally warm with average daytime highs of 15°- 20° C and cool nights. Winter temperatures are cold with overnight lows often below -10° C. Annual precipitation is largely a function of elevation with Gibbonsville at 1355 masl receiving 34 cm, and Moose Creek at 1890 masl receiving 82 cm, mostly as snow between November and March (Carroll, 1996). Snowstorms are frequent, but access routes to the property can be kept open with minimal snow plowing.

5.4 Local Resources and Infrastructure

The town of Salmon has a population of 3,300 people. The economy of Salmon is based on ranching, forestry, mining, and tourism. Salmon is home to the regional offices of the USFS, BLM and Idaho Fish and Game (IDFG) as well as other state and federal agencies. Basic supplies are available, as are food and lodging. Steele Memorial hospital and medical clinic in Salmon provides basic medical needs, but the nearest hospital is in Dillon, Montana, 90 km north of Salmon.

The Lemhi County airport, located 8 km south of town, handles regularly scheduled commuter flights to and from Idaho Falls and Boise as well as charter flights. Salmon has historically provided both skilled and unskilled labor for the mining industry.

The patented mining claims at the Lemhi Gold Project provide adequate area for mine infrastructure. The placer claims of MS 1120 contain 193 ha of gently sloping private land suitable for mine offices, leach pads, a processing plant, and waste dumps. There is no power or other mining infrastructure on the Lemhi Gold Property. A 35.5 kV power line passes through the settlement of North Fork, 16 km by road from the property. Sufficient water for exploration is available from Ditch Creek, which has a good perennial water flow. Two of the patented mining claims carry water rights. Water wells would have to be drilled to provide sufficient water for mining and a processing plant.

Freeman was recently granted a Permit to Appropriate Water (No. 75-15005), which allows for water rights for both potential future mining and domestic use in four sections within the company's patented mining claims. The permit allows for the use of 0.54 m^3/s of water from ground water sources for future processing in a gold operation and 24605 L/d for domestic use. The permit was obtained from the Idaho Department of Water Rights (IDWR). The usage rates are subject to change and the company can apply to amend (increase) the authorization if and when required as the Lemhi Project engineering and economic studies (Freeman News Release dated June 7, 2022).

The permit allows for ground water use in Township 26N, Range 21E sections 28, 29, 32, 33 of 15. 3 L/s and a maximum of 24605 L/d for domestic use. The permit is a final order issued pursuant to Rule 730 of IDWR's Rules of Procedure.

The Lemhi area has a rich history of exploration and metallic mineral mining, the region has available sources of power, water, and mining personnel, and year-round access. Most exploration activities associated with fieldwork and drilling can likely be conducted year-round, although there may be periods in December to March, during which snow conditions may temporarily impede fieldwork. The authors do not see any significant obstacles that would prevent the potential development of a mine on the Lemhi Gold Property.

6 HISTORY

6.1 District and Early Property History

The Lemhi Gold Project is located within the Dahlonega (Gibbonsville) mining district in Idaho, USA.

In the Gibbonsville mining district, placer gold was first discovered in 1867 at Hughes Creek east of the town of Gibbonsville, followed by discoveries in the Dahlonega Creek and Andersen Creeks and the North Fork Salmon River. In 1877, gold-bearing quartz veins were discovered on the slopes of Dahlonega Creek (5 km east of the Lemhi Gold Project) and mining of lode gold deposits ensued (Johnson et al., 1998; Pierson, 2010). The American Development, Mining and Reduction Company (AD&M) purchased the Dahlonega Creek property and erected a 30-stamp mill with amalgamation and chlorination plants in 1895. A fire destroyed the main processing plant in 1907; a 20-stamp mill and cyanidation plant were built the following year. Production from 1901 to 1917 is reported to have been 4,481 oz Au and 755 oz Ag (Kiilsgaard et al., 1989). After a brief hiatus, lode gold mining resumed in the 1930s and continued, with interruptions until 1942. Placer mining continued on and off throughout this period up until 1948 (Kiilsgaard, et al., 1989). The total production of the Gibbonsville Mining district immediately east of the property up to 1913 is estimated at 100,000 oz Au (Johnson et al., 1998 and references therein; Cooper, 1988). The majority of this production was derived from the AD&M mine with reported production of 48,000 oz Au. Production from other notable mines, the Twin Brothers and Clara Morris, is reported at 14,500 oz and 12,000 oz Au, respectively. The remainder of the production was derived from smaller operations for which production values are unavailable (Kiilsgaard,et al., 1989). Figure 6-1 shows the locations of historical mines in the area.

In the Ditch Creek area, overlapping the current Lemhi Gold Project, placer and gold mining commenced in the 1890s, during which time several mining claims were located and patented (Pierson, 2010). Placer mining has been intermittently active in the area over a period of more than 100 years with extensive placer dredge tailings piles still visible today in Hughes Creek. In 1891, a group of six patented lode claims (MS 784A: Beauty Lode, Fraction Load, Atlanta Lode, Ironstone Lode, Chamaleon Lode, Copperstain Lode), was consolidated as the Bull of the Woods Mine. These six patented claims are part of the current Lemhi Gold Property. The Idaho Mining and Lumber Company acquired the Bull of the Woods Mine in 1908. A 100 ton/d stamp mill was built, and the mine produced an unknown amount of gold (Pierson, 2010). Extensive placer mining has been conducted along most of Hughes Creek and many of its tributaries, such as Ditch Creek, which drains north to south through the middle of the Lemhi Project area.

6.2 Modern Exploration History

The modern exploration history below is largely compiled and taken from reports prepared by Cuffney (2011) and Brewer (2019).

6.2.1 Ownership Information

Since the early 1900s, the Gibbonsville district has seen little modern exploration and mining activity until 1984, when FMC Gold Company (FMC) staked claims at Ditch Creek. After conducting regional grass-roots exploration programs in the area, FMC staked additional claims surrounding the Bull of the Woods property (patent claim: MS 784A). FMC leased and purchased some of the key patented claims and accumulated a land package of over 700 unpatented claims surrounding the patented mining claims in the area of the current Lemhi Gold Project. FMC also acquired the Beartrack property, located 48 km southwest of the Lemhi Gold Project.

FMC explored the property from 1984 until 1991, after which American Gold Resources Corporation (AGR) acquired the property and held it until 1996. After 1996, work on the property was limited due to numerous corporate takeovers and downturns in the mining sector. In 2011, LGT, a joint venture between Idaho State Gold Company (ISGC) and Northern Vertex, acquired the newly consolidated Lemhi Gold Project and commenced an aggressive exploration program. The historical LGT Property included the Lemhi (Humbug) Gold Deposit.

FMC explored the Lemhi Gold Property area (known at the time as the Ditch Creek Project, later renamed the Ponderosa Project) between 1984 and 1991. FMC's Ponderosa Project largely overlapped the current Lemhi Gold Project and extended up to Allan Creek west of the current project boundary. During that period FMC completed:

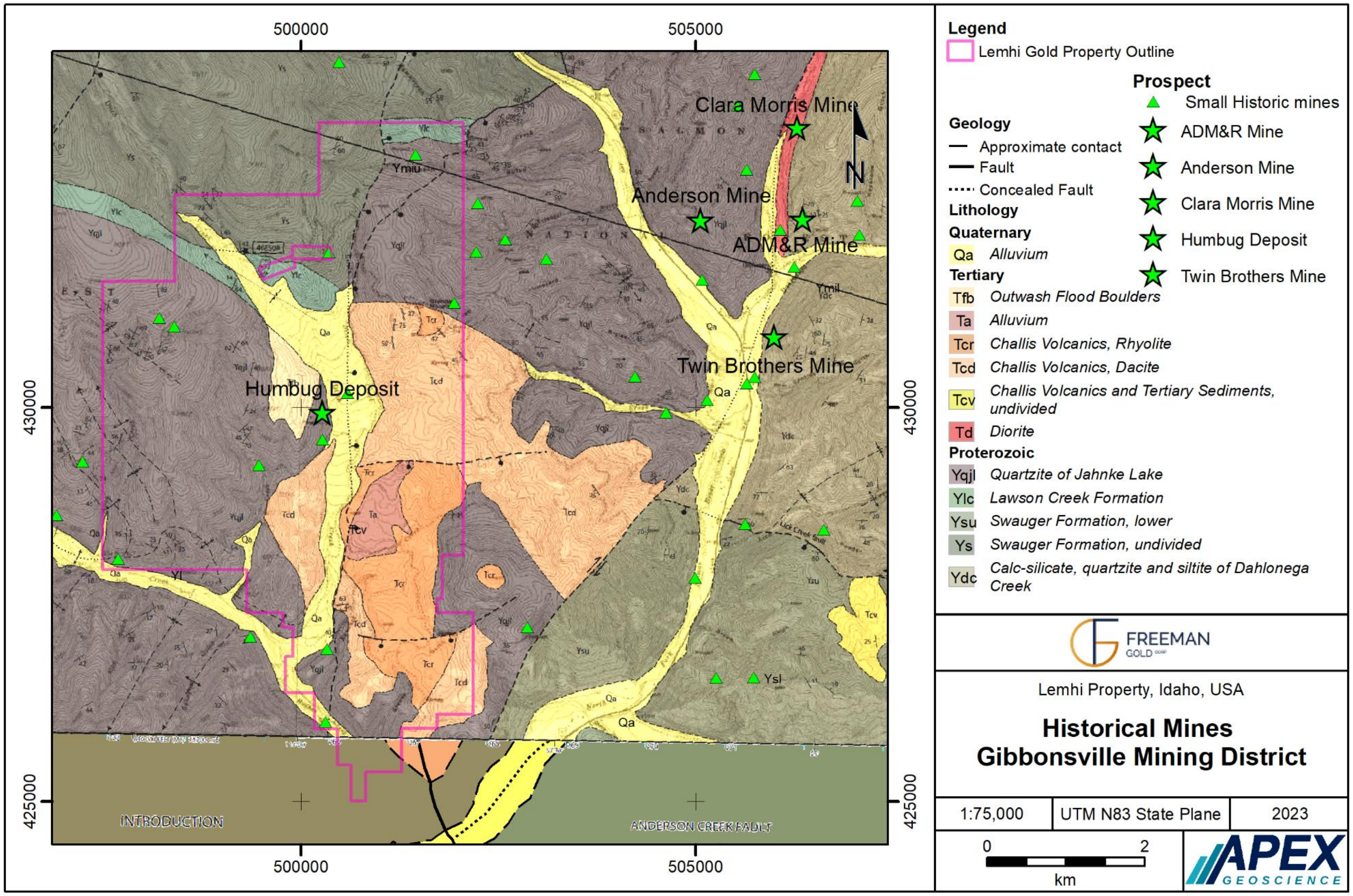
- geological mapping
- geochemical sampling (rock chip, soil, biogeochemical samples)
- geophysical surveying (airborne infrared, IP/resistivity, CSAMT, magnetics)
- trenching
- drilling: 192 RC holes, 177 of which are on the current property and four core holes
- metallurgical testing (cyanide leach tests, bottle roll and column leach tests)
- petrological studies
- deposit modelling and resource estimation.

In 1987, FMC decided to focus on development of their Beartrack deposit near Leesburg. No drilling was conducted in 1988, but drilling resumed in 1989. After completion of the 1989 drilling program, FMC decided to farm out the property. No joint venture (JV) agreements could be reached and in 1991 FMC abandoned the project and dropped the unpatented mining claims (Cuffney, 2011; Brewer, 2019).

In the fall of 1991, after FMC's unpatented claims lapsed, AGR located 94 unpatented claims, the Humbug claim group, to the west of the patented ground. AGR's Humbug Project largely overlapped the current Lemhi Gold Project and extended slightly west to cover Humbug Creek. AGR then consolidated the property by leasing and/or purchasing the patented claims in the area. Exploration work completed by AGR between 1991 and 1996 on the Humbug Property (overlapping the current Lehi Gold Project) included:

- drilling: 159 RC drill holes and nine core holes
- resource calculations (Pincock, Allen and Holt [PAH], 1996: Independent Mining Consultants [IMC], 1996)
- metallurgical testing (Kappes Cassiday, 1994-1996)
- pre-feasibility studies (Kappes Cassiday, 1995, 1996)
- scoping study for permitting
- baseline environmental studies.

Figure 6-1: Historical Mines in The Gibbonsville Mining District



Source: APEX, 2023.

In February 1995, AGR submitted a Conceptual Plan of Operations to the Bureau of Minerals of the Idaho Department of Lands. AGR planned to start an Environmental Impact Statement (EIS) the following year however they were acquired by Ashanti Goldfields Inc. (Ashanti) in May 1996. After the acquisition, Ashanti sold the AGR assets in the Salmon Idaho area, including the Humbug Project, to Meridian Gold Inc. (formerly FMC Gold) in 1997. Meridian was mainly interested in the Arnett Creek and Beartrack projects and completed no additional work on the Humbug Property (Pierson, 2010). Meridian was taken over by Yamana Gold Inc. (Yamana) in 2007. Yamana purchased Meridian for the company's South American assets and sold their North American properties including Humbug. No work was completed by Yamana on the Humbug Property (Cuffney, 2011; Brewer, 2019).

In 2011, the LGT joint venture acquired the consolidated Lemhi Project which consisted of properties from four parties: BHLK, Meridian Gold Inc (Yamana), Joe and Hallie Proksch, and Vineyard Gulch Resources LLC (Cuffney, 2011; Brewer, 2019). The former LGT Property largely overlaps the current Lemhi Gold Project. In 2012, LGT began an aggressive pre-development program consisting of:

- historical data compilation and review
- drilling: 40 core holes and 15 RC holes
- geotechnical work
- petrography
- metallurgical work
- updated geological model and resource
- baseline environmental studies, addressing cultural resources, fisheries, wildlife resources, water rights and right-of-way concerns
- terrestrial vegetation and wetland delineation studies.

In 2013, Northern Vertex decided to focus on the development of their Moss Gold-Silver Mine in Arizona and sold their interest in LGT to the ISGC. No further work has been conducted since 2012-2013 (Brewer, 2019).

6.2.2 Geochemical Surveys

Geochemical surveys completed by FMC between 1984 and 1989 included rock chip sampling, trench sampling, soil sampling, and vegetation sampling. These surveys were conducted across the entire Ponderosa Property including areas overlapping the current Lemhi Gold Project and in areas west of the current project area up to Allan Creek.

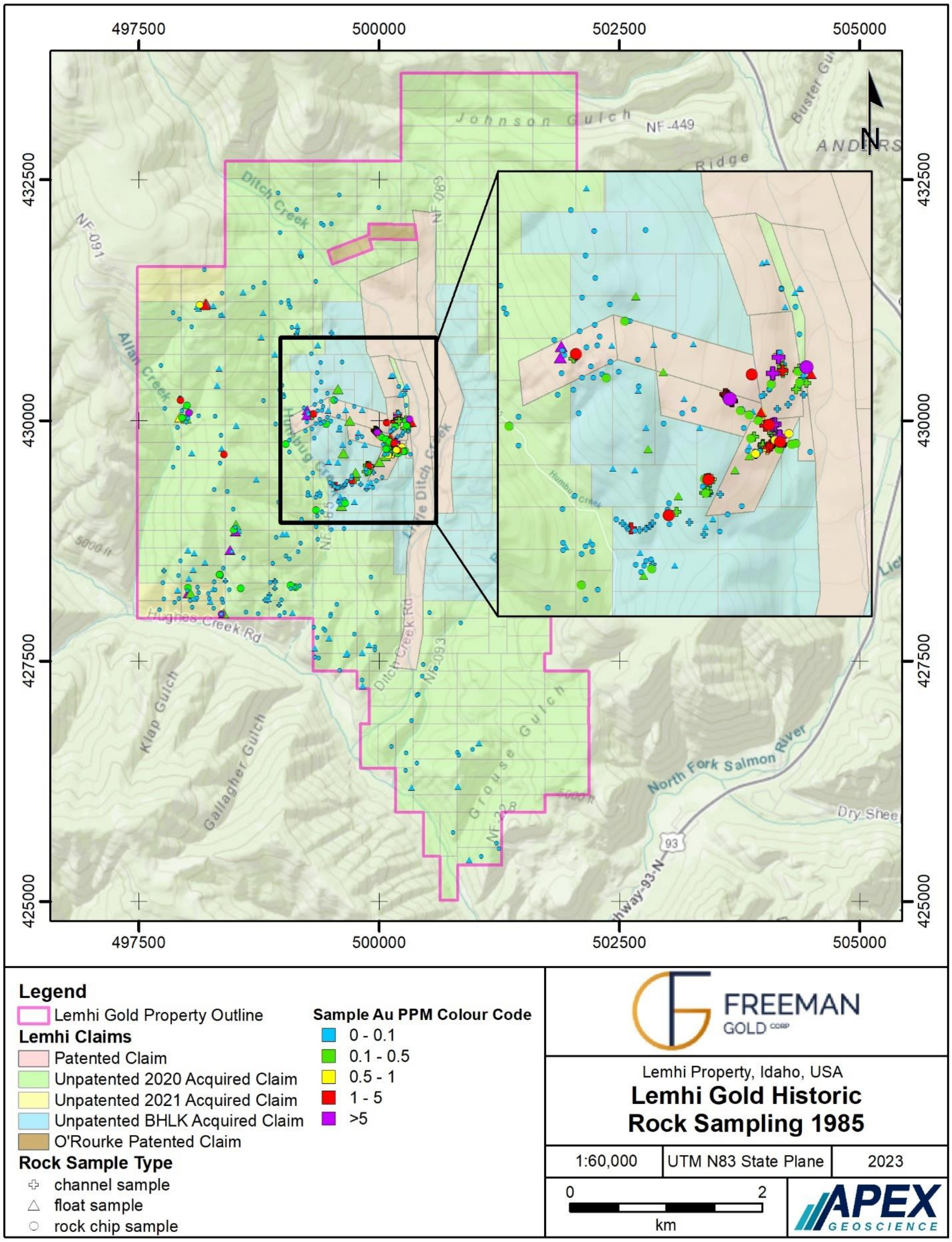
FMC collected 628 float and outcrop rock chip samples from the Ponderosa Property in 1984-1985. Of these, 393 samples have location data and 341 lie within the current Lemhi Gold Project claim boundary as shown in Figure 16-2. In addition, 363, 6.1 m channel samples were collected from all exposures in roads and trenches. A total of 136 of these channel samples have location data and lie within the current Lemhi Gold Project claim boundary. Gold values ranged from less than detection to a high of 160 g/t Au. In the main prospect area along the west slope of Ditch Creek, 630 samples were collected. Anomalous gold >68 ppb Au was found in 58% of the samples; 32% of the samples contained 343 ppb Au and 9% had 1. 71 g/t Au (McCarter, 1985).

FMC also collected 515 soil samples at claim corners and in several small grids surrounding the Lemhi Gold Property. The claim corner soils showed gold anomalies in the main prospect area of Ditch Creek and along the east side of Humbug Creek. The historical interpreted anomaly polygons are shown in Figure 6-3.

A vegetation survey was conducted over the gravels in Ditch Creek. Douglas fir, Spruce and Ponderosa pine twigs and needles were collected. The survey detected four areas of coincident gold, arsenic, and copper anomalies in the gravel-covered valley of Ditch Creek (Huang, 1986).

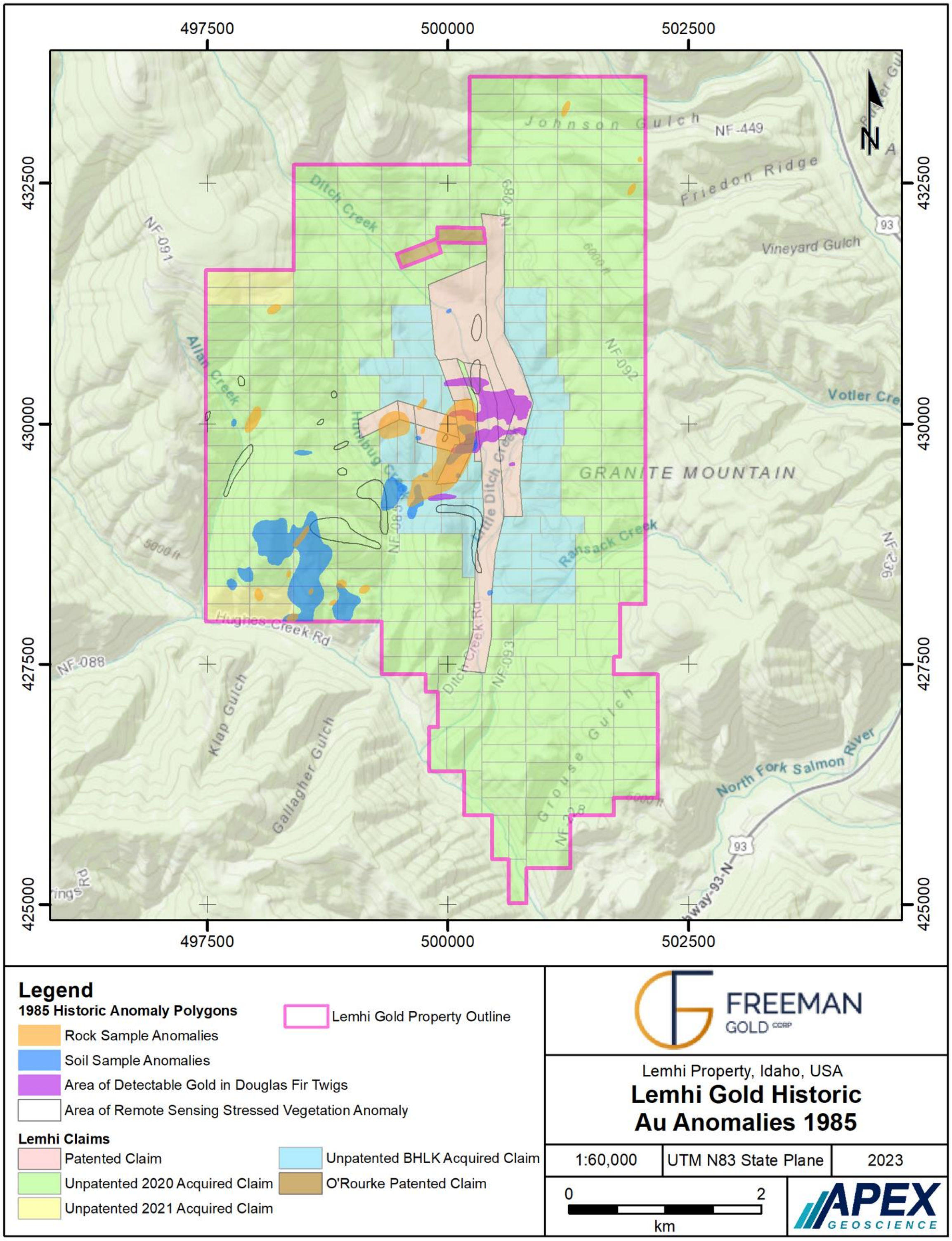
In 1989, a total of 431 rock ship samples, 755 soil samples, and 360 vegetation samples were collected by FMC across five priority areas on the Ponderosa Property. Anomalous gold results in three areas were followed up with RC drilling (McCarter, 1988).

Figure 6-2: Historical Rock Sampling Locations with Assay Results on The Lemhi Gold Project



Source: APEX, 2023.

Figure 6-3: Historical Soil, Rock, and Vegetation Anomaly Polygons on The Lemhi Gold Project



Source: APEX, 2023.

6.2.3 Geophysical Surveys

During the early exploration stage at Ditch Creek, FMC contracted Geophysical Environmental Research Inc. of New York City, NY to fly an aerial infrared remote sensing survey of the area in 1984. The survey was successful in locating a number of vegetation anomalies and spectral anomalies related to argillic alteration (Collins, 1985). Ground magnetic and very low frequency electromagnetic (VLF-EM) surveys were conducted by North American Exploration over the main prospect area centered on Ditch Creek in 1985. Zonge Engineering collected controlled source audio-frequency magnetotelluric (CSAMT) data along six lines across the main prospect area in 1986. Quantech Geoservices performed a dipole-dipole induced polarization (IP)/resistivity survey (5 lines – 7,200 line feet) and a time domain (TDIP) IP/Resistivity survey (54,600 line feet) in 1989 (Morrison, 1989).

Shaubs (1989) noted that drilling geophysical anomalies was not particularly successful. Gold mineralization correlates with intermediate resistivity and conductivity values rather than highs and lows, making targeting IP/resistivity anomalies problematic.

6.2.4 Drilling

Drilling has been conducted on the property by three previous owners from 1984 to 2012: FMC, AGR and LGT. A total of 419 historical drill holes, 366 RC holes (353 inside of the boundaries of the current property) and 53 core holes, have been completed on or near the Lemhi Gold Property. All available historical data pre-2012 was digitized and compiled by LGT and BHLK. The 2012 data was retained by ISGC. Both original and compiled drill data were provided to Lower 48 and Freeman. The availability of historical drill data is variable and summarized in Table 6-1. Thirteen FMC holes are located within the current Lemhi Gold Property Boundary (Figure 6-4). Collars exist for a total of 408 of the 419 holes completed. Assays exist for 411 of the 419 holes completed. Drill logs exist for 405 of the 419 holes completed. A total of 400 holes (including the 13 outside the current property) have collars, assays, and drill logs.

Table 6-1: Summary of Available Historical Drill Hole Data

Company	Year	Total Drill Holes		Collar Data		Assay Data		Drill Log	
		RC	DDH	RC	DDH	RC	DDH	RC	DDH
FMC Gold Corporation	1985	12		12		12		12	
FMC Gold Corporation	1986	74	3	74	3	74	3	74	3
FMC Gold Corporation	1987	84		83		84		83	
FMC Gold Corporation	1989	22	1	16	1	22	1	21	1
American Gold Resources	1993	39	2	39	2	39	2	39	2
American Gold Resources	1994	20	3	20	3	20		20	
American Gold Resources	1995	100	4	96	4	99		95	
Lemhi Gold Trust	2012	15	40	15	40	15	40	15	40
	Total	**366**	**53**	**355**	**53**	**365**	**46**	**359**	**46**

6.2.4.1 FMC Drilling 1985 -1989

FMC conducted drilling on the Ponderosa property in 1985-87 and in 1989. During this period, 192 RC holes (>24,000 m) and four core holes (664 m) were drilled (Figure 6-4). Of these 192 holes, 177 holes with available collar coordinates are located within the current Lemhi Gold Project. Most holes were drilled vertically with the exception of 10 RC holes and one core hole which were drilled at an angle of -45°. These were oriented either to the southwest or northeast across the drill grid. Downhole orientation surveys were not completed. FMC used Lang Exploratory Drilling of Salt Lake City, UT as the primary drill contractor.

6.2.4.2 AGR Drilling 1993-1995

AGR completed drilling on the Humbug property between 1993 and 1995. A total of 159 RC holes and 9 core holes were drilled, totalling>35,000 m of drilling (Figure 6-4). Of these 159 holes, a total of 155 holes with available collar coordinates are located within the current Lemhi Gold Project. AGR drilled vertical holes except for three core holes: DCC 93-1 and 2, and DCC 94-1. In 1995 downhole surveys were completed for several drill holes. AGR drilled three NQ-sized core holes in 1993 for geologic studies. The three holes drilled in 1994 (DCC 94 1-3) were large diameter PQ-sized core drilled to obtain large samples for metallurgical testing. The four core holes in 1995 were drilled with HQ-sized core using a split tube to obtain better core recovery and more intact core for geotechnical studies.

Drill holes ranged from 69 – 305 m in total length (depth). Water was encountered in most holes, but excessive water flow was recorded in only a few holes. The drillers were able to complete holes to over 183 m in depth without the hammer watering out. The Humbug deposit was drilled out on a nominal ~30.5 m x ~30.5 m grid of holes oriented along N-S by E-W lines. Several holes were drilled outside the grid area, following weaker mineralization in the northeast and southwest.

AGR used Lang Exploratory Drilling of Salt Lake City, UT as the primary drill contractor. Target Drilling of Kelowna, BC drilled the core holes for AGR in 1993-1995 using a Longyear 38 drill rig.

6.2.4.3 LGT Drilling 2012

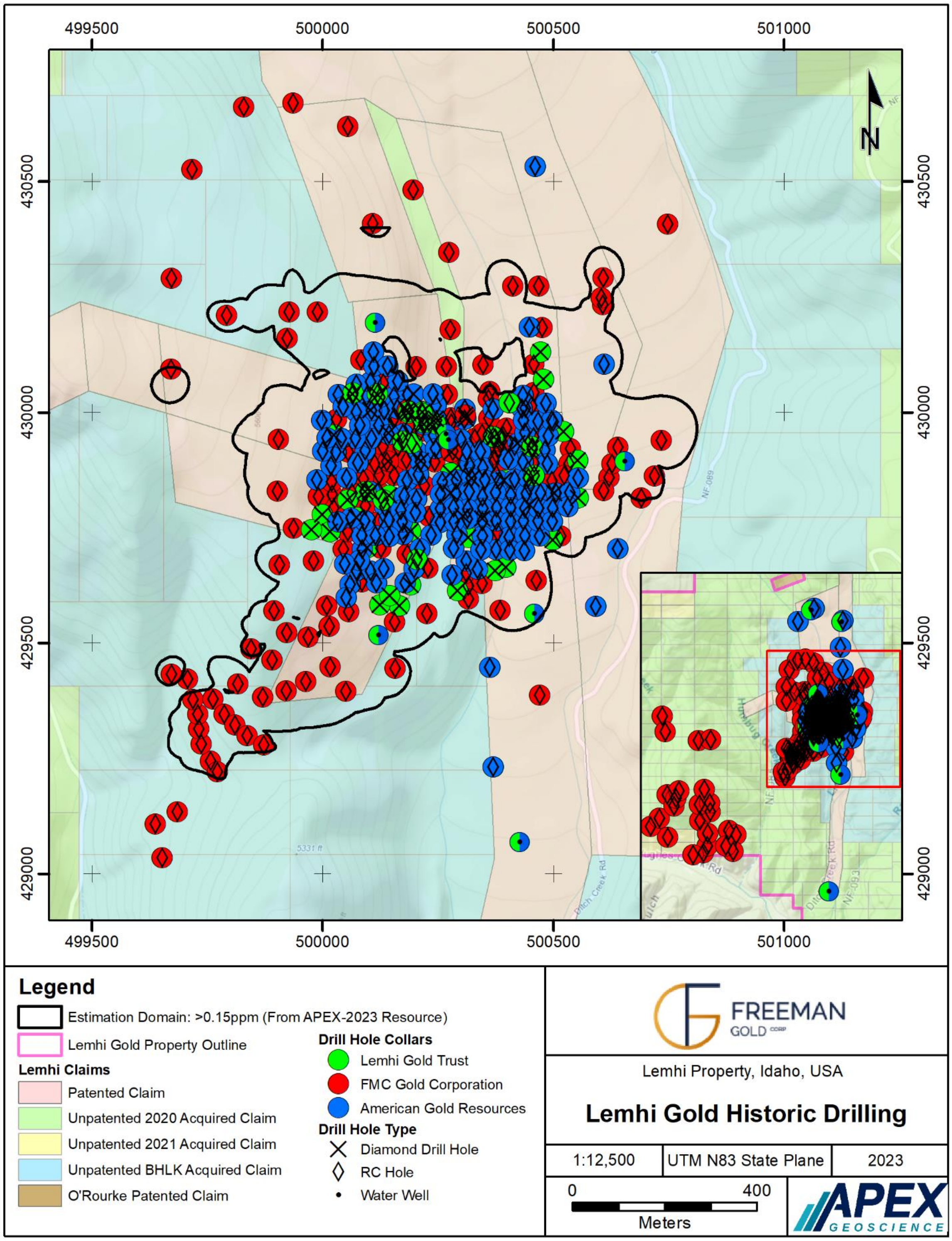
LGT completed an aggressive core and RC drilling program on the property in 2012 (Figure 6-4). A total of 7,860 m of HQ core was drilled in 40 holes throughout the LGT Property. All LGT drill holes are located within the current Lemhi Gold Project. Hole depths ranged from 144 – 245 m. Core holes were drilled as a combination of confirmation ’twin’ holes of historical RC drill holes, and infill and step-out holes of the known deposit. After completing several ’twin’ holes, some significant variation and discrepancies in assay results were identified between the historical RC holes and the recent core holes. To assist in understanding the cause of these discrepancies, LGT completed 15 RC holes totalling 2,672 m. All core and RC holes were drilled vertically. Downhole surveys were completed on all core holes and for three RC holes. LGT used Ruen Drilling of Clark Fork, Idaho as the drill contractor to complete the core drilling and Diversified Drilling of Missoula, MT as the RC drill contractor (Brewer, 2019).

The results of the 2012 drill program indicate that gold mineralization is widespread, even more so than was evident from historical drilling. All 2012 holes encountered at least weak or spotty gold mineralization, with a number of holes providing excellent results with both wide modest grade intercepts, narrower high-grade intercepts, and often multiple downhole intercepts of note (Table 6-2). This is demonstrated in holes such as LGT12-064R which intersected 4.57 m grading 4.35 g/t Au at a depth of 21.34 m followed by 41.15 m grading 1.19 g/t Au. One of the four mineralized intersections in hole LGT12-015C comprises 16.76 m grading 2.62 g/t including 3.05 m grading 12.37 g/t Au. Other wide intercepts include LGT12-029C with 1.2 g/t Au over 32.00 m core length, LGT12-023C with 1.06 g/t Au over 29.87 m and LGT12-66R with 1.53 g/t over 32.00 m hole length (Table 6-2). The LGT drilling program was successful in confirming the historically recognized gold deposit at the LGT Gold Property as well as identifying incremental southward extension of the Main Zone of mineralization (Northern Vertex, 2012; Brewer, 2019).

Twelve of the 40 core holes completed in 2012 were devoted to twinning historical RC holes; some significant discrepancies were identified in the twinning program. A significant effort was spent in an attempt to gain a better understanding of the discrepancies between the historical RC and the 2012 core results, and a number of additional 2012 RC holes were added into the twin drilling program to assist in sorting out the discrepancies. Generally, the mineralized zones between the historical holes and their 2012 twin holes showed excellent continuity, however, there were a number of discrepancies in the gold grades within these mineralized zones. The twin core hole intersections were often lower in grade for the 2012 drilling than the historical RC drilling. Of the twelve 2012 core vs. historical RC hole pairs, five holes yielded a greater than 35% difference in grade over more than 140 m intervals. Six core holes vs historical RC holes yielded marginal differences or were fairly low grade. One core hole, LGT12-011C, yielded a 37% greater grade over 152 m than the historical RC hole (86004) that it was intended to twin (Brewer, 2019).

Nine RC holes were subsequently completed in 2012, in an attempt to twin a number of the 2012 core holes that were drilled to twin historical RC holes. On average, the 2012 LGT core holes yielded the same grade as the 2012 RC holes or slightly higher grades, with the exception of the RC hole (LGT12-069R) drilled to twin LGT12-011C discussed above.

Figure 6-4: Historical Drill Hole Locations on The Lemhi Gold Project



Source: APEX, 2023.

Table 6-2: Drilling Highlights LGT Core and RC Holes

Hole ID	From (m)	To (m)	Intercept		Au	
			Width (m)	Width (ft)	(g/t)	(oz/t)
LGT12-020C	28.22	40.54	12.31	40.4	2.47	0.072
Including	29.81	32.77	2.96	9.7	4.51	0.132
LGT12-021C	156.06	163.98	7.92	26.0	1.74	0.051
LGT12-022C	11.58	23.32	11.73	38.5	1.17	0.034
LGT12-022C	116.59	131.52	14.94	49.0	0.89	0.026
LGT12-024C	107.59	130.61	23.01	75.5	0.81	0.024
LGT12-025C	122.53	157.73	35.20	115.5	0.73	0.021
LGT12-028C	183.19	187.91	4.72	15.5	3.26	0.095
Including	183.18	184.86	1.68	5.5	8.22	0.240
LGT12-028C	194.31	198.12	3.81	12.5	5.47	0.160
Including	195.38	196.75	1.37	4.5	10.85	0.316
LGT12-029C	117.04	149.05	32.00	105.0	1.20	0.035
Including	126.19	131.06	4.88	16.0	3.63	0.106
Including	140.67	146.00	5.33	17.5	2.36	0.069
LGT12-029C	157.43	174.96	17.53	57.5	0.71	0.021
LGT12-014C	61.57	71.93	10.36	34.0	3.46	0.101
Including	67.67	71.93	4.27	14.0	7.24	0.211
LGT12-015C	32.92	49.68	16.76	55.0	2.62	0.076
Including	32.92	35.97	3.05	10.0	12.37	0.361
LGT12-015C	109.12	114.61	5.49	18.0	2.08	0.061
LGT12-017C	92.66	108.66	16.00	52.5	1.60	0.047
Including	92.66	101.80	9.14	30.0	2.40	0.070
LGT12-017C	116.74	167.34	50.60	166.0	0.67	0.020
LGT12-019C	123.75	136.25	12.50	41.0	1.67	0.049
Including	132.28	136.25	3.96	13.0	3.25	0.095
LGT12-023C	137.16	167.03	29.87	98.0	1.06	0.031
Including	138.38	144.48	6.10	20.0	2.37	0.069
LGT12-027C	68.28	77.72	9.45	31.0	1.56	0.046
LGT12-027C	106.83	130.76	23.93	78.5	0.65	0.019
LGT12-027C	136.86	163.83	26.97	88.5	0.75	0.022
LGT12-064R	21.34	25.91	4.57	15.0	4.35	0.127
LGT12-064R	88.39	129.54	41.15	135.0	1.19	0.035
Including	105.16	117.35	12.19	40.0	2.24	0.065
LGT12-064R	149.35	170.69	21.34	70.0	0.68	0.020
LGT12-065R	9.14	35.05	25.91	85.0	0.67	0.019

Hole ID	From (m)	To (m)	Intercept		Au	
			Width (m)	Width (ft)	(g/t)	(oz/t)
LGT12-066R	121.92	153.92	32.00	105.0	1.53	0.045
Including	131.06	135.64	4.57	15.0	3.82	0.111
LGT12-066R	160.02	182.88	22.86	75.0	0.94	0.027
LGT12-073R	30.48	42.67	12.19	40.0	2.06	0.060
Including	32.00	38.10	6.10	20.0	3.58	0.104

Generally, there was pretty good reproducibility between the 2012 twinned core and RC holes. While the cause of the discrepancies with the historical RC holes is not immediately clear, several causes are possible, including the combination of uncertainty of the locations of the historical holes, lack of downhole surveys, as well as other more direct issues, such as nugget effect and/or smearing of gold during drilling, due to poor drilling practices or techniques in the 1980s and 1990s.

6.2.4.4 Quality and Reliability of Drill Data

Historical drilling at the Lemhi Gold Deposit was designed and managed by experienced teams of exploration geologists working for major gold mining companies. Drill contractors and assay labs used by FMC, AGR, and LGT were all established experts in their fields. All work appears to have been done to industry standards at the time. There is no reason to suspect any tampering with samples or other breaches of security during the drilling programs. The author believes that the drilling data is reliable and accurate.

Drill samples from historical drill programs were handled according to industry standards at the time. For the 2012 LGT drill program quality control/quality assurance procedures were implemented that met, or exceeded, all industry standards today (Brewer, 2019).

6.2.4.4.1 Pre-2000 Drilling

This section encompasses drilling completed by FMC from 1985 to 1989 and AGR from 1993 to 1995. The author has relied upon reports from that era and information provided by Mr. Brian Brewer and Mr. Dennis Krasowski who participated not only in the 2012 LGT drilling program but some of the historical programs.

The RC drill holes used a 12.7 cm sized bit. Samples of cuttings were collected continuously on 5-foot intervals and split either using a Gilson splitter for dry drilling or using a rotary splitter for wet drilling. The samples weighed 4.5 – 7 kg. Core drill holes used NQ, HQ, or PQ core diameter drills. Core was logged and then split. One split (one-half of the core) was sent for analysis. The other core split was retained for additional study and sampling but was eventually discarded.

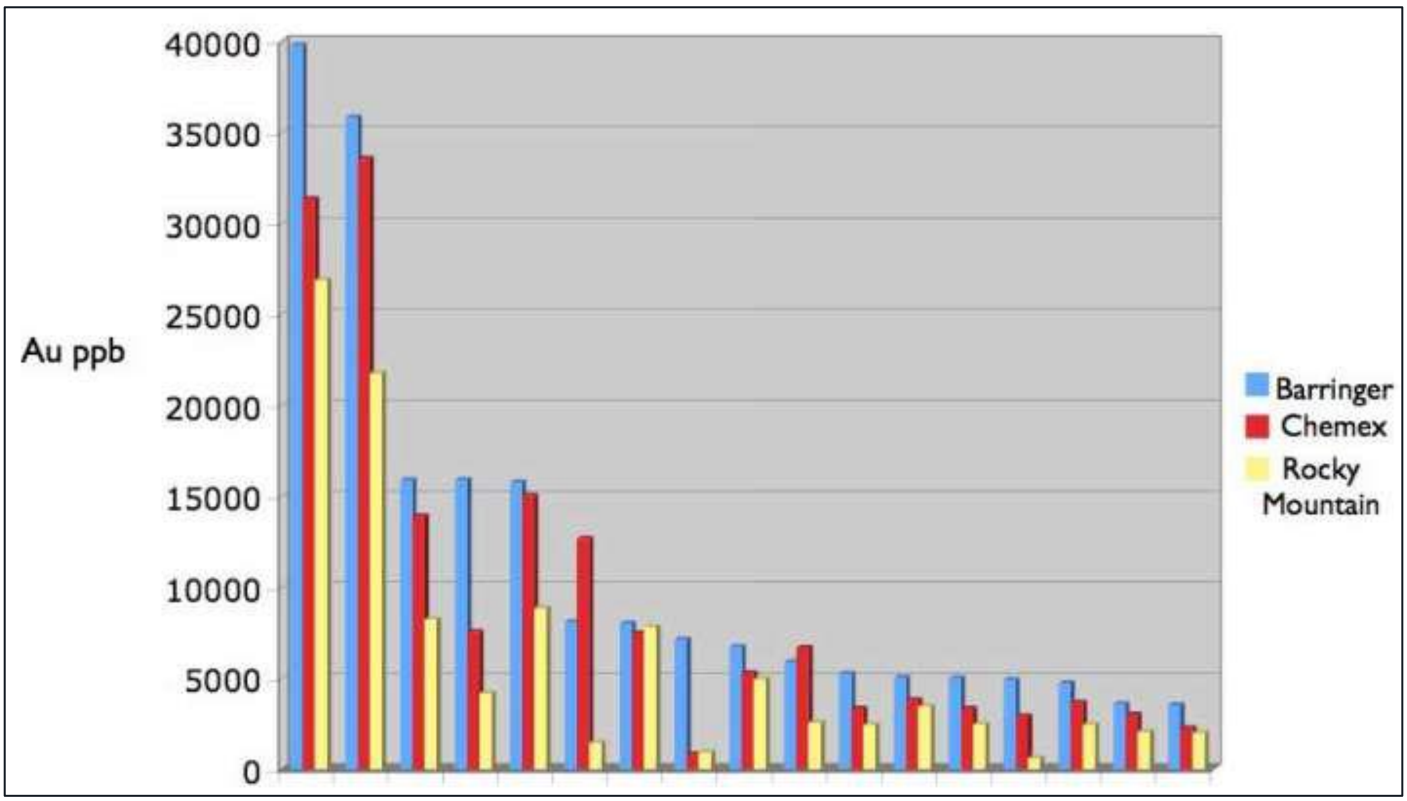
FMC used Intermountain Analytical Services Inc. (Intermountain) of Pocatello, ID for analysis of drill samples, and Bondar Clegg of North Vancouver, B.C. for geochemical analyses. AGR used Bondar Clegg and Barringer Labs (Barringer) of Sparks, NV for analysis of rock and drill samples. These analytical labs were not ISO-registered at the time but were considered reliable assay laboratories.

No standard or blank pulps were inserted into the sample stream in the early drilling by FMC. However, FMC used Bondar Clegg as an umpire lab and routinely sent large numbers of pulps to Bondar Clegg as checks against

Intermountain's assays. In September 1987, FMC started inserting check samples every tenth sample every 15.2 m, beginning with hole 87-047. The author was not able to locate the check assay results or find any discussion of the results. AGR apparently did not insert control samples in the sample stream and relied on the analytical labs' internal quality control procedures. However, AGR did run numerous crosschecks against two umpire labs. Although relying on the labs' internal QA/QC procedure is not ideal, the author considers the combination of the QA/QC protocols of the analytical labs and the umpire lab checks to be acceptable and adequate for the exploration phase of the Lemhi Project.

AGR conducted a series of check assays in 1994. Pulps from samples prepared and assayed by Barringer Labs were assayed at the Rocky Mountain Geochemical (Rocky Mountain) and/or Chemex Laboratory (Chemex). A total of 147 samples were checked by Rocky Mountain and 50 samples were checked by Chemex. Additionally, 28 samples were checked by both Rocky Mountain and Chemex (Figure 6-5). Both mineralized and barren material were check assayed against original assays ranging from <1 part per billion (ppb) Au to 38.29 g/t Au. There was considerable difference in the absolute value of gold assays between the labs. Although some discrepancy can be attributed to nugget effect (erratic distribution of fine gold grains), there is an obvious laboratory bias to the data. Chemex's assays are consistently slightly lower than Barringer's initial assay, but within an acceptable range. Rocky Mountain's assays were consistently and significantly lower than those of both Barringer and Chemex. Figure 6-5 illustrates the difference in assays for the higher-grade samples (17 samples in excess of 3 g/t Au) analyzed by Barringer, Chemex, and Rocky Mountain.

Figure 6-5: Comparison of Check Assays by Chemex and Rocky Mountain Geochemical Against Original Barringer Fire Assays.



Source: Cuffney, 2011
Note: Samples with original assay >3 g/t Au are plotted

The consistent variation in gold assays among the three analytical labs is both striking and puzzling. The average grade of the 17 samples analyzed by Barringer was 11.241 g/t Au. The average grade obtained by Chemex was 9.293 g/t Au, 83% of Barringer's average. Rocky Mountain's assays averaged only 6.155 g/t Au, a mere 55% of Barringer's average grade and only 66% of Chemex's average. The reasons for the discrepancies are not clear and are discussed below.

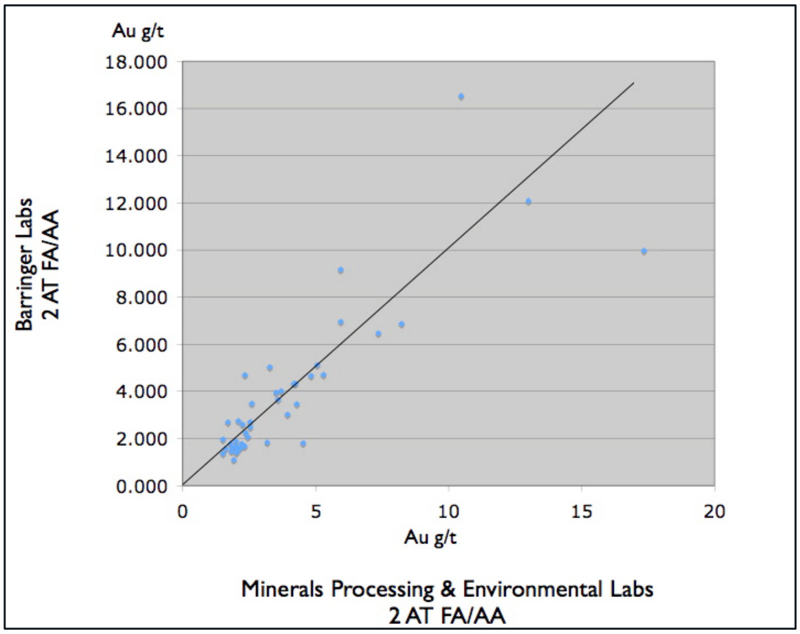
Nugget effect alone would produce variation in gold grades but would be somewhat random. Barringer's and Chemex's assays were two-assay-ton fire assays, which should help reduce the nugget affect by assaying a larger charge (60 g vs 30 g for 1 assay-ton fire assay), whereas Rocky Mountain's assays were smaller one-assay-ton fire assays. Part of the problem may have stemmed from the use of atomic absorption (AA) finish on fire assay fusions. The upper limit for AA finish on fire assays is 10 g/t Au, and at gold grades of >5 g/t the reproducibility of samples is reduced because the solution containing the gold dissolved from the fire assay bead must be diluted, a process which decreases accuracy. Inconsistencies in dilution procedures between labs can produce systematically high or low values between labs. Another possible cause of the discrepancies could be errors in the rolling process and subsequent sub-sampling of pulps, producing inconsistent sample splits (Colwell, 1994). There does seem to be a significant problem with the results from Rocky Mountain, and it was Cuffney's (2011) opinion that those results should be discarded.

A second set of check assays was run by Mineral Processing and Environmental Laboratories Inc. (MPEL) of Sparks, NV acting as the umpire lab. AGR submitted pulps from 154 drill samples for check gold assays. Both Barringer and MPEL utilized 2-assay-ton fire assays with an AA finish. Although there was considerable difference between individual analysis by Barringer and MPEL, the difference was random and was greater at high gold grades, as would be expected from a nugget effect. For 81 samples exceeding 1 g/t Au the average grade of the Barringer assays was 3.065 g/t Au, whereas the average of MPEL's assays was 3.222 g/t Au, a difference of only 5%. For 73 samples containing less than 1 g/t Au Barringer's average grade was 635 ppb, whereas MPEL's average was 556 ppb Au (Figure 6-6).

MPEL conducted duplicate check assays on several of the Barringer pulps and found a similar variance in gold grades, thus confirming that the irreproducibility of gold assays is a function of erratically distributed fine gold grains, likely the nugget effect. The nugget effect is surprising, given the fine-grained nature of gold particles (5-25 µm) found both in petrographic and metallurgical studies. It is likely that a coarser fraction of gold particles is present in fair abundance, perhaps associated with some of the late veins. Krasowski (1994) mentions observing visible gold in core holes drilled in 1993 and 2012. Cuffney (2011) has observed visible gold in outcrop on the property substantiating the presence of coarse gold, at least locally within the deposit. This is supported also by the presence of actual placer gold accumulations in the local creeks.

FMC drilled three core holes (C-1, C-2, C-3) as twin holes of RC holes in 1987. The core holes were located within 3 m of the RC collars and drilled in the same vertical orientation. All three core holes showed significant variation in grade from the RC holes. Individual ~1.5 m sample intervals rarely correlate, yet the tops and bottoms of broad mineralized zones (and thickness of zones) are fairly consistent. There is also significant variation in the average grade of the mineralized zones. Figure 6-7 illustrates the differences between core hole C-2 and RC hole 86-014.

Figure 6-6: Barringer Lab's gold assays vs. MPEL's assays for 34 samples containing > 1. 5 g/t in initial samples.



Source: Apex, 2023.
Note: Although there is considerable scatter, it is random and increases with gold grade. Best fit line through the data approximates a 45° slope (Cuffney, 2011).

Cuffney, 2011 concluded:

> *"The differences in gold grades are likely due to one or more of four potential factors, irregular distribution of quartz veining and gold mineralization within the deposit, a nugget effect, core loss through the mineralized zones, and hole deviation. Although the variation in grade on detailed scale is significant, overall mineralization holds together and the effect on tons and grade should be minimal."*

Also, the wet drilling conditions for the RC drill program might have concentrated the gold mineralization, due to lost sericite and associated light minerals; however, this would not explain the higher grades encountered in core drilling vs. RC drilling. Reverse-circulation drilling in highly broken or friable rocks with high-water flows can lead to downhole contamination, particularly if free gold is present. Typically, for holes with such contamination, gold values will gradually tail off downhole from a high-grade intercept and/or will spike every ~6.1 m downhole at rod changes (when material can fall downhole). These patterns were not observed in the RC drill logs. FMC's drill logs were checked for notes on water levels and water flows, and high-water flows were mentioned in only a few holes. Significant water was usually not encountered until at least 123-183 m in most holes. There does not seem to be a downhole contamination problem with RC drilling at the Lemhi Gold Deposit.

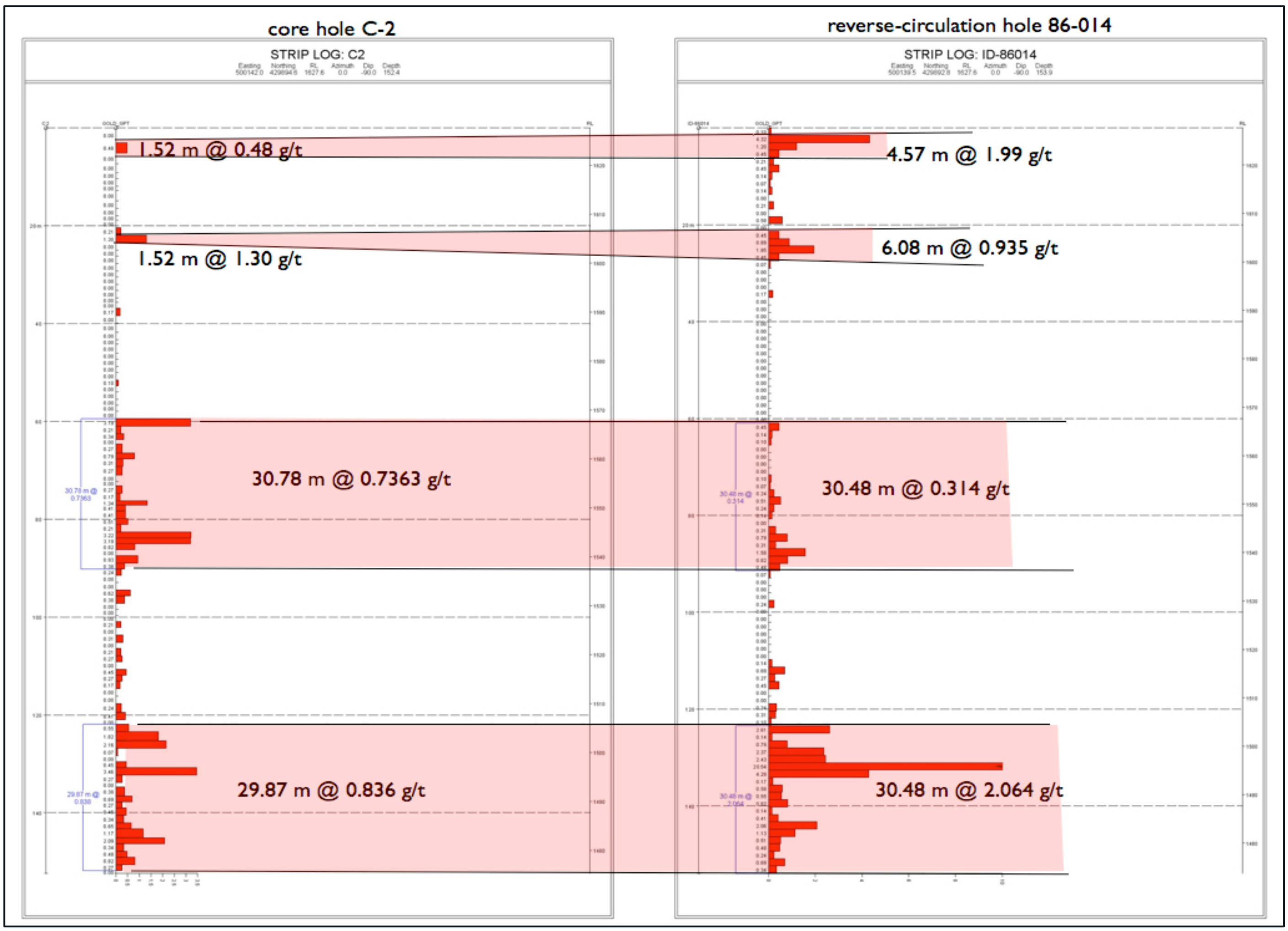
FMC did not perform downhole orientation surveys on its drill holes. AGR started surveying holes near the end of the drilling program in 1995. Fairly significant downhole deviation was noted in some of the holes. Given the tight (nominal 30.5 m) spacing of the holes, the actual location of gold intercepts at depth and the relationship of intercept between adjacent holes in both AGR's and FMC's drilling is somewhat questionable.

In conclusion, the historical pre-2000 drilling completed on the Lemhi Gold Project for FMC and AGR was conducted by experienced professionals using industry best practices at the time. Excepting the lack of downhole surveys, the work conducted was adequate for mineral resources calculations. The gold mineralization within the Lemhi Gold Deposit is erratically distributed and a nugget effect is plausible due to the presence of fine to moderately coarse free gold, which causes difficulties in replication of individual gold assays.

6.2.4.4.2 LGT Drilling 2012

Core drilling was completed using HQ core diameter drills. Drill core was securely stored at the drill site and the core logging/office facility in Salmon, Idaho. The core processing entailed cleaning of the core, geotechnical and geological logging, photographing, and sampling. During the geological logging process, the geologist identified and clearly marked all sample descriptions and intervals along with placement of all QA/QC samples (Brewer, 2019).

Figure 6-7: Strip log comparison of core holes C-2 and RC hole 86-014 (Cuffney, 2011)



Source: Cuffney, 2011.

QA/QC samples included analytical reference standards and blanks along with duplicates. Analytical reference standards of varying geochemical grades were inserted into the normal sample stream at a frequency of no less than one per 30 normal core samples. Coarse blank material samples were inserted immediately after or within a presumed mineralized interval and at a frequency of no less than one per 30 normal core samples randomly and at an alternating frequency to the analytical reference standard (Brewer, 2019).

The core was split utilizing either a 36 cm diamond core saw or a hydraulic core splitter. Samples of one-half core were submitted for assay to ALS in Reno Nevada, an ISO-accredited laboratory. Standard analytical methodology for all LGT core samples included a standard fire assay for gold with a 30 g nominal charge weight and 61 element four acid “near total” digestion ICP-AES. Subsequent analysis included metallic screen fire assay for all samples that initially reported 4.0 g/t gold or greater.

Because the 2012 RC drilling was initiated to try to better understand the discrepancies between the historical RC drilling and the more recent core drilling, the 2012 RC sampling process tried to achieve 100% sample collection from the drill rig. This included collecting all water, as well as drill cuttings. Samples were collected in 5 gallon buckets, which at times resulted in numerous buckets per 1.5 m sample, with many containing only water, minimal cuttings, and slimes. All samples were treated with flocculant, water was decanted, and solid material with slimes were combined for one sample per 1.5 m of drilling (Brewer, 2019). All samples were securely shipped to ALS laboratory in Reno, NV. The RC samples were analyzed using the same methods as the core samples.

Downhole surveys for all core holes were conducted every 15 m utilizing a single shot survey camera. Minimal deviation was detected in core holes. All RC holes were left open upon completion and attempts were made to case the holes with PVC in order to facilitate downhole surveys. International Directional Services was contracted to complete the downhole surveys of the RC holes, however, only three holes (LGT12-60R, 69R and 72R) were stable enough to achieve any meaningful downhole survey. LGT12-60R had the most deviation at 6° in 213 m. Holes LGT12-69R and LGT12-72R had 2° in 160 m and 3.25° in 181 m, respectively. All drill collars were adequately marked and preserved after hole abandonment and were subsequently surveyed by a professional land surveyor upon completion of the drill program.

6.3 Historical Resource and Reserve Estimates

This section contains historical information on resource estimates made prior to Freeman entering into an agreement to acquire the Lemhi Gold Project. Historical resource estimates from the 1980s and 1990s were completed prior to the implementation of NI 43-101 and the construction of the CIM Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines, dated November 23rd, 2003 and its recent update, dated November 29, 2019 along with the most recent CIM Definition Standards on Mineral Resources & Mineral Reserves dated May 10th, 2014. These historical resource estimates use resource categories different from those defined by the CIM Definition Standards. In addition, even the most recent resource estimates that were completed on behalf of LGT in 2012 and 2013, were informal estimates that were not properly documented in any NI 43-101 technical reports and were completed prior to the most recent CIM Guidelines of 2019, and CIM Definition Standards of 2014. A brief synopsis of the history of the resource calculations on the Lemhi Gold Project is presented in Table 6-3 and is discussed below. The authors of this report, independent QPs, have not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves. They are presented to assist in describing the extent of gold mineralization at the project and to outline the exploration potential. The historical resources presented are superseded by the drilling conducted in 2020-

2022 by Freeman and the MRE presented in this technical report. The following summary is based on Dufresne (2020), Brewer (2019), and Cuffney (2011).

Table 6-3: Historical Resource Estimates Lemhi Gold Deposit[1]

Source	Category[1]	Grade (g/t)[3]	Tons (Tonnes)	Cut-off (g/t)***	Ounces[1]
1987 FMC (Disbrow, 1987)	"Geological Reserve"	0.057 (1.95)	3,006,595 (2,727,537)	0.035 (1.20)	171,375
1989 FMC (Mine Reserve Associates)	"Reserves"	0.055 (1.89)	623,700 (565,811)	0.032 (1.10)	34,304
		0.044 (1.51)	1,014,400 (920,248)	0.024 (0.82)	44,634
1996 AGR (Pincock Allen Holt PAH – Sandefur, 1996)	"Geological Resource"	0.0375 (1.29)	32,361,539 (29,357,894)	0.003 – 0.012 (0.1 – 0.4)	1,217,704
	"In-pit Geological Resource"	0.0385 (1.32)	13,649,974 (12,383,048)	0.003 – 0.012 (0.1 – 0.4)	525,938
1996 AGR (Independent Mining Consultants)	"In-pit Potential Mineable Resource"	0.036 (1.23)	15,031,000 (13,635,894)	0.011 (0.38)	542,620
2012 LGT (Practical Mining Swanson et al. 2012)[2]	Indicated	0.025 (0.87)	21,003,440 (19,054,000)	0.004 (0.14)	529,300
	Inferred	0.020 (0.69)	14,083,130 (12,776,000)	0.004 (0.14)	281,400
2013 LGT (Practical Mining)[2]	Measured and Indicated	0.024 (0.81)	24,222,402 (21,974,200)	0.006 (0.20)	569,631
	Inferred	0.018 (0.61)	13,781,831	0.006 (0.20)	268,959
2013 LGT (Practical Mining)[2]	Unconstrained Pit Resource	0.020 (0.68)	23,461,740 (21,284,138)	0.006 (0.21)	464,480
	Patent Constrained Pit Resource	0.020 (0.67)	10,796,117 (9,794,075)	0.006 (0.21)	211,648

Note:
1. All resources are considered historical in nature. Resources completed in or prior to 2013 either do not use categories as set out in in the CIM Definition Standards on Mineral Resources & Mineral Reserves (2014), and/or are outdated due to subsequent drilling.
2. The authors of this technical report do not have enough information to verify the 2012 or 2013 Practical Mining resource estimates (which were internal estimates with no formal technical reports) as current mineral resources, therefore they are considered historical in nature and are superseded by the MRE presented in this technical report.
3. g/t = grams per metric tonne.

6.3.1 1987 FMC Resource

In 1987, FMC reported that the Ponderosa property (i.e., Lemhi Gold Deposit) contained an in-house "drill indicated mineral inventory" of 2,727,537 tonnes grading 1.95 g/t Au, using a cut-off grade of 1.20 g/t Au as shown in Table 6-3. The historical estimate was reported by FMC in a 1987 internal company report titled "Ponderosa Reserve Evaluation" (Disbrow, 1987). Historical "drill indicated reserves" were estimated using a block model of 15 m x 15 m x 3 m and varying gold prices. The "reserves" ranged from 237,000 tons grading 2.61 g/t Au for 18,000 oz Au at $350/oz gold to 989,300 tons grading 1.99 g/t Au for 57,000 oz Au at $550/oz gold. Disbrow (1987) concluded, "although the deposit displays significant geologic reserves, the surface mineable reserve potential Is very limited." The 1987 study also stated, "A brief economic analysis indicated that the reserves, as defined, would not support the capital required".

FMC's "mineral inventory" and "reserve" estimates were calculated prior to implementation of NI 43-101 and use categories other than those defined by the CIM Definition Standards on Mineral Resources & Mineral Reserves (2014). The term "mineral inventory" was an in-house term used by FMC to indicate mineralization defined by wide- spaced drilling, but for which no economic assessments had been made. The confidence level of FMC's "mineral inventory" approximates that of inferred resources as defined by the CIM Definition Standards, however, the lack of economic consideration prevents the mineralization from being considered a resource estimate by current definitions. FMC's term "reserve" does not equate to any reserve category as defined by the CIM Definition Standards on Mineral Resources & Mineral Reserves (2014). Given that FMC's own preliminary economic analysis indicated that the gold mineralization was uneconomic, the "reserve" estimate would not qualify as a reserve. This 1987 estimate was superseded by revised estimates in 1989 and estimates made for AGR in 1996 after additional drilling.

The authors of this report have not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves.

6.3.2 1989 FMC Resource

In 1989 Mine Reserve Associates updated the geologic model and resource estimate for the Ponderosa Property on behalf of FMC. The modelling produced resource estimates termed "reserves" at the time ranging from 565,811 t grading 1.89 g/t Au at a cut-off grade of 1.10 g/t Au to 920,248 t at 1.51 g/t Au at a cut-off grade of 0.82 g/t Au (Table 6-3). The historical estimate was reported in a 1989 company report titled "Ponderosa re-evaluation: unpublished intra-company memorandum" prepared for FMC Gold Corporation (Disbrow, 1989). The 1989 update was based on reinterpretation of the geometry of the mineralization, revised production costs and grade estimation parameters which allowed the high-grade zones to be estimated separately from the surrounding low-grade material. An economic analysis concluded that the project would be a break-even proposition, but was sensitive to the price of gold, operating costs, and heap-leach recoveries. Disbrow therefore recommended conducting additional drilling and heap-leach testing. FMC's 1989 resource estimate used resource categories not allowable as defined by the CIM Definition Standards on Mineral Resources and Mineral Reserves. The historical estimate was superseded by estimates made in 1996 and 2012–2013, which encompassed additional drilling.

The authors of this report have not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves.

6.3.3 1996 PAH Resource

In 1996, Pincock, Allen, and Holt (PAH) developed a geological model and calculated resources for the Lemhi Gold (Humbug) Deposit. PAH estimated "geological resources" (measured, indicated, and inferred) as 29.36 Mt grading 1.29 g/t Au containing 1.217 Moz of Au. The historical estimate was reported in "Geologic and Resource Model of the Humbug Deposit" an unpublished report prepared for American Gold Resources Inc. (Sandefur, 1996).

The PAH geological and grade block models were based on 277 RC drill holes totalling 47,854 m of drilling. The resource was developed utilizing a multiple indicator kriged model with a cut-off grade varying from 0.1 g/t Au to 0.4 g/t. PAH used a block size of 7.6 m x 7.6 m x 3 m for the block model. PAH classified resources as "measured and indicated" if they were within ~23 m of a drill hole. The PAH estimates had a very high in-situ waste to resource ratio of 45. 7:1 for the measured and indicated geological resources and 29. 2:1 for the larger measured, indicated, and inferred geological resource shown in Table 6-4. Given that the drilling in these areas was largely confined to a narrow NW-SE corridor, the author's opinion is the large amount of rock classified as waste was likely a function of the lack of drilling, rather than rock actually verified as barren by drilling.

Table 6-4: Humbug Geological Resources calculated by PAH 1996

American Gold Resources Humbug Project Undiluted Geologic Resources				
Rock Type	Tonnage		Grade (oz /ton Au)	Contained Ounces
	Waste	Resources		
Measured, indicated, and inferred (Au>=0.0122 grade multiple indicator model with 0.72 factor)				
1	241,339	580,661	0.0433	25,141
2	5,834,569	14,438,431	0.0433	624,692
10	6,226,000	0	0.0000	0
20	161,000	0	0.0000	0
50	44,978,472	1,991,528	0.0291	57,994
61	887,882,954	15,355,046	0.0329	505,813
All	945,328,461	32,361,539	0.0375	1,213,704
Measured, Indicated (Distance = 75. 0 ft) (Au>=0.0122 grade multiple indicator model with 0.72 factor)				
1	292,714	529,286	0.0422	22,324
2	6,961,748	13,311,252	0.0435	579,408
10	6,226,000	0	0.0000	0
20	161,000	0	0.0000	0
50	45,856,811	1,113,189	0.0296	32,996
61	897,276,629	5,961,371	0.0308	183,719
All	956,767,434	20,922,566	0.0391	818,424
Tonnage and ounces given to one's unit and grade given to 0.0001 oz/t for comparative purposes only				

Source: Sandefur, 1996.

PAH's resource estimates were made prior to implementation of NI 43-101 and use categories other than those defined by the CIM Definition Standards on Mineral Resources & Mineral Reserves (2014). Drill spacing of ~23 m for PAH's measured and indicated geological resource, given the variography of the deposit, is sufficient to meet current confidence levels equivalent to measured and indicated resources. However, PAH's combined resources were not in any way constrained by any kind of economic model in order to demonstrate a reasonable prospect for future economic extraction. The resource estimate has been superseded by 2012–2013 resource estimates, the 2021 maiden MRE and the current 2023 MRE reported herein, along with additional drilling conducted during 2012 and 2020-2022, therefore the resource estimate is considered historical in nature.

PAH made a separate estimate of "in-pit resources" based on a floating cone pit design. Several different grade indicator models were used, but all produced similar results, ranging from 525,938 in-situ oz of Au to 584,205 in-situ oz of Au at more reasonable waste to resource strip ratios of 4.73:1 to 4.78:1. PAH's table of resource calculations is reproduced below (Table 6-5).

Table 6-5: Humbug In-pit Measured and Indicated Resources calculated by PAH 1996

American Gold Resources Humbug Project Comparative In-Pit Resources					
Rock Type	Tonnage		Contained		SR (tons/tons)
	Waste	Resource	Grade oz/ton Au	Ounces	
Au>=0.0112 Grade Single Indicator Model					
1	70,965	379,035	0.0358	13,564	0.19
2	2,374,317	11,980,683	0.0419	504,696	0.20
10	3,821,000	0	0.0000	0	0.00
20	19,000	0	0.0000	0	0.00
50	10,521,013	169,987	0.0242	4,106	611. 89
61	48,282,975	1,238,025	0.0251	31,084	39.00
All	65,087,742	13,768,258	0.0400	550,439	4.73
Au>=0.0112 Grade Multiple Indicator Model					
1	71,685	378,315	0.0397	15,035	0.19
2	2,401,592	11,953,409	0.0446	533,360	0.20
10	3,821,000	0	0.0000	0	0.00
20	19,000	0	0.0000	0	0.00
50	10,534,911	156,089	0.0258	4,027	67.49
61	48,362,209	1,158,791	0.0274	31,797	41.74
All	65,206,026	13,649,974	0.0428	584,205	4.78
Au>=0.0112 Grade Multiple Indicator Model with 0.72 Factor					
1	71,685	378,315	0.0366	13,845	0.19
2	2,401,592	11,953,409	0.0400	477,941	0.20
10	3,821,000	0	0.0000	0	0.00
20	19,000	0	0.0000	0	0.00
50	10,534,911	156,089	0.0248	3,875	67.59
61	48,362,209	1,158,791	0.0261	30,292	41.74
All	65,206,026	13,649,974	0.0385	525,938	4.78
Au>=0.0112 Grade Multiple Indicator Model with 0.84 Factor					
1	71,685	378,315	0.0379	14,355	0.19
2	2,401,592	11,953,409	0.0420	501,691	0.20
10	3,821,000	0	0.0000	0	0.00
20	19,000	0	0.0000	0	0.00
50	10,534,911	156,089	0.0252	3,941	67.59
61	48,362,209	1,158,791	0.0267	30,936	41.74
All	65,206,026	13,649,974	0.0404	550,912	4.78
Based on 0.0112 single indicator floating cone pit					

Source: Sandefur, 1996.

PAH used the 0.72 indicator factor as the best fit to the model. This produces an in-pit resource of 525,938 oz Au at a grade of 1.32 g/t Au and a waste to resource strip ratio of 4.78:1. PAH's resource estimates were made prior to implementation of NI 43-101. This estimate was superseded by a revised estimate made later in 1996 by Independent Mining Consultants (IMC) and more recent estimates made by Practical Mining, LLC in 2012 and 2013 and the MRE reported in this technical report.

The authors of this report have not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves.

6.3.4 1996 IMC Resource

Later in 1996, IMC calculated a revised resource estimate applying realistic mining criteria at the time. IMC used PAH's block model and a floating cone algorithm based on $400/oz Au to design the pit. The final pit design was used to calculate "potential mineable resources" at cut-off grades ranging from 0.34 g/t to 1.03 g/t Au. IMC settled on a plan using a 0.38 g/t Au cut-off grade. The in-pit "potential mineable resource" was reported as 13,635,894 t at 1.23 g/t containing 542,620 oz Au. The strip ratio was 4.90:1. IMC's table of calculations at different cut-off grades is reproduced below in Table 6-6. The historical mineral resource estimate was reported by Independent Mining Consultants in a 1996 internal company report titled "American Gold Resource Corporation Humbug project, Idaho, scoping study, mine plan, capital and operating costs" prepared for American Gold Resources Inc. (IMC, 1996).

Table 6-6: Humbugs Historical "Potential Mineable Resource"

Potential Mineable Resources by Gold Cut-off Grades				
Gold Cut-off	Ore (ktons)	Gold (opt)	Gold (koz)	Strip Ratio
0.010	15,070	0.036	544.03	4.89
0.011	15,031	0.036	542.62	4.90
0.013	14,735	0.037	539.30	5.02
0.015	14,200	0.037	531.08	5.25
0.020	12,956	0.039	510.47	5.85
0.025	11,012	0.042	465.81	7.06
0.030	8,730	0.046	403.33	9.16
Total Ktons Contained in The Pit 88,723				

Source: Estimated IMC, 1996.

IMC used a multiple indicator kriging partial block model for the resource study. Due to multiple correction factors and dilution considerations, IMC found the kriging method to be unstable and prone to yielding a wide range of resource estimate results, depending on the correction factors applied. The study concluded that the use of a better estimation method than indicator kriging would be preferable. IMC classified the historical estimates as "potential mineable resources", which is a term not defined by the CIM Definition Standards on Mineral Resources & Mineral Reserves (2014). This historical estimate has been superseded by more recent resource estimates constructed in 2012 and 2013 by Practical Mining, LLC and by more recent drilling and the MRE presented in this technical report.

The authors of this report have not done sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves.

6.3.5 2012-2013 LGT Resources

Prior to the initiation of the 2012 drill program, LGT commissioned Practical Mining, LLC (Practical) of Elko, Nevada to complete an informal but modern mineral resource estimate based on all the historical drilling data. The historical estimate was reported by Practical in a 2012 internal unpublished company report entitled "Lemhi Project Geologic Model and Resource Estimate January 2012" prepared for LGT (Swanson, et al., 2012). Practical utilized grade shells, a 0.14 g/t Au cut-off and modeled the mineralization with a 2 m x 2 m x 2 m block model, which gave a resolution that maintained a reasonably accurate volume of the stacked mineralization shapes and accurately fit the modeled geology. Only samples within the grade shell were used to estimate the blocks within the grade shells, which prevented both dilution and over-estimation. Practical delineated a mineral resource (not pit constrained) of 19,054,000 t at 0.87 g/t Au for 529,300 oz Au in indicated, and 12,776,000 t at 0.69 g/t for 281,400 oz Au inferred using a lower cut-off of 0.14 g/t Au.

Despite having a cut-off grade that is 38% lower than that used by PAH, Practical's resource has 33% fewer ounces than the 1996 PAH estimate. The lower average grade is, at least partly, attributed to the lower cut-off grade, however the decrease in total ounces is more difficult to reconcile. One explanation for the difference is that PAH overestimated the tons and grade by using a block size that did not have a high enough resolution to accurately model the geology and mineralization shapes, and by populating those blocks with grades derived from a more restrictive sample population than that depicted by the 7.6 Mx 7.6 m x 3 m blocks used in the PAH model. Practical Mining validated their block model by comparing the average grades obtained by each of the three grade estimation techniques and comparing this grade to the average grade of all drill composites within the 0.14 g/t grade shells (Table 6-7).

Table 6-7: Practical Mining's Grade Estimation Comparisons

Grade Estimation Method	Tonnes (000's)	Grade (g/t)	Ounces (000's)
Ordinary Kriging	32,420	0.781	814. 1
Inverse Distance Cubed	32,420	0.785	818. 2
Nearest neighbour	32,420	0.778	810.9
3M Drill Composites	-	0.861	-

Source: Swanson, et al., 2012.

Subsequently, Practical Mining re-blocked the 2 m x 2 m x 2 m blocks to 6 m x 6 m x 8 m blocks to represent the SMU. The average grade of the larger block was calculated as the weighted average grade of the smaller blocks within each large block. Lerchs-Grossman optimal pits were calculated using the large blocks at a range of gold prices including $1,275/oz and $1,400/oz, see Table 6-8.

Table 6-8: Practical Mining's Lemhi Open Pit Resource

Cut-off	Indicated			Inferred		
	Tonnes (000's)	G/t	Ounces (000's)	Tonnes (000's)	G/t	Ounces (000's)
$1,275 Pit (0.025 g/t)	13,758	0.85	355.2	3,177	0.68	66.0
$1,400 Pit (0.023 g/t)	15,218	0.82	379.0	3,667	0.65	72.8

Source: Swanson et al., 2012.

The Practical estimates appear to have employed industry standard methodologies and statistical treatments being used today. However, the 2012 mineral resource estimate was not formalized in a NI 43-101 technical report, nor did it utilize the 2012 core and RC drilling, therefore the resource is considered historical in nature and has been superseded by their 2013 estimate and the current MRE presented in this technical report.

In 2013, after the completion of the 2012 drill program, Practical provided an informal update mineral resource that included all the 2012 drill results and included an estimate using all the data and an alternative estimate downgrading the historical drill assay results. There is no technical report or summary report provided by Practical that supports and provides the methodologies and assumptions for the 2013 resource estimates even though the appropriate categories are utilized as set forth in the CIM Definition Standards on Mineral Resources & Reserves (2014). A summary is provided in a 2019 unpublished internal report prepared for LGT by Brewer (2019). Practical delineated a mineral resource (not pit constrained) of 24.2 Mt (21,974,200 t) at 0.81 g/t for 569,631 oz Au in measured and indicated and 15. 19 Mt (13,781,831 t) at 0.61 g/t for 268,959 oz Au inferred using a cut-off of 0.24 g/t Au (Table 6-9). They also provided a pit constrained resource of 23. Mt (21,284,138 t) at 0.68 g/t for 464,480 oz Au using a lower cut-off of 0.21 g/t Au, see Source: Brewer, 2019

Table 6-10. The authors of this technical report are treating the 2013 mineral resource estimates as historical in nature and not a as a current mineral resource due to the fact that they were not supported by a technical report and little detail is available on the exact assumptions, methodology, or parameters employed to calculate the resource estimates, including the logic and reasons for downgrading the historical assay data. Accordingly, the 2013 mineral resource estimates presented below use the appropriate mineral resource categories as per CIM Definition Standards on Mineral Resources & Reserves (2014), however, the authors have not done sufficient work to verify these resource estimates as current mineral resources, as per the CIM Estimation of Mineral Resources & Mineral Reserves Best Practices Guidelines (2019), therefore they are considered historical in nature, and Freeman are not treating any of the historical resource estimates as current mineral resources or mineral reserves.

The historical resource estimations discussed in Section 6. 3, including all associated subsections are relevant in that they were prepared and calculated by reputable companies that were intimately familiar with, and knowledgeable about, the property and the geology and resource potential of the Lemhi Gold Property. These historical resources do provide an indication of the extent of mineralization identified by previous operators at the project. The authors of this technical report have not done sufficient work to classify any of the historical estimates in this section as current mineral resources, therefore, none of the historical estimates are being treated as current resources. Further work, including but not limited to infill and confirmatory drilling with appropriate standard reference materials and QA/QC protocols, along with additional metallurgical work is required. This work was completed in 2020-2022 by Freeman and is discussed and outlined in Sections 10, 11, 12, 13, and 14.

Table 6-9: Practical Mining's Revised 2013 Resource Calculation with 2012 Drill Results and 36% Downgrading of All Historical Drill Data

Technique	Cut-off (g/t)	Measured			Indicated			Inferred		
		Tonnes (000's)	g/t	Ounces (000's)	Tonnes (000's)	g/t	Ounces (000's)	Tonnes (000's)	g/t	Ounces (000's)
Ordinary Kriging	0.2	193,147	1.18	7,309	21,781,053	0.8	562,322	13,781,831	0.61	268,959
Inverse Distance Cubed	0.2	186,470	1.24	7,434	21,479,485	0.82	566,277	12,875,448	0.64	264,931
Nearest Neighbour	0.2	160,993	1.43	7,402	17,396,039	0.98	548,110	9,688,084	0.8	249,183

Source: Brewer, 2019

Table 6-10: Practical Mining's 2013 Comparison Between Global Mineral Resource at the LGT Property (Unconstrained) and Pit Constrained to The Patented Property (Patented) at a $1500 Gold Price

Pit Class	Ore > 0.49 g/t			Low Grade 0.21 – 0.49 g/t			Waste Tonnes
	Tonnes	Grade g/t	Ounces	Tonnes	Grade g/t	Ounces	
Unconstrained $1,500							
Measured	103,155	1.39	4,593	23,704	0.33	252	2,978
Indicated	9,476,520	0.99	302,164	4,299,732	0.36	49,369	827,885
Inferred	2,241,411	0.77	55,410	5,139,616	0.32	52,691	14,732,567
Nrm		0.00			0.00		75,720
None		0.00			0.00		37,886,893
Unconstrained $1,500 Total	11,821,086	0.95	362,167	9,463,052	0.34	102,313	53,526,042
Patented $1,500							
Measured	56,338	1.31	2,379	8,700	0.31	87	2,213
Indicated	3,901,749	1.05	132,013	2,165,361	0.35	24,047	598,024
Inferred	1,063,425	0.78	26,789	2,598,502	0.32	26,333	9,471,869
None							24,286,637
Patented $1,500 Total	5,021,512	1.00	161,181	4,772,564	0.33	50,467	34,358,743

Source: Brewer, 2019.

6.4 Historical Processing and Metallurgical Testing

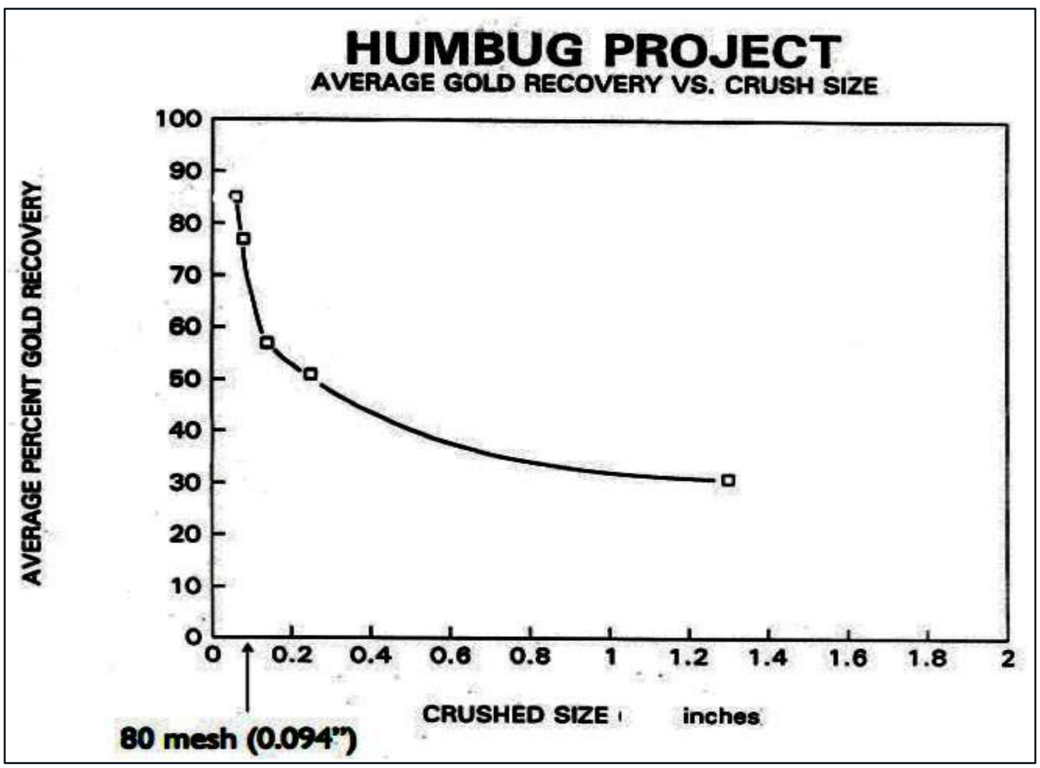
FMC analyzed drill cuttings for gold both by fire assay methods and cyanide leach analyses. Cyanide leach values varied widely from the fire assay values.

Hazen Research Inc. of Golden, Colorado performed cyanide leach tests in 1986 and performed bottle roll tests, column leach tests, agitated leach tests and flotation, and concentrate leach tests on five composite samples of drill cuttings in 1987. Head grades of the samples ranged from 0.032 to 0.104 opt Au, 0.071 to 0.22 opt Ag, and 0.023% to 0.127% Cu. Bottle roll results were disappointing with only 39.5% to 47.9% gold dissolution. Hazen's test results showed that fairly fine grinding was necessary to liberate fine-grained gold and achieve high gold recoveries. Gold in -100 mesh shake leach test residues occurred as very fine (10-12 µm free gold grains and 5 µm gold inclusions in pyrite). Hazen also concluded that the mineralized rocks had poor permeability characteristics, even after agglomeration (Shaw, 1987).

AGR commissioned Kappes, Cassiday and Associates (KCA) of Reno, NV to perform metallurgical testing on core and RC samples in 1994 and 1995. KCA was also contracted to prepare a "pre-feasibility" report on the Lemhi Gold (Humbug) Project to guide mine design. KCA conducted seven bottle roll tests and 22 column leach tests on mineralized core and cuttings. No information is provided in the KCA report regarding the nature of the samples (i.e. oxide vs. sulfide, location of samples within deposit), but it is assumed that material representative of the overall deposit was used. Column leach tests were performed on 10 sample types (three core composites, one quartzite sample, one phyllite sample, a quartz vein sample, one unidentified core sample, and a "mixed" sample). Column test sample size ranged from 20 kg to 90.7 kg (44-200 lbs). Head grades of the samples ranged from 0.01 opt to 0.188 opt (Defilippi, 1996).

The primary goal of the study was to determine the optimum crush size for heap leaching. Column leach tests were performed on agglomerated crushed samples. Samples were crushed and tested at five different sizes: - 2 inches, - ½ inch, - 4 mesh, as-received drill cuttings, and -8 mesh (particles <2. 38 mm. across). Gold recoveries ranged from 31% to 85% as a direct function of crush size (Figure 6-8). Gold recovery was found to increase with finer crush sizes down to -16 mesh. Crushing to smaller than -16 mesh yielded only minimal improvement in gold recovery.

Figure 6-8: Gold Recovery vs. Crush Size



Source: Defilippi, 1996.

Defilippi (1996) concluded:

> *"Metallurgical test results indicated that ore is amendable to cyanide heap leaching at a crush size of 90 percent minus 8 mesh with agglomeration at an average of 8. 5 pounds of cement per ton of ore. Gold recovery is projected at 80%. It is estimated that sodium cyanide consumption will be 1. 0 pounds per ton of ore."*

Silver head grades were not reported and no recoveries for silver were calculated. It is assumed that silver recovery would be insignificant.

Defilippi (1996) recommended additional column tests using a lower amount of cyanide to determine the effects of lower cyanide levels on gold recovery, leach time, and cyanide consumption.

6.5 Production History

Placer gold was produced from Ditch Creek between 1867 and 1877. There are no records of the amount of gold extracted. There is a significant amount of dredge tailings along Hughes Creek suggesting that placer gold was extracted along Hughes Creek in gold dredging and hydraulic operations likely in the 1890s to early 1900s. No information or records are available for this period of mining.

The Bull of Woods mine on the MS 784A patented claims produced an unknown amount of lode gold between 1891 and the early 1900s. There are no production records for the operation.

7 GEOLOGICAL SETTING AND MINERALIZATION

The following sections of Geology and Mineralization have been modified or taken directly from previous reports by Brewer (2019) and Cuffney (2011).

7.1 Regional Geology

7.1.1 Stratigraphy and Geologic Units

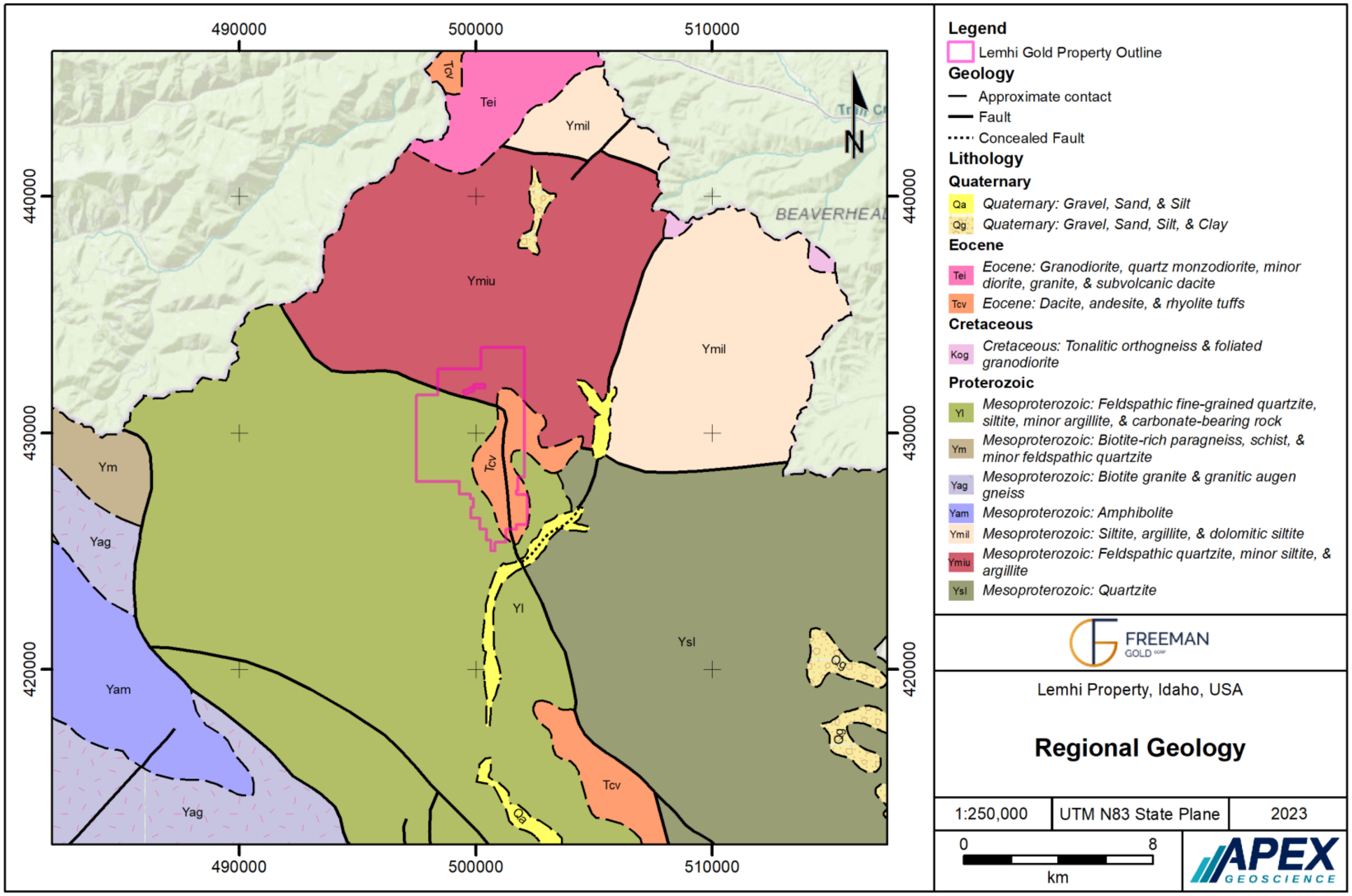
Bedrock in most of the Clearwater Mountains and Salmon River Mountains of east-central Idaho, is composed of Precambrian siliciclastic metasediments tentatively correlated with the Mesoproterozoic Belt Supergroup of Montana and southern B.C. (Figure 7-1; Link et al., 2007). These rocks were deposited in a large intracratonic basin between 1,470 and 1,400 million years ago (Ma).

The stratigraphy of the Belt Supergroup has been extensively studied. However, the similarity between the (rather monotonous) lithologies within the various formations makes identification of formations and units difficult. Lonn and McFaddan (1999) described the problem succinctly:

> *"Differentiating Belt Supergroup units in the field is difficult and describing them so others can recognize them is even more problematic. Neither grain size, color, nor mineralogy can be used to distinguish the formations. Argillite, siltite, and quartzite are found in virtually every formation; color is mostly a diagenetic or metamorphic feature, and mineralogy is quite uniform."*

Stratigraphic correlations are made even more difficult by the use of informal local unit names, lack of fossils for dating and correlation, and structural complexities due to folding and several periods of high-angle and low-angle faulting. As a result, there is often disagreement in geologic maps produced by different geologic mappers.

Figure 7-1: Regional Geology of the Lemhi Gold Project

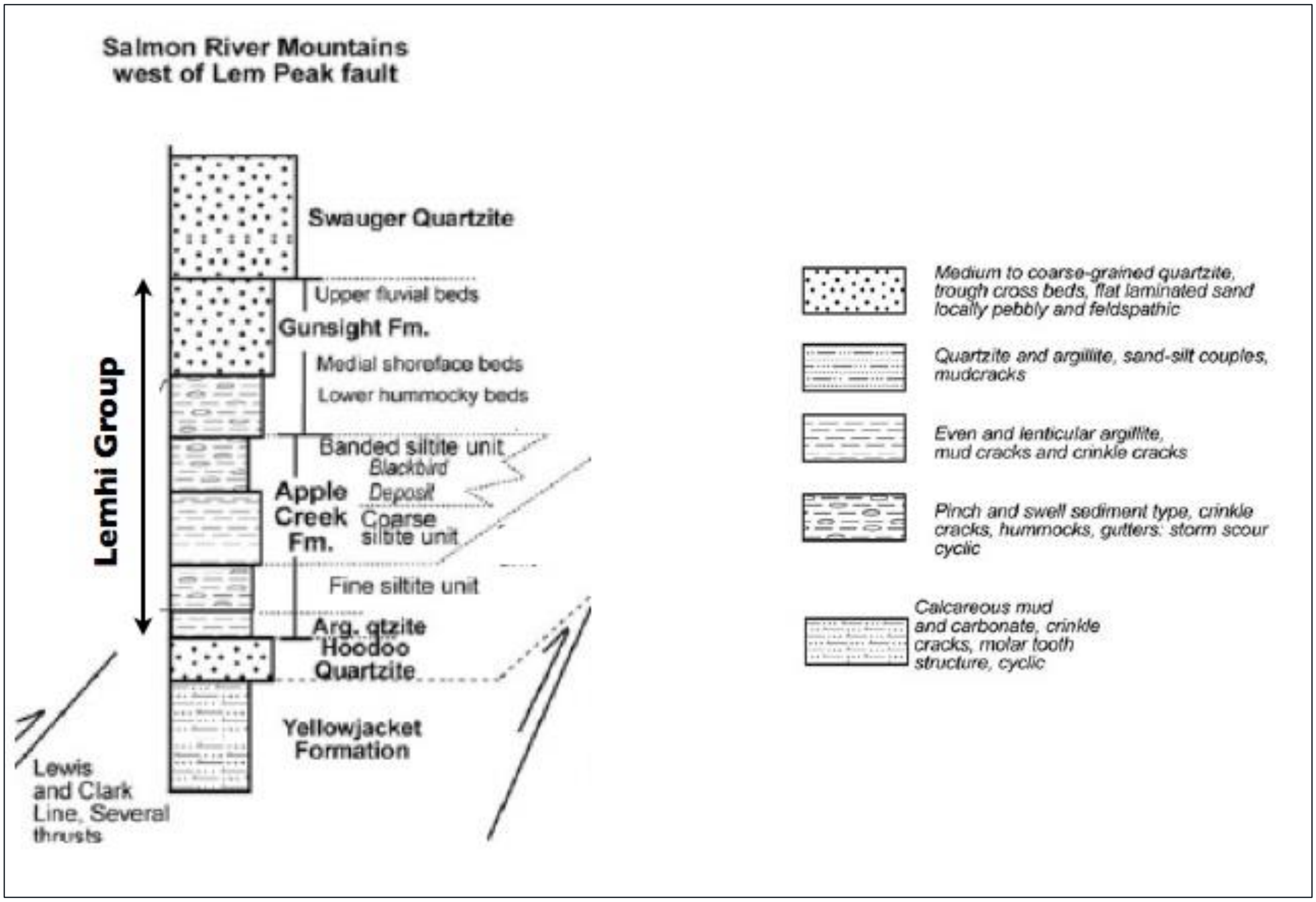


Source: APEX, 2023.

Lopez (1982) mapped the Gunsight, Apple Creek and Big Creek Formations of the Lemhi Group and the older Yellowjacket Formation in the Gibbonsville quadrangle.

Immediately to the west, in the adjacent Allan Mountain quadrangle, Stewart et al. (2009) mapped the Quartzite of Hughes Creek (Lemhi Group) and the Quartzite of Allan Mountain. Tysdal et al. (2003) and Lund et al. (2003) mapped the Gunsight Formation, Apple Creek Formation, Helena, and Empire Formations, and an unnamed feldsphathic sandstone (quartzite) unit within the Proterozoic package in the Gibbonsville and Allan Mountain areas. A stratigraphic section is presented below in Figure 7-2.

Figure 7-2: Regional Stratigraphy of the Lemhi Group



Source: Cuffney, 2011.

Intrusive rocks of Precambrian to Tertiary age cut the Mesoproterozoic rocks and Paleozoic rocks south and west of the Gibbonsville area. A large Middle Proterozoic granite pluton intrudes the metasediments to the southwest of the Lemhi Gold Project and is flanked on its northeast side by a body of amphibolite as shown in Figure 7-1. Two Cretaceous granite to granodiorite bodies lie to the west-southwest of the area. The Painted Rock pluton, a small granitic batholith of Eocene age, is exposed ~35 km to the west of the project. Cretaceous to Eocene diorite to granodiorite of the Chief Joseph plutonic complex is exposed in the headwaters of the North Fork of the Salmon River, and small dikes and sills of similar age and composition occur throughout the Gibbonsville area.

Mapped thrust relationships are questionable because faults locally place younger Eocene volcanic rocks of the Challis Volcanic Group over the Precambrian rocks. The volcanic rocks consist of intermediate to mafic lava flows and intermediate to felsic pyroclastic rocks. The Challis volcanics were derived from several large calderas located to the southwest of the Lemhi Gold Property, including Thunder Mountain, Van Horn Peak, and the Twin Peaks calderas, and from the Mount Withington caldera, located south of Salmon. The volcanic rocks are very thick within the calderas, but thin rapidly away from the effusive centers. Smaller volcanic centers, which sourced local intermediate-composition lava flows, occur throughout Lemhi County.

7.1.2 Structure

East-central Idaho lies within the Cordilleran fold and thrust belt, a wide zone of folding and thrust faulting produced by east-northeast/west-southwest compression during the Cretaceous Sevier orogeny. Several northwest-trending regional thrust faults, including the Poison Creek fault and the Brush Creek fault, have been mapped by the USGS (Evans and Green, 2003). These faults are regional in nature and generally thrust the Yellowjacket or equivalent units over the younger Lemhi Group. Large areas of brecciated quartzite and siltite are often associated with the fault zones. Recent geological investigations have shown that many of these mapped faults are actually stratigraphic contacts, zones of normal faulting, decollements associated with folds, or deformation zones produced during bedding-parallel shearing (Link and Janecke, 1999). Winston et al. (1999) note that most geologists tend to map all contacts between the Lemhi Group and the older Yellowjacket Formation as a thrust fault, when in many places the contact is conformable. The Medicine Lodge thrust fault is often mapped as Lemhi Group over Yellowjacket Formation (younger-over-older). Winston et al. (1999) interpret that relationship to actually be the normal stratigraphic relationship of Gunsight Formation on top of Apple Creek Formation.

The Gibbonsville area lies along the Trans-Challis fault system, a broad (20-30 km-wide) system of en-echelon northeast-trending structures extending from Idaho City, Idaho for more than 270 km to the northeast to the Idaho-Montana border (Kiilsgaard,et al., 1986). The Trans-Challis fault system is one of many structures within the Idaho-Montana porphyry belt which parallels the contact between the Cordilleran Fold and Thrust belt and the Idaho batholith and corresponds to a zone of strike-slip faults and northeast-trending magnetic features. The Idaho-Montana porphyry belt encompasses a wide northeast-trending alignment of porphyry-related deposits (Figure 7-3; Hildebrand et al., 2000).

Basin-and-range style extensional faulting has broken much of the area into north-northwest-trending horsts and grabens or half grabens. Extensional faulting was initiated between 17 Ma and 5 Ma and continues to this day, as evidenced by the magnitude 7. 3 Borah Peak earthquake of 1983.

7.1.3 Regional Mineralization

The Lemhi Project lies within the Idaho-Montana porphyry belt, a northeast-trending alignment of metallic deposits related to granitic porphyry intrusions that extend north-easterly across Idaho from the Boise Basin in west-central Idaho to the Little Belt Mountains of central Montana, see Figure 7-3. Within the mineral belt, much of the mineralization is related to the Trans-Challis fault system, a broad (20-30 km-wide) system of en-echelon northeast-trending structures extending from Boise Basin more than 270 km to the Idaho-Montana border (Kiilsgaard et al., 1986). Mineralization related to the Trans-Challis fault system includes porphyry molybdenum deposits (Thompson Creek, Napoleon Hill), epithermal and intrusion related gold-silver veins and stockworks (Silver City, Stibnite), uranium and thorium veins, stratiform copper-cobalt deposits (Blackbird), and fluorite vein and breccia deposits.

Gold deposits in the north part of the Trans-Challis belt, located south of the Lemhi Gold Project include the Beartrack mine from which FMC produced 650,000 oz Au between 1995 and 2000 (Hatch, 2008); and the Grouse Creek gold deposit mined by Hecla, see Figure 7 3.

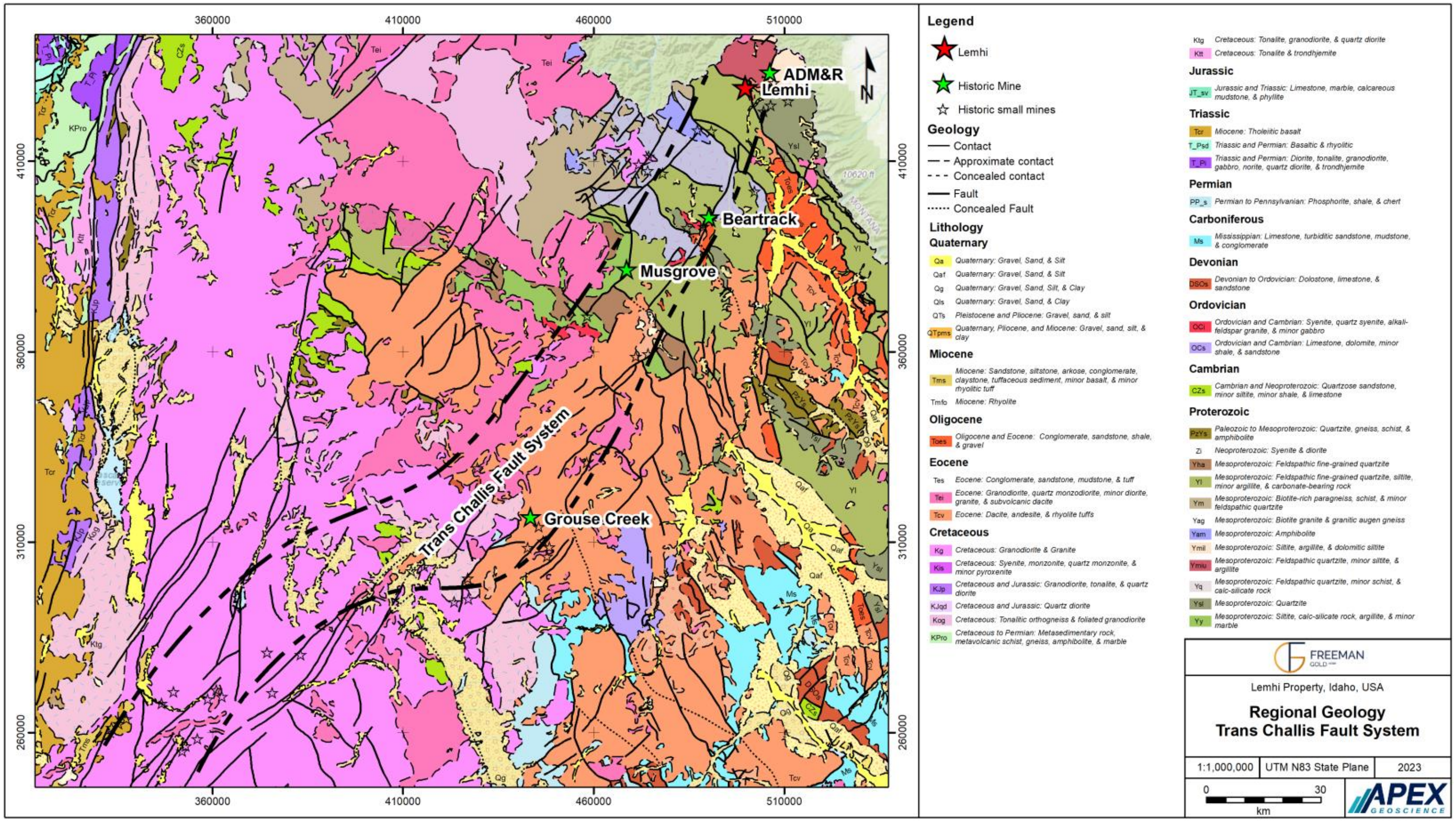
These gold deposits are not “adjacent properties” to the Lemhi Gold Project and are noted only to indicate that the Lemhi Property lies within an important mineral belt. The authors have not verified the published production figures for the Beartrack and Grouse Creek mines nor do they mean to imply any size or grade relationship between these deposits and the Lemhi Property. This information is not necessarily indicative of the mineralization known or to be expected on the Lemhi Property.

7.2 Property Geology

The Lemhi Property is largely underlain by quartzites and phyllites of the Mesoproterozoic age. Porphyritic dacite flows of the Eocene Challis volcanics are preserved in down-dropped fault blocks on the east side of Little Ditch Creek and the south end of Ditch Creek valley. The valley between Ditch Creek and Little Ditch Creek is filled with coarse Quaternary gravels composed of subround quartzite cobbles and boulders. Drilling has determined that the gravels are up to 20 m thick in places. Perched gravels of similar composition lie along the ridge on the west side of Ditch Creek. Cobbles and boulders of quartzite derived from the gravels mantle the hillside down-slope of the perched gravel deposits.

Evans and Green (2003) mapped a west-northwest-trending thrust fault passing through the northern part of Ditch Creek. The thrust fault places Mesoproterozoic metasediments of the Gunsight Formation of the Lemhi Group over an unnamed feldsphathic quartzite unit in the upper reaches of Ditch Creek, shown in Figure 7-4. Stewart et al. (2009) mapped a similar fault but interpreted the relationship as the Quartzite of Hughes Creek thrust over the Quartzite of Allan Mountain. The Quartzite of Hughes Creek is a local unit interpreted to be part of the Lemhi Group. The Quartzite of Allan Mountain is also a local map unit, which is interpreted to correlate with the slightly younger Missoula Group, thus the fault relationship would be a thrust with older rocks over younger ones.

Figure 7-3: Location of Lemhi Gold Project Within Trans-Challis Fault System and Related Gold Deposits

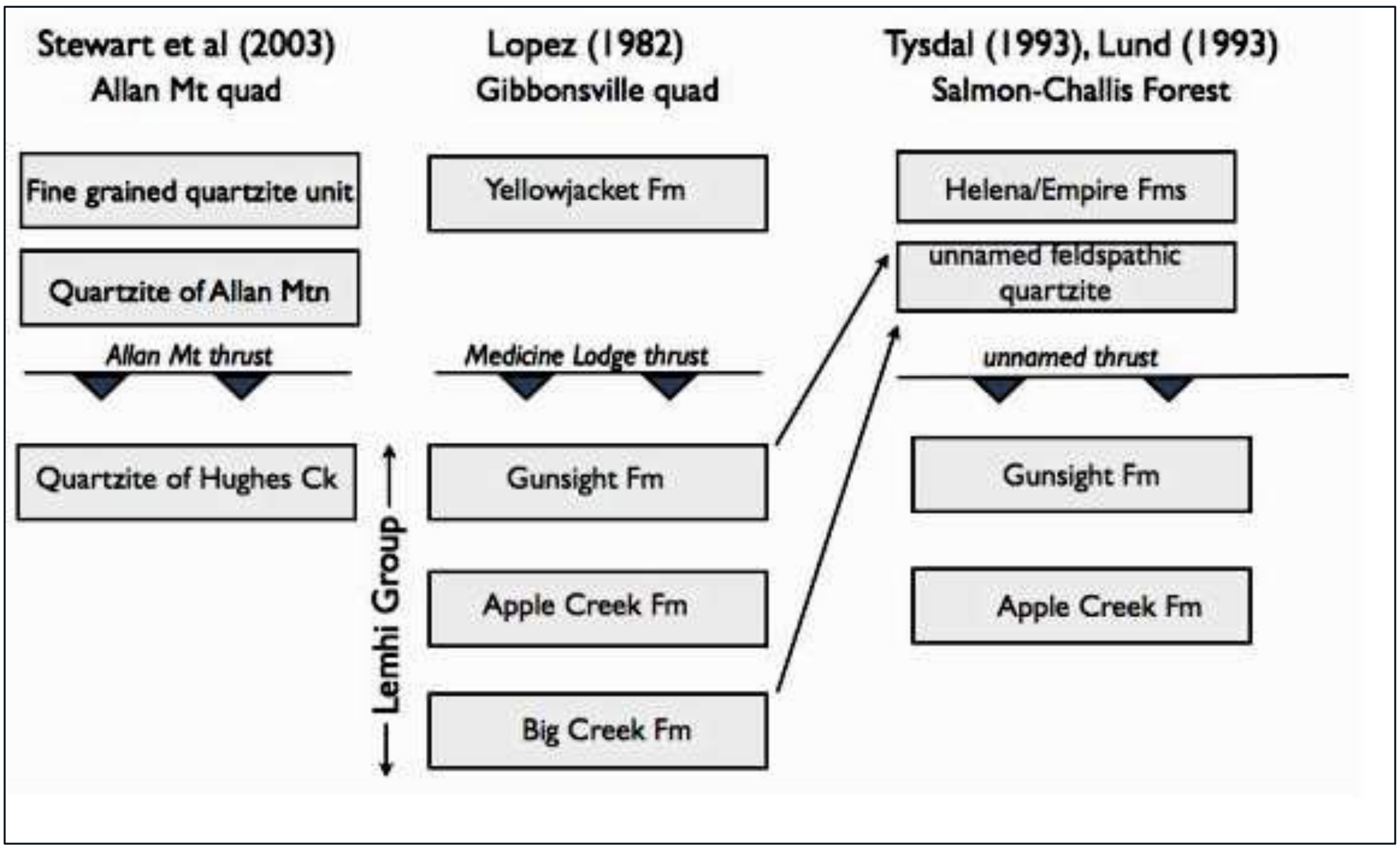


Source: APEX, 2023.

The project lies within a structurally complex region defined simplistically by an east-westerly striking thrust fault (southerly dipping), which is offset by north-northeast-trending, high-angle basin-and-range normal faults (block faulting) to the east and west. Numerous intrusions of late Cretaceous to early Tertiary age are also widespread throughout the area (Brewer, 2019).

Lopez (1982) mapped the Medicine Lodge thrust fault to the east of Ditch Creek. However, Lopez mapped the Big Creek and Gunsight Formations of the Lemhi Group in the upper plate of the Medicine Lodge thrust, in areas where Tysdal et al. (1993) mapped the unnamed feldspathic unit. Lopez mapped Yellowjacket Formation in the lower plate, which would produce a younger-over-older relationship. Given the similarity of lithologies in the Belt Supergroup it may well be that the assignment of rocks in the upper plate of the thrust to the Yellowjacket Formation by Lopez and FMC is incorrect, and the older-over-younger thrust relationship noted by Stewart et al. (2009) may be correct. Figure 7-5 attempts to correlate the various units mapped in the Gibbonsville area.

Figure 7-4: Correlation of Mapped Units in the Gibbonsville Area



Source: Apex, 2023

Regardless of stratigraphic nomenclature and relative movement of fault blocks, it is obvious that a large low-angle fault passes through Ditch Creek and has produced a wide zone of sheared and brecciated rock (Cuffney, 2011). Stewart et al. (2009) divided the rocks in the Ditch Creek area into three structural domains of which two domains are separated by the thrust fault (a low-angle ductile shear): Domain 3 in the hanging wall, characterized by south to northwest-plunging fold axes; and Domain 2 in the footwall, characterized by strongly folded rocks with northwest-trending sub-horizontal fold axes.

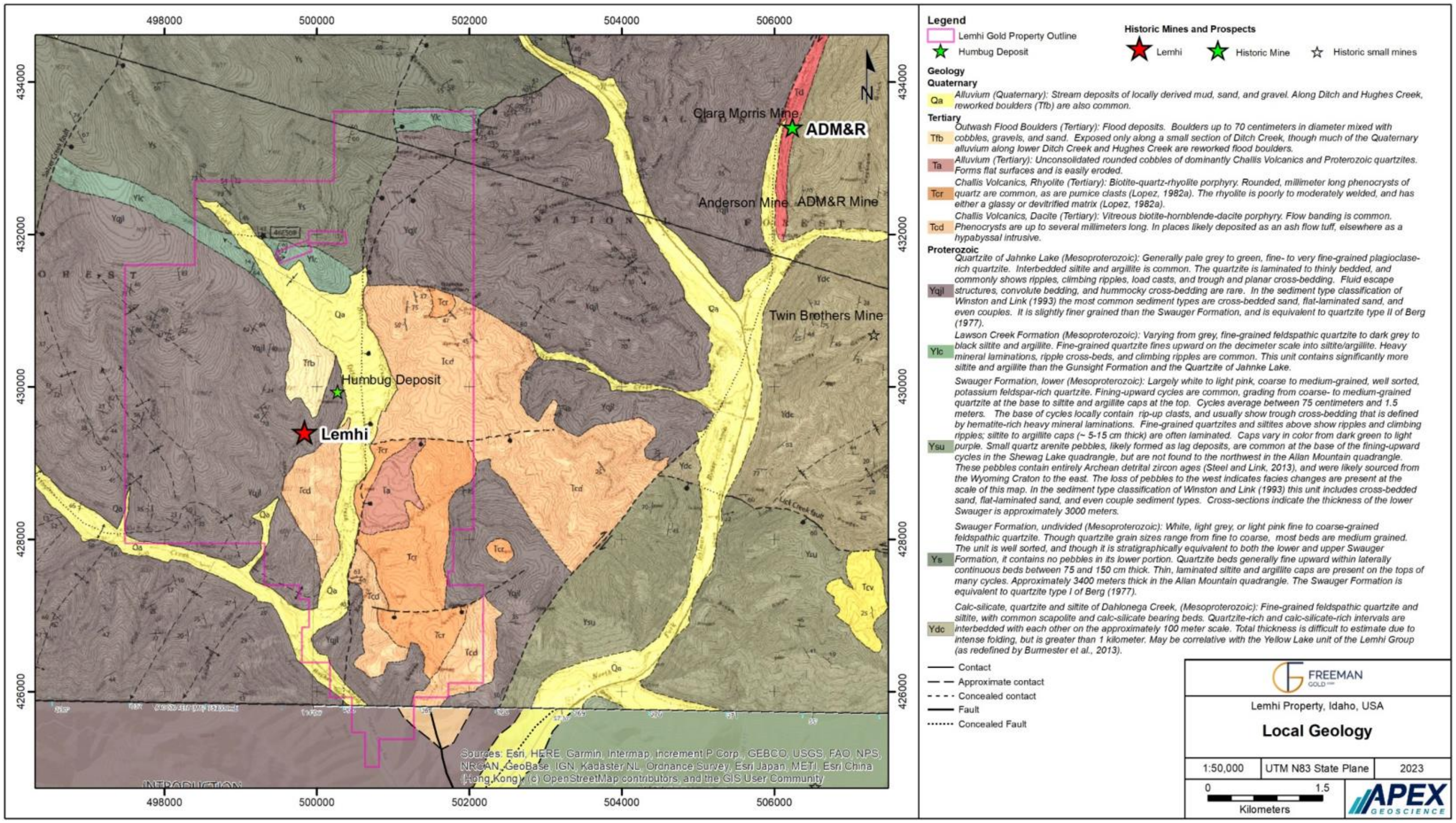
Cuffney (2011) summarizes mapping completed by Evans and Green (1993), which shows that the thrust fault is offset and rotated from a northwest orientation to a west-northwest orientation across Ditch Creek. The trace of the fault is lost in Quaternary gravels in Ditch Creek and it is not known if the fault merely bends to the north or is offset by a normal fault.

Geologists working for FMC and AGR interpreted the unit in the upper plate of the thrust fault at Ditch Creek to be correlative with the Lemhi Group (Gunsight and Apple Creek Formations) and the rocks in the lower plate to be part of the Yellowjacket Formation (Bertram, 1996), in accordance with mapping of the Gibbonsville quadrangle by Lopez (1982), which was the only published geologic mapping in the area at the time. Mineralization is interpreted to be hosted by phyllitic quartzite of the Apple Creek, as mapped by Lopez along the west side of Ditch Creek.

Geologists working for Freeman Gold Corp have identified and described a unit in the northeast and eastern portion of the main deposit area as an interbedded brecciated siltstone and purple quartztie. This package is clearly distinct from the quartzite and siltite units belonging to the Yellowjacket Formation and sits in an apparent-conformable hangingwall position to the Yellowjacket package. Rare, to no quartz, veins occur in the brecciated siltstone and purple quartzite unit. No anomalous gold has been detected in the siltstone/purple quartzite unit. Its lower contact is often gouge mixed with brecciated porphyry, Yellowjacket, or mineralized quartz veins. The lower contact of this unit may correspond to the thrust fault mapped by Evans and Green (1993) and summarized by Cuffney (2011).

The low-angle fault at Ditch Creek created a sub-horizontal zone of shattering and brecciation, along which sill-like bodies of quartz-feldspar porphyry intruded. The intrusive rocks are not exposed on the surface but have been encountered in numerous drill holes. The porphyry is described as a quartz diorite but is compositionally and texturally a dacite or rhyodacite. The groundmass and feldspar phenocrysts in the porphyry have largely been altered to kaolinite and sericite. The groundmass was initially very fine-grained to glassy, suggesting high-level hypabyssal emplacement (Cuffney, 2011 and references therein). Geist (1995) studied thin sections of the volcanic rocks at Lemhi and concluded that they were all welded ash-flow tuffs derived from distant sources, based on eutaxitic fabric of the rocks.

Figure 7-5: Property Geology of the Lemhi Gold Project



Source: APEX, 2023.

7.3 Mineralization

Gold deposits in the Dahlonega mining district consist of two types of mineralization: gold-bearing lodes (quartz veins and stockworks), and placer gold deposits derived from weathering of the veins, which were mined in drainages a short distance downstream from the lode deposits. Gold occurs both as lodes (Lemhi Gold Deposit) and placers on the Lemhi project. Extensive placer mining took place in the drainage of Ditch Creek in the late 1800s to early 1900s. Current exploration is targeting the lode gold mineralization.

Previous interpretation of the mineralization by Cuffney (2011) states that:

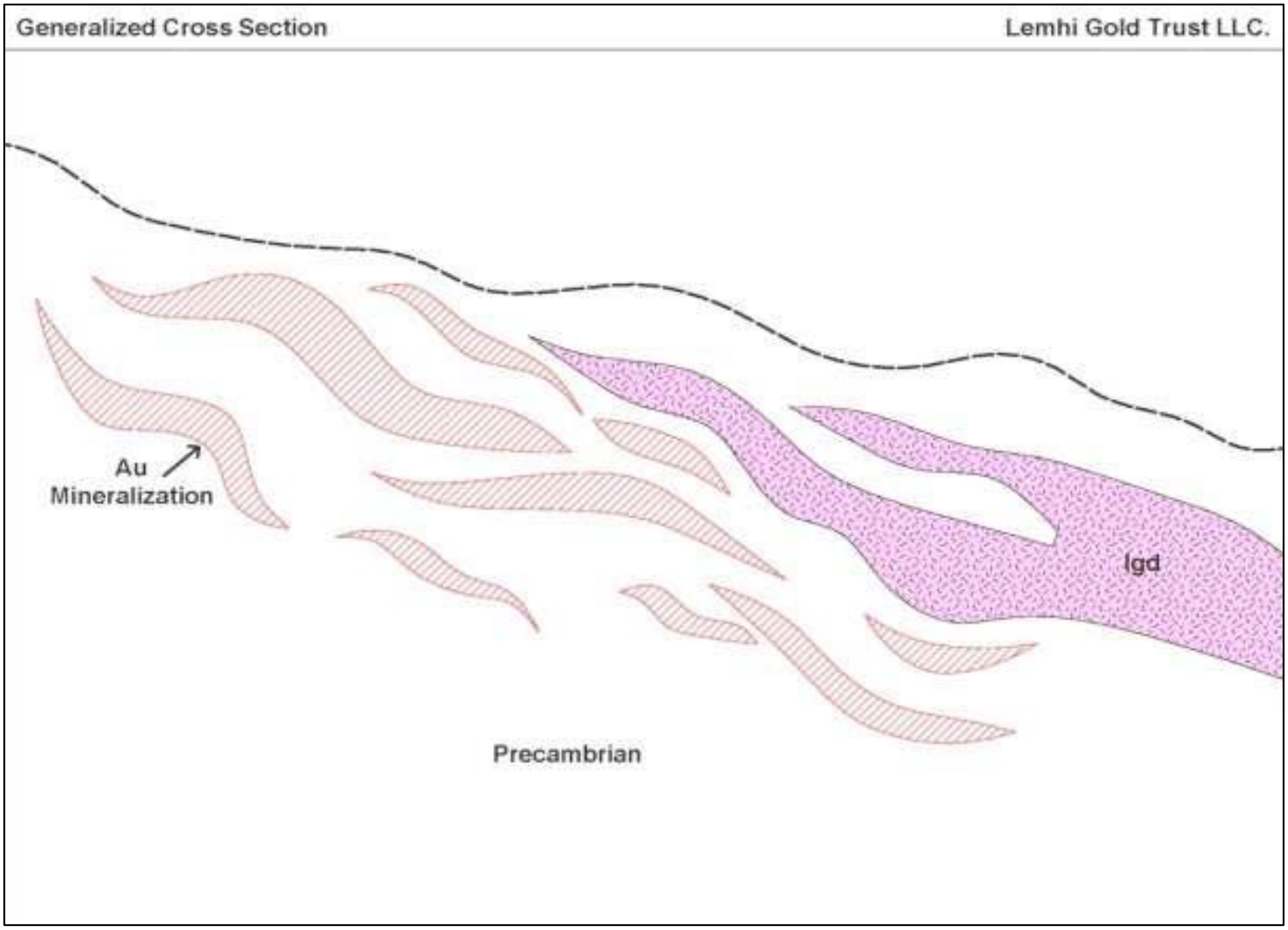
> *"…gold accompanied by minor silver and copper mineralization is spatially and likely genetically related to sub-horizontal dikes/sills that intrude quartzites and phyllites of the Lemhi Group (Gunsight and Apple Creek Formations) in the hangingwall of a low-angle (thrust?) fault. Mineralization occurs as swarms of gold-bearing quartz veins and silicified zones. Quartz veining, silicification, and gold mineralization occur in low-angle zones of sheared/cataclastic phyllite generally dipping gently up to 25° to the southeast. Mineralization more or less surrounds the quartz porphyry intrusions. Thicker and higher-grade gold mineralization occurs in the footwall of the low-angle dike/sill, whereas mineralization above the intrusion is thinner and lower grade. Mineralization is also concentrated along the western terminus of the main intrusive. Minor precious metals mineralization occurs within the intrusions, suggesting that they are pene-contemporaneous with the mineralization."*

Reinterpretation based on the results of the 2012 core drilling program suggested that the deposit is a structurally controlled hydrothermal deposit associated with varying amounts of sulfides in a quartz-carbonate gangue hosted by late-Proterozoic metasediments within the structurally complex Trans-Challis fault system (Brewer, 2019). It is further suggested that gold mineralization was introduced during a tectonically active period and is likely temporally related to intrusive activity associated with the Idaho Batholith. Gold mineralization has a strong association with base metal copper (Cu) and molybdenum (Mo) mineralization and occurs as multiple hydrothermal (epithermal – mesothermal) silica replaced structures resembling multiple flat-lying veins, see Figure 7-6.

7.3.1 Mineralogy

Precious metals mineralization at the Lemhi Project occurs within a gangue of quartz and minor carbonate (magnesite or ferroan dolomite). Bartlett (1986) identified magnesite as the most abundant alteration mineral, occurring as veinlets cutting both quartz veins and wall rocks. Overall gold mineralization has a low sulfide content, normally less than 2%, but pockets of high sulfide concentration have been noted. Pyrite and lesser chalcopyrite and molybdenite are the dominant sulfide species. Bornite, digenite, and traces of galena, sphalerite, pyrargyrite, and arsenopyrite have also been identified. Silver is present in small amounts with silver to gold ratios usually less than 1:1.

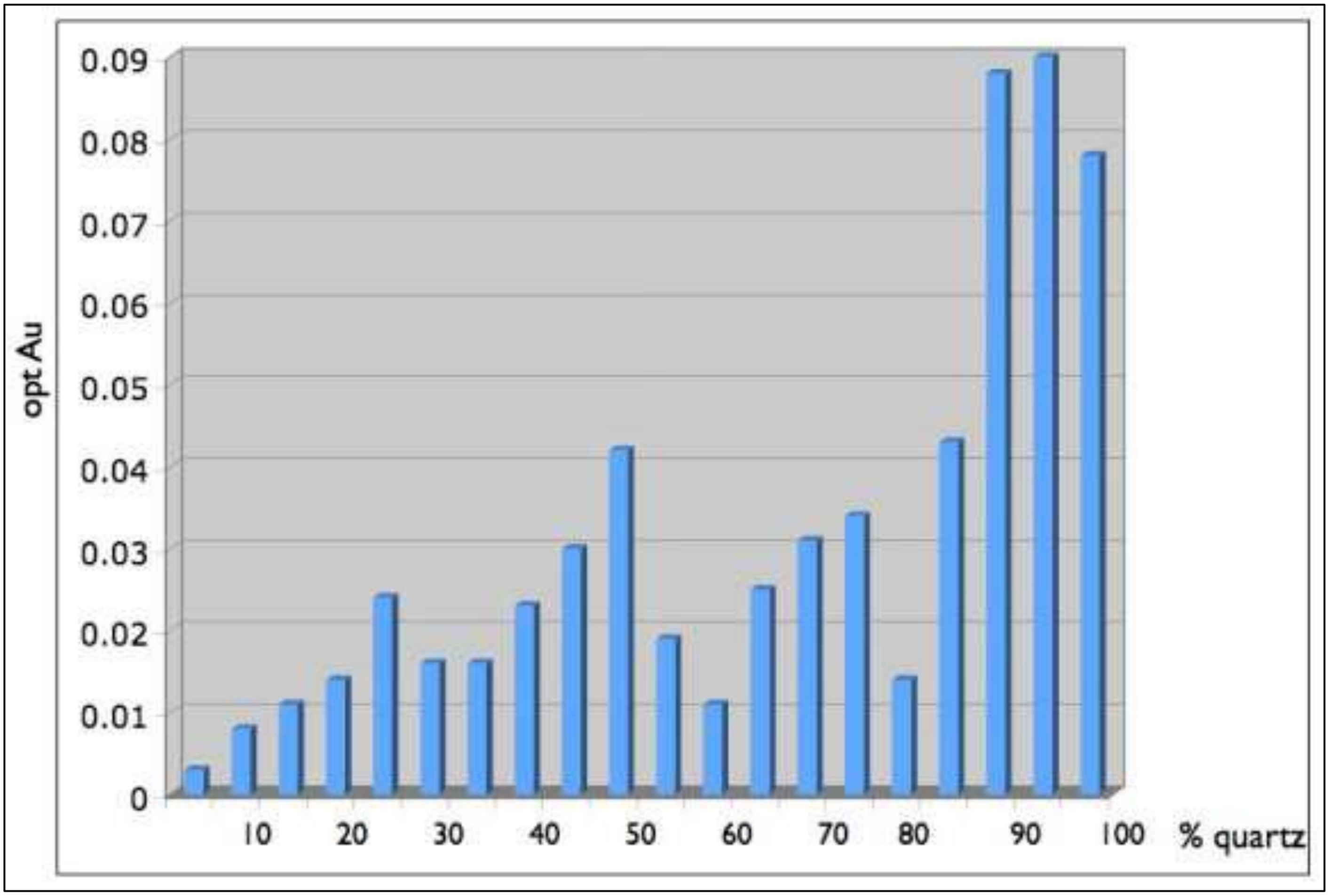
Figure 7-6: Generalized Cross-Section of Mineralization at Lemhi



Source: Brewer, 2019.

Gold is nearly always associated with quartz veining or quartz flooding. McCarter (1985) observed that gold intercepts in drilling correlated with zones of >20% quartz. Sandefur et al. (1994) performed a statistical analysis of mean gold grade vs lithology and alteration codes (9,723 code entries) from the 1985-1994 drilling database (Figure 7-7). Gold grades were found to be closely related to quartz veining. Two peaks in gold grade were found at quartz concentrations of 35%-50% and at 85%-95%. The latter range averaged 2.65 g/t Au. However, high quartz intervals account for only a small percentage of the volume of the deposit and the bulk of gold mineralization contains 15%-45% quartz. Intervals with <15% quartz veining contained little or no gold.

Figure 7-7: Graph of Gold Grade (Opt Au) vs Quartz Content Based on 9. 723 Alteration Codes Compiled by PAH

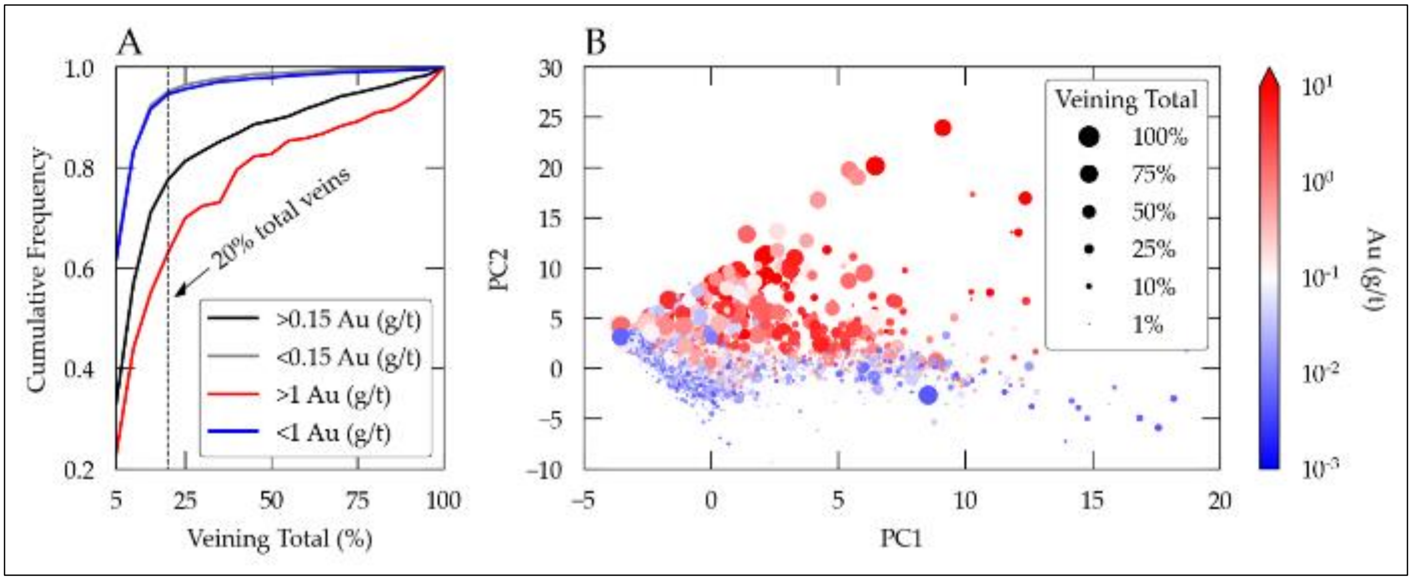


Source: Sandefur et al., 1994.

A 2023 review of assayed samples from all drilling generations with logged veining percentages (21,657 entries) agrees with the >20% veining correlation observed by McCarter (1985). A current study of samples >2.65 g/t Au supports the findings of Sandefur et al. (1994) see Figure 7-7; however, the new lower peak is centered closer to 15-20% quartz vein concentration. Plotting the cumulative frequency of sample groups with varying gold concentrations reveals that quartz veining is more prolific at higher grades, see Figure 7-8. Furthermore, groupings of samples at both <0.15 g/t and <1 g/t Au largely have a total vein concentration of <20%. Conversely, 20-40% of the higher Au concentration groups have total vein concentrations >20%.

Principle component analysis of samples from 2020-2022 Freeman drilling with multi-element geochemistry revealed a similar association between vein content and Au concentration. It should be noted that the first two principal components, PC1 and PC2, explain 33% of the dataset variation. While PC1 assigns positive weights to a mix of chalcophile and siderophile elements (e. g., In, Sc, Fe, V, Zn, Co, Ca, Cr, and Al), both PC1 and PC2 assign high negative scores to the lithophile elements La, Th, and Ce. Unlike PC1, PC2 is weighted towards chalcophile elements, giving positive scores to Bi, Ag, S, and Cd. Plotting of PC1 against PC2 reveals a grouping of Au concentration and total vein percentage. In Figure 7-8, a total vein percentage >20-25% and an Au concentration >0.1g/t are correlated to a PC2 value >4. This correlation indicates a positive relationship between high vein density (>20%), higher Au concentration (>0.1g/t Au), and path-finder elements Bi, Ag, and Te.

Figure 7-8: Distribution of Gold Grade and Total Vein Density at Lemhi.



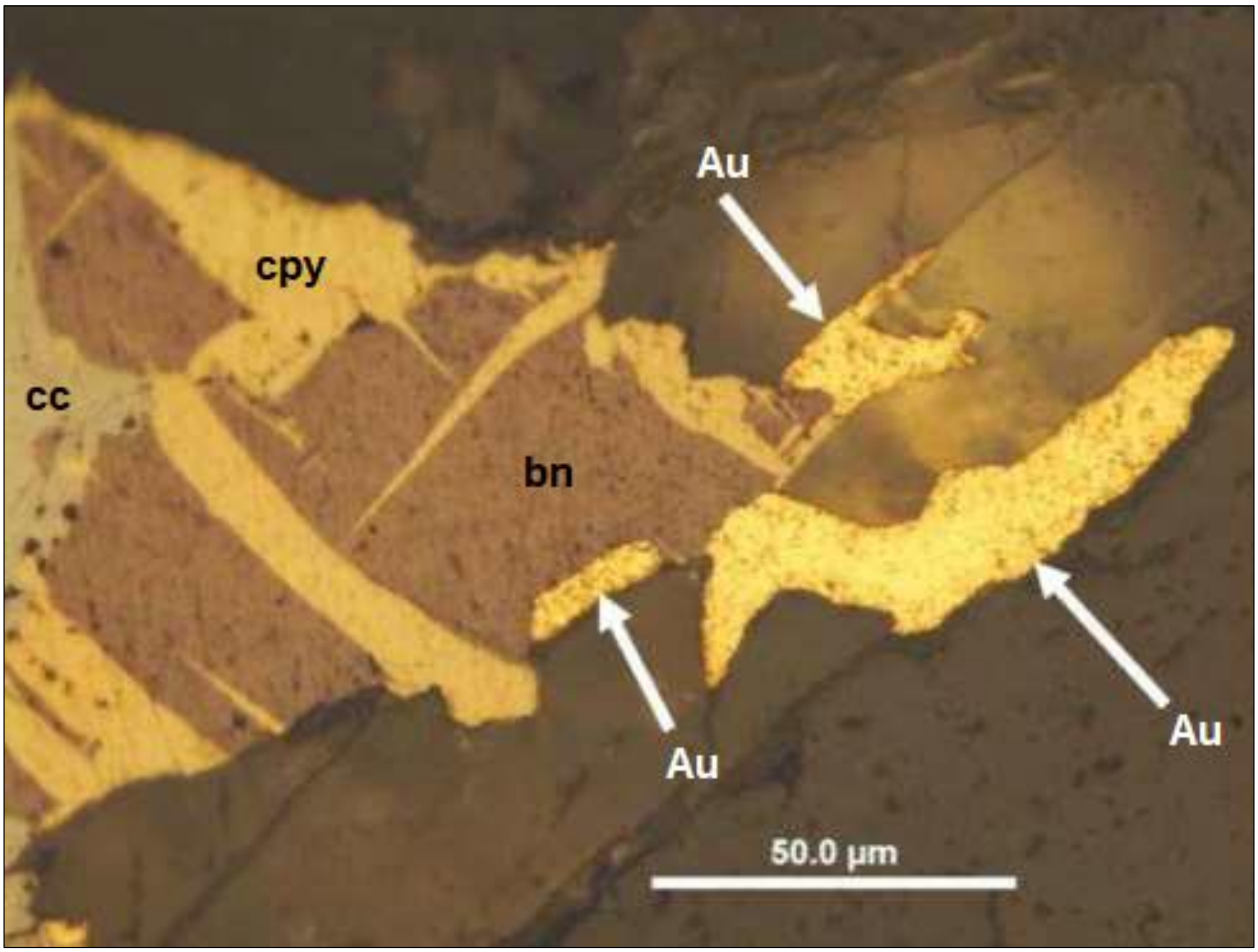
Source: APEX, 2023.
Note:
A) Cumulative frequency distribution Using a 0.15 g/t and 1 g/t Au grade cut-off for all drilling generations.
B) Principal component analysis of freeman gold corp drilling. PC1 plotted against PC2, sized by the percentage of total veining, and coloured by Au concentration (g/t).

Oxidation generally extends 30-50 m below the surface (Bertram, 1996). Gold occurs largely as free gold in the oxide zone. Gold grade below the redox zone correlates with sulfide content, suggesting that gold in primary occurs as auriferous pyrite or within/on copper sulfides. Paster (1986) studied polished sections of mineralized core and established that gold occurs as irregular blebs in bornite and as small blebs intergranular to dolomite veinlets, in some cases in association with chalcopyrite. Gold found by Paster was fine-grained (2 to 25 µm) but one sample contained a gold grain that was 70 µm across. Petrographic work performed by Hazen Research on -100 mesh leach residues found fine (10-20 µm) gold as free grains and very fine (5 µm) gold as inclusions within fine-grained pyrite (Shaw, 1987). Gold and associated base metal sulfides were emplaced late in the paragenetic sequence.

In 2012, LGT sent 13 core samples consisting of four igneous and 9 metasedimentary rocks, for petrographic analysis. All samples exhibited some degree of hydrothermal alteration, notably carbonate veining/flooding. Thompson (2012) noted sample number 005C-168 was a calcareous quartzite with ferroan calcite-sulfide alteration see Figure 7-9 and Figure 7-10. Thomson (2012) described the sample as containing:

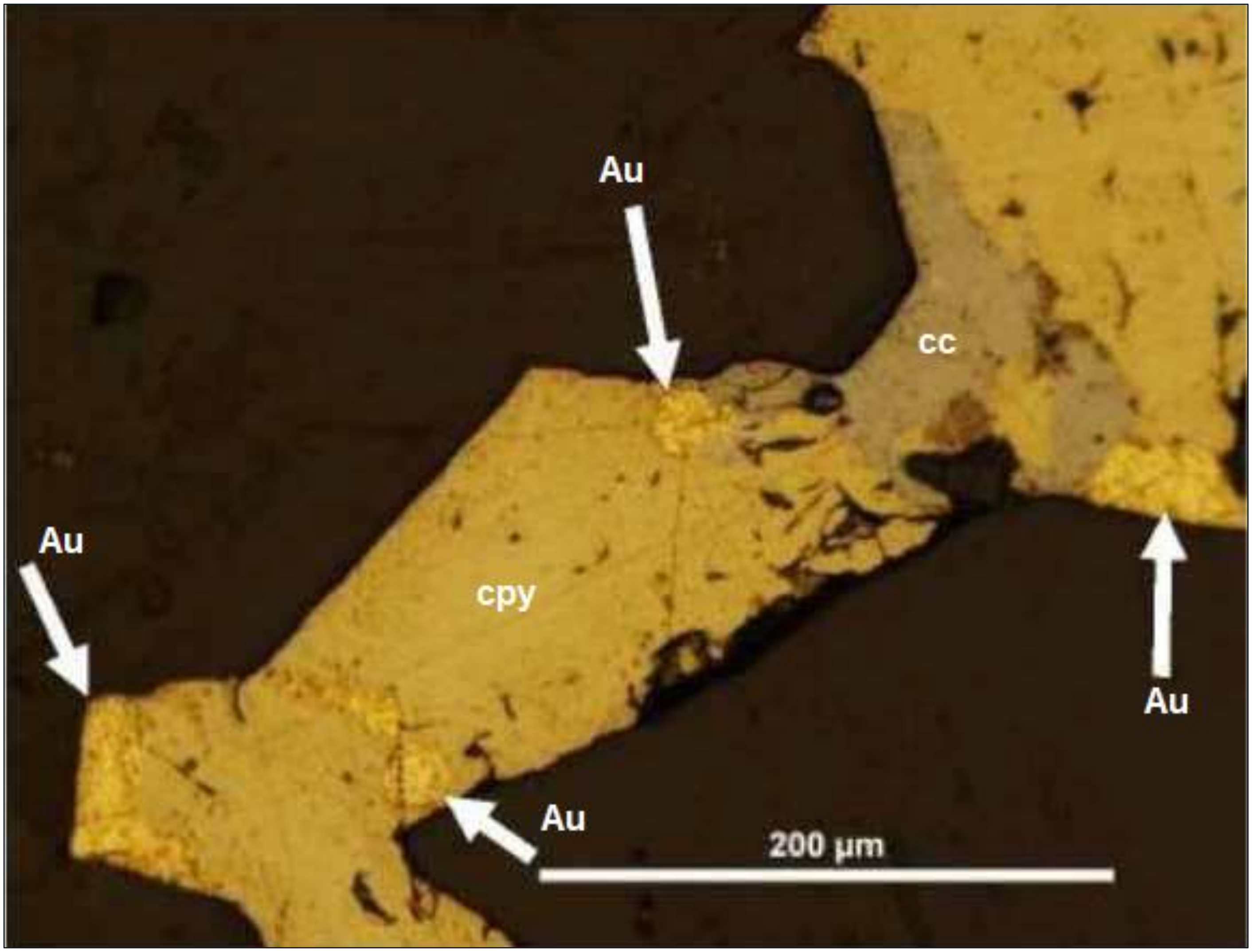
"Sutured quartz grains with veinlet-bedding-controlled sulfides-ferroan calcite. Some sulfides appear styolitic in form or along sutured quartz veins. Sulfides present in order of formation are pyrite→chalcocite→bornite→ chalcopyrite→gold. Locally, vugs with a quartz druse have the sulfide assemblage coating the quartz. Pyrite is invariably brecciated and cemented by later Fe-Cu sulfides. Traces of monazite are present in this sample. The sulfide paragenetic relationships are consistent with early pyrite that is broken followed by chalcocite, bornite, chalcopyrite and gold. There is invariably close spatial association between the sulfides and ferroan calcite."

Figure 7-9: Petrographic Sample 005C-168 Showing Gold Along Boundary of Sulfide Minerals



Source: APEX, 2023

Figure 7-10: Petrographic Sample 005C-168, Vug Infilled by Sulfide Minerals, Crosscut or Rimmed by Gold



Source: APEX, 2023

7.3.2 Deposit Character and Geometry

The Lemhi Gold Deposit is exposed in road cuts and trenches on the slope along the west side of Ditch Creek. The eastern one-half of the deposit lies under cobble to boulder gravels in the Ditch Creek valley and has no surface expression. Mineralization exposed in artificial cuts is characterized by intense brecciation, quartz veining, and flooding, and abundant hematite staining. Brecciation and silicification are so intense that protoliths are difficult to determine. The rocks are microbrecciated to the point of being cataclasites. Williams (1984) studied thin sections of mineralized material and mentioned that he had difficulty identifying the protolith due to intense crushing and alteration. Several of the samples were described as mylonitized breccias. In 1989 FMC drilled a -45° core hole (C-4) across the deposit in hopes of obtaining oriented core for structural analysis. Schaubs (1989) laments that, "the section sampled by the core hole was crushed and broken to such a degree that oriented core was impossible to complete." Schaubs also noted that, "These sediments have been contorted and brecciated, and shears and fault zones are present throughout the length of the hole."

The Lemhi Gold Deposit has a footprint of 1000 m east-west by 1100 m north-south and is defined to a depth of 240 m below surface, based on grade x thickness plots. Higher-grade mineralization in the northern part of the deposit has a strong west-northwest alignment. McCarter (1988) describes this high-grade zone as 395 m long by 75 m wide and up to 30 m thick. West-northwest high-angle structures were noted in trenches and road cuts during the fieldwork conducted at the time, and are probably responsible for this trend. A strong northeast trend (035°) and a weaker parallel northeast trend further to the east are also indicated by the grade-thickness contours, see Figure 7-11. Both the west-northwest and northeast high-grade zones are interpreted to be mineralization concentrated at intersections of high-angle structures with the broad low-angle fault zone. In the core of the deposit, the low-grade envelope of mineralization is greater than 200 m thick.

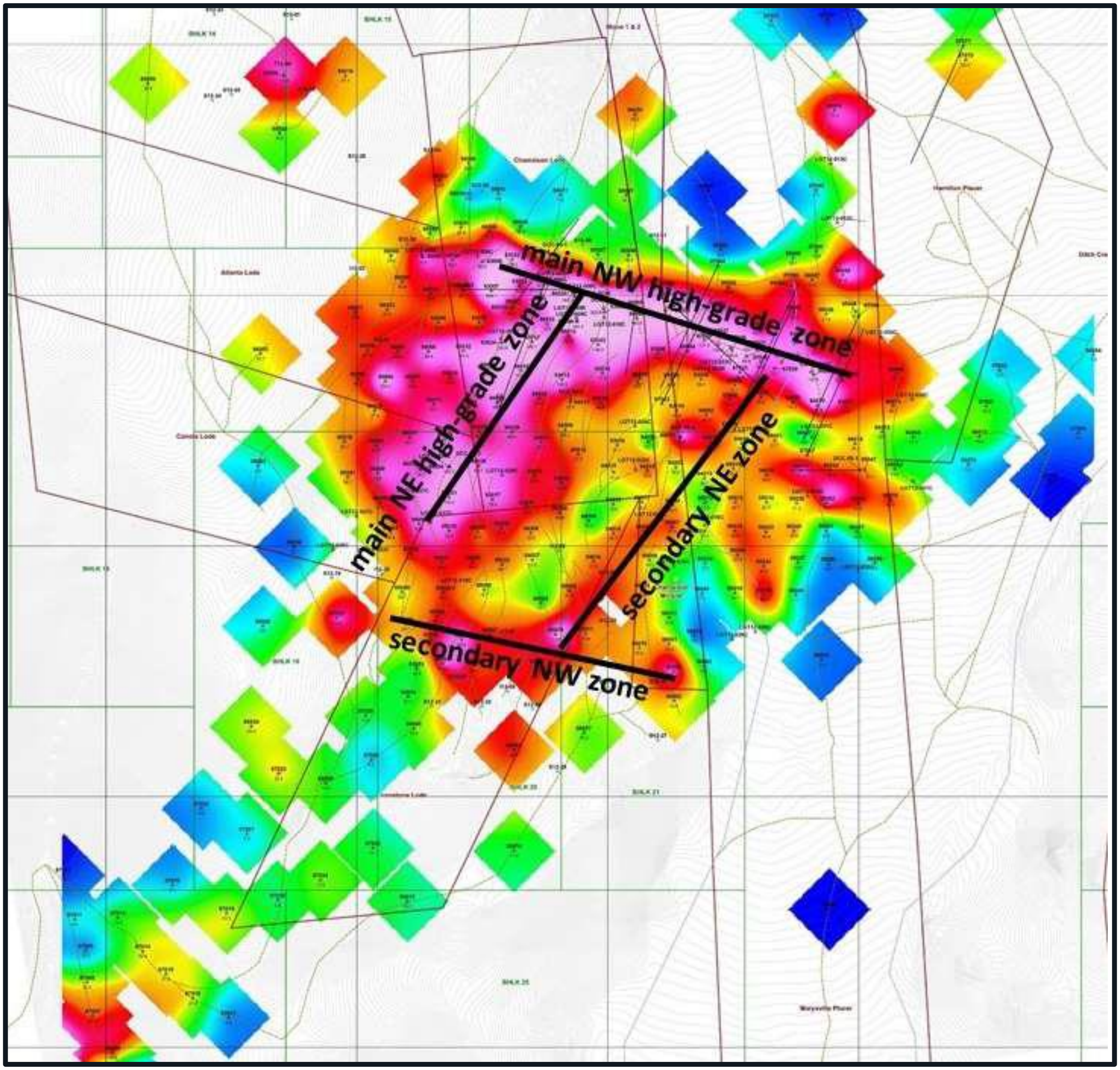
The current developing geologic model for the gold mineralization at the Lemhi Project is of a structurally controlled hydrothermal deposit associated with varying amounts of sulfides in a quartz-carbonate gangue hosted by late-Proterozoic metasediments within the structurally complex Trans-Challis fault system. It is further suggested that gold mineralization was introduced during a tectonically active period and is likely temporally related to intrusive activity associated with the Idaho Batholith. Gold mineralization has a strong association with base metal (Cu and Mo) mineralization and occurs as multiple hydrothermal (epithermal – mesothermal) silica replaced structures resembling multiple flat-lying veins.

Cross-sections produced by FMC, AGR, and ISGC show good lateral correlation of mineralized zones, but grades that often change rapidly from hole to hole. Sandefur (1996) noted that variograms developed in modelling by PAH showed that the deposit was significantly more variable in plan than in vertical dimension. The structural complexity of the Lemhi Gold Deposit along with the abundant Quaternary alluvium, glacial till and outwash flood boulders and the overlying Tertiary volcanics (Challis Formation) have hindered the full understanding of the deposit character and geometry.

The relative role of high-angle vs. low-angle structures as mineralization controls remains unresolved. Two sets of prominent high-angle fracture sets with related quartz veining (N75°W 85°SW, N45°E 75°NW) and a low-angle fracture set (N30°E 45°SE) were observed by Cuffney (2011), who suggested "It is quite possible that the low-angle stacked ore pods are in fact sub-horizontal zones of nested high-angle veins".

Given the flat-lying nature of the mineralized zones, the gold intercepts from the vertical drill holes on average approximate true thickness.

Figure 7-11: Grade Contour (g/t Au x metre) Map of the Humbug Deposit with Suggested Mineralized Trends



Source: Brewer, 2019.

8 DEPOSIT TYPES

The following section has been slightly modified or taken directly from previous summary and/or technical reports by Brewer (2019) and Cuffney (2011).

8.1 Structurally Controlled Hydrothermal Gold

Kiilsgaard et al. (1986) note that gold deposits in the Gibbonsville area do not extend to depth and they assumed that the deposits were epithermal in nature. However as noted by Cuffney (2011) "a mesothermal level of emplacement is evidenced by the following: 1) lack of open space filling, 2) crystalline quartz and lack of very fine-grained or chalcedonic quartz, 3) copper - molybdenum association, 4) coarse-grained sulfides, 5) associated bismuth, 6) low arsenic, antimony and mercury and 7) spatial association with the porphyritic intrusions. "

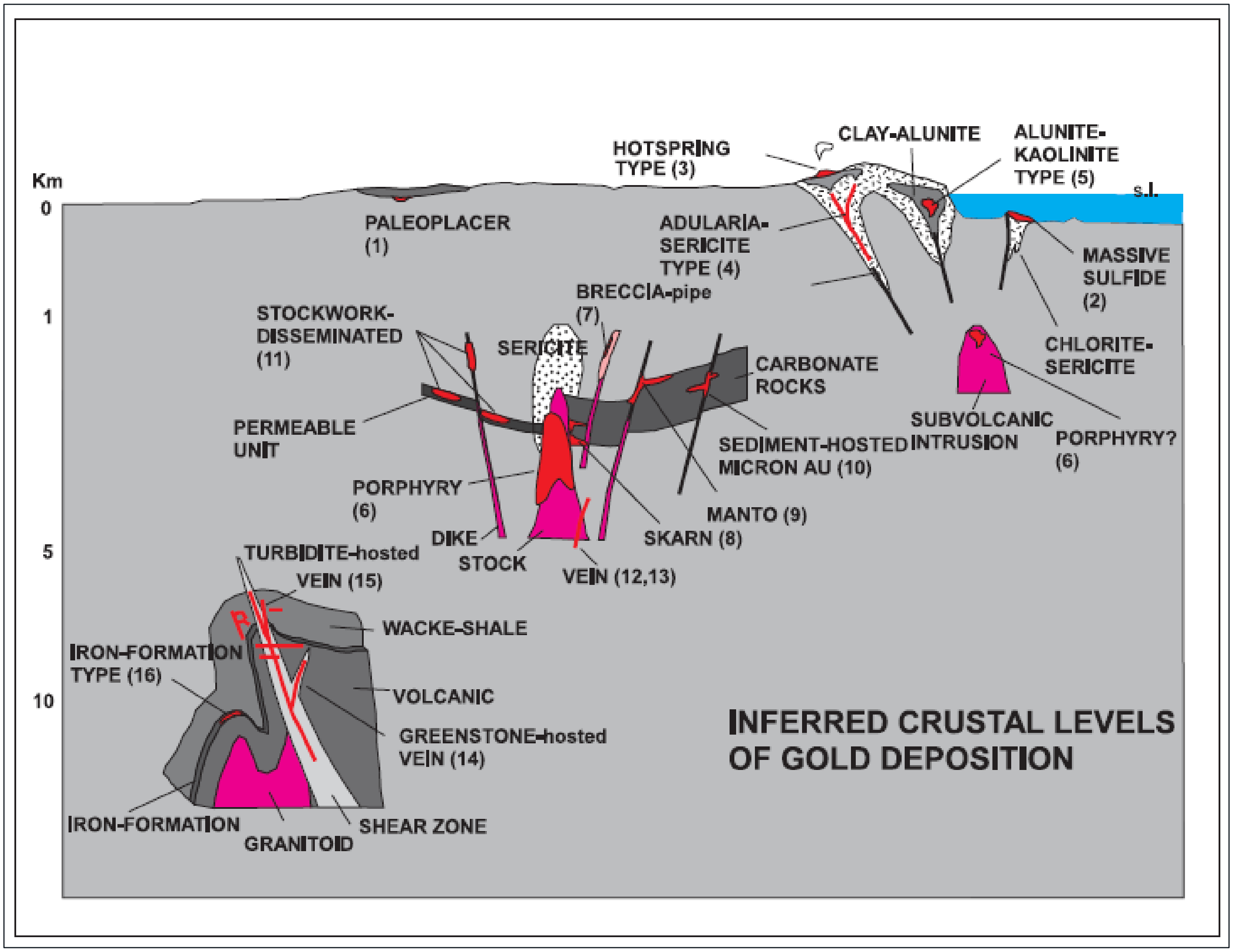
The Lemhi Gold Deposit is localized within a major low-angle shear zone and is spatially associated with a high-level porphyritic intrusion. Precious metals mineralization at Lemhi has historically been classified as shear-hosted porphyry-style mineralization. Both FMC and AGR recognized this deposit type and used a porphyry-related model to guide their exploration programs. Key elements of the exploration model were major structures (structural permeability); high-level intrusions (source of heat and fluids); alteration consisting of silicification and sericitization; and gold, copper, and molybdenum geochemical anomalies.

However, based on the 2012 core drilling, an alternate deposit model was suggested of a structurally controlled hydrothermal deposit associated with varying amounts of sulfides in a quartz-carbonate gangue hosted by late-Proterozoic metasediments within the structurally complex Trans-Challis fault system. It was suggested that gold mineralization was introduced during a tectonically active period and was likely temporally related to intrusive activity associated with the Idaho Batholith. The observed gold mineralization is strongly associated with base metal copper and (Cu) and molybdenum (Mo) mineralization and occurs as multiple hydrothermal (epithermal – mesothermal) silica replaced structures resembling multiple flat-lying veins.

The gold deposit on the Lemhi Gold Property shares many similarities with the Beartrack mine, 35 km to the southwest, and the Musgrove deposit, 25 km further southwest. Both Beartrack and Musgrove are quartz stockworks hosted within major shear zones cutting the Apple Creek Formation.

Geochron Labs measured K-Ar ages on two samples of quartz-veined and sericitic altered quartzite from core hole C- 4.Sericite believed to be a product of hydrothermal alteration yielded an age date of 65.5 ± 2.5 Ma. The age date is close to that of the Beartrack deposit (68 Ma) and Napoleon Hill porphyry molybdenum deposit (Schaubs, 1990). These dates in conjunction with the observed geological characteristics are suggested to be indicative of a structurally controlled orogenic mesothermal gold deposit with potentially a link to a late Cretaceous intrusive event as the heat source for the hydrothermal activity along the Trans-Challis fault system. Intrusion and potentially structurally controlled mesothermal gold deposits and their characteristics formed during the Phanerozoic are described by a number of authors including Robert et al. (1997), Robert et al. (2007) and Groves and Santosh (2015). Figure 8-1 below shows the model of formation and crustal levels of gold deposit formation for intrusion and orogenic mesothermal gold deposits at 1 to greater than 5 km depth.

Figure 8-1: Schematic Representation of the Crustal Levels Inferred for Gold Deposition for Commonly Recognized Deposit Types



Source: Robert, 1997
*Note: The depth scale is approximate and logarithmic and numbers beside named deposit types coincide with those used in Robert et al. (1997).

9 EXPLORATION

Prior to 2020, the most recent exploration was completed by LGT in 2012 and included revaluation of the historical data, additional petrographic analysis, geochemical analysis and evaluation, and baseline environmental studies. A summary of these activities is provided in the Section 6 History and Section 7 Property Geology Section.

In 2020, Freeman commenced a surface exploration program at the Lemhi Gold Project. This first phase of exploration consisted of the following methods:

- Soil orientation survey (conventional soil, IL, and mobile metal ions (MMI))
- Rock and chip sampling
- Ground magnetic survey
- 3D Induced polarization survey.

In 2021, Freeman commenced a second surface exploration program at the Lemhi Gold Project. This second phase of exploration consisted of the following methods:

- Regional and targeted soil sampling (IL)
- Rock and chip sampling
- Ground magnetic survey
- Geological mapping.

9.1 Soil Survey (Orientation, Regional, and Targeted)

With no modern soil samples taken on the property, a soil orientation survey was completed in 2020 to determine the method most suitable for the Lemhi Gold Property. Across the property variable soil profiles are observed with areas of moderate-low organics to areas of significant glacial or glacial-fluvial cover, specifically north of the deposit. Of the methods chosen, two utilized partial extraction techniques, mobile metal ions (MMI) and ionic leach (IL), with the third technique comprising a conventional soil sample. An orientation survey consists of a single transect over a known target, with dense site spacing. Multiple samples are collected from each sample pit. The primary reasons for performing this survey are to:

- Determine the appropriate method to identify mineralization
- Determine a site spacing that is sufficiently dense to identify mineralization
- Identify which elements characterize the mineralized zone
- Establish the appropriate depth below live organic material at which to collect samples

- Determine whether to do the complete elemental suite or establish the appropriate elements to use in a reduced elemental package.

The orientation survey methodology, results, and interpretation are outlined by Dufresne et al. (2021). Based upon the results of the orientation soil survey, IL, and MMI are preferred over conventional soil geochemistry since they have less noise and pick up underlying mineralization when plotting the response ratio. Results for IL and MMI were similar, however, due to cost of analysis and the fact that IL provides a larger suite of elements analyzed, IL was the recommended method for future soil programs at Lemhi.

In May 2021, a regional sampling program was undertaken with soils collected at a depth of 30 cm. Samples were taken at a 100 m spacing and targeted soil samples at a spacing of 25 m. Following the results from the soil orientation program, IL was employed for analysis of the soils.

At each sample site tools were brushed and flushed to eliminate residue from the previous sample. Organic matter (~5 – 10 cm), if present, was removed and included decomposed leaf matter, rootlets, and hairs. Organic matter will not adversely affect the IL analysis, but large rocks and twigs were removed by hand from each sample. The “Zero Datum” depth where organics decompose and start to see soil formation was recorded. IL samples must be taken at a constant depth near surface at the various depth profiles below.

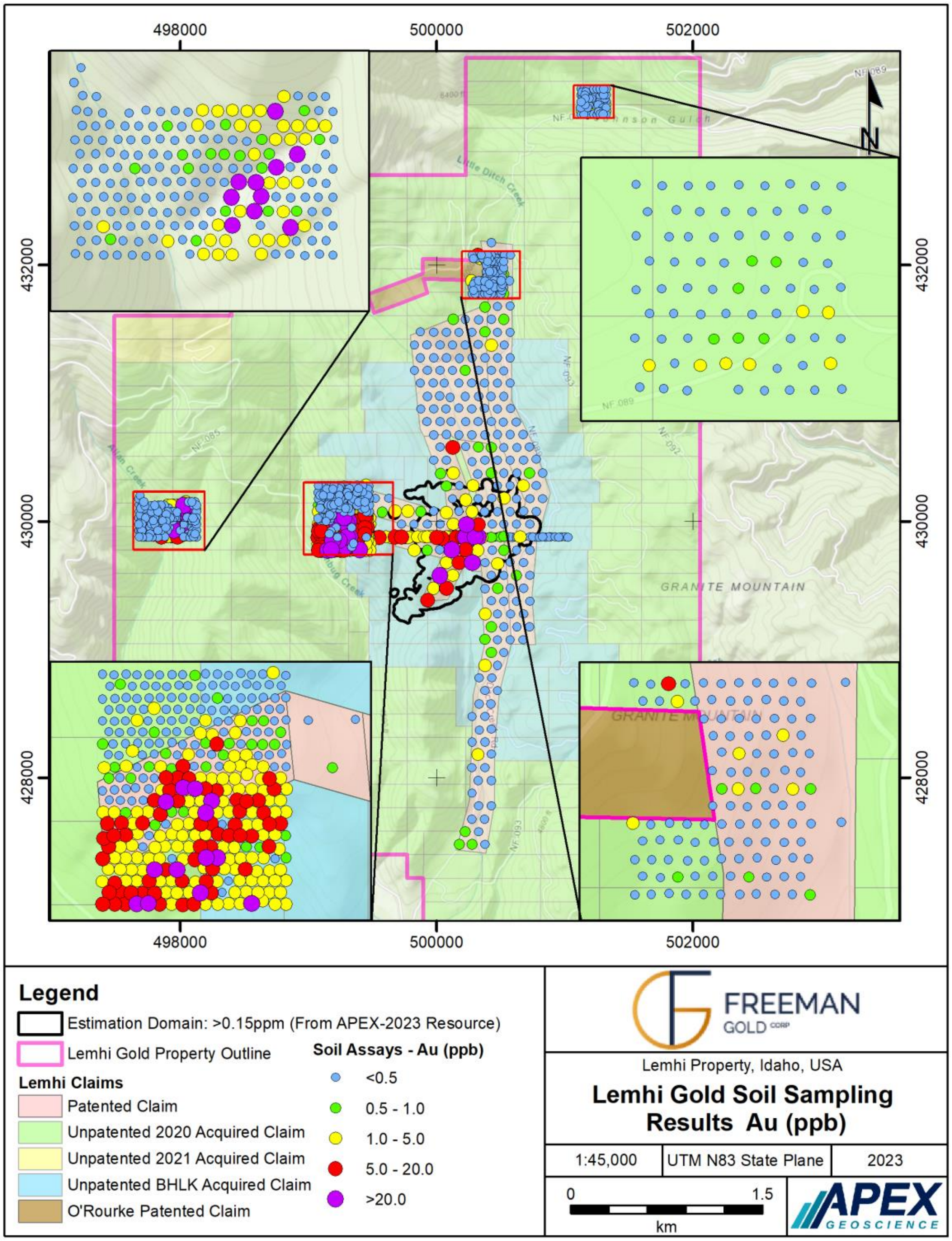
In skeletal soils with sub-crop/outcrop, samples were collected nearer to the surface where deeper profiles did not exist (30 – 40 cm). If a soil profile was atypical of the survey area, then it was recorded as this may influence the data interpretation stage. Conventional soils were taken from the B-horizon where soil profile exists (C-horizon when no B-horizon is present) with the depth of sample recorded. All field data was recorded within a custom-built Fulcrum App. IL samples were placed in a labeled snap and seal bag. Excess air was removed from the snap and seal bags preserving volatile elements (Hg, I, and Br) in the sample. Excess water, when present, was immediately decanted from the sample bags at site. IL samples were not allowed to air dry.

This regional sampling program was partially completed on the patented lands before it was temporarily suspended. Freeman intends to resume the soil program on the unpatented lands and add additional targeted soil grids in areas with anomalous rock grabs identified in the 2021 surface program.

This regional sampling program delineated several areas warranting follow-up exploration. Best results were surrounding the Beauty target, and from the grid to the west of the main Lemhi deposit area, depicted in Figure 9-1. The western soil grid has defined an anomalous area trending north-south with anomalous assays >1 ppb Au. This trend could be extended by further soil sampling to the north and south and could be further extrapolated with mapping in the area prior to a drilling program to test the anomaly at depth.

Strong anomalism was delineated by the Beauty soil grid (trends consistently >5 ppb Au). Two parallel areas trending southwest away from the Beauty showing terminate at the edge of the sampling grid. These southwest anomalies are considered to be false anomalies due to extensive hydraulic mining that occurred over the showing in the late 1800s. At Beauty, geologist mapped the hydraulic washes which extend 500-600 m and are up to 50 m wide and 30 m deep. The true anomaly runs northwest to southeast, in addition to the southwest corner of the grid where no washes exist. Follow-up soils are planned to further delineate these anomalies. Additional drilling is planned to follow-up the significant anomalous results from soils to the southeast of the Beauty target.

Figure 9-1: Soil Sample Sites Across the Lemhi Gold Deposit



Source: APEX, 2023.

Given the significant topography in the project area, the effects of colluvial, alluvial and other transported materials should be considered when interpreting the results of the soil sampling, particularly at Beauty, where extensive hydraulic mining has occurred over a known outcrop of significant mineralization within the soil grid on a topographic high.

9.2 Rock and Chip Sampling

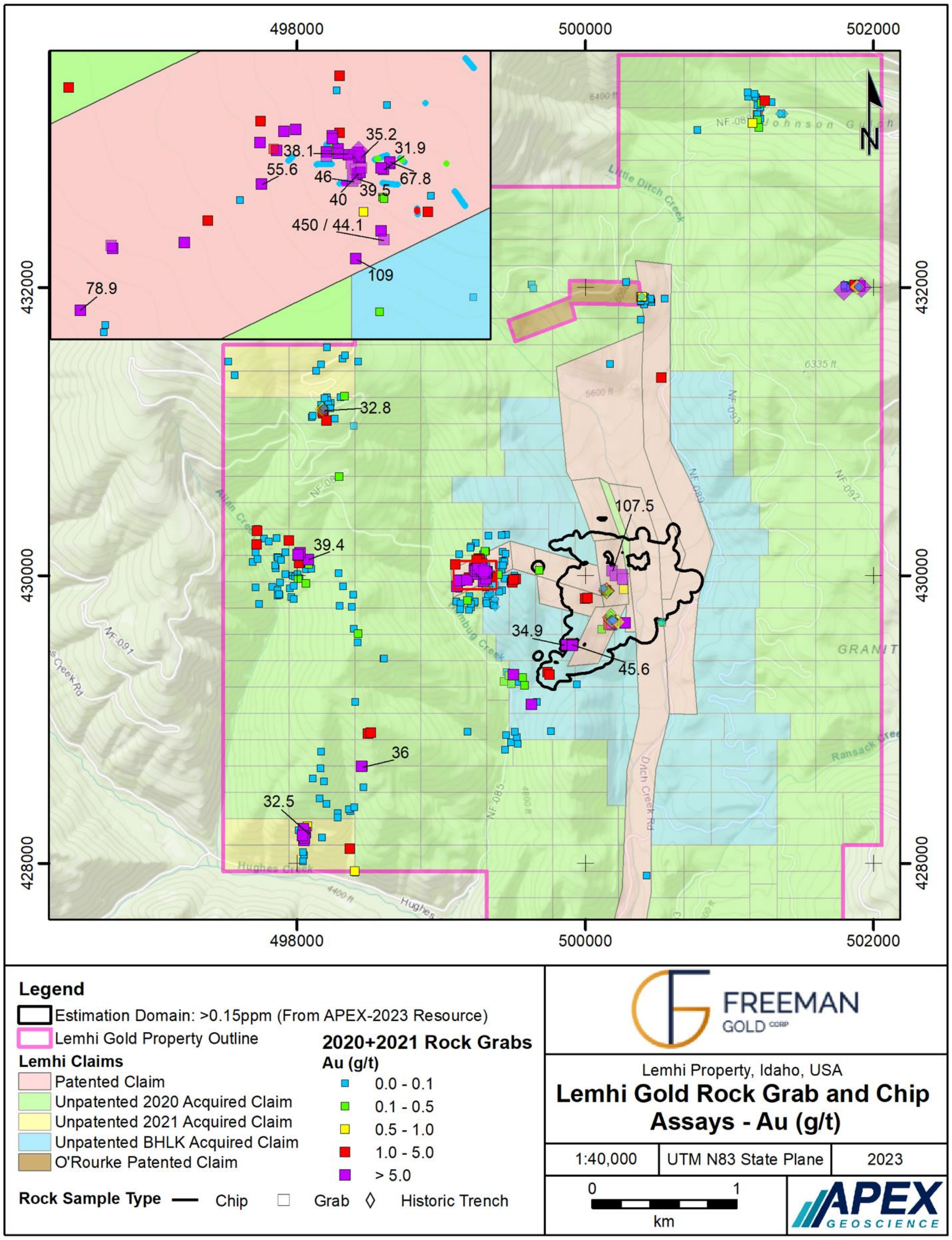
Rock grab and chip sampling at the project consisted of prospecting and chip sampling various exposed rock faces and trenches across the property. Prospecting consisted of following up previous high-grade rock samples and visiting numerous adits and old workings to evaluate continuity, confirm mineralization, and assess potential for drill targeting. In 2020, a total of 145 samples (see Figure 9-2; transparent points) were collected, including blanks, standards, and duplicates. In 2021, a total of 548 samples (see Figure 9-2, opaque points) were collected, including blanks, standards, and duplicates. Samples in both programs ranged from 0.5 – 2. 5 kgs. Chip samples were taken from outcrop, historical trenches and new road cuts. They were taken at 50 cm intervals in 2020 and ~1 m intervals in 2021 across a given rock face.

In 2020, a total of five locations were selected for chip sampling, from these locations 69 samples were collected. Grab samples were collected from historical trenches, outcrop, or float for a total of 55 samples. The remaining 21 samples were standards and blanks. Of the samples collected, 54 returned assay values greater than 1 g/t Au and 20 greater than 5 g/t Au (up to 450 g/t). Of the rock samples collected, 27 samples contain greater than 10 g/t Ag (up to 219 g/t).

In 2021, a total of 13 locations were selected for chip sampling, from these locations 105 samples were collected. Grab samples were collected from historical trenches, outcrop, or float for a total of 283 samples. The remaining 55 samples were standards and blanks. Of the chip and grab samples collected, 2+75, respectively, returned assay values greater than 1 g/t Au and 1+44, respectively, greater than 5 g/t Au (up to 109 g/t). Of the chip and grab samples collected, 1+60, respectively, samples contain greater than 10 g/t Ag (up to 281 g/t).

Across the property, mineralization was within phyllites, quartzite and quartz veins, and appears similar in nature to that of known mineralization encountered in core drilling at Lemhi.

Figure 9-2: Rock Grab and Chip Sample Locations. Transparent Points 2020 Grabs and Chips, Opaque Points 2021 Grabs and Chips. All Rocks Grabs >30 g/t are Labeled.



Source: APEX, 2023.

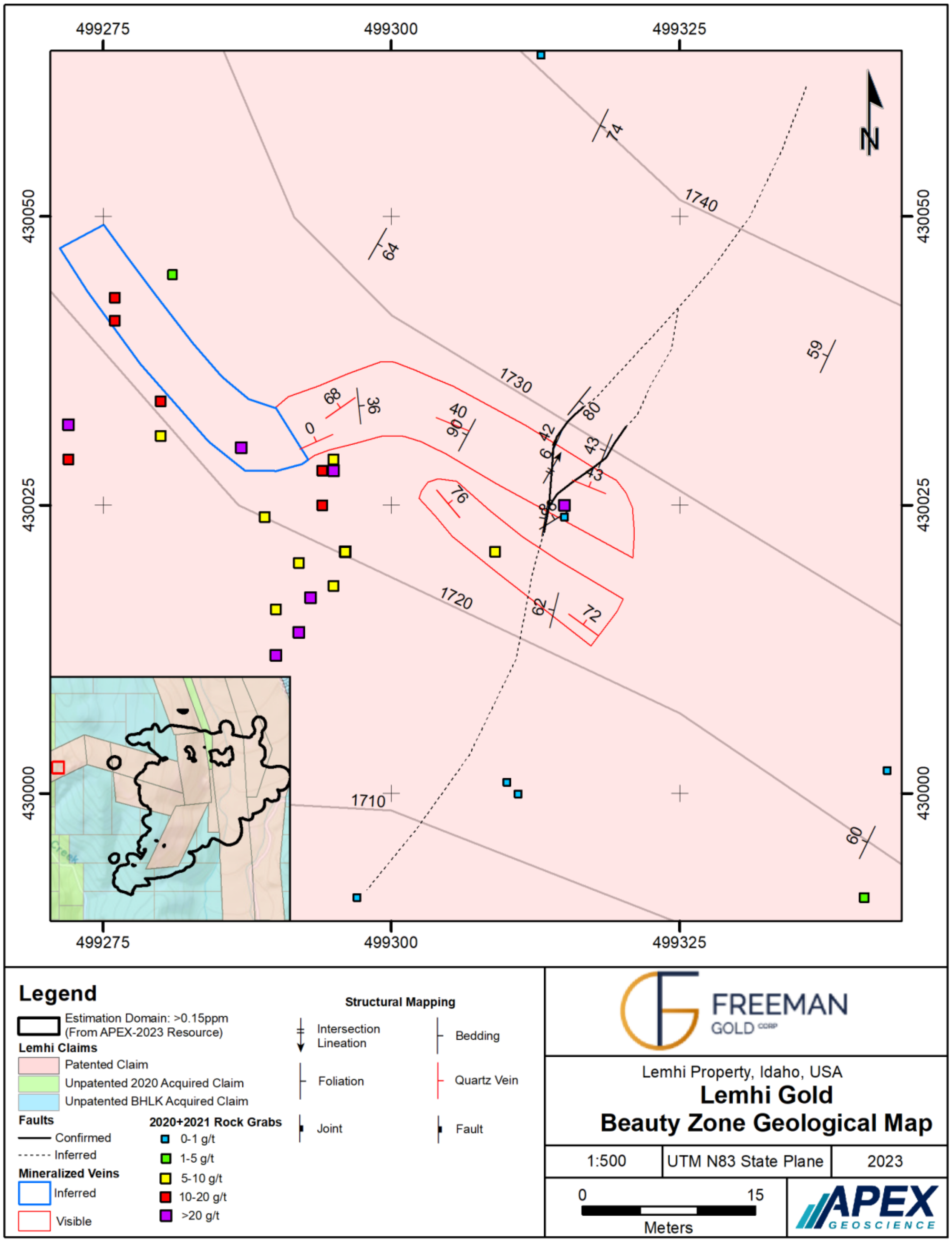
9.3 Geological Mapping

Mapping was completed at the Beauty target to help guide exploration drilling. Additional mapping was completed at other exploration targets to guide drill hole locations for permitting at various targets across the property. Beauty mapping was the main focus of geologically mapping due to the structurally complex nature of this target, anomalous rocks and soils. Limited outcrop at Lemhi results in few mapping points. A total of 69 mapping points were collected. Of these 55 have strike and dip data, with 38 mapping points taken at Beauty (550 m east-west by 400 m north-south).

The Beauty grid yielded 52 rock grab samples with >1 g/t Au from a total of 105 total samples collected with numerous samples originating from or proximal to a mineralized outcrop in the center of the targeted soil grid. Most of the samples were actual composite grab samples collected systematically across a few 10 of centimeters to a couple of meters a few meters apart across the outcrop including vein and wall rock material. Disturbed material composite samples generally involved collecting several different representative pieces of the available material from a pile of disturbed rocks. Rock grab samples are by nature selective and can represent biased data. However, care was taken at the Beauty grid to collect representative samples where possible, particularly from outcrops on the grid.

The Beauty showing is located in the middle of a set of hydraulic washes. At some point following the hydraulic mining, the showing appeared to be blasted with dynamite. Several mineralized veins and foliation structural measurements were taken at the showing and across the soil grid (Figure 9-3). The northwestern portion of the Beauty grid contains outcrop that dips to the southeast while the southeast portion of the grid contains outcrop that dips to the northwest. In addition, a southwest-northeast-trending fault was identified at the showing. Structure measurements of quartz 30 cm in width showed veins dipping into the hill to the northeast with veins folding. Numerous vein sets were identified with several thin veins sets near vertical. To support drilling Freeman cut a road down the mountain through the showing. This road cut revealed several faults of varying size with some cuts showing gouge intervals up to 1 m in size. Smaller centimeter scale faults are identified by hematite slicks. Several faulted blocks sit against each other, each with variable foliation orientations. Currently, geologists believe this mineral showing is the result of a complex fault zone, however the possibility of a faulted isoclinal fold has been considered. It is clear that this structurally complex target was a fluid pathway for high-grade gold with rock grabs containing up to 450 g/t Au and one trench sample contained 68. 23 g/t Au and 40.18 g/t Ag over 6 m.

Figure 9-3: Beauty Geological Map with Rock Grab Locations.



Source: APEX, 2023.

9.4 Ground Magnetic Survey

Freeman commissioned a ground magnetic (MAG) survey over the entire Lemhi Gold Property. The MAG survey was completed between September 20 to December 10, 2020. An additional MAG survey was completed in May 2021 to cover the eight additional claims staked by Freeman.

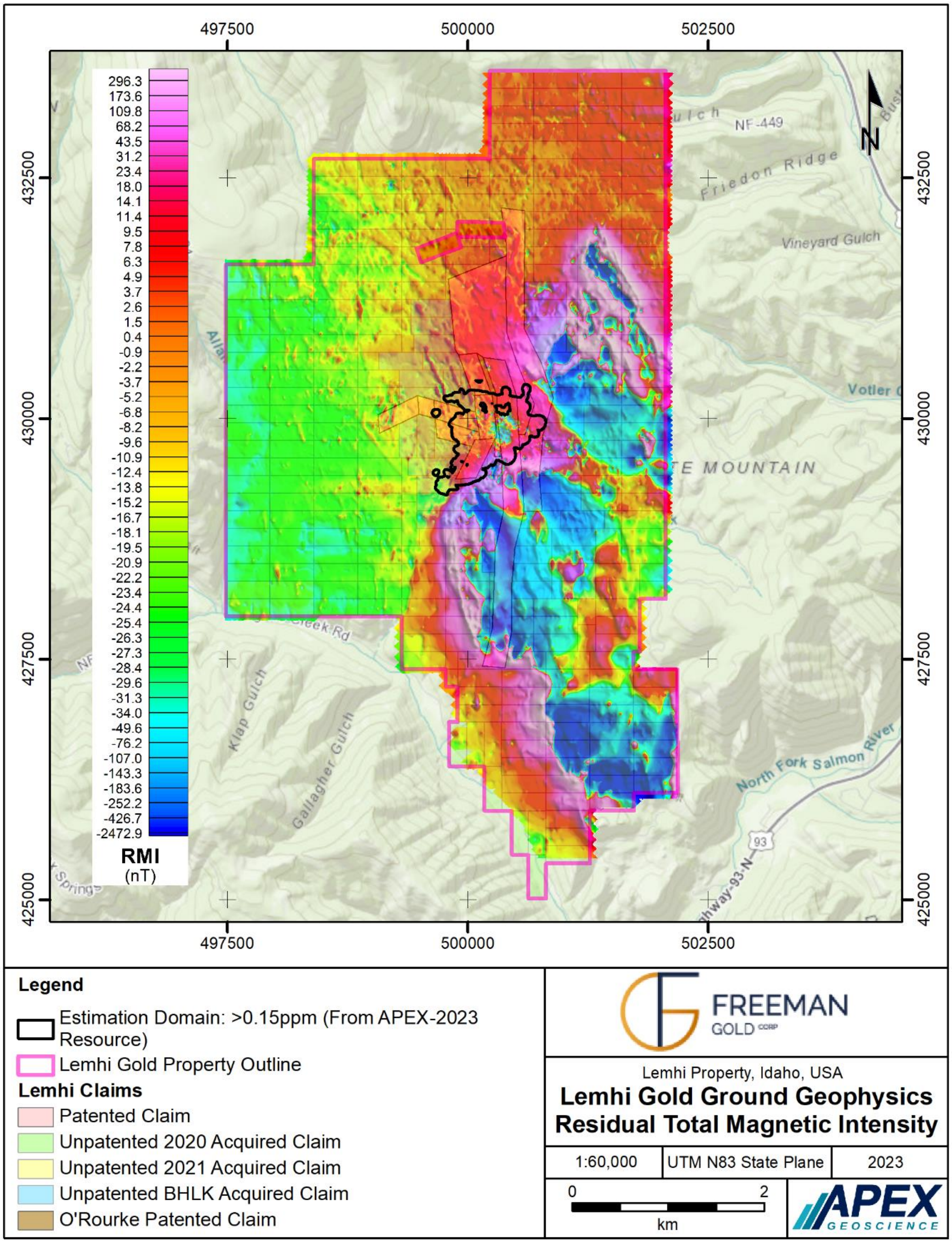
The survey grid encompassed an area covering 2,675 ha and consisted of 246 traverse lines oriented E-W, spaced 12.5 m apart over the 1.44 km^2 deposit area (1.2 x 1.2 km), and 50-100 m apart over the rest of the property. During the survey, preliminary interpretations of the unlevelled data were carried out at a regional scale. Based on the results of these interpretations, it was deemed appropriate to decrease sampling from 50 to 100 m line spacing in order to increase productivity along the northern, western, and southern extents of the property. Survey lines ranged in length from 180 to 4,570 m. In total, 559.4 line km of MAG data was collected in 2020 and 8.0 line km of MAG data was collected in 2021.

Ground magnetic measurements were obtained using GEM GSM-19V Overhauser magnetometers, which are equipped with an integrated global navigation satellite system (GNSS) receiver. The MAG data was recorded as total magnetic intensity readings, with a cycle time of 1 s, while the GEM unit was in walk mode and collecting continuous measurements along the traverse lines.

The processing and levelling of ground magnetic data has been completed, and interpretation and target generation have followed. The levelled ground magnetic data image is presented in Figure 9-4, which clearly identifies the contact of the porphyry as a magnetic high. Although the ground magnetic amplitude within the project area is generally weak, the magnetic high provides a clear indication of the porphyry location. The Lemhi Gold Deposit is located between two distinct bodies with a magnetic low in the centre of the estimation domain outline. Based on the ground magnetic survey data, a clear contact of the porphyry body is identified as a magnetic high, which is significant since gold is typically associated with this contact. This contact, therefore, warrants additional follow-up exploration to evaluate the potential for gold mineralization in the area.

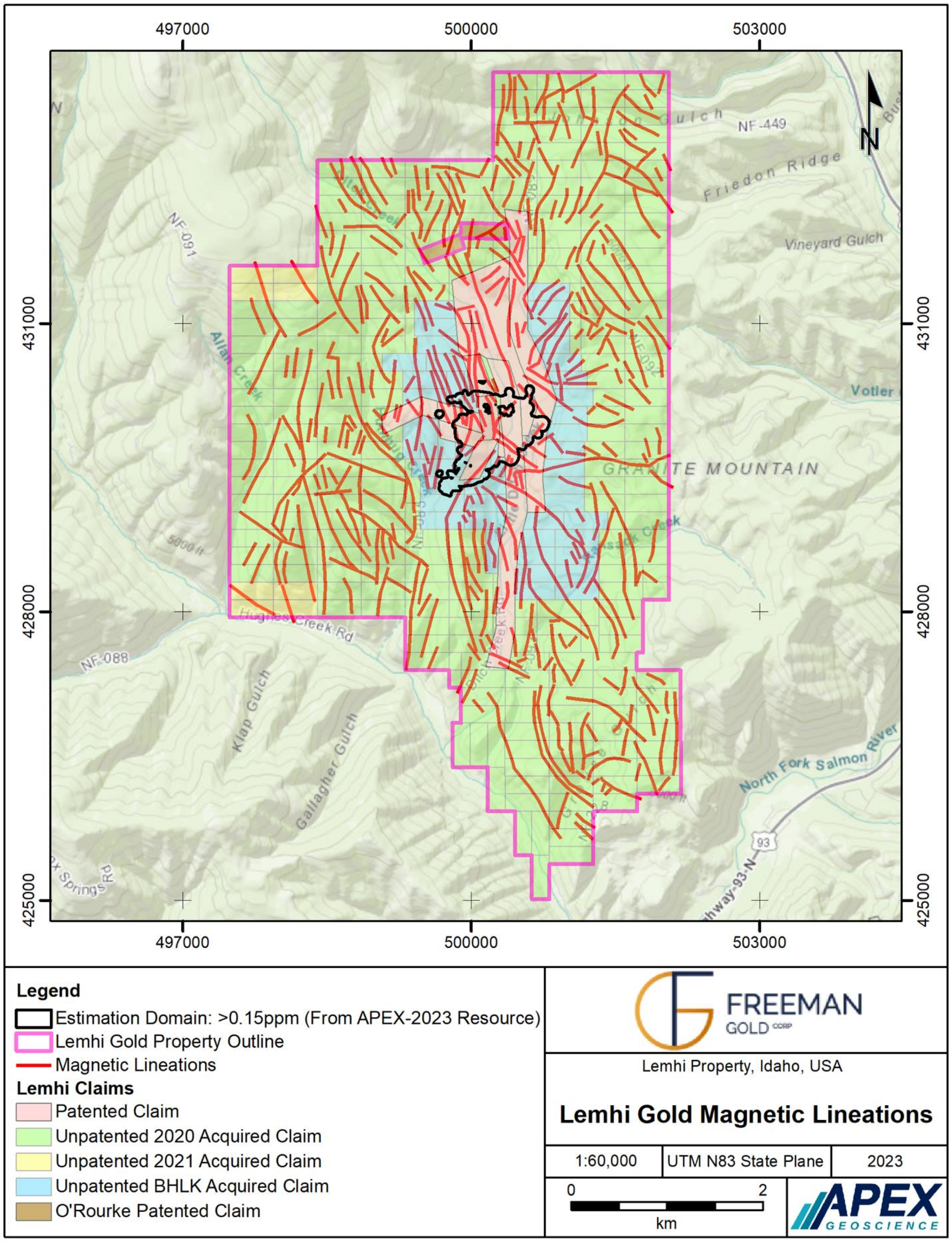
A lineation study was conducted, and the results are shown in Figure 9-5. The study identified several structures that are worth exploring further. The contact of the intrusion is clearly visible, as are other large-scale features that could be associated with faults or dykes that may be linked to gold mineralization. Additionally, Figure 9-6 highlights two anomaly groups associated with the Lemhi Gold Deposit, MAG lows, and MVI highs. Areas where these two anomalies overlap represent high-priority targets for additional follow-up exploration.

Figure 9-4: Leveled Ground Magnetic Data Image.



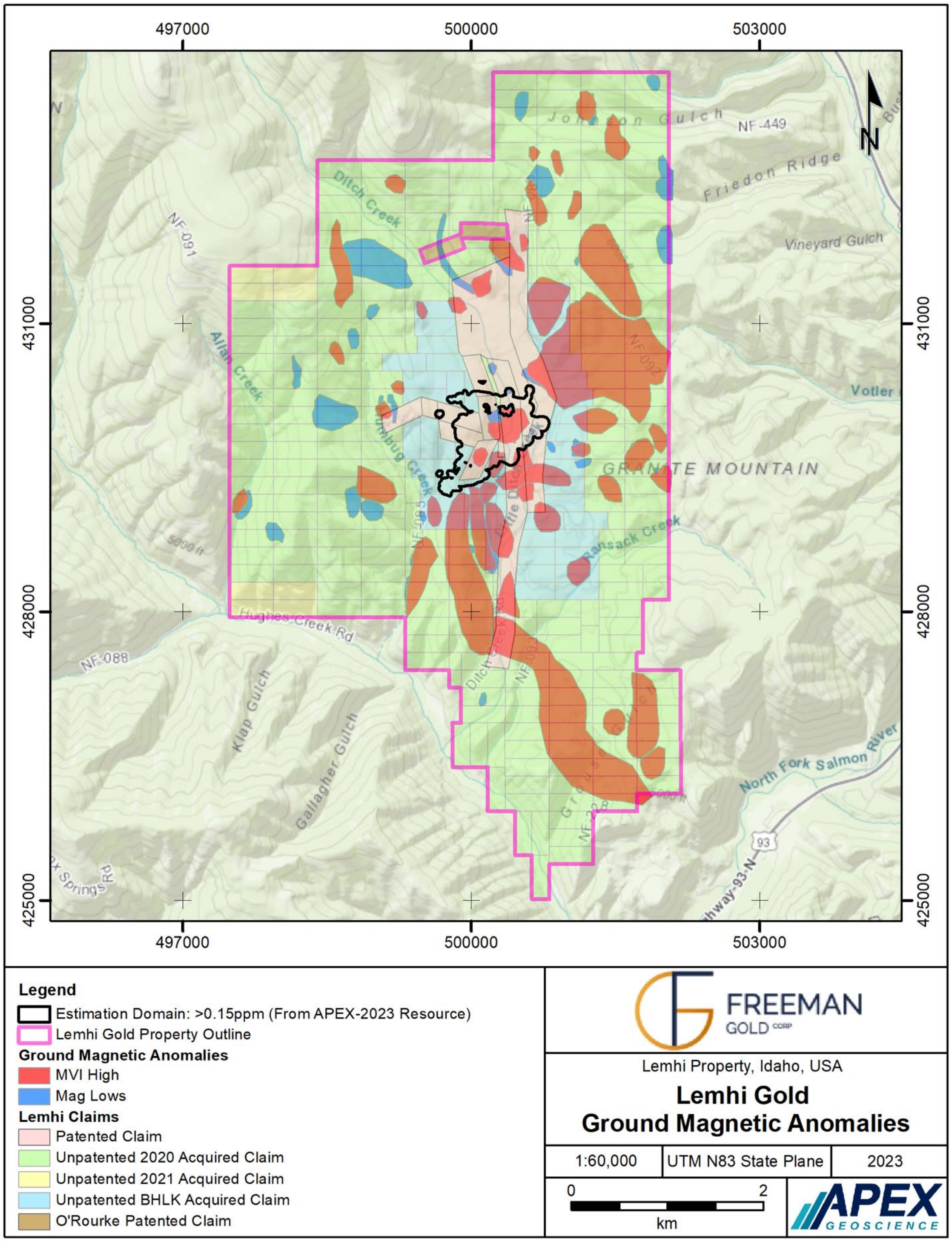
Source: APEX, 2023.

Figure 9-5: Magnetic Lineations Across the Property



Source: APEX, 2023.

Figure 9-6: Magnetic Targets for Future Exploration



Source: APEX, 2023.

9.5 3D Induced Polarization Survey

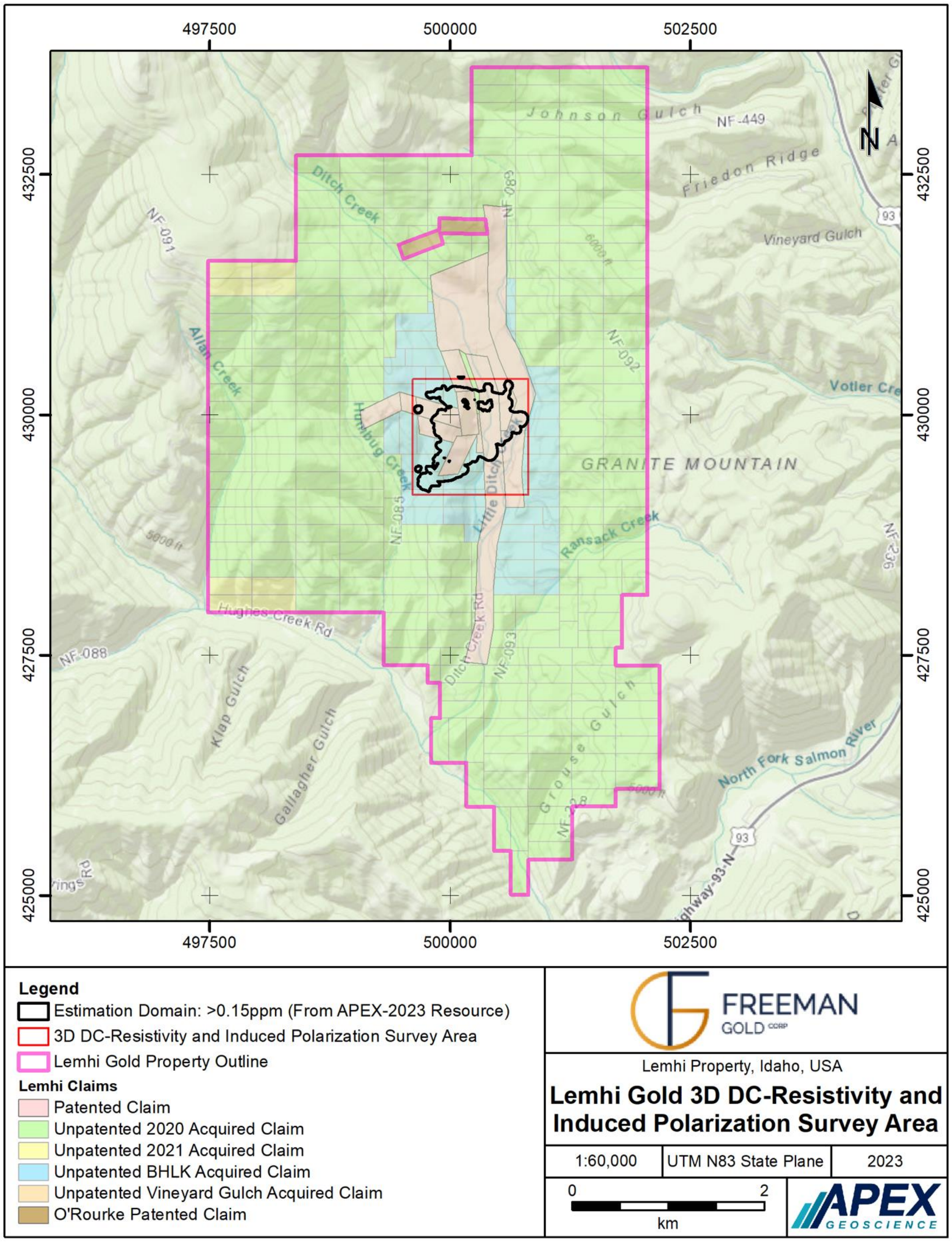
Between September 23 and October 9, 2020, Dias Geophysical Limited (Dias) carried out a 3D DC-resistivity and induced polarization (DCIP) survey on the Lemhi Gold Property using the DIAS32 system. This geophysical program was designed to detect the electrical resistivity and chargeability signatures associated with potential targets of interest. This was achieved using the DIAS21 acquisition system in conjunction with on GS5000 transmitter. The survey was completed using a rolling distributed partial 3D DCIP array with a pole-dipole transmitter configuration. The survey covered an area spanning 1.44 km^2 (1.2 x 1.2 km) over the deposit with thirteen 100 m spaced lines that were 1.2 km in length, see Figure 9-7.

Additional information regarding methodology and procedures, data processing, and presentation and 3D inversion modelling is included in a report by Dias Geophysical Ltd (2020). The survey was designed to characterize the geophysical signature of the deposit and possibly define new areas of gold mineralization and extensions of the known mineralized zones delineated by drilling. Cross-sections displaying chargeability in Figure 9-8 and resistivity in Figure 9-9 from the 3D IP survey plotted along drill section 430000N with results of FG20-001C and FG20-002C.

From the 3D IP results two major contacts have been interpreted: the strongest one follows an east-northeast curvi-linear trend where chargeabilities are generally low and resistivities are very low to the south-southeast. The contact is also coincident with a magnetic high trend. The second major contact is also coincident with a magnetic high trend and trends to the north-south, located on the west side of the survey bock and is characterized by low chargeability coincident with low resistivity.

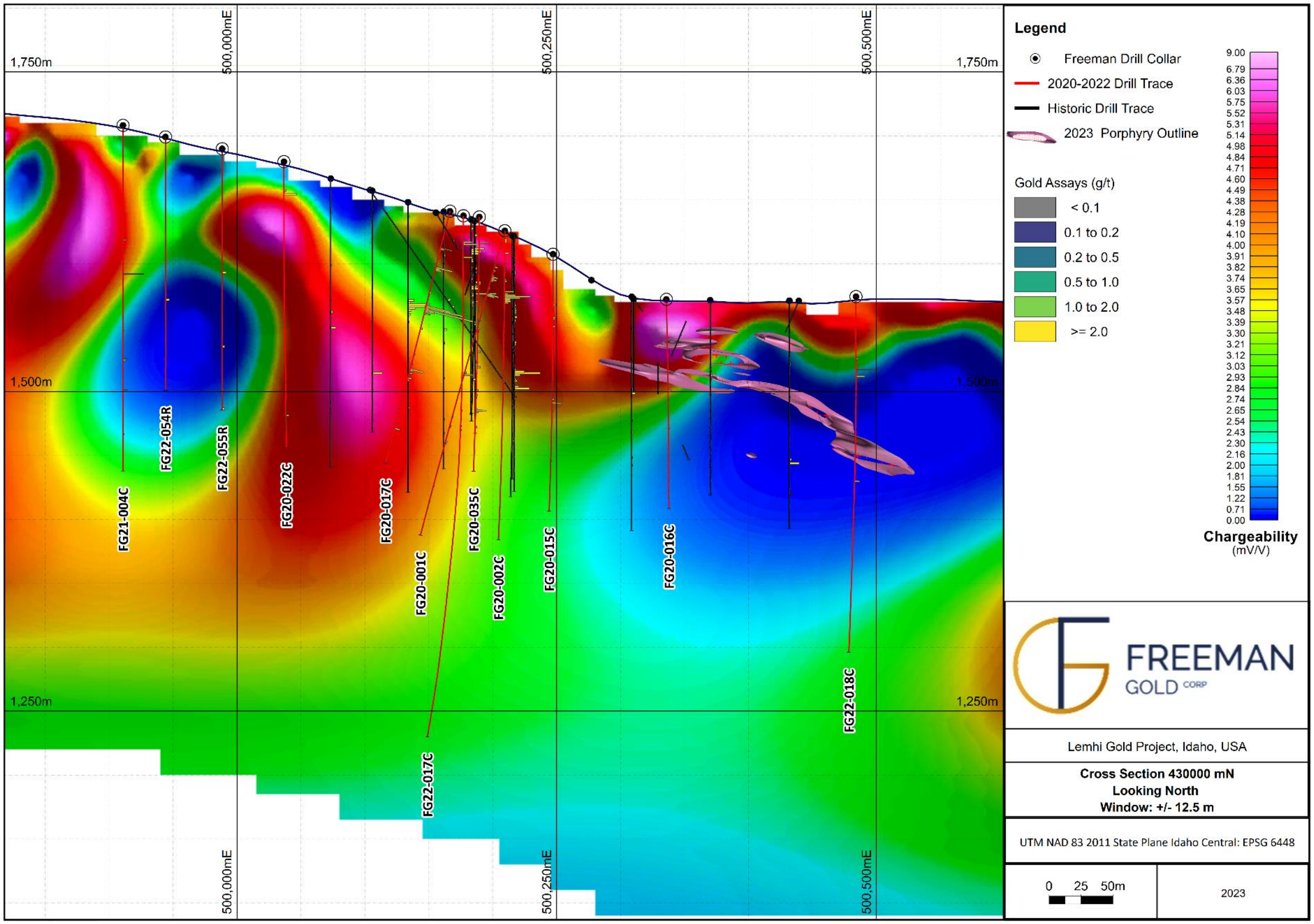
Three high priority and two moderate priority anomalies have been defined and shown in Figure 9-10. The first high priority is an area of elevated resistivity that is partially coincident with the northern limit of the gold grade zone. The second is a large north-south trending zone of high resistivity and high chargeability located at the western boundary of the survey block that is unbounded to the west. The third is a zone of high chargeability located at the eastern border of the survey block and unbounded to the east. The first moderate priority is a north-south trending zone of high resistivity and high chargeability adjacent to the northwestern boundary of the gold grade zone that is only seen in the shallow depth slices. The second moderate priority is a zone of high chargeability that straddles the southwestern portion of the mineralized zone and is seen only in the deep depth slices. The anomalies require drill testing and are shown on Figure 9-10. If additional gold mineralization is intersected, the IP survey should be extended to define the extent of the anomalies. Also, 3D IP could then be used as an important exploration tool in other areas with coincident anomalies to better define buried mineralization.

Figure 9-7: 3D DC-Resistivity and Induced Polarized Survey Area



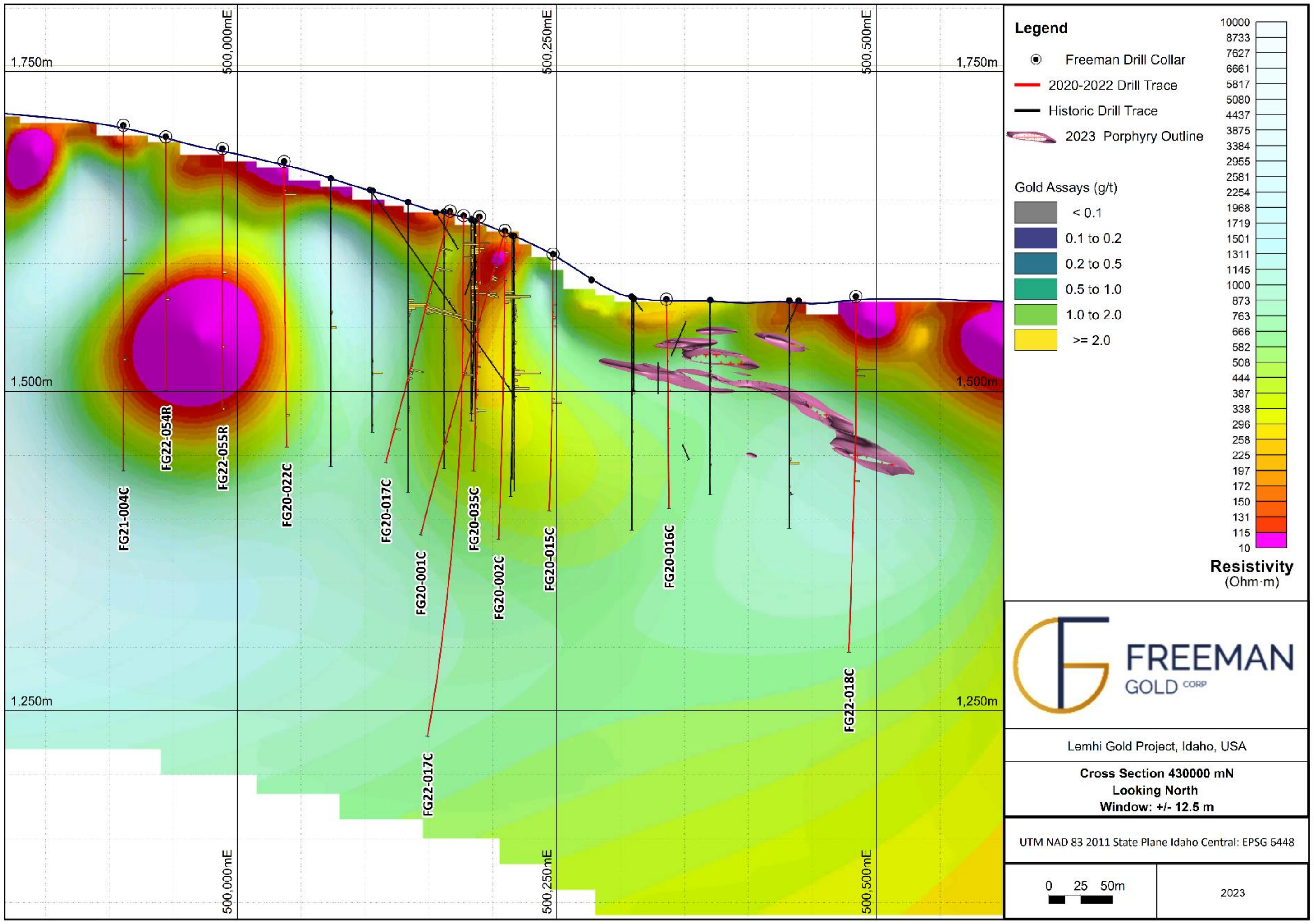
Source: APEX, 2023.

Figure 9-8: Cross-section of Chargeability from the 3D IP Survey Plotted Along Drill Section 430000 with Results of 2020-2022 Drilling



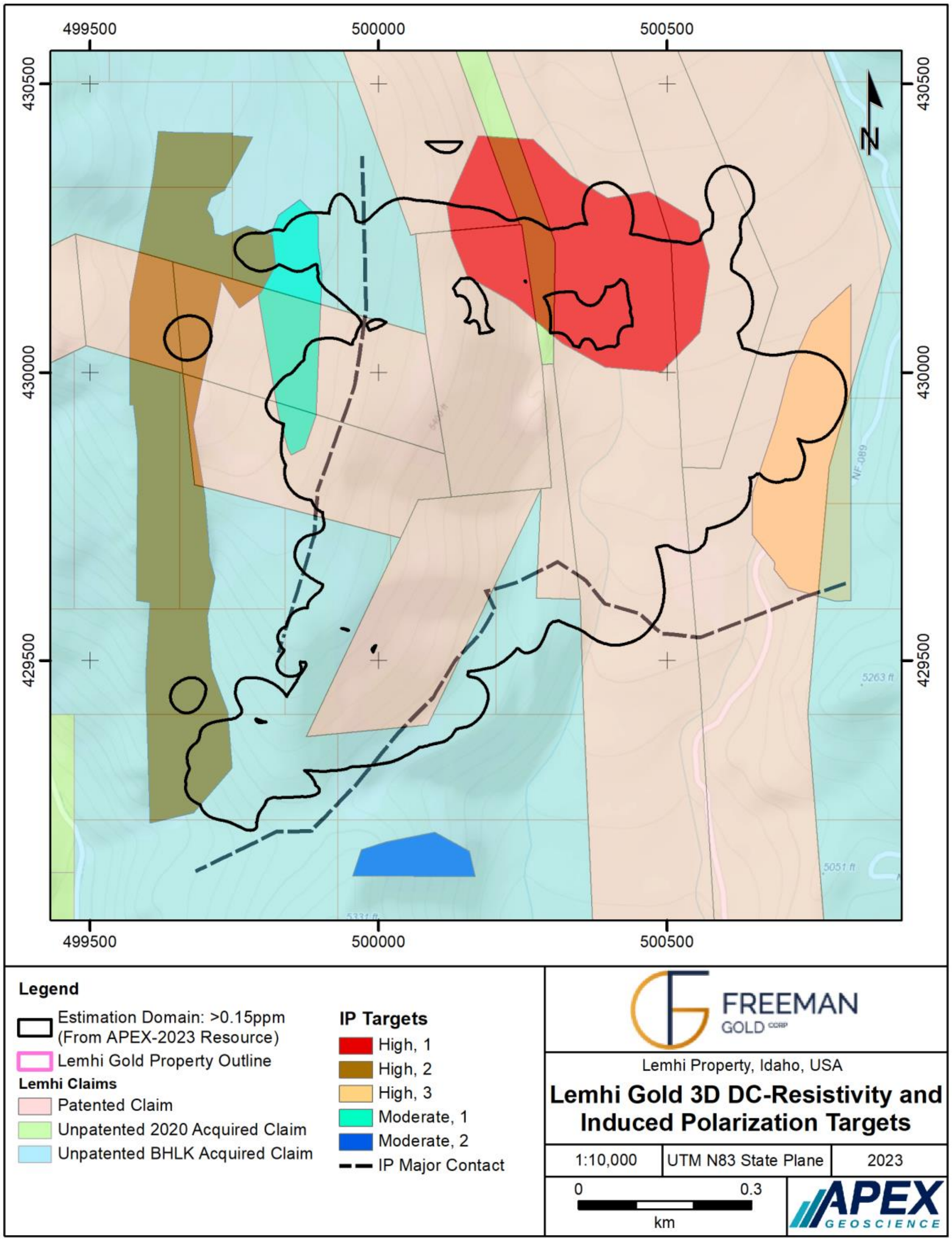
Source: APEX, 2023.

Figure 9-9: Cross-section of Resistivity from the 3D IP Survey Plotted Along Drill Section 430000 with Results of 2020-2022 Drilling



Source: APEX, 2023.

Figure 9-10: Targets and Major Contacts Interpreted from the 3D DC-Resistivity and Induced Polarized Survey



Source: APEX, 2023.

10 DRILLING

During 2020-2022, Freeman drilled 106 core and reverse-circulation (RC) holes at the Lemhi Gold Project, on the Lemhi and Beauty prospects. The holes are outlined in Table 10-1 and Figure 10-1.

Table 10-1: Summary of Drilling at Lemhi by Freeman Gold in 2020-2022

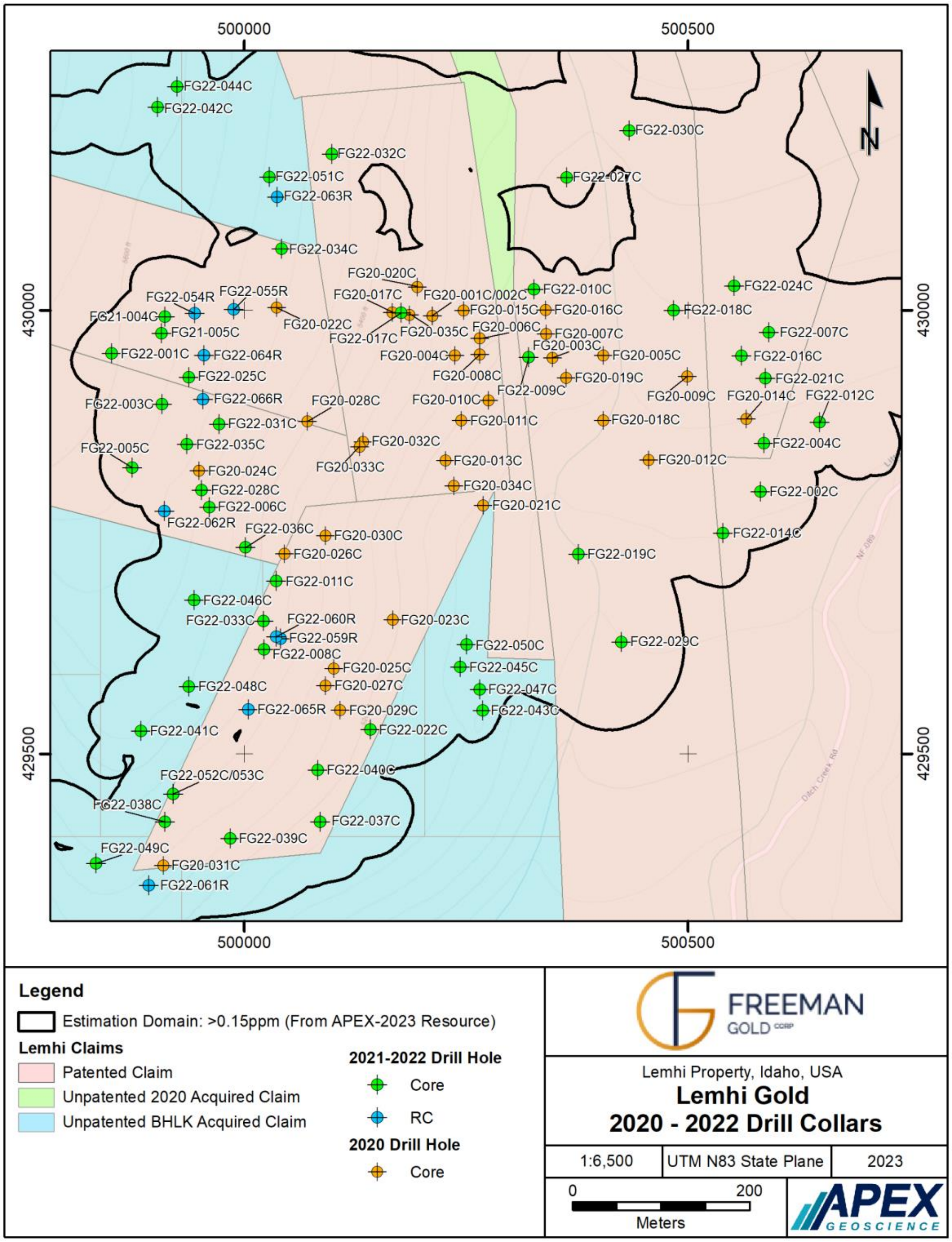
Prospect	Hole ID	Easting (NAD83 SPIC)	Northing (NAD83 SPIC)	Elevation (m)	Depth (m)	Dip (deg)	Azi (deg)	Drilling Company	Hole Type
Lemhi	FG20-001C	500210	429995	1626	247.19	-75	270	Major	HQ
Lemhi	FG20-002C	500210	429995	1626	241.71	-90	0	Major	HQ
Lemhi	FG20-003C	500347	429946	1566	188.37	-90	0	Major	HQ
Lemhi	FG20-004C	500239	429954	1609	222.98	-75	300	Major	HQ
Lemhi	FG20-005C	500403	429948	1566	209.71	-90	0	Major	HQ
Lemhi	FG20-006C	500263	429967	1596	213.06	-75	270	Major	HQ
Lemhi	FG20-007C	500336	429975	1569	181.66	-90	0	Major	HQ
Lemhi	FG20-008C	500264	429950	1596	183.64	-90	0	Major	HQ
Lemhi	FG20-009C	500501	429925	1567	196.90	-90	0	Major	HQ
Lemhi	FG20-010C	500273	429899	1579	173.43	-90	0	Major	HQ
Lemhi	FG20-011C	500245	429875	1584	172.82	-90	0	Major	HQ
Lemhi	FG20-012C	500455	429826	1555	264.26	-90	0	Major	HQ
Lemhi	FG20-013C	500223	429830	1585	184.46	-90	0	Major	HQ
Lemhi	FG20-014C	500565	429875	1561	284.99	-90	0	Major	HQ
Lemhi	FG20-015C	500247	429999	1607	201.17	-90	0	Major	HQ
Lemhi	FG20-016C	500336	430003	1572	163.68	-90	0	Major	HQ
Lemhi	FG20-017C	500167	430001	1641	203.45	-75	270	Major	HQ
Lemhi	FG20-018C	500401	429876	1560	178.46	-90	0	Major	HQ
Lemhi	FG20-019C	500364	429927	1565	170.08	-90	0	Major	HQ
Lemhi	FG20-020C	500197	430029	1634	201.17	-90	0	Major	HQ
Lemhi	FG20-021C	500269	429781	1557	170.08	-90	0	Major	HQ
Lemhi	FG20-022C	500037	430000	1680	223.42	-90	0	Major	HQ
Lemhi	FG20-023C	500164	429644	1595	212.29	-90	0	Major	HQ
Lemhi	FG20-024C	499945	429818	1686	222.20	-90	0	Major	HQ
Lemhi	FG20-025C	500104	429596	1608	238.35	-90	0	Major	HQ
Lemhi	FG20-026C	500043	429726	1651	226.62	-90	0	Major	HQ
Lemhi	FG20-027C	500091	429573	1607	235.15	-90	0	Major	HQ

Prospect	Hole ID	Easting (NAD83 SPIC)	Northing (NAD83 SPIC)	Elevation (m)	Depth (m)	Dip (deg)	Azi (deg)	Drilling Company	Hole Type
Lemhi	FG20-028C	500072	429875	1650	198.12	-90	0	Major	HQ
Lemhi	FG20-029C	500108	429549	1606	248.87	-90	0	Major	HQ
Lemhi	FG20-030C	500094	429750	1634	213.66	-90	0	Major	HQ
Lemhi	FG20-031C	499911	429376	1613	228.30	-90	0	Major	HQ
Lemhi	FG20-032C	500134	429849	1621	70.41	-90	0	Major	HQ
Lemhi	FG20-033C	500129	429850	1621	199.34	-90	0	Major	HQ
Lemhi	FG20-034C	500236	429800	1567	182.27	-90	0	Major	HQ
Lemhi	FG20-035C	500190	429997	1637	199.03	-90	0	Major	PQ
Beauty	FG21-001C	499332	430056	1743	114.91	-90	0	Cabo	HQ
Beauty	FG21-002C	499332	430056	1743	106.68	-65	120	Cabo	HQ
Beauty	FG21-003C	499332	430056	1743	106.98	-65	300	Cabo	HQ
Lemhi	FG21-004C	499911	429993	1708	270.6	-90	0	Cabo	HQ
Lemhi	FG21-005C	499907	429974	1708	272.80	-90	0	Cabo	HQ
Lemhi	FG22-001C	499851	429952	1712	254.20	-90	0	Cabo	HQ
Lemhi	FG22-002C	500582	429796	1552	398.68	-90	0	Major	HQ
Lemhi	FG22-003C	499907	429894	1704	280.42	-90	0	Cabo	HQ
Lemhi	FG22-004C	500586	429851	1557	356.01	-90	0	Major	PQ/HQ
Lemhi	FG22-005C	499874	429823	1700	249.94	-90	0	Cabo	HQ
Lemhi	FG22-006C	499961	429778	1674	278.89	-90	0	Cabo	HQ
Lemhi	FG22-007C	500592	429975	1569	287.73	-90	0	Major	PQ/HQ
Lemhi	FG22-008C	500022	429618	1645	255.73	-68	270	Cabo	HQ
Lemhi	FG22-009C	500321	429947	1568	229.51	-90	0	Major	PQ/HQ
Lemhi	FG22-010C	500327	430024	1573	202.69	-90	0	Major	PQ/HQ
Lemhi	FG22-011C	500036	429695	1648	251.46	-70	270	Cabo	HQ
Lemhi	FG22-012C	500649	429874	1557	332.69	-90	0	Major	PQ/HQ
Beauty	FG22-013C	499346	430108	1757	147.52	-70	255	Cabo	HQ
Lemhi	FG22-014C	500540	429749	1548	351.74	-90	0	Major	PQ/HQ
Beauty	FG22-015C	499346	430108	1757	209.31	-76	300	Cabo	HQ
Lemhi	FG22-016C	500560	429949	1568	250.85	-90	0	Major	PQ/HQ
Lemhi	FG22-017C	500177	429998	1637	409.19	-90	0	Major	PQ/HQ
Lemhi	FG22-018C	500484	430000	1574	278.43	-90	0	Major	PQ/HQ
Lemhi	FG22-019C	500377	429726	1545	232.87	-90	0	Major	PQ/HQ
Beauty	FG22-020C	499346	430108	1757	160.02	-74	275	Cabo	HQ
Lemhi	FG22-021C	500587	429924	1564	247.95	-90	0	Major	PQ/HQ
Lemhi	FG22-022C	500142	429528	1586	159.26	-80	90	Major	PQ/HQ

Prospect	Hole ID	Easting (NAD83 SPIC)	Northing (NAD83 SPIC)	Elevation (m)	Depth (m)	Dip (deg)	Azi (deg)	Drilling Company	Hole Type
Beauty	FG22-023C	499302	430094	1741	92.05	-65	300	Cabo	HQ
Lemhi	FG22-024C	500553	430027	1576	297.03	-90	0	Major	PQ/HQ
Lemhi	FG22-025C	499938	429925	1701	268.99	-90	0	Major	PQ/HQ
Beauty	FG22-026C	499302	430094	1741	113.54	-80	230	Cabo	HQ
Lemhi	FG22-027C	500364	430150	1585	225.70	-90	0	Major	PQ/HQ
Lemhi	FG22-028C	499952	429798	1679	289.56	-90	0	Major	PQ/HQ
Lemhi	FG22-029C	500425	429627	1537	297.48	-90	0	Major	PQ/HQ
Lemhi	FG22-030C	500434	430203	1588	226.47	-90	0	Major	PQ/HQ
Lemhi	FG22-031C	499972	429872	1681	252.98	-90	0	Major	PQ/HQ
Lemhi	FG22-032C	500099	430176	1668	221.89	-90	0	Major	PQ/HQ
Lemhi	FG22-033C	500022	429650	1648	204.98	-74	270	Major	PQ/HQ
Lemhi	FG22-034C	500042	430069	1677	221.89	-90	0	Major	PQ/HQ
Lemhi	FG22-035C	499936	429849	1690	258.32	-90	0	Major	PQ/HQ
Lemhi	FG22-036C	500002	429733	1663	235.31	-75	270	Major	PQ/HQ
Lemhi	FG22-037C	500086	429424	1576	244.75	-90	0	Major	PQ/HQ
Lemhi	FG22-038C	499911	429424	1623	226.92	-59	270	Major	PQ/HQ
Lemhi	FG22-039C	499985	429405	1599	191.41	-90	0	Major	PQ/HQ
Lemhi	FG22-040C	500083	429482	1593	222.35	-65	90	Major	PQ/HQ
Lemhi	FG22-041C	499883	429526	1658	163.68	-85	270	Major	PQ/HQ
Lemhi	FG22-042C	499902	430229	1721	192.63	-90	0	Major	PQ/HQ
Lemhi	FG22-043C	500269	429549	1558	169.47	-90	0	Major	PQ/HQ
Lemhi	FG22-044C	499925	430252	1717	203.30	-90	0	Major	PQ/HQ
Lemhi	FG22-045C	500244	429598	1561	218.54	-90	0	Major	PQ/HQ
Lemhi	FG22-046C	499944	429674	1674	229.51	-90	0	Major	PQ/HQ
Lemhi	FG22-047C	500266	429573	1555	165.20	-90	0	Major	PQ/HQ
Lemhi	FG22-048C	499938	429577	1670	221.44	-90	0	Major	PQ/HQ
Lemhi	FG22-049C	499833	429377	1603	200.71	-80	270	Major	PQ/HQ
Lemhi	FG22-050C	500251	429624	1564	222.35	-90	0	Major	PQ/HQ
Lemhi	FG22-051C	500029	430150	1685	214.12	-90	0	Major	PQ/HQ
Lemhi	FG22-052C	499920	429455	1626	9.60	-90	0	Major	PQ/HQ
Lemhi	FG22-053C	499920	429455	1626	221.89	-90	0	Major	PQ/HQ
Lemhi	FG22-054R	499944	429997	1699	198.12	-90	0	Specialized	RC
Lemhi	FG22-055R	499989	430001	1690	204.22	-90	0	Specialized	RC
Beauty	FG22-056R	499332	430060	1743	198.12	-70	25	Specialized	RC
Beauty	FG22-057R	499330	430122	1750	158.50	-70	65	Specialized	RC

Prospect	Hole ID	Easting (NAD83 SPIC)	Northing (NAD83 SPIC)	Elevation (m)	Depth (m)	Dip (deg)	Azi (deg)	Drilling Company	Hole Type
Beauty	FG22-058R	499305	430100	1741	152.40	-67	5	Specialized	RC
Lemhi	FG22-059R	500041	429630	1640	82.30	-90	0	Specialized	RC
Lemhi	FG22-060R	500036	429633	1640	167.64	-80	270	Specialized	RC
Lemhi	FG22-061R	499892	429352	1608	161.54	-90	0	Specialized	RC
Lemhi	FG22-062R	499910	429774	1693	161.54	-90	0	Specialized	RC
Lemhi	FG22-063R	500037	430128	1680	152.40	-60	270	Specialized	RC
Lemhi	FG22-064R	499955	429950	1698	195.07	-90	0	Specialized	RC
Lemhi	FG22-065R	500005	429551	1636	176.78	-90	0	Specialized	RC
Lemhi	FG22-066R	499954	429900	1693	121.92	-90	0	Specialized	RC

Figure 10-1: Drill Collar Locations for the 2020 and 2021-2022 Programs



Source: APEX, 2023.

10.1 2020 Drilling

From September 13 to December 5, 2020, Freeman conducted a 7,149 m drill program consisting of 35 core holes on the Lemhi Project, see Figure 10-1. The focus of this program was to confirm historical drill results completed by LGT in 2012, reported in detail in Brewer (2019), and other historical drilling summarized in the History section of this report.

In addition to confirming historical mineralization, the objective of the 2020 Phase 1 drill program was designed to allow the use of 385 historical drill holes in a current and inaugural mineral resource estimate (MRE). The drill program focused on infill and step-out drilling within the known mineralized body to increase confidence and maximize the potential resource.

Through the Phase 1 drilling, Freeman confirmed and extended the presence of a number of stacked mineralized structures over a 600 mx 700 m area from surface down to over 260 m in depth. As of January 27, 2021, all geological logging of the core was completed, and samples were submitted to ALS Geochemistry – Vancouver. Of the 35 holes drilled, 23 yielded visible gold within the various mineralized zones (Figure 10-2). Highlights of the results (presented as average grade over drill hole length) are summarized below in Table 10-1. Most of the drill holes intersected shallow high-grade oxide gold, and several highlighted intersections are displayed in Figure 10-3, Figure 10-4, Figure 10-5, and Figure 10-6. Of note, the high-grade zones lie within broader lower grade mineralized envelopes, such as 1.1 g/t Au over 189.1 m (FG20-006C); 2.5 g/t Au over 174.26 m (FG20-017C), and 0.50.54 g/t Au over 189.35 m (FG20-035C).

Figure 10-2: Visible Gold Hosted in Quartz Vein from Drill Hole FG20-002C at 47.25 m, the Sample C375828 from 47 – 48 m Returned 14.45 g/t Au.

Source: APEX, 2023

Table 10-2: 2020 Significant Drill Results

Drill Hole	Depth (m)	DIP	Azimuth	Depth (m)		Interval (m)	Grade (g/t Au)	Highlight
				From	To			
FG20-001C	247	-75	247	28.0	53.0	25.0	3.3	25.0m @ 3.3 g/t Au
Including				32.0	41.0	9.0	4.0	
Including				46.0	53.0	7.0	5.	7.0m @ 5.4 g/t Au
FG20-002C	242	-90	360	6.4	58.0	51.6	3.4	51.6m @ 3.4 g/t Au
Including				47.0	57.0	10.0	14.0	10.0m @ 14 g/t Au
FG20-003C	185	-90	360	40.0	96.0	56.0	1.2	56.0m @ 1.2 g/t Au
Including				81.4	96.0	14.6	3.2	14.6m @ 3.2 g/t
FG20-004C	223	-75	298	0	27.43	27.43	0.4	
				93.03	167.03	74	0.7	
Including				93.03	107.23	14.2	1.8	14.2m @ 1.8 g/t Au
				208.18	209.85	1.67	5.2	
FG20-005C	210	-90	360	42.99	57.07	14.08	2.	
Including				49.03	57.07	8.04	3.5	8.04m @ 3.5 g/t Au
				66.85	123.6	56.75	0.5	
FG20-006C	213	-75	213	12.9	202.1	189.2	1.1	189.2m @ 1.1 g/t Au
Including				37.0	129.0	92.0	1.8	
Including				81.5	89.2	7.7	8.7	7.7m @ 8.7 g/t Au
Including				81.5	85.8	4.3	15.1	
FG20-007C	182	-90	360	7.4	181.66	174.26	0.8	174.26m @ 0.8 g/t Au
Including				15.8	36.01	20.21	2.2	
Including				89.97	97.5	7.53	6.3	7.3m @ 6.3 g/t Au
				14.89	100.85	85.96	1.6	
FG20-008C	184	-90	360	9.36	183.64	174.28	0.9	174.28m @ 0.9 g/t Au
Including				64.7	71.78	7.08	3.8	
Including				82.05	100.58	18.53	3.9	
FG20-009C	197	-90	360	16.46	183.1	166.64	0.3	
Including				155.06	161.98	6.92	2.6	6.92m @ 2.6 g/t Au
FG20-010C	173	-90	360	100.01	136.94	36.93	0.6	
Including				108.02	113.06	5.04	1.7	
FG20-011C	173	-90	360	12.08	153.02	140.94	0.3	
Including				118.1	121.95	3.85	5	3.85m @ 5 g/t Au
Including				118.1	132.02	13.92	1.9	
FG20-012C	264	-90	360	56.86	99.6	42.6	1.2	

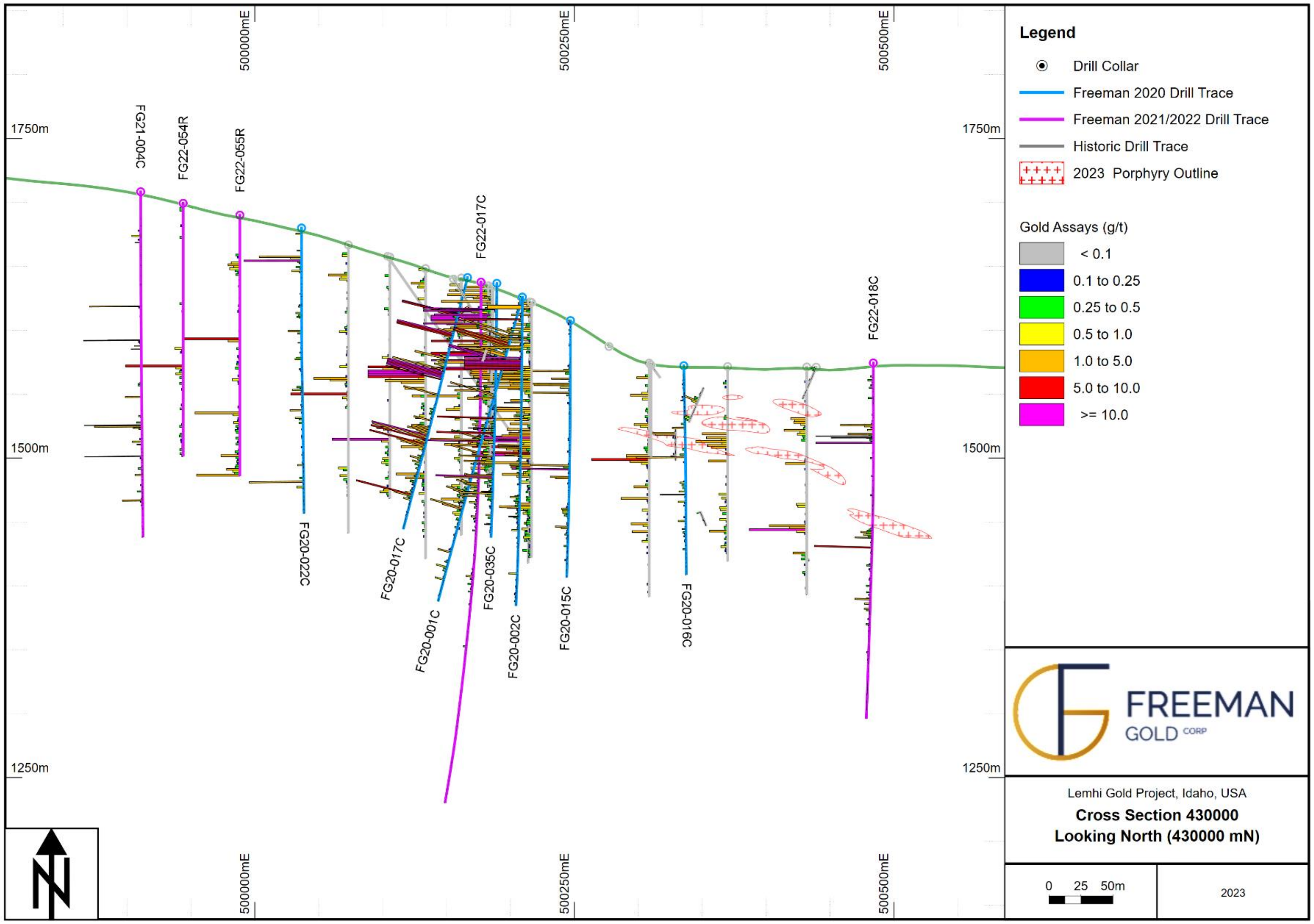
Drill Hole	Depth (m)	DIP	Azimuth	Depth (m)		Interval (m)	Grade (g/t Au)	Highlight
				From	To			
Including				56.86	70.03	13.17	2.5	13.17m @ 2.5 g/t Au
				139.6	234.53	94.93	0.4	
Including				139.6	149.97	10.37	2.1	10.37m @ 2.1 g/t Au
Including				143.69	148.13	4.44	4.	4.44m @ 4.2 g/t Au
FG20-013C	184	-90	360	106.92	127.21	20.29	2.1	
Including				109.12	118.57	9.45	3.	9.45m @ 3.5 g/t Au
Including				110.2	116.89	6.69	4.3	
FG20-014C	286	-90	360	70.02	75.04	5.02	1.1	
				157.87	179.68	21.81	1.2	21.81m @ 1.2 g/t Au
Including				159	163	4	2	
FG20-015C	201	-90	360	35	59	24	1	
Including				49	51	2	4.8	
				113	124	11	2.1	11m @ 2 1 g/t Au
Including				113	117	4	4.9	4m @ 4.9 g/t Au
				146	168	22	0.3	
FG20-016C	164	-90	360	64.8	101.09	36.29	0.25	
Including				71	72	1	4.3	1m @ 4.3 g/t Au
FG20-017C	203	-75	270	29	180	151	2.5	151m @ 2.5 g/t Au
Including				29	33.07	4.07	4.9	4.07m @ 4.9 g/t Au
Including				45	48	3	14.5	3m @ 14.5 g/t Au
Including				74	82.7	8.7	25	8.7m @ 25 g/t Au
Including				121	137	16	3.35	16m @ 3.35 g/t Au
Including				127	131	4	8.3	4m @ 8.3 g/t Au
Including				175	177	2	5.26	2m @ 5.26 g/t Au
FG20-018C	178	-90	360	12	47	35	0.3	
				112.32	163	50.68	0.4	
Including				112.32	124	11.68	1	11.68m @ 1 g/t Au
FG20-019C	170	-90	360	52	56	4	1.2	
				78	127.05	49.05	0.9	49.05m @ 0.9 g/t Au
Including				78	81	3	2.3	3m @ 2.3 g/t Au
Including				101.92	105	3.08	2.9	3.08m @ 2.9 g/t Au
FG20-020C	201	-90	360	75	110	35	0.3	35m @ 0.3 g/t Au
Including				83	84	1	4.2	1m @ 4.2 g/t Au
Including				109	110	1	3.6	1m @ 3.6 g/t Au
FG20-021C	170	-90	360	32.92	57.9	24.98	0.6	24.98m @ 0.6 g/t Au

Drill Hole	Depth (m)	DIP	Azimuth	Depth (m)		Interval (m)	Grade (g/t Au)	Highlight
				From	To			
Including				32.92	34	1.08	3.	1.08m @ 3.1 g/t Au
Including				47	53	6	1.7	6m @ 1.7 g/t Au
				129.1	133	3.9	1.3	
FG20-022C	223	-90	360	4	34.14	30.14	1	30.14m @ 1 g/t Au
Including				22	28	6	4.6	6m @ 4.6 g/t Au
				198	203.32	5.32	1.1	
FG20-023C	212	-90	360	2.13	26.64	24.51	0.5	24.51m @ 0.5 g/t Au
Including				24.91	26.64	1.73	3.	1.73m @ 3.5 g/t Au
				95	98.05	3.05	0.9	3.05m @ 0.9 g/t Au
				120.3	122.8	2.5	1.1	2.5m @ 1.1 g/t Au
				174.45	194.4	19.95	0.6	19.95m @ 0.6 g/t Au
FG20-024C	222	-90	360	143	215	72	0.4	72m @ 0.4 g/t Au
Including				180	181	1	10.15	1m @ 10.15 g/t Au
Including				205.05	208	2.95	1.4	2.95m @ 1.4 g/t Au
FG20-025C	238	-90	360	17.75	69	51.25	0.3	51.25m @ 0.3 g/t Au
Including				26	28	2	1.	2m @ 1.9 g/t Au
				116	127	11	0.6	11m @ 0.6 g/t Au
				189.57	206	16.43	0.5	16.43m @ 0.5 g/t Au
FG20-026C	227	-90	360	21.34	38.06	16.72	0.8	16.2m @ 0.8 g/t Au
Including				22	23	1	5.65	1m @ 5.65 g/t Au
				101	173.37	72.37	0.9	72.37m @ 0.9 g/t Au
Including				139	160.1	21.1	2.1	21.1m @ 2.1 g/t Au
Including				141	149.85	8.85	4.1	8.85m @4.1 g/t Au
Including				171.29	173	1.71	5	1.71m @ 5 g/t Au
FG20-027C	235	-90	360	9	72.54	63.54	0.5	63.54m @ 0.5 g/t Au
Including				63	72.54	9.54	1.9	9.54m @ 1.9 g/t Au
Including				68	72.54	4.54	2.8	4.54m @ 2.8 g/t Au
				192.05	212	19.95	0.5	19.95m @ 0.5 g/t Au
FG20-028C	197	-90	360	20	21	1	2	1m @ 2 g/t Au
				76	77	1	1.2	1m @ 1.2 g/t Au
				95	192	97	0.5	97m @ 0.5 g/t Au
Including				149	174	25	1.1	25m @ 1.1 g/t Au
Including				155	156	1	10.85	1m @ 10.85 g/t Au
FG20-029C	249	-90	360	48	66	18	1.1	18m @ 1.1 g/t Au

Drill Hole	Depth (m)	DIP	Azimuth	Depth (m)		Interval (m)	Grade (g/t Au)	Highlight
				From	To			
				202	215	13	0.4	13m @ 0.4 g/t Au
Including				202	203.4	1.4	1.3	1.4m @ 1.3 g/t Au
FG20-030C	214	-90	360	4	123	119	0.4	119m @ 0.4 g/t Au
Including				72.97	95	22.03	1	22.03m @ 1 g/t Au
Including				75.81	78.1	2.29	2.9	2.29m @ 2.9 g/t Au
				109.15	123	13.85	1.1	13.85m @ 1.1 g/t Au
				145	150.86	5.86	1	5.86m @ 1 g/t Au
				167	173.13	6.13	0.9	6.13m @ 0.9 g/t Au
FG20-031C	228	-90	360	39	87.15	48.15	0.4	48.15m @ 0.4 g/t Au
Including				71.17	74	2.83	2.4	2.83m @ 2.4 g/t Au
				179.98	188.05	8.07	2.	8.07m @ 2.1 g/t Au
FG20-032C	70	-90	360					NSR–- LOST HOLE
FG20-033C	199	-90	360	112.25	161	48.75	1.4	48.75m @ 1.4 g/t Au
Including				116	138	22	2.1	22m @ 2.1 g/t Au
Including				155.75	160.32	4.57	4	4.57m @ 4 g/t Au
FG20-034C	182	-90	360	102.32	109.95	7.63	2.3	7.63m @ 2.3 g/t Au
				132	141	9	1.5	9m @ 1 5 g/t Au
Including				133.01	135	1.99	4	1.99m @ 4 g/t Au
FG20-035C	199	-90	360	8.65	189	180.35	0.54	189.35m @ 0.54 g/t Au
Including				20	23	3	3.9	3m @ 3.9 g/t Au
				49.95	53	3.05	2.7	3.05m @ 2 7 g/t Au
				128.47	167	38.53	1.1	38.53m @ 1.1 g/t Au
				149.46	153	3.54	6.6	3.54m @ 6.6 g/t Au

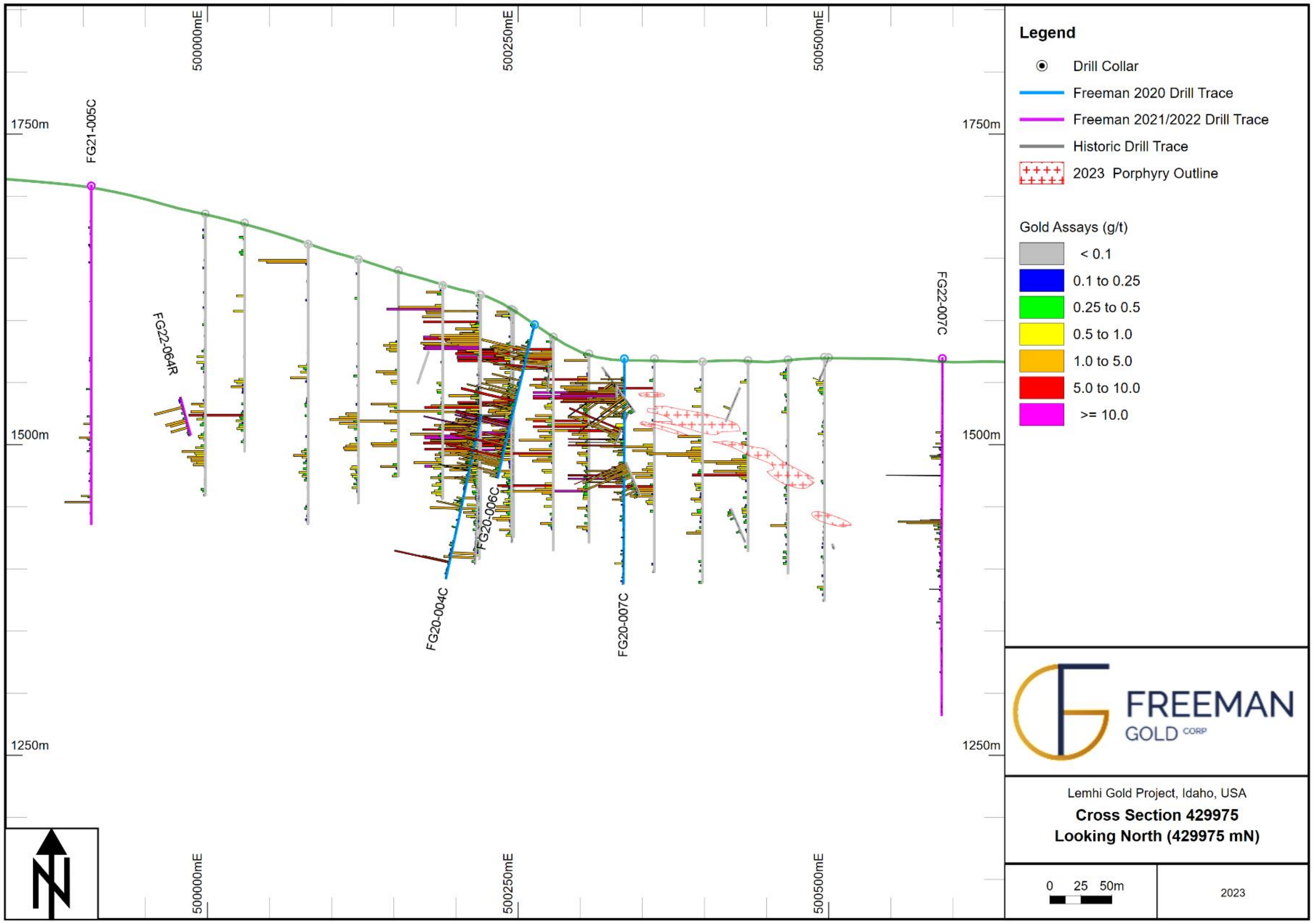
Note: True thickness is estimated to be 70 to 100% of drill interval thickness.
Source: APEX, 2023.

Figure 10-3: Drill Section 430000 with Highlighted Results of FG20-001C, FG20-002C, FG20-017, FG20-035C and FG22-017C, Among Other Recent Results



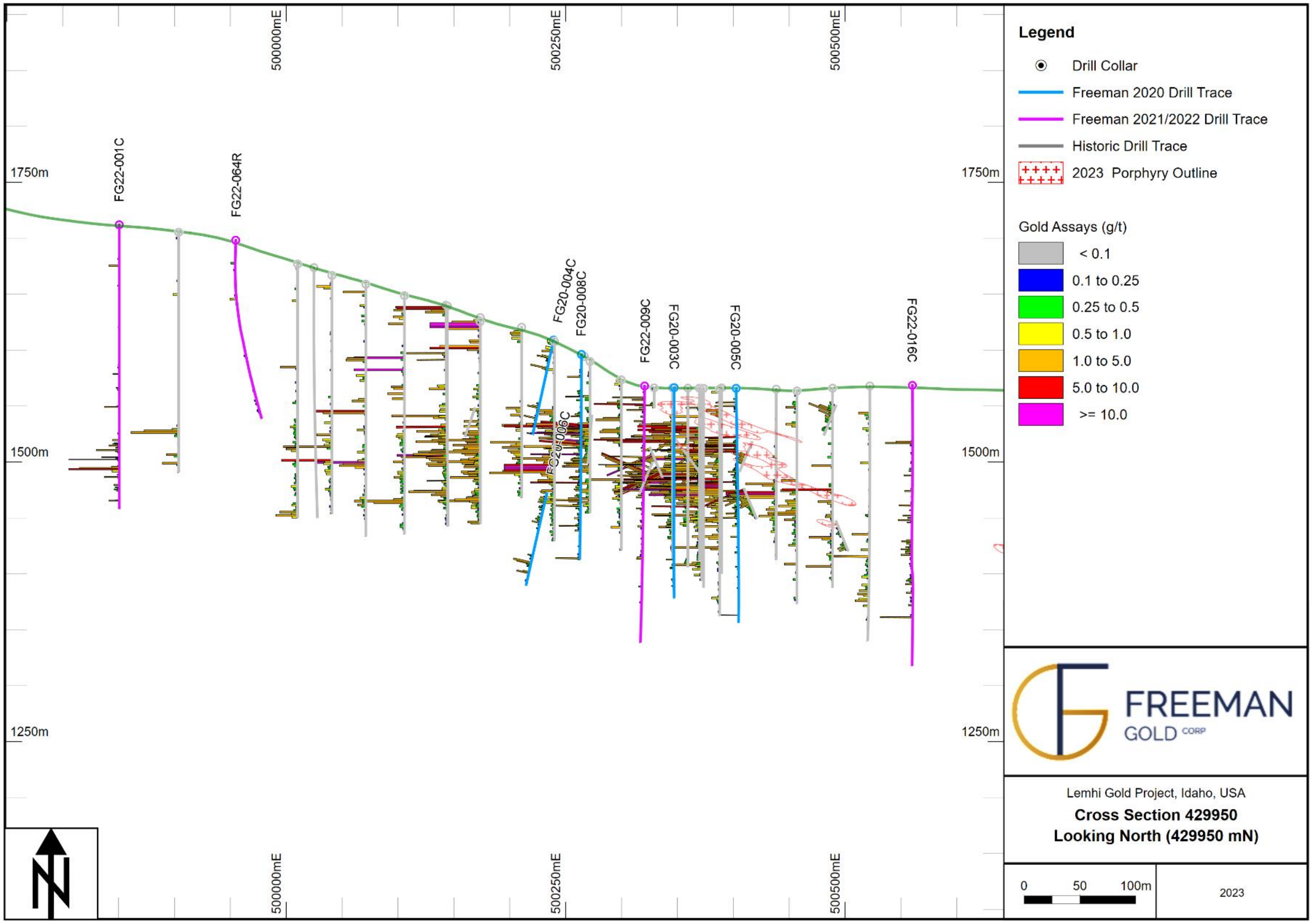
Source: APEX, 2023.

Figure 10-4: Drill Section 429975 with Highlighted Results of FG20-007C and FG20-008C



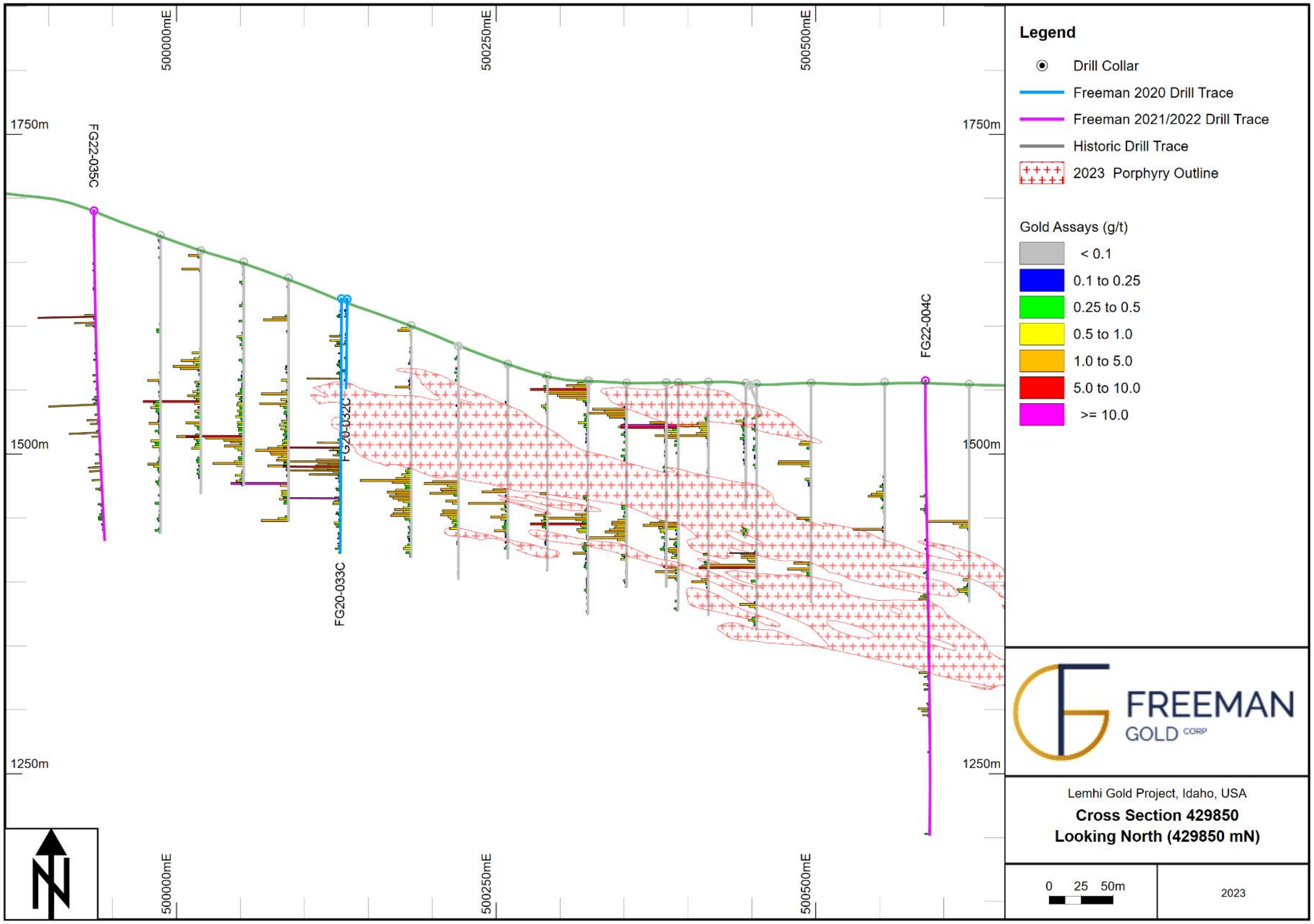
Source: APEX, 2023.

Figure 10-5: Drill Section 429950 with Highlight Results of FG20-003C and FG20-008C



Source: APEX, 2023.

Figure 10-6: Drill Section 429850 with Highlight Results of FG20-033C



Source: APEX, 2023.

10.2 2021 – 2022 Drilling

From November 14, 2021, to November 21, 2022, Freeman conducted a 15,351 m drill program consisting of 58 core holes and 13 RC holes on the Lemhi Project, see Figure 10-1 and Table 10-2. The aim of this program was to expand and infill the existing resource at Lemhi, and to provide increased confidence in areas with less reliable historical drilling results.

Core drilling was completed by Cabo Drilling Corp. of New Westminster, BC and by Major Drilling Group International, of Moncton, NB. Core drilling completed by Cabo was run at HQ size. Major Drilling completed the first few holes at HQ size. HQ core at the top of the first few holes drilled slow with low recoveries. Major began coring PQ size to top of fresh rock before casing and running HQ to depth. Drill core was oriented wherever possible, however the friable and broken nature of the rocks at Lemhi impeded regular and reliable orientation of sequential core runs. Holes were quick-logged by geologists on site, before being shipped to the core shack in Edmonton, AB for detailed geological and geotechnical logging.

Vertical core holes were surveyed downhole by a north-seeking Reflex EZ-Gyro every 15.2 m. Continuous surveys were also run on angled core holes in addition to the 15.2 m multi-shot captures. The survey data was examined, and problematic data removed where the tools recorded errors or flawed measurements. The core holes were generally straight, and the surveys are considered to be a reasonable representation of the hole paths.

RC drilling was conducted by Specialized Drilling Corp. of San Diego, CA, using a Fraste Multidrill XL rig and XRVS portable compressor with 1,000 cfm x 500 psi air capacity. The rig ran a 7.6 cm face-sampling hammer with a 3.8 cm diameter rod string.

RC chip returns were collected into clean buckets directly from a rig-mounted cyclone at 1.52 m intervals and fed through a riffle splitter. The bulk of the return was retained and split samples of 0.5 – 1 kg were sent for analysis to ALS in barcoded calico sample bags. Chips were sieved from each 1.52 m retention sample on-site and shipped in Kraft bags to the core shack in Edmonton AB, where they were cleaned and logged by geologists for various attributes including lithology, alteration, mineralization, and veining. On-site geologists logged information for each sample interval including size (indicative of recovery), condition, clay proportion and any obvious presence of contamination.

Downhole surveying of RC holes was completed using a referential Inertial Sensing SlimGyro. Collar setup references were collected by a north-seeking Reflex TN14 Gyrocompass. Downhole surveys were collected as continuous runs with capture stations every 15.2 m. The RC downhole survey paths demonstrated high deviation over short distances, and the data was variable with repeated surveys. It is considered that the relatively soft rocks of the Lemhi sedimentary package, combined with strong preferential fabrics, dense structural influence, and the small-diameter rod string could all have contributed to high deviation of the RC drill holes. Given these considerations and the variability of the data from the SlimGyro, the resultant hole paths are determined to be largely acceptable, though regarded with some caution.

All 2021-2022 drillhole collars were captured using a Topcon HiPer VR differential GPS, accurate to ±0.10 m. The data were corroborated by the planned coordinates and by handheld Garmin GPS accurate to ±5 m.

As of December 8, 2022, all geological logging of the core and RC chips was complete, and all samples were submitted to ALS Geochemistry – Vancouver. All core and RC sample results were received by January 21, 2023, and select significant intersections are summarized below in Table 10-3.

Drillhole geological logs were inspected upon receipt of assay results to verify the correlation of anomalous intersections with expected and appropriate lithologies, alteration and mineralization. Assay data was directly reported by the issuing laboratory and no adjustment of the data was undertaken.

Where possible, drillholes were drilled vertically to intersect mineralization. While the strike of the mineralization is variable and vertical holes are not always perpendicular to the strike of the deposit, this orientation conforms with the dense historical drilling and has been accepted as the optimal drill orientation for the project. Anomalous intersections reported herein are expressed as downhole widths as opposed to true width intersections. Drillhole spacing was designed in conjunction with historical hole spacing and close enough to confirm continuity of mineralization.

The 2021-2022 drilling expanded the mineralization at Lemhi down- and up-dip of the existing Lemhi resource. Best intersections were encountered in the north-central portion of the deposit, and along south-east extensions to the deposit. Examples of these intersections include:

- 3.7 m at 10.2 g/t Au from 20.27 m in FG22-017C
- 10.7 m at 3.0 g/t Au from 74.68 m in FG22-061R
- 15 m at 2.1 g/t Au from 93 m in FG22-050C
- 7.2 m at 3.8 g/t Au from 121.31 m in FG22-022C.

These and other anomalous intersections are displayed in Figure 10-7, Figure 10-8, Figure 10-9, and Figure 10-10.

Drilling at the Beauty target (Figure 10-10) confirmed the continuation of mineralization at depth from the veins mapped and sampled at surface. Best intersections included:

- 5.2 m at 78.7 g/t Au from 57.8 m in FG21-003C
- 3.0 m at 4.4 g/t Au from 134.1 m in FG22-056R
- including 1.5 m at 8.0 g/t Au from 135.6 m in FG22-056R.

Select anomalous intersections from Beauty are presented in Table 10-3.

Table 10-3: Selected Significant Intersections (>0.5 g/t Au) from 2021-2022 Drilling at Lemhi

Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
FG21-004C	89.31	90.00	0.69	4.48	0.7 m at 4.5 g/t Au	3.1
	115.80	116.43	0.63	23.11	0.6 m at 23.1 g/t Au	14.6
Including	116.00	116.43	0.43	32.50	0.4 m at 32.5 g/t Au	14.0
	183.00	185.10	2.10	2.34	2.1 m at 2.3 g/t Au	4.9
FG21-005C	202.10	203.30	1.20	1.06	1.2 m at 1.1 g/t Au	1.3
	249.00	250.00	1.00	0.90	1.0 m at 0.9 g/t Au	0.9
	254.00	255.00	1.00	2.36	1.0 m at 2.4 g/t Au	2.4
FG22-001C	201.00	202.00	1.00	2.28	1.0 m at 2.3 g/t Au	2.3
	209.70	210.00	0.30	7.41	0.3 m at 7.4 g/t Au	2.2
	216.00	221.00	5.00	2.82	5.0 m at 2.8 g/t Au	14.1
Including	218.00	219.06	1.06	7.56	1.1 m at 7.6 g/t Au	8.0
FG22-002C	230.30	231.88	1.58	0.96	1.6 m at 1.0 g/t Au	1.5
	340.60	342.00	1.40	1.32	1.4 m at 1.3 g/t Au	1.8
	345.90	346.95	1.05	1.54	1.1 m at 1.5 g/t Au	1.6
FG22-003C	161.00	164.00	3.00	1.70	3.0 m at 1.7 g/t Au	5.1
	197.00	198.00	1.00	3.51	1.0 m at 3.5 g/t Au	3.5
	200.00	201.00	1.00	2.50	1.0 m at 2.5 g/t Au	2.5
FG22-004C	170.00	171.26	1.26	0.82	1.3 m at 0.8 g/t Au	1.0
	227.69	229.00	1.31	0.87	1.3 m at 0.9 g/t Au	1.1
	255.88	257.00	1.12	0.97	1.1 m at 1.0 g/t Au	1.1
FG22-005C	119.00	120.00	1.00	5.09	1.0 m at 5.1 g/t Au	5.1
	138.00	139.00	1.00	2.01	1.0 m at 2.0 g/t Au	2.0
	155.00	156.00	1.00	1.21	1.0 m at 1.2 g/t Au	1.2
FG22-006C	129.00	130.00	1.00	0.92	1.0 m at 0.9 g/t Au	0.9
	153.00	154.00	1.00	1.91	1.0 m at 1.9 g/t Au	1.9
	188.00	189.00	1.00	0.98	1.0 m at 1.0 g/t Au	1.0
FG22-007C	79.56	81.00	1.44	0.96	1.4 m at 1.0 g/t Au	1.4
	94.12	94.51	0.39	5.45	0.4 m at 5.5 g/t Au	2.1
	129.55	133.00	3.45	2.72	3.4 m at 2.7 g/t Au	9.4
FG22-008C	61.00	62.00	1.00	2.69	1.0 m at 2.7 g/t Au	2.7
	152.20	153.54	1.34	1.38	1.3 m at 1.4 g/t Au	1.9
	177.00	178.00	1.00	1.36	1.0 m at 1.4 g/t Au	1.4
FG22-009C	16.00	17.00	1.00	5.44	1.0 m at 5.4 g/t Au	5.4
	48.00	49.00	1.00	8.46	1.0 m at 8.5 g/t Au	8.5
	79.00	83.00	4.00	1.60	4.0 m at 1.6 g/t Au	6.4

Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
FG22-010C	48.00	49.17	1.17	1.44	1.2 m at 1.4 g/t Au	1.7
	62.00	64.00	2.00	4.61	2.0 m at 4.6 g/t Au	9.2
Including	62.00	63.00	1.00	8.18	1.0 m at 8.2 g/t Au	8.2
	70.00	71.00	1.00	2.29	1.0 m at 2.3 g/t Au	2.3
FG22-011C	40.00	41.00	1.00	2.50	1.0 m at 2.5 g/t Au	2.5
	150.00	154.08	4.08	1.74	4.1 m at 1.7 g/t Au	7.1
	158.00	161.00	3.00	4.97	3.0 m at 5.0 g/t Au	14.9
Including	158.50	160.30	1.80	7.76	1.8 m at 7.8 g/t Au	14.0
FG22-012C	128.08	128.63	0.55	8.24	0.5 m at 8.2 g/t Au	4.5
	136.00	137.00	1.00	1.12	1.0 m at 1.1 g/t Au	1.1
	161.00	162.39	1.39	1.04	1.4 m at 1.0 g/t Au	1.4
FG22-014C	115.00	121.00	6.00	1.83	6.0 m at 1.8 g/t Au	11.0
Including	119.00	120.00	1.00	5.62	1.0 m at 5.6 g/t Au	5.6
	128.83	130.00	1.17	7.64	1.2 m at 7.6 g/t Au	8.9
	145.00	146.00	1.00	6.77	1.0 m at 6.8 g/t Au	6.8
FG22-016C	50.00	52.00	2.00	2.17	2.0 m at 2.2 g/t Au	4.3
	138.00	140.00	2.00	0.76	2.0 m at 0.8 g/t Au	1.5
	207.00	208.00	1.00	3.22	1.0 m at 3.2 g/t Au	3.2
FG22-017C	20.27	24.00	3.73	10.24	3.7 m at 10.2 g/t Au	38.2
Including	20.27	21.00	0.73	13.65	0.7 m at 13.7 g/t Au	10.0
Including	22.00	23.00	1.00	25.00	1.0 m at 25.0 g/t Au	25.0
	32.00	33.00	1.00	14.25	1.0 m at 14.2 g/t Au	14.3
	122.95	126.87	3.92	5.51	3.9 m at 5.5 g/t Au	21.6
Including	124.06	124.97	0.91	17.10	0.9 m at 17.1 g/t Au	15.6
FG22-018C	57.00	57.30	0.30	32.40	0.3 m at 32.4 g/t Au	9.7
	62.00	63.00	1.00	13.60	1.0 m at 13.6 g/t Au	13.6
	144.00	145.00	1.00	8.09	1.0 m at 8.1 g/t Au	8.1
FG22-019C	70.00	72.00	2.00	1.07	2.0 m at 1.1 g/t Au	2.1
	73.00	75.00	2.00	2.97	2.0 m at 3.0 g/t Au	5.9
Including	73.00	74.00	1.00	5.04	1.0 m at 5.0 g/t Au	5.0
	91.00	93.00	2.00	1.42	2.0 m at 1.4 g/t Au	2.8
FG22-021C	139.00	144.00	5.00	1.68	5.0 m at 1.7 g/t Au	8.4
	146.00	148.00	2.00	1.75	2.0 m at 1.7 g/t Au	3.5
	196.00	197.00	1.00	1.86	1.0 m at 1.9 g/t Au	1.9
FG22-022C	52.00	53.00	1.00	0.70	1.0 m at 0.7 g/t Au	0.7
	121.31	128.47	7.16	3.80	7.2 m at 3.8 g/t Au	27.2

Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
Including	121.31	121.80	0.49	6.63	0.5 m at 6.6 g/t Au	3.2
Including	126.00	127.00	1.00	5.24	1.0 m at 5.2 g/t Au	5.2
	128.63	130.45	1.82	5.18	1.8 m at 5.2 g/t Au	9.4
Including	128.63	129.00	0.37	9.58	0.4 m at 9.6 g/t Au	3.5
FG22-024C	151.00	152.00	1.00	1.19	1.0 m at 1.2 g/t Au	1.2
	155.00	158.00	3.00	1.88	3.0 m at 1.9 g/t Au	5.6
	177.00	178.00	1.00	0.67	1.0 m at 0.7 g/t Au	0.7
FG22-025C	95.00	96.00	1.00	2.47	1.0 m at 2. 5 g/t Au	2. 5
	110. 00	112. 00	2. 00	1. 39	2. 0 m at 1. 4 g/t Au	2. 8
	160. 00	161. 00	1. 00	3. 37	1. 0 m at 3. 4 g/t Au	3. 4
FG22-027C	18. 00	19. 00	1. 00	1. 53	1. 0 m at 1. 5 g/t Au	1. 5
	102. 00	103. 00	1. 00	0. 82	1. 0 m at 0. 8 g/t Au	0. 8
	111. 20	112. 00	0. 80	1. 84	0. 8 m at 1. 8 g/t Au	1. 5
FG22-028C	35. 00	36. 00	1. 00	1. 32	1. 0 m at 1. 3 g/t Au	1. 3
	147. 00	149. 00	2. 00	1. 88	2. 0 m at 1. 9 g/t Au	3. 8
	186. 00	186. 94	0. 94	0. 81	0. 9 m at 0. 8 g/t Au	0. 8
FG22-029C	223. 35	224. 85	1. 50	4. 77	1. 5 m at 4. 8 g/t Au	7. 2
	229. 00	230. 00	1. 00	1. 26	1. 0 m at 1. 3 g/t Au	1. 3
	277. 00	279. 00	2. 00	0. 61	2. 0 m at 0. 6 g/t Au	1. 2
FG22-030C	134. 35	135. 34	0. 99	1. 80	1. 0 m at 1. 8 g/t Au	1. 8
	146. 98	148. 00	1. 02	5. 21	1. 0 m at 5. 2 g/t Au	5. 3
	162. 00	164. 00	2. 00	14. 41	2. 0 m at 14. 4 g/t Au	28. 8
Including	163. 00	164. 00	1. 00	27. 50	1. 0 m at 27. 5 g/t Au	27. 5
FG22-031C	139. 00	143. 00	4. 00	2. 25	4. 0 m at 2. 2 g/t Au	9. 0
	154. 00	155. 00	1. 00	3. 15	1. 0 m at 3. 1 g/t Au	3. 2
	177. 00	182. 00	5. 00	1. 38	5. 0 m at 1. 4 g/t Au	6. 9
FG22-032C	12. 00	13. 00	1. 00	4. 34	1. 0 m at 4. 3 g/t Au	4. 3
	64. 05	65. 08	1. 03	1. 55	1. 0 m at 1. 6 g/t Au	1. 6
	148. 00	149. 00	1. 00	2. 05	1. 0 m at 2. 0 g/t Au	2. 1
FG22-033C	7. 00	8. 00	1. 00	1. 94	1. 0 m at 1. 9 g/t Au	1. 9
	176. 55	176. 94	0. 39	9. 96	0. 4 m at 10. 0 g/t Au	3. 9
	187. 67	189. 00	1. 33	1. 78	1. 3 m at 1. 8 g/t Au	2. 4
FG22-034C	134. 00	135. 00	1. 00	6. 79	1. 0 m at 6. 8 g/t Au	6. 8
	171. 00	174. 00	3. 00	4. 22	3. 0 m at 4. 2 g/t Au	12. 7
Including	171. 00	172. 00	1. 00	9. 97	1. 0 m at 10. 0 g/t Au	10. 0
	175. 00	177. 00	2. 00	10. 52	2. 0 m at 10. 5 g/t Au	21. 0

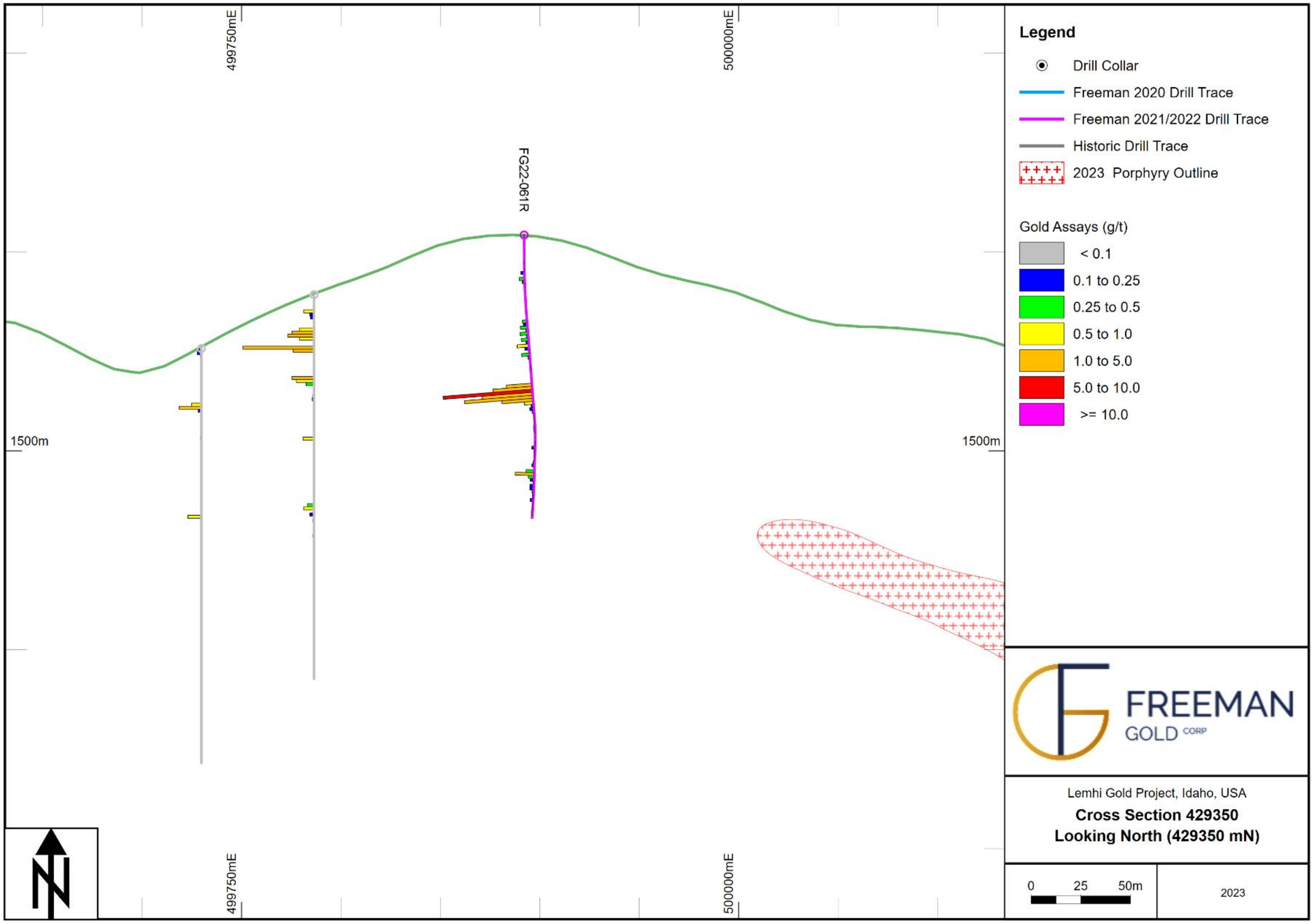
Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
Including	175. 00	176. 00	1. 00	20. 40	1. 0 m at 20. 4 g/t Au	20. 4
FG22-035C	81. 00	83. 21	2. 21	2. 84	2. 2 m at 2. 8 g/t Au	6. 3
Including	82. 30	83. 21	0. 91	5. 56	0. 9 m at 5. 6 g/t Au	5. 1
	151. 00	152. 00	1. 00	4. 31	1. 0 m at 4. 3 g/t Au	4. 3
	173. 00	174. 00	1. 00	2. 62	1. 0 m at 2. 6 g/t Au	2. 6
FG22-036C	18. 00	20. 00	2. 00	3. 73	2. 0 m at 3. 7 g/t Au	7. 5
Including	19. 00	20. 00	1. 00	6. 13	1. 0 m at 6. 1 g/t Au	6. 1
	125. 00	126. 80	1. 80	4. 39	1. 8 m at 4. 4 g/t Au	7. 9
Including	125. 00	126. 00	1. 00	6. 54	1. 0 m at 6. 5 g/t Au	6. 5
	142. 00	147. 68	5. 68	2. 49	5. 7 m at 2. 5 g/t Au	14. 1
Including	146. 15	147. 68	1. 53	5. 31	1. 5 m at 5. 3 g/t Au	8. 1
FG22-037C	93. 00	102. 00	9. 00	1. 83	9. 0 m at 1. 8 g/t Au	16. 5
Including	94. 00	95. 00	1. 00	5. 33	1. 0 m at 5. 3 g/t Au	5. 3
	138. 00	141. 00	3. 00	1. 82	3. 0 m at 1. 8 g/t Au	5. 5
	227. 00	229. 00	2. 00	2. 04	2. 0 m at 2. 0 g/t Au	4. 1
FG22-038C	62. 00	64. 00	2. 00	3. 28	2. 0 m at 3. 3 g/t Au	6. 6
	78. 00	79. 00	1. 00	0. 98	1. 0 m at 1. 0 g/t Au	1. 0
	213. 00	214. 00	1. 00	1. 06	1. 0 m at 1. 1 g/t Au	1. 1
FG22-039C	119. 00	120. 00	1. 00	0. 83	1. 0 m at 0. 8 g/t Au	0. 8
	122. 00	123. 00	1. 00	0. 88	1. 0 m at 0. 9 g/t Au	0. 9
FG22-040C	102. 00	103. 00	1. 00	0. 96	1. 0 m at 1. 0 g/t Au	1. 0
	138. 00	139. 20	1. 20	1. 17	1. 2 m at 1. 2 g/t Au	1. 4
FG22-041C	3. 00	4. 00	1. 00	6. 07	1. 0 m at 6. 1 g/t Au	6. 1
	14. 00	15. 00	1. 00	0. 90	1. 0 m at 0. 9 g/t Au	0. 9
	63. 00	64. 00	1. 00	2. 08	1. 0 m at 2. 1 g/t Au	2. 1
FG22-042C	72. 00	73. 00	1. 00	1. 43	1. 0 m at 1. 4 g/t Au	1. 4
	140. 00	141. 00	1. 00	1. 43	1. 0 m at 1. 4 g/t Au	1. 4
	163. 00	164. 74	1. 74	0. 76	1. 7 m at 0. 8 g/t Au	1. 3
FG22-043C	80. 00	81. 00	1. 00	1. 80	1. 0 m at 1. 8 g/t Au	1. 8
	136. 00	138. 00	2. 00	1. 03	2. 0 m at 1. 0 g/t Au	2. 1
	152. 00	155. 00	3. 00	1. 73	3. 0 m at 1. 7 g/t Au	5. 2
FG22-044C	43. 30	43. 82	0. 52	3. 08	0. 5 m at 3. 1 g/t Au	1. 6
	158. 00	159. 60	1. 60	5. 92	1. 6 m at 5. 9 g/t Au	9. 5
Including	158. 65	159. 60	0. 95	8. 51	0. 9 m at 8. 5 g/t Au	8. 1
	180. 00	181. 00	1. 00	1. 24	1. 0 m at 1. 2 g/t Au	1. 2
FG22-045C	89. 00	90. 00	1. 00	1. 04	1. 0 m at 1. 0 g/t Au	1. 0

Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
	91. 00	93. 00	2. 00	1. 38	2. 0 m at 1. 4 g/t Au	2. 8
	97. 00	99. 00	2. 00	2. 17	2. 0 m at 2. 2 g/t Au	4. 3
FG22-046C	185. 00	187. 00	2. 00	0. 82	2. 0 m at 0. 8 g/t Au	1. 6
	188. 00	189. 89	1. 89	0. 60	1. 9 m at 0. 6 g/t Au	1. 1
	191. 00	192. 00	1. 00	1. 04	1. 0 m at 1. 0 g/t Au	1. 0
FG22-047C	118. 87	121. 00	2. 13	1. 30	2. 1 m at 1. 3 g/t Au	2. 8
	140. 00	148. 00	8. 00	1. 99	8. 0 m at 2. 0 g/t Au	15. 9
Including	142. 00	143. 00	1. 00	5. 41	1. 0 m at 5. 4 g/t Au	5. 4
FG22-048C	86. 00	88. 00	2. 00	1. 52	2. 0 m at 1. 5 g/t Au	3. 0
	89. 00	92. 00	3. 00	2. 60	3. 0 m at 2. 6 g/t Au	7. 8
	139. 75	140. 82	1. 07	3. 38	1. 1 m at 3. 4 g/t Au	3. 6
FG22-049C	25. 00	26. 00	1. 00	3. 24	1. 0 m at 3. 2 g/t Au	3. 2
	51. 00	53. 00	2. 00	0. 87	2. 0 m at 0. 9 g/t Au	1. 7
	71. 00	72. 00	1. 00	1. 44	1. 0 m at 1. 4 g/t Au	1. 4
FG22-050C	93. 00	108. 00	15. 00	2. 13	15. 0 m at 2. 1 g/t Au	32. 0
	112. 00	116. 59	4. 59	2. 30	4. 6 m at 2. 3 g/t Au	10. 5
Including	115. 00	116. 00	1. 00	5. 40	1. 0 m at 5. 4 g/t Au	5. 4
	176. 75	178. 00	1. 25	2. 59	1. 2 m at 2. 6 g/t Au	3. 2
FG22-053C	107. 00	109. 00	2. 00	2. 71	2. 0 m at 2. 7 g/t Au	5. 4
	112. 00	114. 00	2. 00	1. 14	2. 0 m at 1. 1 g/t Au	2. 3
	179. 00	182. 00	3. 00	1. 85	3. 0 m at 1. 8 g/t Au	5. 5
FG22-054R	112. 78	114. 30	1. 52	0. 73	1. 5 m at 0. 7 g/t Au	1. 1
	126. 49	129. 54	3. 05	3. 33	3. 0 m at 3. 3 g/t Au	10. 1
Including	126. 49	128. 02	1. 52	6. 02	1. 5 m at 6. 0 g/t Au	9. 2
	132. 59	134. 11	1. 52	0. 64	1. 5 m at 0. 6 g/t Au	1. 0
FG22-055R	96. 01	99. 06	3. 05	3. 72	3. 0 m at 3. 7 g/t Au	11. 3
Including	96. 01	97. 54	1. 52	6. 42	1. 5 m at 6. 4 g/t Au	9. 8
	153. 92	156. 97	3. 05	2. 25	3. 0 m at 2. 2 g/t Au	6. 9
	202. 69	204. 22	1. 52	3. 76	1. 5 m at 3. 8 g/t Au	5. 7
FG22-060R	60. 96	62. 48	1. 52	0. 76	1. 5 m at 0. 8 g/t Au	1. 2
	132. 59	134. 11	1. 52	1. 78	1. 5 m at 1. 8 g/t Au	2. 7
	155. 45	161. 54	6. 10	1. 06	6. 1 m at 1. 1 g/t Au	6. 5
FG22-061R	54. 86	56. 39	1. 52	0. 67	1. 5 m at 0. 7 g/t Au	1. 0
	74. 68	85. 34	10. 67	3. 04	10. 7 m at 3. 0 g/t Au	32. 4
Including	77. 72	79. 25	1. 52	8. 76	1. 5 m at 8. 8 g/t Au	13. 4
	120. 40	121. 92	1. 52	1. 07	1. 5 m at 1. 1 g/t Au	1. 6

Hole ID	From (m)	To (m)	Width (m)	Au (g/t)	Intersection	Au (g x m)
FG22-062R	27. 43	28. 96	1. 52	2. 14	1. 5 m at 2. 1 g/t Au	3. 3
FG22-063R	27. 43	28. 96	1. 52	0. 71	1. 5 m at 0. 7 g/t Au	1. 1
	123. 44	131. 06	7. 62	2. 52	7. 6 m at 2. 5 g/t Au	19. 2
Including	126. 49	128. 02	1. 52	7. 72	1. 5 m at 7. 7 g/t Au	11. 8
FG22-064R	170. 69	172. 21	1. 52	2. 62	1. 5 m at 2. 6 g/t Au	4. 0
	181. 36	184. 40	3. 05	1. 70	3. 0 m at 1. 7 g/t Au	5. 2
	187. 45	188. 98	1. 52	1. 60	1. 5 m at 1. 6 g/t Au	2. 4
FG22-065R	80. 77	82. 30	1. 52	1. 02	1. 5 m at 1. 0 g/t Au	1. 5
	128. 02	129. 54	1. 52	1. 21	1. 5 m at 1. 2 g/t Au	1. 8
	166. 12	169. 16	3. 05	0. 83	3. 0 m at 0. 8 g/t Au	2. 5
FG22-066R	67. 06	70. 10	3. 05	0. 69	3. 0 m at 0. 7 g/t Au	2. 1
	73. 15	74. 68	1. 52	1. 58	1. 5 m at 1. 6 g/t Au	2. 4
	77. 72	80. 77	3. 05	0. 93	3. 0 m at 0. 9 g/t Au	2. 8

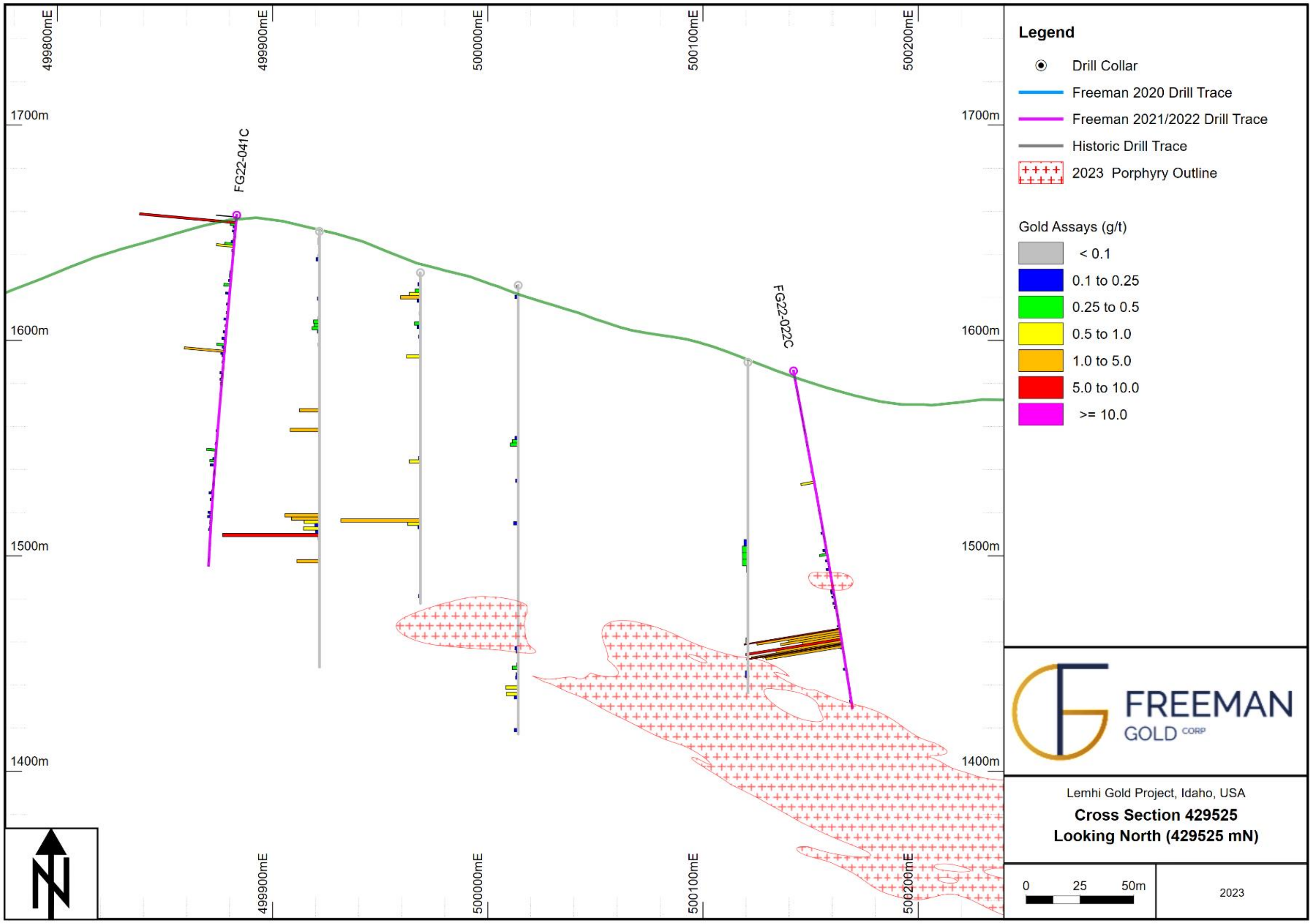
Note: True thickness is estimated to be 70 to 100% of drill interval thickness.
Source: APEX, 2023.

Figure 10-7: Drill Section 429350 N Highlighting Results of FG22-061R



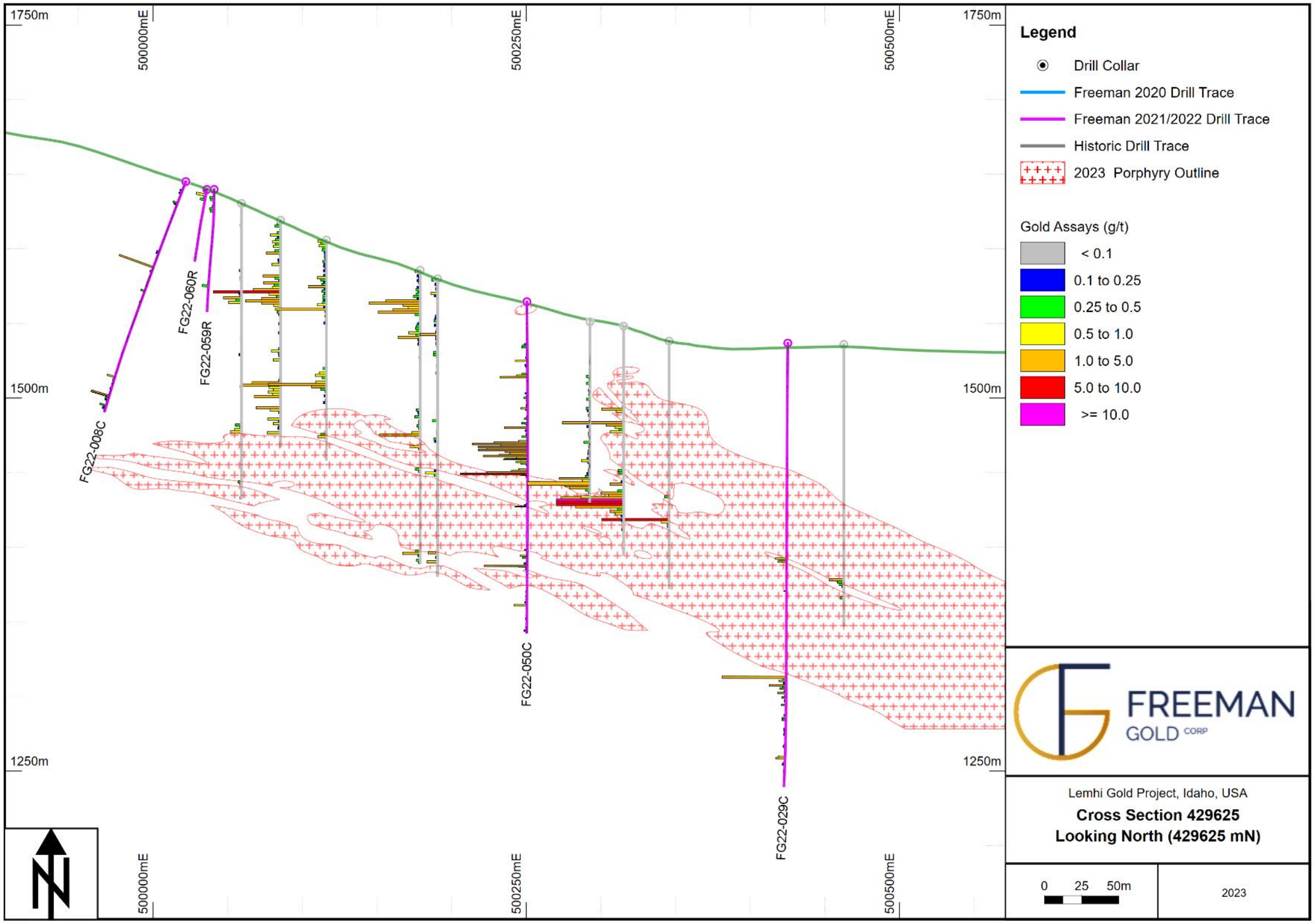
Source: APEX, 2023.

Figure 10-8: Drill Section 429525 N Highlighting Results of FG22-022C and Other 2021-2022 Anomalous Intersections



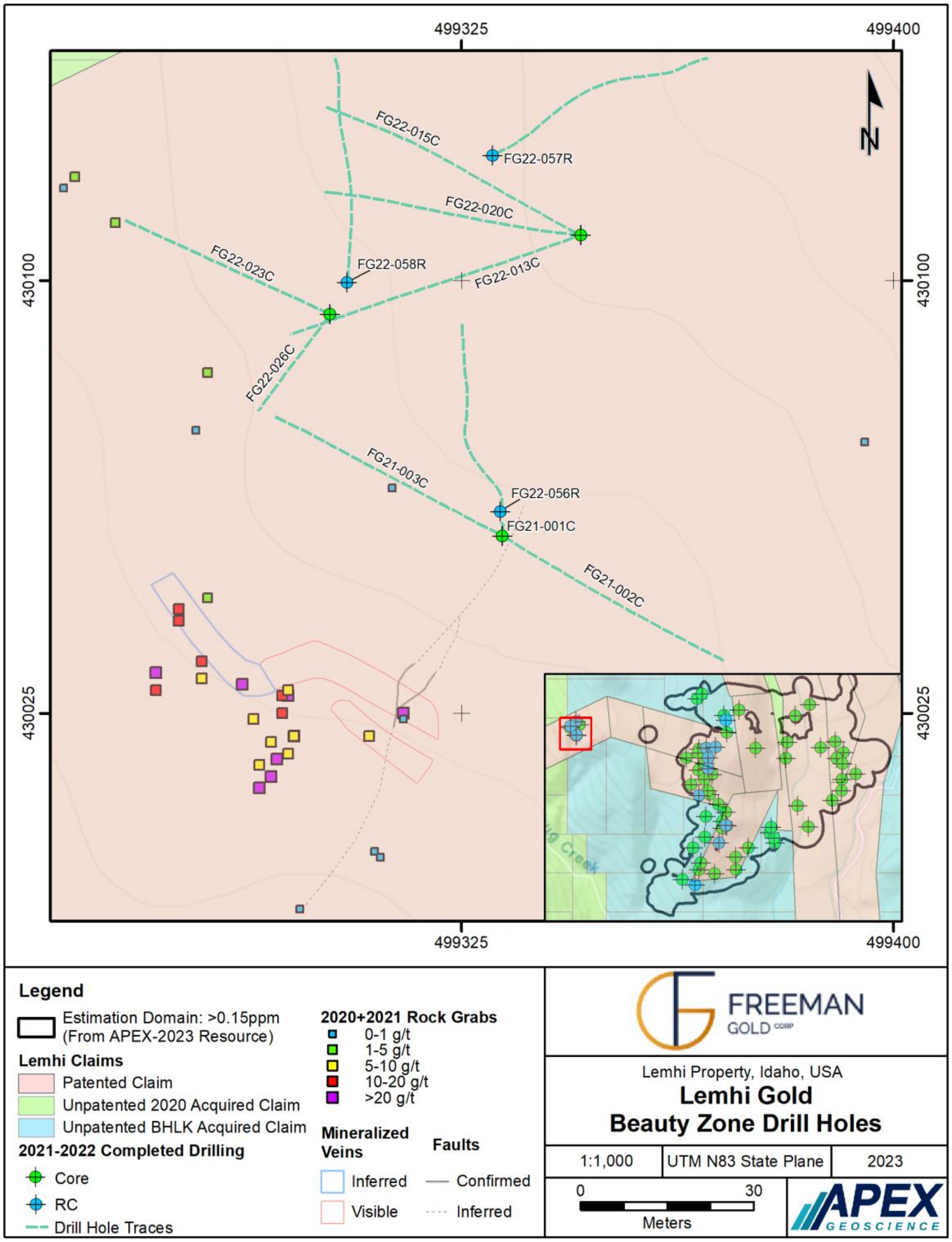
Source: APEX, 2023.

Figure 10-9: Drill Section 429625 N Highlighting Results of FG22-050C and Other 2021-2022 Anomalous Intersections



Source: APEX, 2023.

Figure 10-10: 2021 – 2022 Beauty Zone Drill Holes



Source: APEX, 2023

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Sample Collection, Preparation and Security

A total of 7,215 drill core samples, 145 rock samples, and 633 soils samples were collected during the 2020 exploration program. Of the soil samples, 291 were submitted to SGS Mineral Services – Burnaby (SGS) for MMI analysis, all remaining core, rock and soil samples were submitted to ALS Geochemistry – Vancouver (ALS). A total of 12,144 drill core samples, 1,432 RC chip samples, 394 rock samples, and 1,006 soil samples were collected during the 2021-2022 exploration program. All core, chip, rock, and soil samples were submitted to ALS.

During the 2021-2022 program no conventional soils or MMI analysis on soils were completed. All other sample preparation, analyses and security methods remained the same between the 2020 and 2021-2022 programs. As samples were collected, they were recorded within custom built Fulcrum Apps and relevant information such as sample location, geological information and photographs of the sample and site were recorded within the apps. Sample locations were recorded with a handheld GPS and input into the apps.

Soil and rock sample shipments were prepared on the property. Individual samples were placed in rice bags and sealed with tamper-proof security seals. These samples were then placed on a pallet and shipped to their respective labs via semi-truck for analysis.

Once geological logging of diamond core was completed, samples of 1 m were selected for analysis. Sample intervals were chosen so that they did not cross significant changes in lithology, alteration, or mineralization. Drill core samples were cut along cut lines drawn down the long axis of the core tube. The left half of the core was placed in its respective sample bag while the right half was placed back into the core box. Duplicate samples consisted of a quarter sample of the remaining core leaving a quartered core segment in the box. The same approach of shipping drill core samples was applied, placing several samples in rice bags and tying with tamper-proof security seals before shipping to ALS.

RC chip returns were collected into clean buckets directly from a rig-mounted cyclone at 1.52 m intervals and fed through a riffle splitter. The bulk of the sample was retained and split samples of 0.5 – 1 kg were sent for analysis to ALS in calico sample bags. RC recovery was generally poor, particularly above the water table, and the drilling struggled with frequently wet samples. Wet samples were not riffle split to avoid contamination, and analytical samples were collected via scoop with attention paid to collecting a cross-section of each interval for best sample representation. RC sample shipments were prepared on the property, where sample bags were placed into numbered rice bags and tied with tamper-proof security seals before transiting to ALS in palleted mega-bags.

11.2 Analytical Procedures

The core, RC chip, soil, and rock chip samples were assayed at ALS Global Vancouver, BC, Canada (ALS) or SGS Mineral Services Burnaby, BC, Canada (SGS), both of which are entirely independent of APEX and Freeman. ALS is certified with ISO 9001:2015 for survey/inspection activity and ISO/IEC 17025:2017 UKAS ref 4028 for laboratory analysis. SGS is also certified with ISO 9001:2015.

A subset of the soil samples were measured for MMI, which is a partial extraction proprietary method offered by SGS. MMI measures metal ions that travel upward from mineralization to unconsolidated surface materials such as soil, till, and sand. Utilizing careful sampling strategies (discussed in Section 9. 1), sophisticated chemical ligands, and ultra-sensitive instrumentation, SGS measures metals ions through a partial dissolution of the sample. Targeted elements are extracted using weak solutions of organic and inorganic compounds rather than conventional aggressive acid or cyanide-based digests. MMI solutions contain strong ligands, which detach and hold metal ions that were loosely bound to soil particles by weak atomic forces in an aqueous solution. The MMI solutions are the chemically active or 'mobile' component of the sample. Because these loosely bound complexes are in very low concentrations, measurement is by conventional ICP-MS and the latest evolution of this technology ICP-MS Dynamic Reaction CellTM (DRC IITM). The MMI complete package returns values for 53 elements.

The two other sub-sets of soil samples were IL and conventional soil geochemical methods completed by ALS. IL is a proprietary method offered by ALS similar to the MMI method offered by SGS. Ionic leach is a partial extraction technique for surface samples that relies on complexing agents to selectively extract and hold ionic species from soil, stream and organic rich sediment samples in the leachant solution. The leachant solution is introduced directly into the ICP-MS instrument. Using advanced sample introduction technology and ultra-low sub-ppb detection limits, this technique routinely achieves 'natural background' levels and enhances 'signal to noise' ratios. This helps identify often subtle, but significant responses from mineralization, geology, and alteration that can be diagnostic of numerous mineral systems. The IL complete package returns values for 61 elements.

Conventional soil geochemical analysis was completed by ALS. Analysis consisted of the preparation code PREP-41 and the analytical methods ME-MS41L and Au-AA23. The soil and sediment preparation package PREP-41 comprises drying the sample at <60°C/140°F, sieve sample to -180 µm (80 mesh) and both fractions are retained. The super trace gold and multi-element in soils and sediment method (ME-MS41L) consist of an aqua regia digestion with super trace ICP-MS finish. This method utilizes 0.5 g of sample thus gold determinations are semi-quantitative. ME-MS41L package returns values for 53 elements. Gold was determined via Au-AA23 which is a 30 g fire assay with an atomic absorption spectroscopy (AAS) finish for a 0.005 ppm lower detection limit.

Rock sample, RC chip, and drill core analysis was completed by ALS. Analysis consisted of preparation code PREP-31BY and analytical methods ME-MS41 and Au-AA24. The rock preparation package PREP-31BY is a crusher/rotary splitter combo. The sample is crushed to 70% less than 2 mm and 1 kg is rotary split off, the split is pulverized to better than 85% passing 75 µm. The method ME-MS41 is an aqua regia digestion with an ICP-MS finish. This method utilizes 0.5 g of sample, thus gold determinations are semi-quantitative. The ME-MS41 packages returns values for 51 elements. Gold was determined via Au-AA24 which is a 50 g fire assay with an AAS finish for a 0.005 ppm lower detection limit.

11.3 Quality Assurance – Quality Control (QA/QC)

During the 2020 program, a total of 7,993 drill samples were submitted for analysis. This total includes 875 QA/QC samples (10.9 %) which falls within the industry standard of at least 10% QA/QC samples for ongoing quality control and future resource work. Known standards were inserted after every 20 unknown analyses, duplicates after every 20 unknown analyses and coarse blanks were inserted after predicted high-grade intersections. Six different Certified Reference Materials (CRMs) were selected from CDN Resource Laboratories Ltd. The selected CRMs include: CDN-BL-10, CDN-CM-40, CDN-GS-6F, CDN-GS-P4J, CDN-ME-1705 and CDN-CGS-28.

Overall, the 2020 dataset shows both high precision and accuracy with only a few analyses falling outside of three standard deviations and the vast majority within two standard deviations. This further demonstrates the high reliability of ALS and validity of the 2020 core sample dataset. The 2020 core sample data is considered suitable for use in the 2023 MRE presented in this technical report. The 2020 QA/QC program is discussed in detail by Dufresne et al., 2021 and not considered material for the report.

During the 2021-2022 program a total of 13,062 drill core and 1,573 RC chip samples were submitted for analysis. This includes 1,593 QA/QC samples (12.2 %) for drill core and 188 QA/QC samples (12.0 %) for RC chip samples, which falls within the industry standard of at least 10% QA/QC samples for ongoing quality control and future resource work. Known standards were inserted after every on samples ending in 00, 10, 20, 40, 50, 70, and 80 or ~7% of analyses. Duplicates on samples ending with 30, 60, and 90 or ~3% of analyses and coarse blanks were inserted on samples ending with 25, 75 and after predicted high-grade intersections or ~2% of analyses. Eight different CRMs were selected from CDN Resource Laboratories Ltd. The selected CRMs include:

- CDN-BL-10
- CDN-CM-40
- CDN-GS-P4J
- CDN-ME-1705
- CDN-CGS-22
- CDN-CGS-27
- CDN-CM-44
- CDN-ME-2104.

Re-assays were completed when certified reference materials plotted outside three standard deviation (SD), pulp blanks >2*LOD or coarse blank >3*LOD. Five natural samples on either side of the failed standard were re-assayed. A total of 25 CRMs from 19 drill holes failed. The database utilized for Section 11 includes all the re-assays and is the final 2021-2022 assay database used in the MRE in this report.

11.3.1 Coarse and Pulp Banks

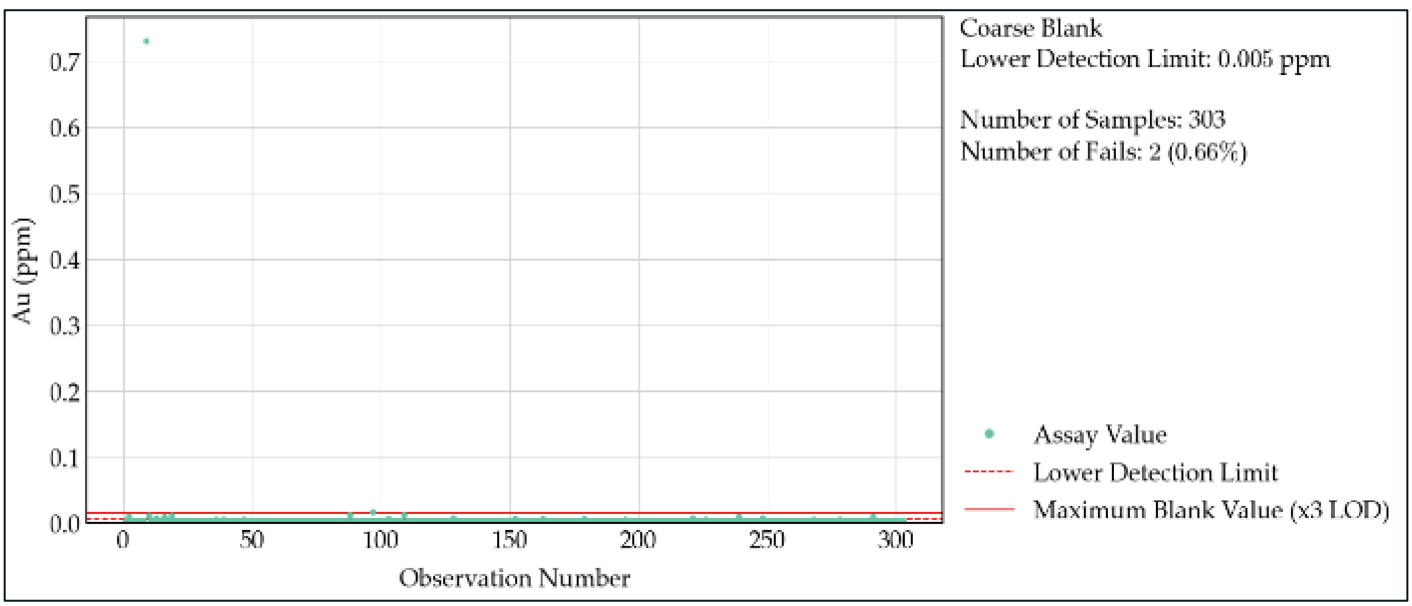
Coarse blanks were inserted regularly at sample numbers ending in 25 or 75 and by the logging geologist after predicted high-grade samples or zones. For the core samples, a total of two coarse blank failures of the 303 samples submitted, while the RC samples contained zero failures of the 31 samples, shown in Figure 11-1 and Figure 11-2, respectively. The two coarse blank failures and the preceding high-grade samples are displayed in Table 11-1. The carryover amount of gold during crushing is calculated based on a barren material with an equivalent weight as the high-grade sample. Contamination of the preparation facility through carryover from a high-grade sample processed at the same station is a known risk in the industry. Both the blank samples yielded calculated carryover of less than 0.1%. One of the failures returned 0.016 ppm Au, this is well below any potential economic threshold for a mineral resource or mining, therefore is not considered material. The other failure reported a grade of 0.72 ppm Au. This sample is above a lower cut-off grade for a resource. However, it was prepped after a sample from the Beauty zone containing 441ppm sample and has less than 0.1 % carryover. ALS was notified for all coarse blank failures and five samples on either side were re-assayed and returned similar results. ALS recognizes this risk and actively mitigates it by conducting actions listed below:

- Carrying out the crushing and pulverizing in custom-made plenum with sufficient ventilation to remove and reduce the dust generated in operation
- Utilizing pre-tested barren materials to clean the equipment between two batches of samples or more frequently as needed
- Implementing vigorous standard operating procedures (SOPs), which require a thorough cleaning of the sample preparation equipment using compressed air before processing each sample.

With the above measures, ALS is confident that the carryover, if any, will not exceed the target of 1% of the previous sample processed at the same station. Due to the carryover <1%, these blank failures are not statistically significant and do not pose any concern in confidence in the lab. Another potential source of uncertainty is the lower volume of material used for each of the coarse blank samples (~500 g of blank). Given these factors and ALS' mitigating procedures, coarse blank failures of under 20% are deemed acceptable. However, ALS was made aware of these failures and both of these samples were selected for re-assay and returned similar results.

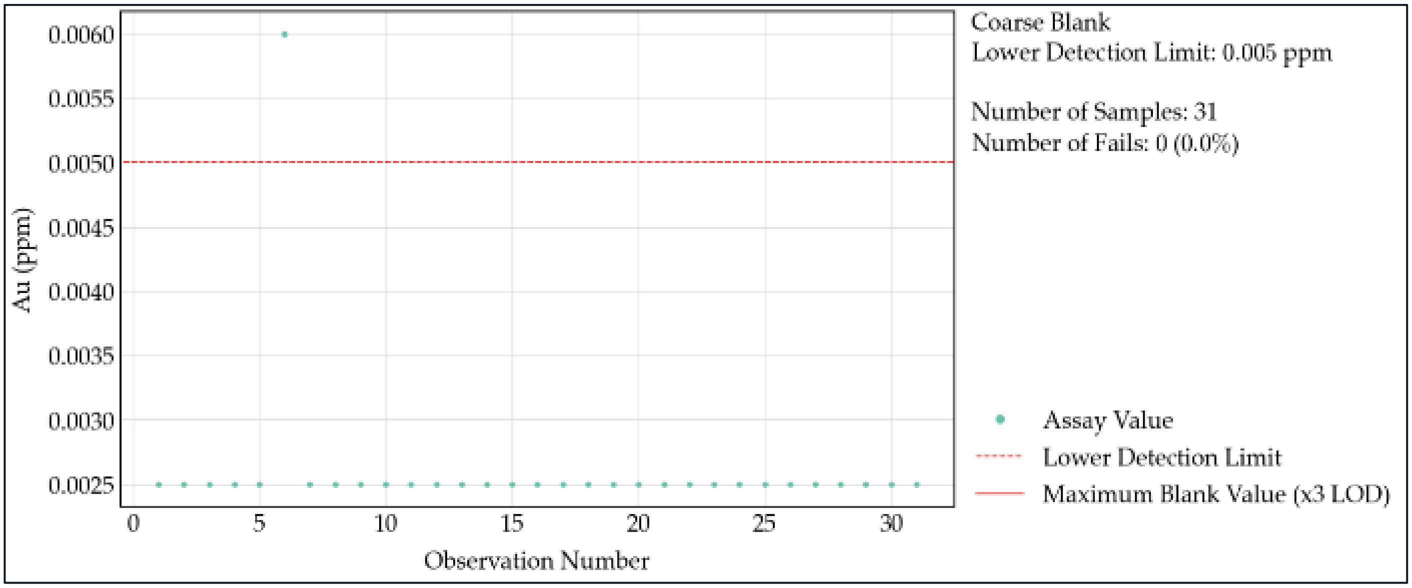
The pulp blank, CDN-BL-10, had no sample failures in either drill core or RC chips, shown in Figure 11-3 and Figure 11-4, respectively. Of the passed pulp blanks 22 of the total 294 (7. 5%) returned assays greater than LOD.

Figure 11-1: Drill Core Coarse Blank Au Concentration



Source: APEX, 2023.

Figure 11-2: RC Chips Coarse Blank Au Concentration Results



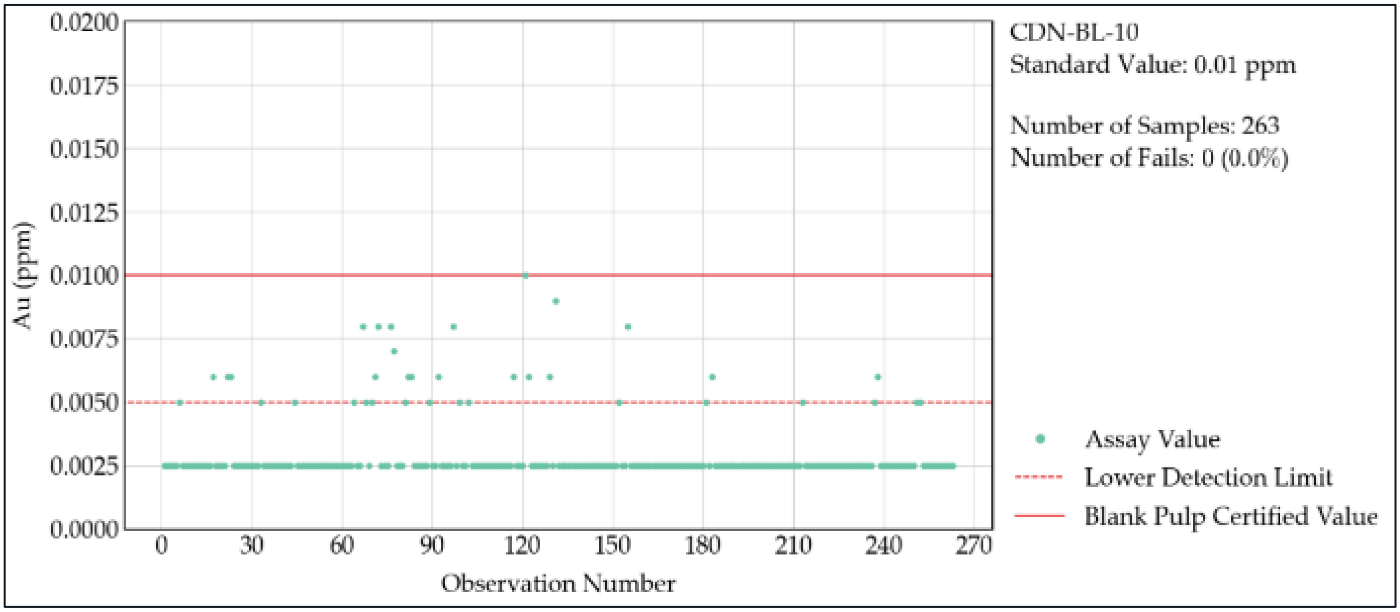
Source: APEX, 2023.

Table 11-1: Failed Coarse Blank Analysis with the Previous High-grade Sample

Certificate	Barren Material			High-Grade Sample			Carryover*
	Sample ID	Sample Weight (kg)	Au Result (ppm)	Sample ID	Sample Weight (kg)	Au Result (ppm)	
VA21351916	D240205	0.72	0.73	D240204	1.32	441	0.090%
VA22154258	G195260	0 48	0.016	G195259	1.62	6.56	0.077%

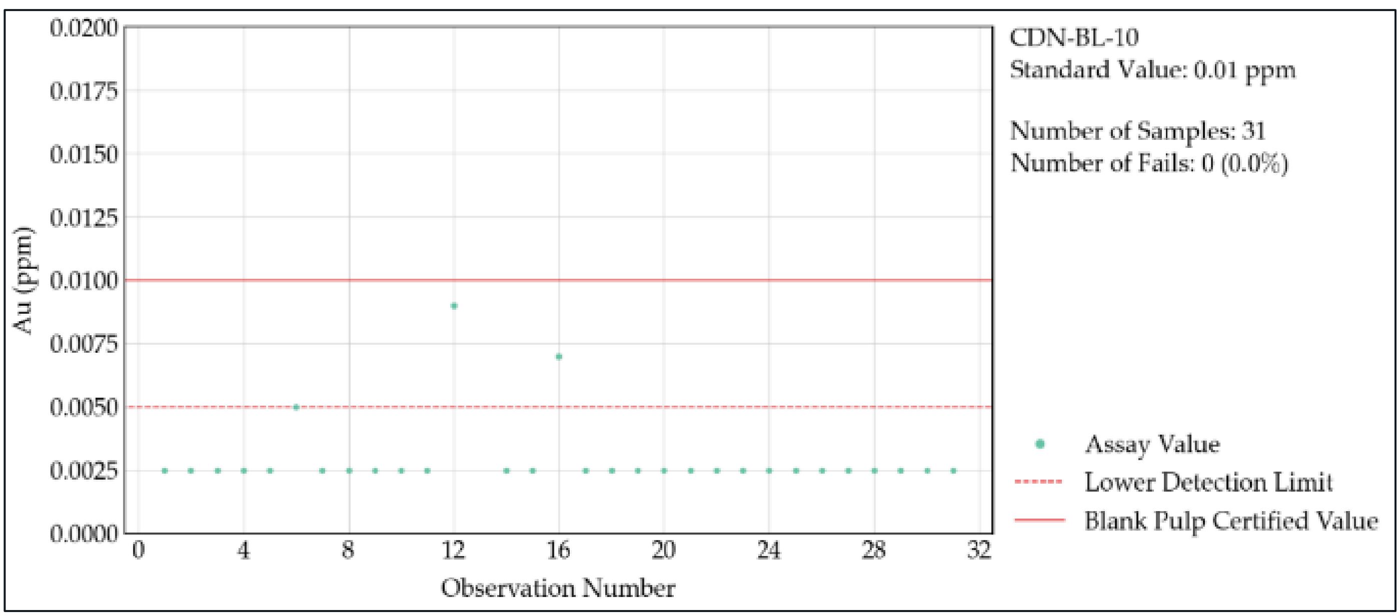
*Carryover % = (Blank Au result * Blank Sample weight)/ (High-grade Au result * High-grade sample weight) * 100%.

Figure 11-3: Drill Core Pulp Blank CDN-BL-10 Au Concentrations



Source: APEX, 2023.

Figure 11-4: RC Chips Pulp Blank CDN-BL-10 Au Concentrations



Source: APEX, 2023.

11.3.2 Certified Reference Materials

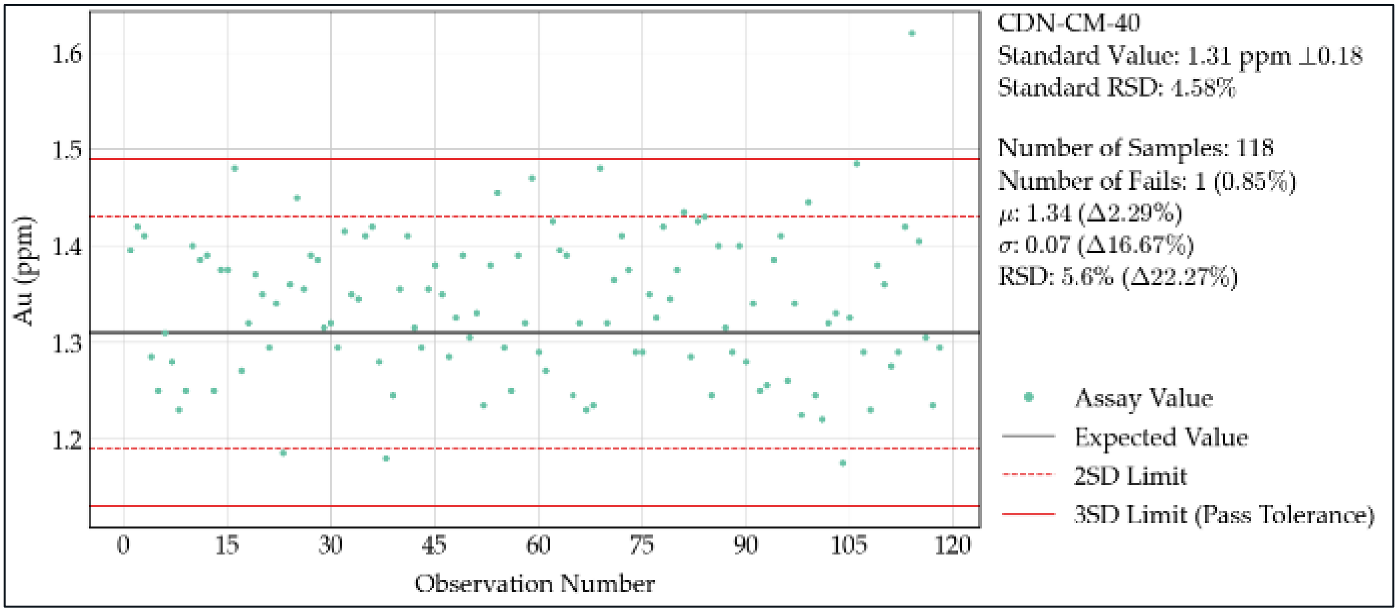
Gold standards prior to the re-assays had 17 failures of a total 782 assays (2.2% failure rate). Each CRM had the following number of failures:

- CDN-CM-40 (6 failures)
- CDN-GS-P4J (3 failures)
- CDN-ME-1705 (3 failures)
- CDN-CGS-22 (0 failures)
- CDN-CGS-27 (3 failures)
- CDN-CM-44 (0 failure-core and 1 failure-RC)
- CDN-ME-2104 (1 failure-core and 0 failure-RC).

Gold standards plotted against gold are displayed below: CDN-CM-40 (Figure 11-5), CDN-GS-P4J (Figure 11-6), CDN-ME-1705 (Figure 11-7), CDN-CGS-22 (Figure 11-8), CDN-CGS-27 (Figure 11-9), CDN-CM-44 (Figure 11-10 and Figure 11-11) and CDN-ME-2104 (Figure 11-12 and Figure 11-13). Three CRMs plot outside of three standard deviations (3SD) of the expected value. Each of the failed CRM in CDN-CM-40, CDN-GS-P4J and CDN-CGS-27 were selected for re-assay but either had already been re-assayed by ALS internal QAQC due to fusion issues or the re-assay failed. These three failures occur around in natural samples that are low grade and are considered not material for this report.

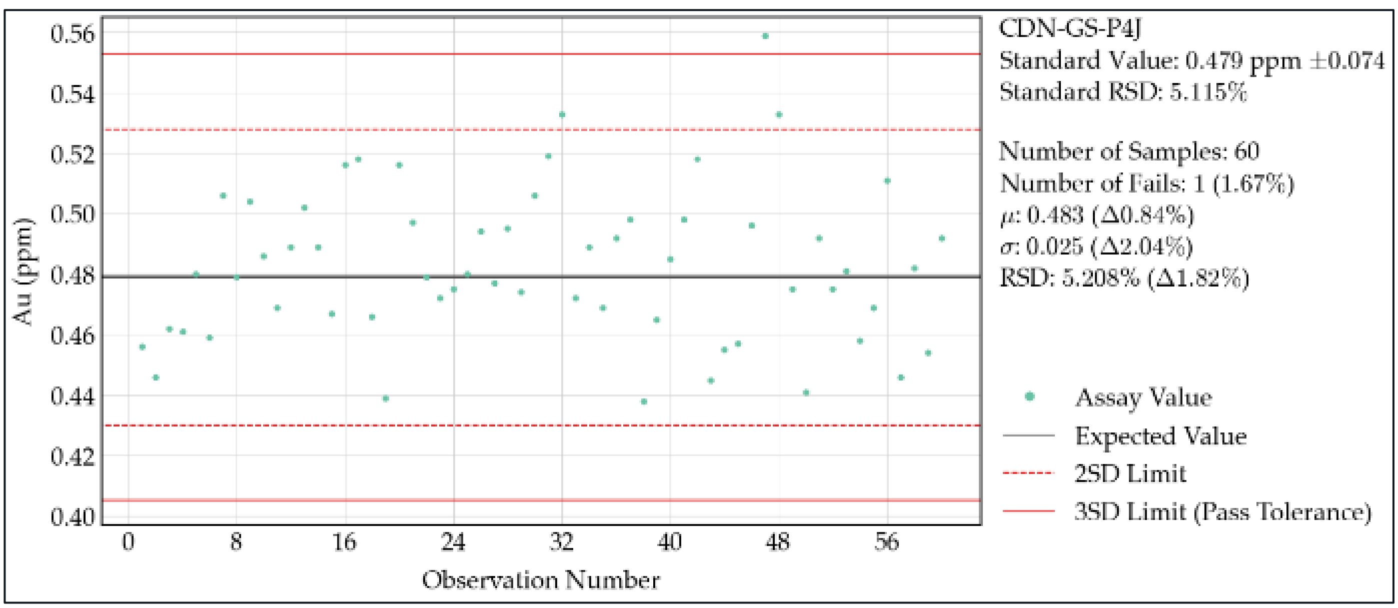
CDN-CM-40 (Figure 11-5) shows adequate accuracy with a mean slightly higher than expected; precision shows more variability than expected. CDN-GS-P4J (Figure 11-6) performed as expected with adequate accuracy and precision. CDN-ME-1705 (Figure 11-7) performed as expected adequate high accuracy and precision. CDN-CGS-22 (Figure 11-8) shows moderate accuracy with a mean higher than expected; precision would be high but two low outliers make it as expected. CDN-CGS-27 (Figure 11-9) performed as expected with adequate accuracy and precision. CDN-CM-44 (Figure 11-10 and Figure 11-11) both datasets show shows moderate accuracy with a mean higher than expected; precision is as expected. CDN-ME-2104 (Figure 11-12 and Figure 11-13) both datasets show adequate accuracy with means slightly lower than expected; precision is higher than expected on core and as expected for RC chips.

Figure 11-5: CDN-CM-40 Au Concentration Results



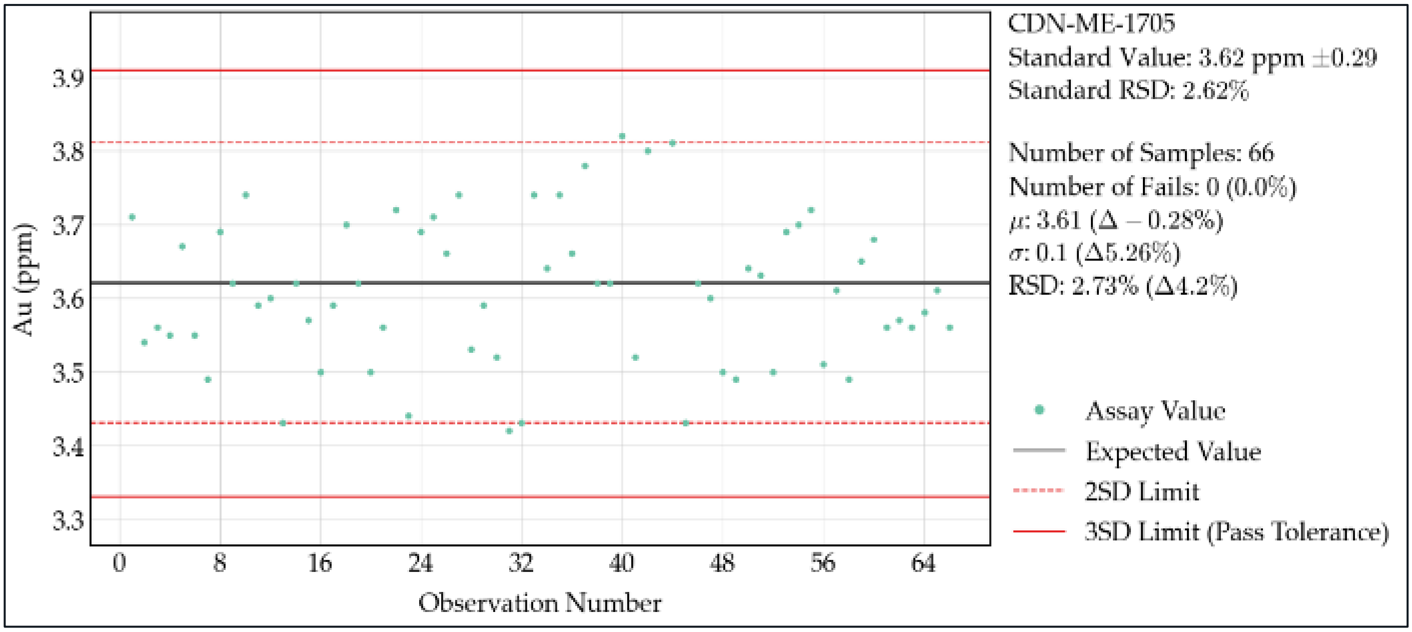
Source: APEX, 2023.

Figure 11-6: CDN-GS-P4J Au Concentration Results



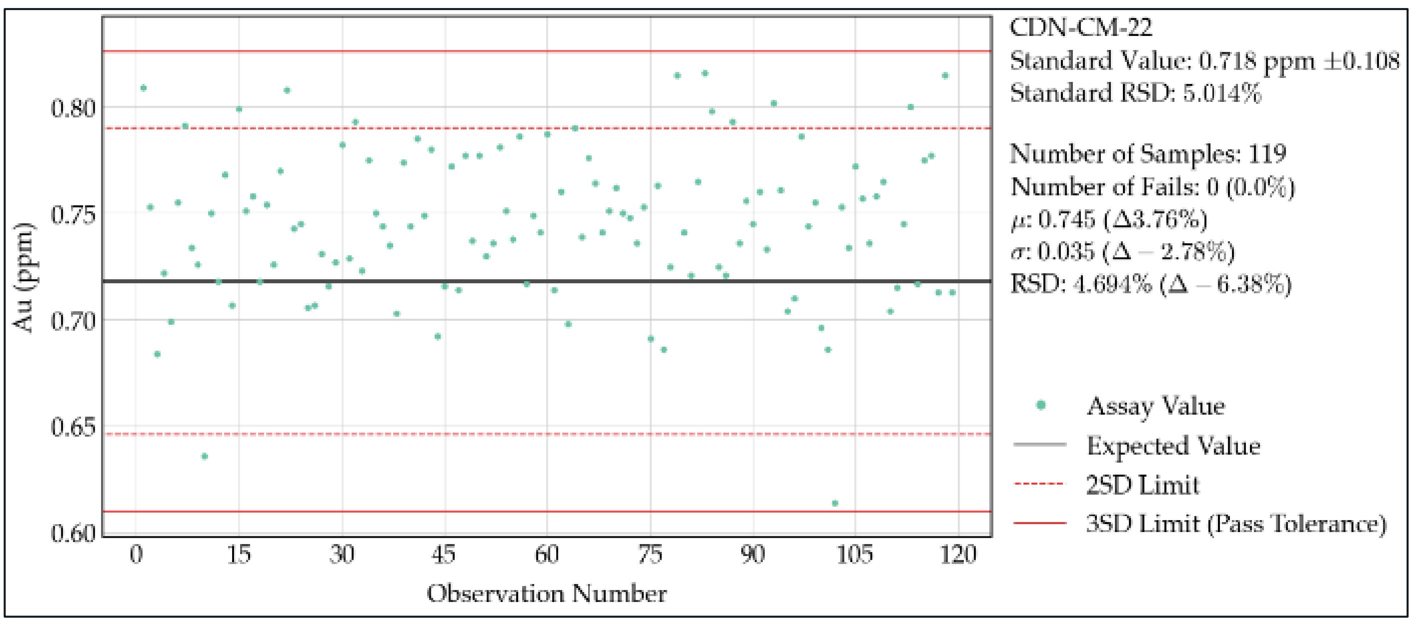
Source: APEX, 2023.

Figure 11-7: CDN-ME-1705 Au Concentration Results



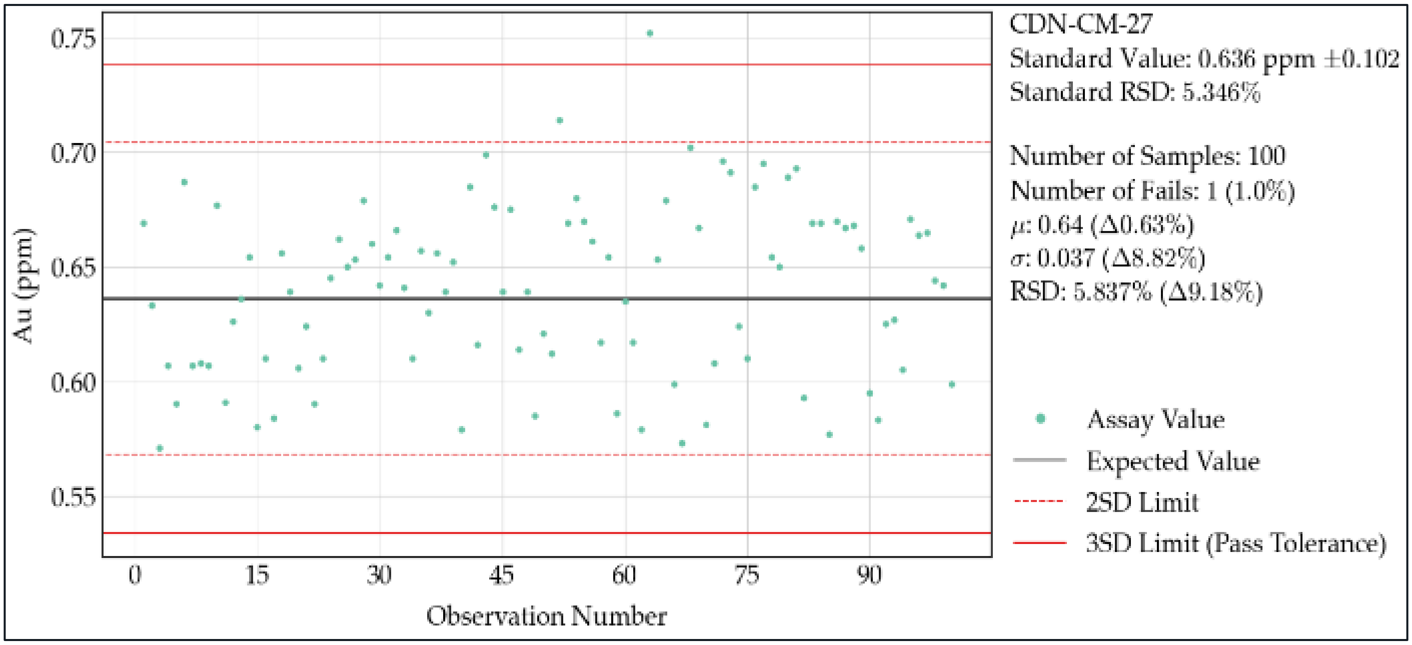
Source: APEX, 2023.

Figure 11-8: CDN-CGS-22 Au Concentration Results



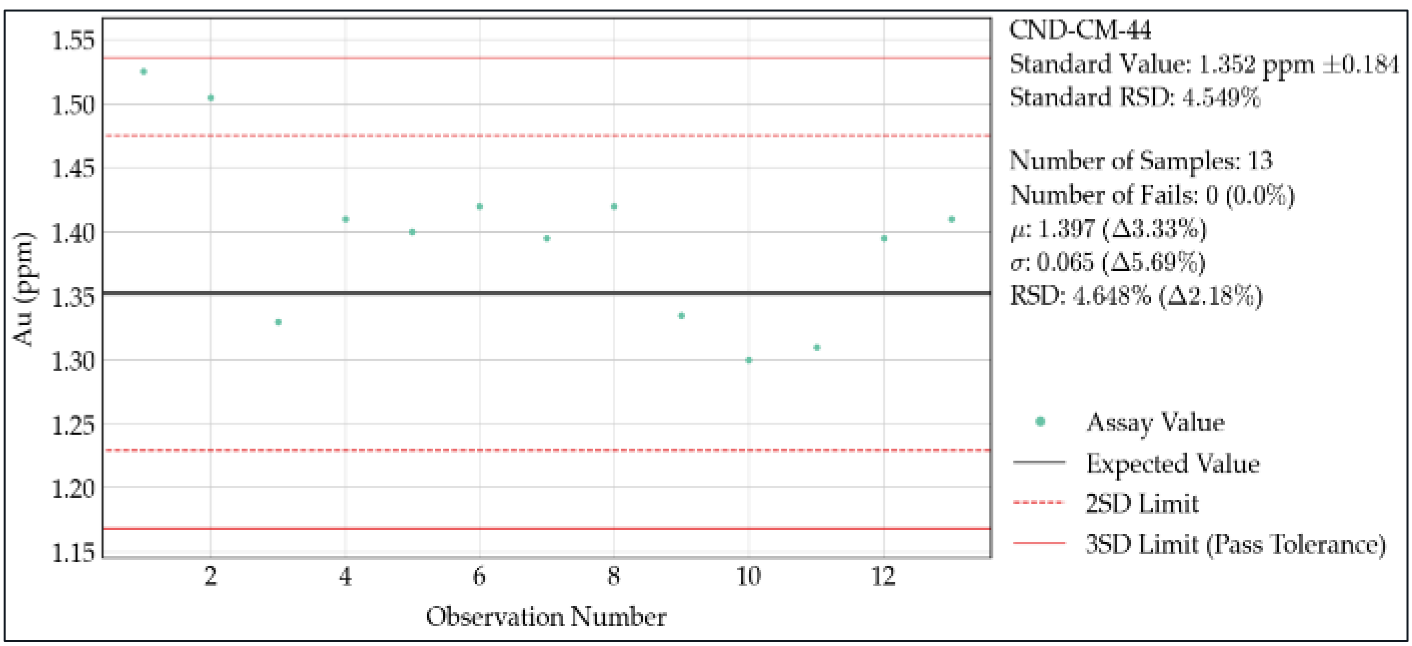
Source: APEX, 2023.

Figure 11-9: CDN-CGS-27 Au Concentration Results



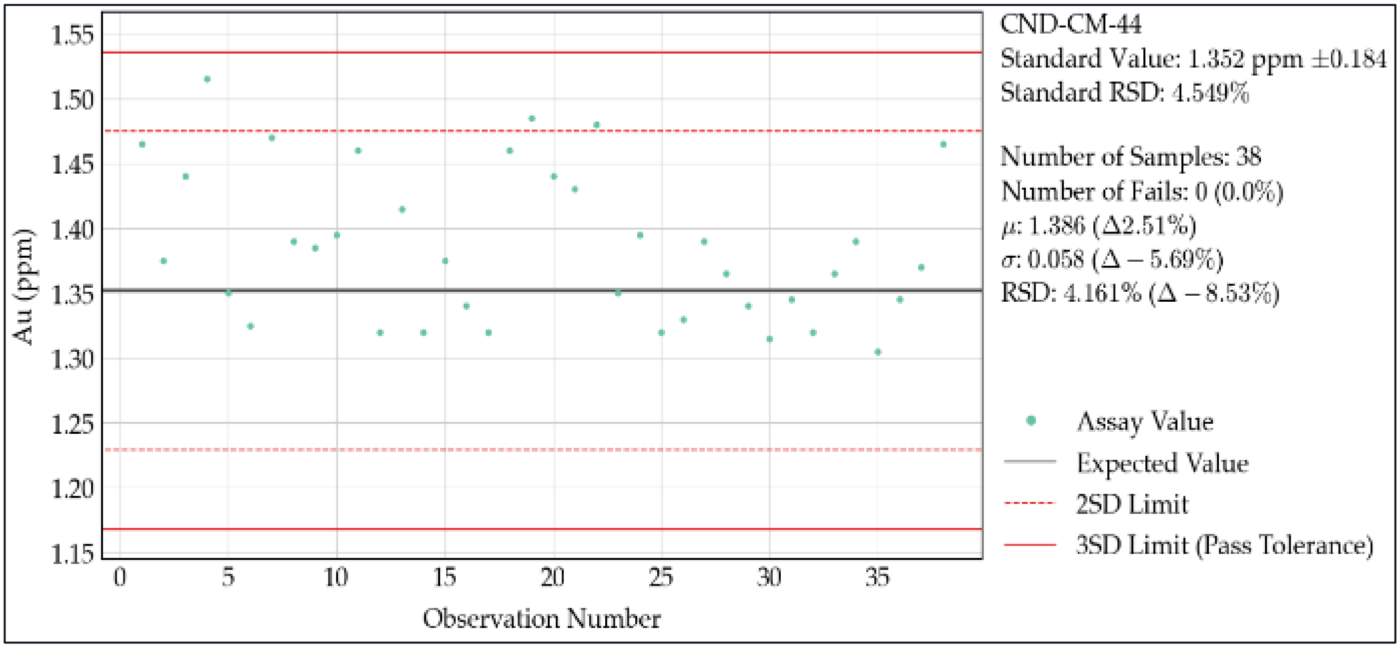
Source: APEX, 2023.

Figure 11-10: CDN-CGS-44 Au Concentration in Core Results



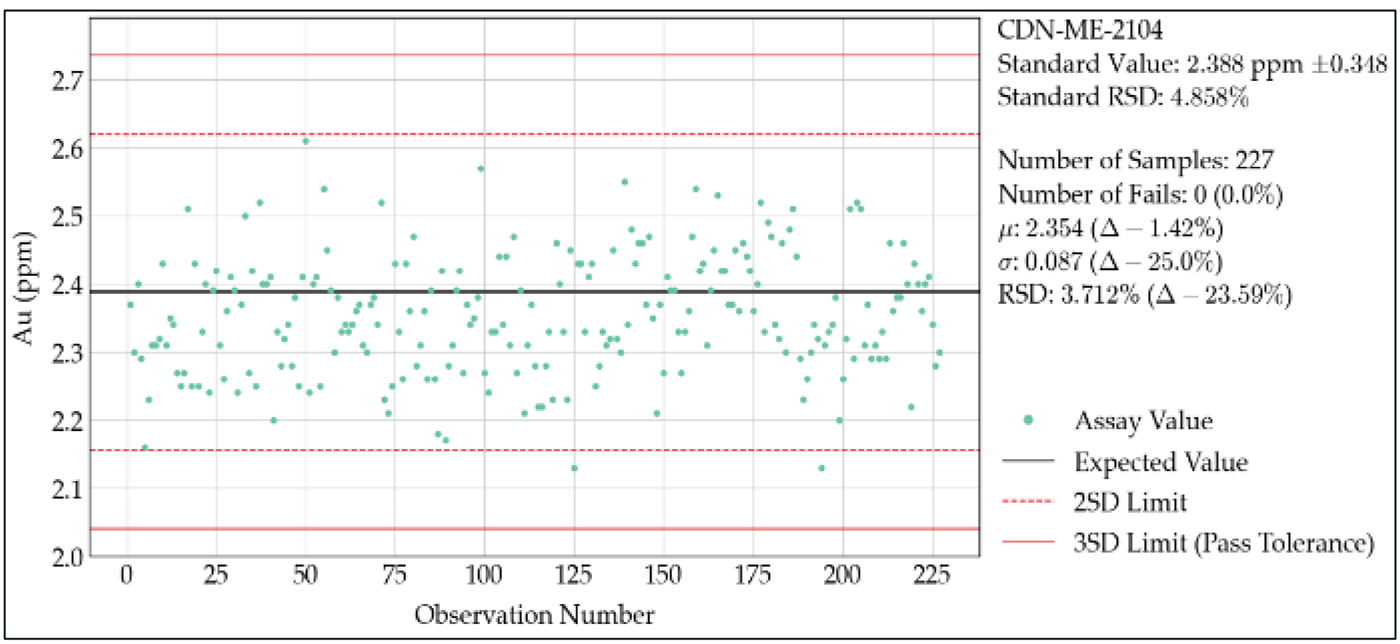
Source: APEX, 2023.

Figure 11-11: CDN-CGS-44 Au Concentration in RC Chips Results



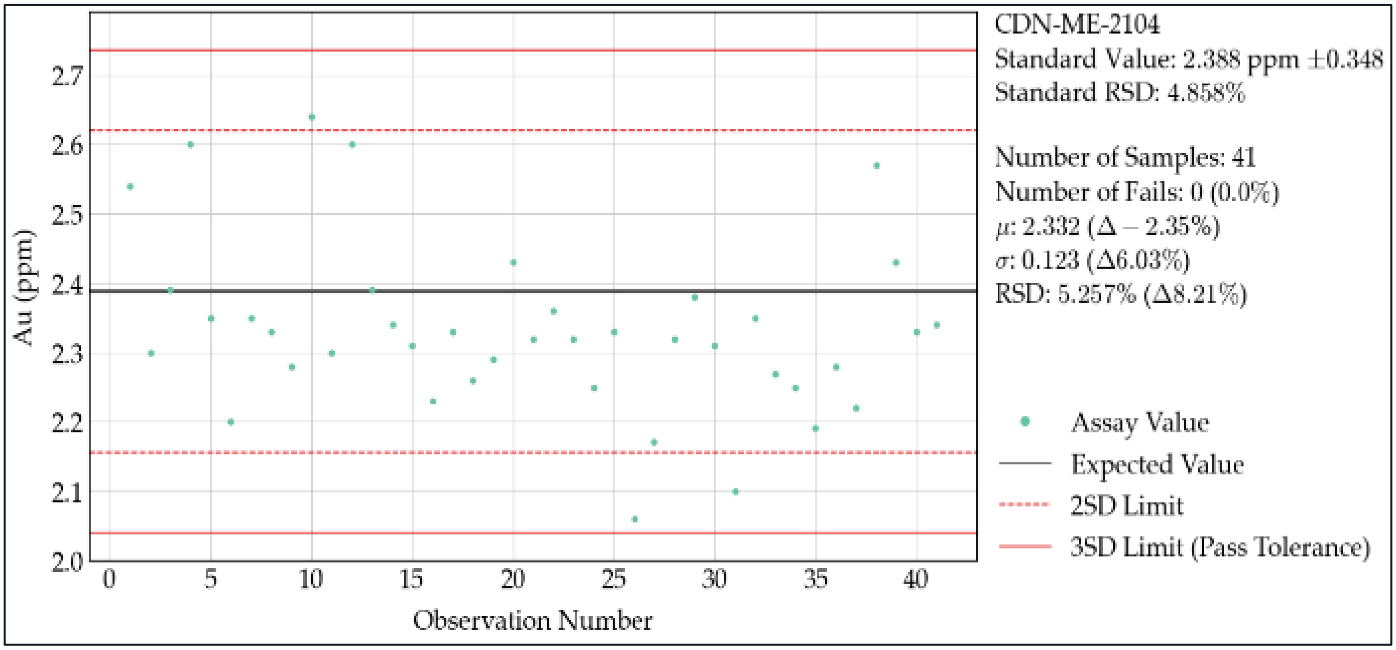
Source: APEX, 2023.

Figure 11-12: CDN-ME-2104 Au Concentration in Core Results



Source: APEX, 2023.

Figure 11-13: CDN-ME-2104 Au Concentration in RC Chips Results



Source: APEX, 2023.

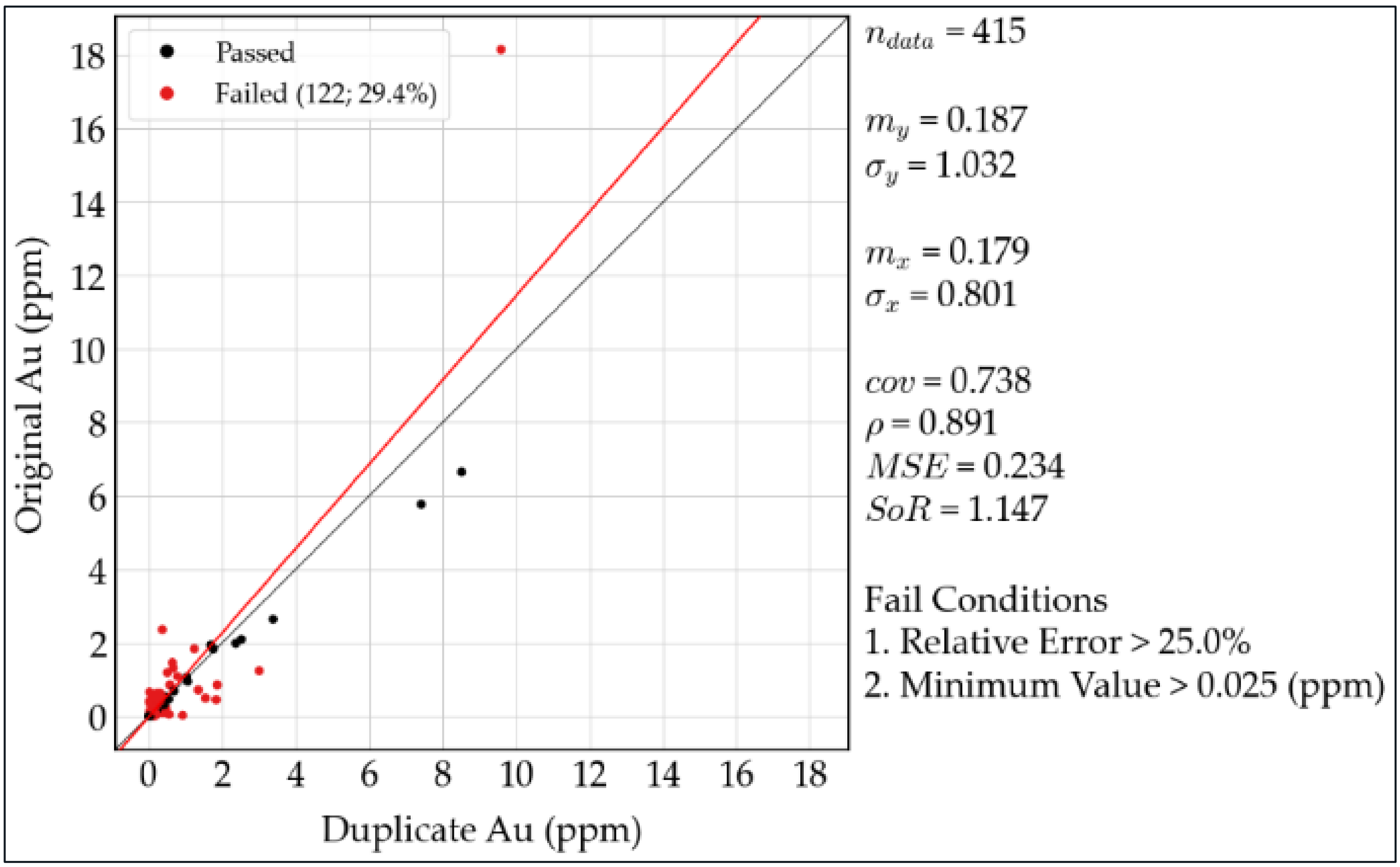
Overall, the dataset shows both high accuracy and precision with only a few analyses falling outside of three standard deviations and the vast majority within two standard deviations. This further demonstrates the high degree of confidence placed in the reliability of ALS and validity of the 2021-2022 core sample dataset. The 2021-2022 core sample data is considered suitable for use in the 2023 MRE presented in this technical report.

11.3.3 Duplicates

Three core duplicates were collected for every 100 samples. Regular and parent samples were obtained by cutting the core in half, with one half going to the lab, and the other half returning to the core box. Duplicate samples were collected from the same sampling interval as the parent, where the half core in the box was quartered, with one quarter going to the lab as a core duplicate and the remaining quarter returned to the box. RC duplicates were taken similar to core with three duplicated collected every 100 samples. As regular samples the parent was taken in the same matter. Once the sample was passed through the riffle splitter, 0.5-1 kg of the split portion was placed in the parent bag. The remaining split portion of the bag was placed in the duplicate bag. If insufficient material remained in the split portion a transect of the retention bag was taken with a hand-held scoop to fill the duplicate bag.

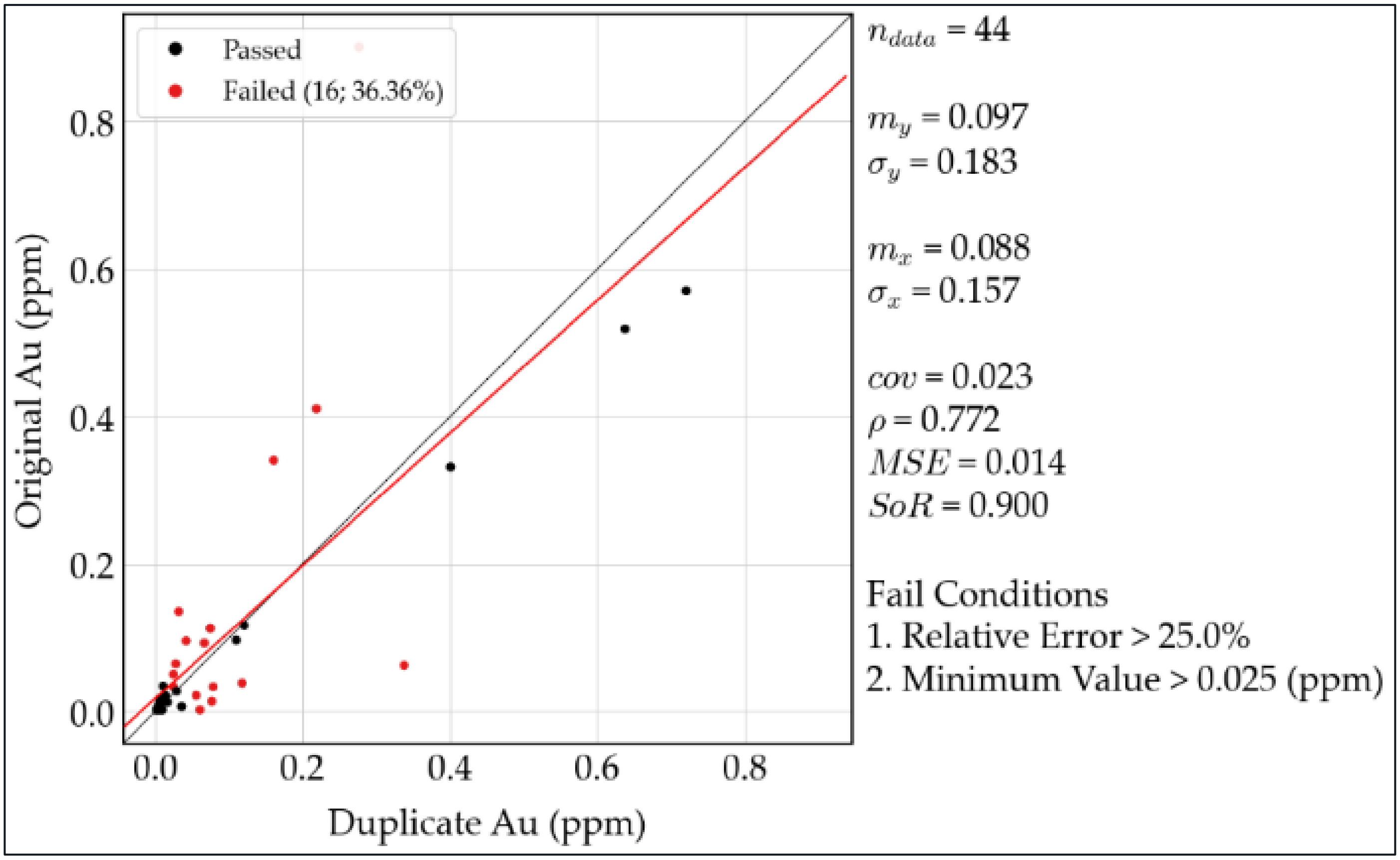
Duplicate analysis results to date are considered poor, with 25 of 29 samples >0.025 ppm Au yielding a >25% difference in grade. Several factors are considered to account for this poor reproducibility. The volume of rock in the duplicate submitted is half of the parent sample which further exacerbates the potential distribution of coarse gold and blebs of sulfide in veins, known as the 'nugget' effect. Historically, this discrepancy has been observed in the drill hole 'twinning' programs conducted at Lemhi, discussed in Section 6, which is attributed to the uneven distribution of veins across the deposit. Although grade reproducibility has been a challenge in historical twinning programs, horizons or zones of high-grade values are consistent. This feature of the Lemhi Gold Deposit is observed on a smaller scale in the duplicate samples shown in Figure 11-14 (core) and Figure 11-15 (RC). Duplicate sampling procedure could also contribute to the discrepancy between parent and duplicate sample returns. Due to the fissile nature of the host rock, cutting the drill core proved to be challenging. Although extreme care was taken to ensure all core pieces returned to the box or sample bag, the fissile rock would often strongly fragment during cutting which may have resulted in a non-uniform volume of rock split between the duplicate and parent samples, and the remaining material in the core box. These factors could be further skewing the duplicate data but should not show any kind of systematic bias.

Figure 11-14: Drill Core Duplicate Analyses, Original (Half Core) and Duplicate (Quartered Core)



Source: APEX, 2023.

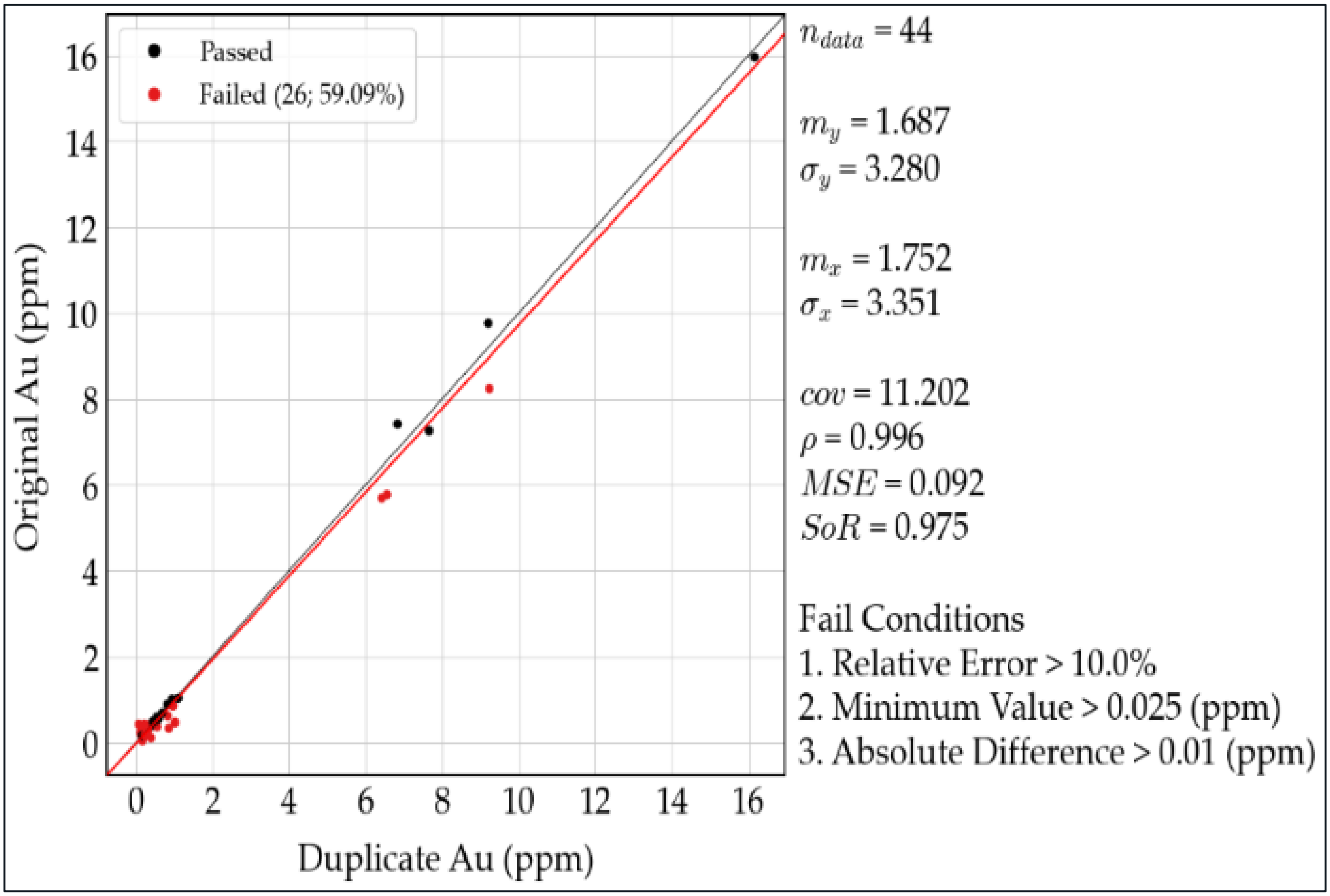
Figure 11-15: RC Sample Splits Duplicate Analyses, Original (First Split) and Duplicate (Second Split)



Source: APEX, 2023.

Each of the failed CRM's along with five natural samples above and below the failed standards were selected for re-assay. The original values along with the re-assays, with at least one assay above the low-grade cut-off (0.15 ppm), are displayed by Figure 11-16. The re-assays of the natural sample suites associated with a failed standard returned both higher and lower values with no trends observed in the dataset. Greater scatter is observed in samples with lower concentrations while samples >0.5 ppm plot tighter to the 1:1 reference line. These pulp re-assays display a similar trend to the pulp re-assays completed by Dufresne (2019) and by Freeman (Dufresne et al., 2021). All the re-runs plotted (>0.15 ppm) had an average increase of 13.0% in samples. Three samples have greater than 100% change with originals and re-assays being 0.071 ppm to 0.429 ppm (504 2% change), 0.115 ppm to 0.236 ppm (105.2% change) and 0.126 ppm to 0.27 ppm (114.3% change). Removing these three outliers the average of the assays changed by -3.7%. The maximum and minimum percent difference in natural samples was 97.7% and -76.5%, respectively. The sample re-assays were issued on corrected certificates.

Figure 11-16: Re-assayed Samples (Pulp Duplicates) Near Failed QA/QC Samples



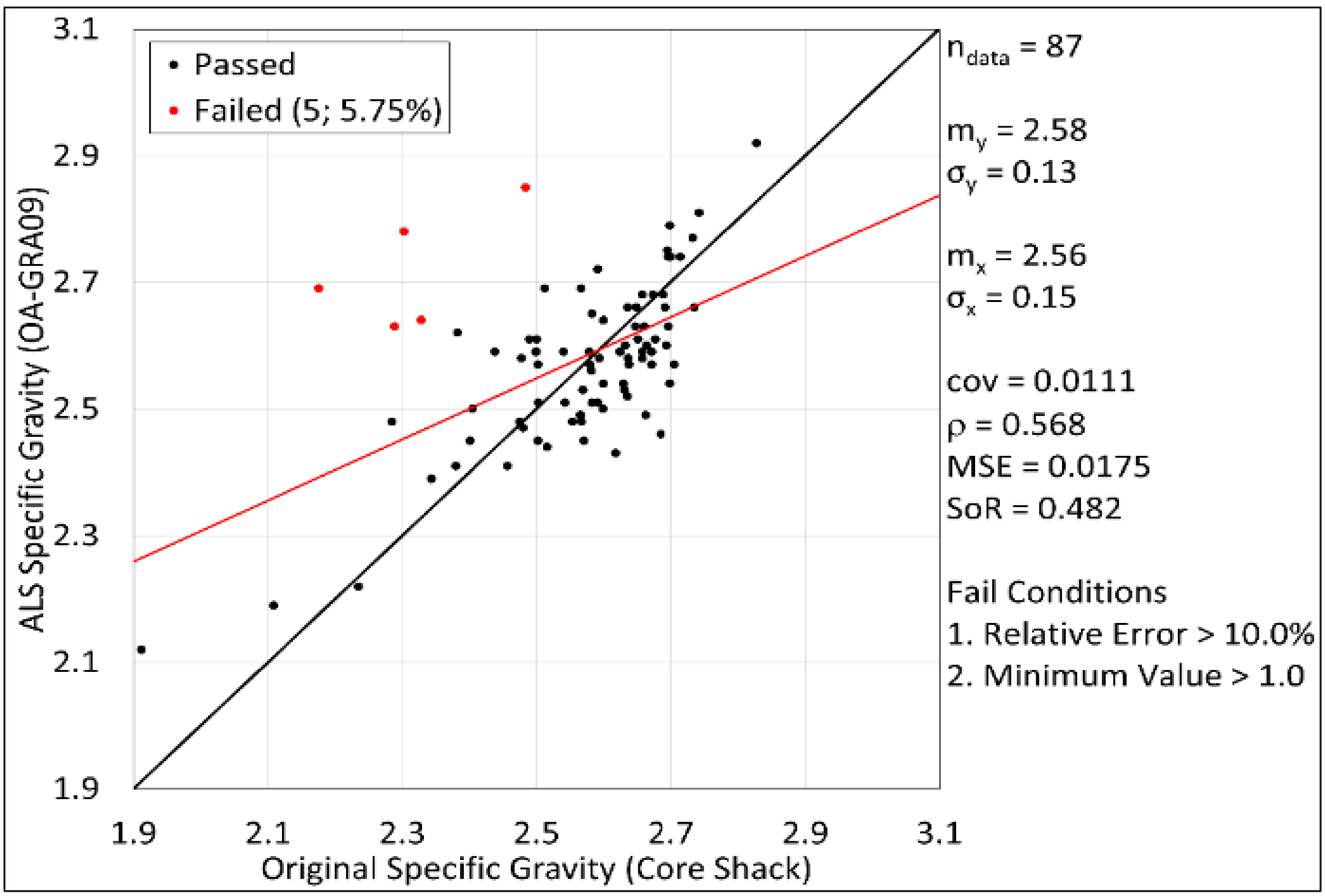
Source: APEX, 2023.

11.3.4 Specific Gravity

Specific gravity measurements (SG), collected by geologists in the core shack, had their own QA/QC protocol. An SG measurement was taken within every sample interval (~1 m samples) in all 2020, 2021 and up to an including FG22-027C except for FG22-024C in 2022. Holes following FG22-027C, including FG22-024C, had an SG measurement completed on samples ending in 4 or 9 (~4-5 m intervals). In 2020, a total of 6,578 SG measurements were taken and in 2021-2022 a total of 7,766 SG measurements were taken. In the core shack, a geologist selected a sample of core ~10 cm in length from within each sample interval. The sample was weighed dry then weighed a second time in a wire basket suspended in a bucket of water. From these measurements, the SG was calculated (dry weight)/(dry weight - wet weight). A subset of SG measurement samples (87) was sent to ALS geochemistry to undergo OA-GRA09 (bulk density by water displacement) and OA-GRA09A (bulk density after wax coating). A total of 87 samples was measured by OA-GRA09 and 26 samples by OA-GRA09A. These results are displayed in Figure 11-17, Figure 11-18, and Figure 11-19.

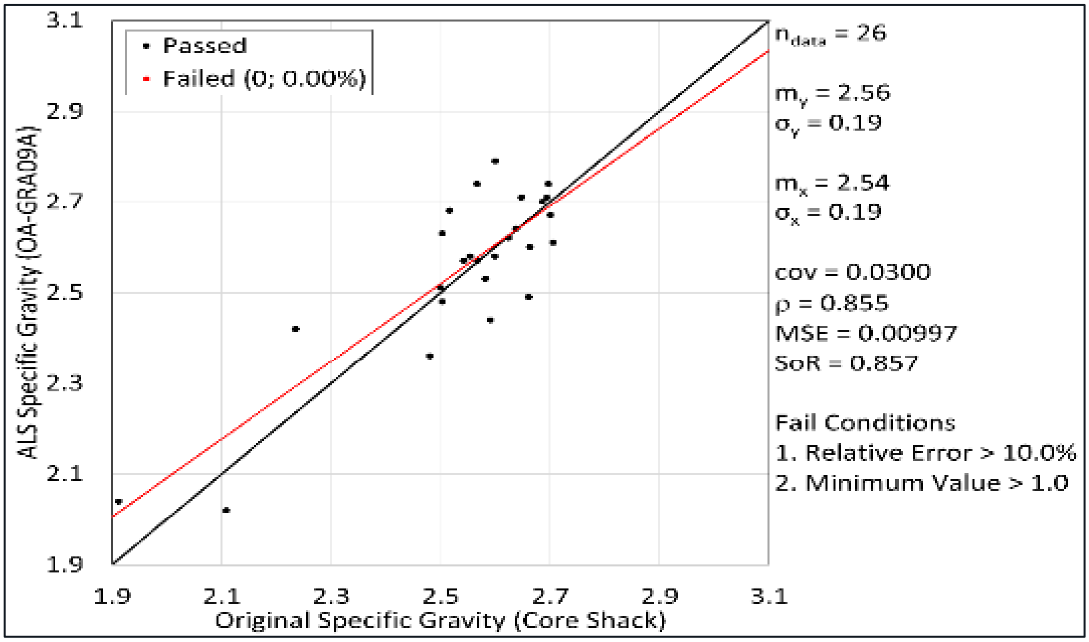
The core shack SGs and OA-GRA09 display similar results with samples plotting both higher and lower than the 1:1 ratio showing little to no bias in the dataset. OA-GRA09 results display lower variability (standard deviation) than the core shack measurements (Figure 11-17). Five (5. 75%) of the core shack versus OA-GRA09 measurements have >10% change with and average relative error of -14. 77%. The average relative change of the entire core shack and OA-GRA09 dataset is -0.60% When comparing the core shack measurement and OA-GRA09 to wax coated OA-GRA09A (Figure 11-18 and Figure 11-19, respectively) these show similar trends to one another. The average relative error between core shack versus OA-GRA09A and OA-GRA09 versus OA-GRA09A are -0.46% and -1.43%, respectively. Figure 11-17, Figure 11-18 and Figure 11-19 appear to have a shotgun distribution; however, this is due to the relatively small variation in SGs record at Lemhi. Each of the three comparisons have low change in average relative errors (-0.60%, 0.46% and 1.43%) demonstrating each SG method compares well with the other; thus, all core shack measurements are considered acceptable for use in the MRE.

Figure 11-17: Specific Gravity QA/QC. Core Shack Measurement as a function of ALS Measurement (OA-GRA09).



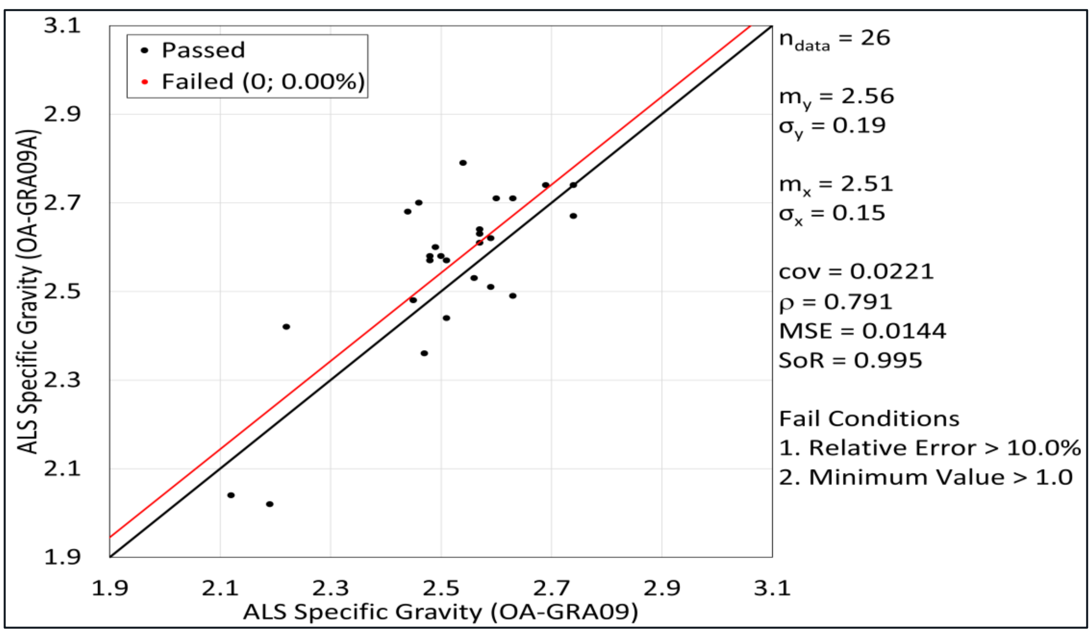
Source: APEX, 2023.

Figure 11-18: Specific Gravity QA/QC. Core Shack Measurement as a Function of ALS Wax Coating Measurement (OA-GRA09A)



Source: APEX, 2023.

Figure 11-19: Specific Gravity QA/QC. ALS Measurement (OA-GRA09) as a Function of ALS Wax Coating Measurement (OA-GRA09A).



Source: APEX, 2023.

11.3.5 Soil and Rock Samples

Both the soil orientation and rock sample programs had their own QA/QC protocols. The 2021 rock grab sample program had a total of 444 rock grabs, which included 29 CRMs, 9 blanks, and 14 duplicate samples (11.7% QA/QC). The standard suite consisted of the same CDN standards utilized in the drilling program. All the CRMs plotted within 3SD and two of the nine coarse blanks contained >0.015 ppm Au. The two failed coarse blanks plot come after high-grade samples and are not considered material for the validity of the rock sample database.

The total rock database (392 natural samples) contains significant high-grade material with 89 (22 %) > 1ppm and 124 (32 %) > 0.015ppm. The majority of the duplicate samples were at low grade but a similar variation in duplicates is observed in both core and rock grabs. The soil samples QA/QC consisted of duplicates and blanks. Of the 56 duplicate soils taken ten samples contain a significant increase (>100%) in their result. The remaining 46 duplicates show both higher and lower values with an average change of -3 %. All of the outliers in the soil duplicate dataset return higher results than the original value. Eight of the ten outliers contained less than <0.5 ppb Au and are considered not significant. The two others went from 0.26ppb to 3.41ppb and 0.13ppb to 1.32ppb. This is likely due to the first sample being taken at a shallower depth (~25cm) while the duplicate was taken deeper.

The orientation survey identified the optimal depth for an IL soil sample to be ~30cm. Of the 53 blank samples, five samples returned higher than 3x the method detection limit (MDL). Both failures were in the IL method (MDL = 0.02 ppb). Four of the five failures contain <0.5 ppb Au and are considered statistically insignificant. Silica sand material was sourced from a local hardware store. This remaining failure contained 2.31 ppb Au and could be attributed to the sand being derived from a gold-bearing source or possible contamination in the field or at the lab. This one outlier in the blank sand dataset is considered statistically insignificant.

11.4 Adequacy of Sample Preparation, Security and Analytical Procedures

Based upon the review of the prior and recent sample methodologies for surface sampling, RC and core sampling, the observations with respect to the QA/QC data and the security measures along with chain of custody for the recent work the Authors accept the data and deem it suitable for the purposes used herein.

12 DATA VERIFICATION

Mr. Michael Dufresne, M. Sc., P. Geol., P. Geo. made a site visit from September 10-17, 2020, in which he confirmed the locations of several historical collars on the property in preparation for drilling and observed core from the first two holes completed in 2020. Mr. Dufresne also conducted a visit to Freeman's core facility on February 26, 2021, and reviewed a number of the 2020 core intersections with significant values of gold. The most recent site visit was conducted on February 18, 2022, where Mr. Dufresne visited two active drill pads, viewed core being quick-logged by the on-site geologist, and visited the Beauty Zone mineral occurrence showing and several Beauty drill pads.

Freeman provided APEX with the Lemhi Gold Property's drill hole database (DHDB) consisting of analytical, geological, density, collar survey and downhole survey information. APEX personnel compiled a DHDB containing the historical and 2020 drilling databases and incorporated the new 2021-2022 drilling by Freeman. The database includes collar, downhole survey, assay, geology, structural, and geotechnical data. The 2023 DHDB was validated 100% by APEX personnel and the validation work consisted of:

- Updates to historical collar metadata including company, drill dates, assay certificates, downhole survey types etc.
- Incorporation of additional historical geological drill logs as needed
- Normalization of historical geology logs to the current geological model
- Compilation and validation of the current geological logs for incorporation into the database
- Rectification of any problems with the survey and collar files.

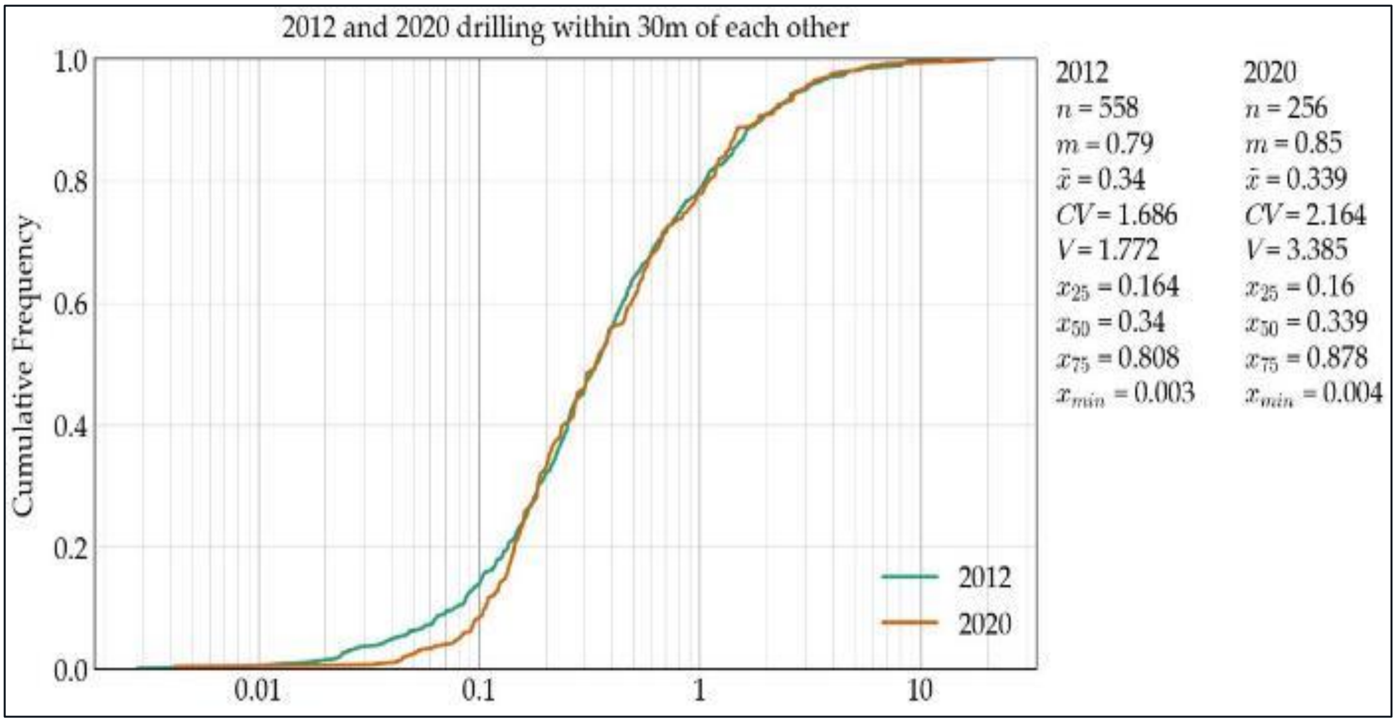
12.1 Adequacy of the LGT Post-2000 Data

During Mr. Dufresne's initial site visit in 2019, a total of 128 pulps were selected from the existing 2012 drill core sample pulps for re-assay. The pulps were submitted to ALS labs in Vancouver, BC, Canada. Pulps were selected from drill intersections that covered mineralized zones to confirm the 2012 assay results. The 2019 pulp re-assays returned values which have close correlation with the original assays for these samples confirming the validity of the 2012 assay results (Dufresne, 2020).

To further validate the reliability of the LGT drill hole data for mineral resource estimation, an analysis of the LGT 2012 database was completed comparing it to the 2020 drill hole data completed by Freeman. The drill hole data was first treated as if it was going into a mineral resource estimation. Compositing of data, composite orphan analysis, and capping of gold grades were completed on all the drill hole data, 2012 LGT samples and 2020 Freeman samples together. See Section 14 for the general drill hole data preparation workflow that was completed for mineral resource estimation. This process normalized the samples from both the LGT and Freeman datasets to the same volume of support. The LGT and Freeman datasets were pared down to only samples within the currently constructed mineralized domains. Spatially similar data were then compared based upon certain distances from each other. The LGT composites within 30 m of Freeman composites were compared to the respective Freeman nearby composites.

Figure 12-1 shows the cumulative histograms of the LGT and Freeman composites withing 30 m of each other. The LGT data compares favourably to the Freeman data with the main discrepancies being in the low-grade portion of the dataset below 0.2 g/t. The mean of the 2012 LGT dataset is slightly lower than the Freeman data, while the medians of the of the two datasets are nearly the same.

Figure 12-1: Cumulative Histograms of 2012 LGT Composites Within 30 m of 2020 Freeman Composites and Vice-Versa.



Source: APEX, 2023.

Based on the results of the 2019 re-assays (Dufresne, 2020), as well as the comparison of 2012 LGT composite data to nearby 2020 Freeman composite data, it is Mr. Dufresne's opinion that the 2012 LGT drill hole data is suitable for the purposes used in the technical report. It is also Mr. Dufresne's opinion that the data is suitable for future work, including mineral resource estimations.

12.2 Adequacy of the Pre-2000 Data

Pre-2000 drill hole data encompasses drilling completed by FMC from 1985 to 1989 and AGR from 1993 to 1995. Mr. Dufresne has reviewed reports from that era and information provided by Mr. Brian Brewer and Mr. Dennis Krasowski who participated not only in the 2012 LGT drilling program but some of the older historical programs, see Section 6.2.4. It is the QP's opinion that the historical pre-2000 drilling completed on the Lemhi Gold Project for FMC and AGR was conducted by experienced professionals using industry best practices at the time. Visual comparisons showed no major discrepancies in the pre-2000 era drill hole data in terms of capturing mineralization zones. In general, cross-section reviews showed that where post-2000 era data showed relative high grade, lower grade, or waste zone, the pre-2000 era data also showed similar relative high grade, lower grade, or waste zones. Discrepancies within the pre-2000 drill hole assay results were noticed and discussed in Section 6.2.4.4.

The pre-2000 drill hole data is not deemed to be as reliable as drill hole data undertaken with current industry best practices for sample preparation, analyses, QA/QC, and security. The discrepancies in the pre-2000 era dataset included lower accuracy in collar location due to collar coordinates often being based on rectified collar location maps, and discrepancies between check assays and umpire assay results based on review of previous reports. Previous industry best practices for sample preparation, assay, and security standards did not include adequate QA/QC of lab assay results and therefore confidence in pre-2000 assay results is lower than current assay results.

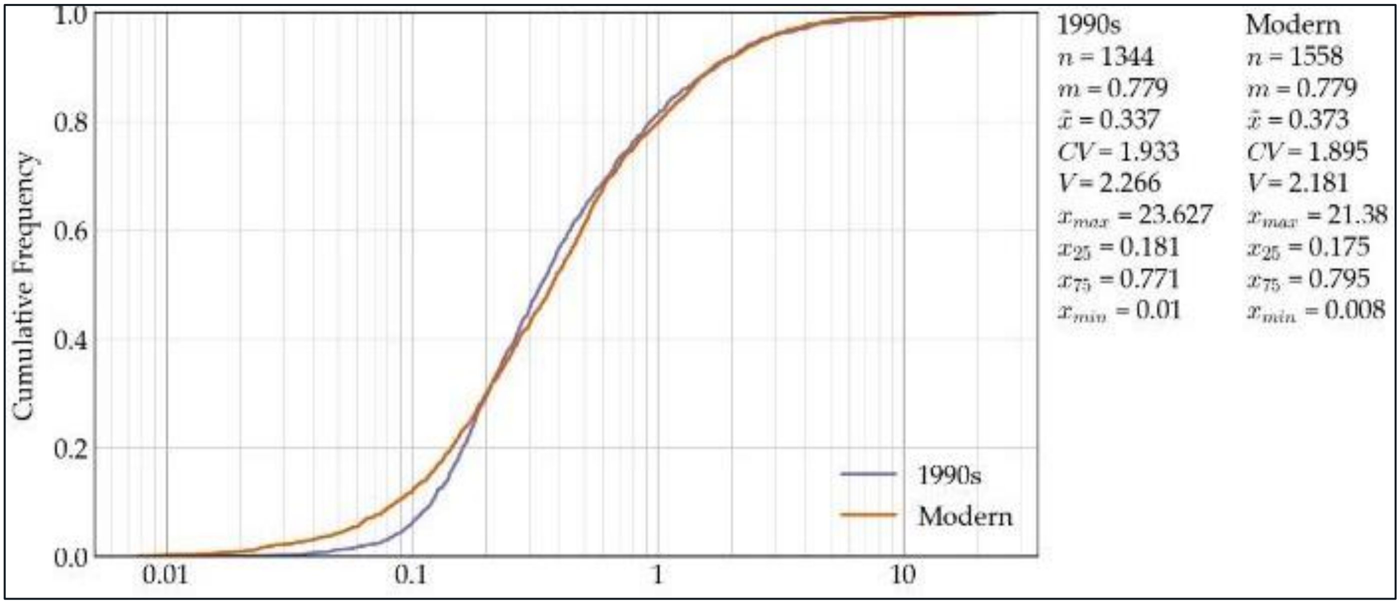
12.2.1 Comparing Pre-2000 Drill Hole Data to Post-2000 Drill Hole Data for Bias

An analysis was completed on pre-2000 era drill hole data by comparing it to modern drill hole data in the form of the LGT and Freeman drilling undertaken using current industry QA/QC best practices, in order to qualify the confidence in the pre-2000 data. Drill holes within close proximity to each other and within the mineralized zone were compared for potential systematic bias in assay results. The main intent of this analysis was to determine confidence in the pre-2000 drill hole data that would be used to estimate gold grades in the MRE.

To qualify the reliability of the pre-2000 drill hole data for mineral resource estimation, the drill hole data was first treated as if it was going into a mineral resource estimation. Compositing of data, composite orphan analysis, and capping of gold grades were first completed on all the drill hole data, both pre-2000 and post-2000 era samples together. See Section 14 for the general drill hole data preparation workflow that is completed for a mineral resource estimation. This process normalized the samples from both pre-2000 and post-2000 era datasets to the same volume of support. The drill hole database was then split into a pre-2000 sample dataset, and a post-2000 sample dataset. The pre-2000 dataset was further split into an FMC dataset and an AGR dataset, with each dataset consisting of all composites within the mineralized zone from the drill holes completed by either FMC or AGR respectively. Spatially similar data were then compared. FMC composites within 30 m of post-2000 composites were compared to the respective post-2000 nearby composites. Similarly, AGR composites within 30 m of post-2000 composites were compared to the respective post-2000 nearby composites.

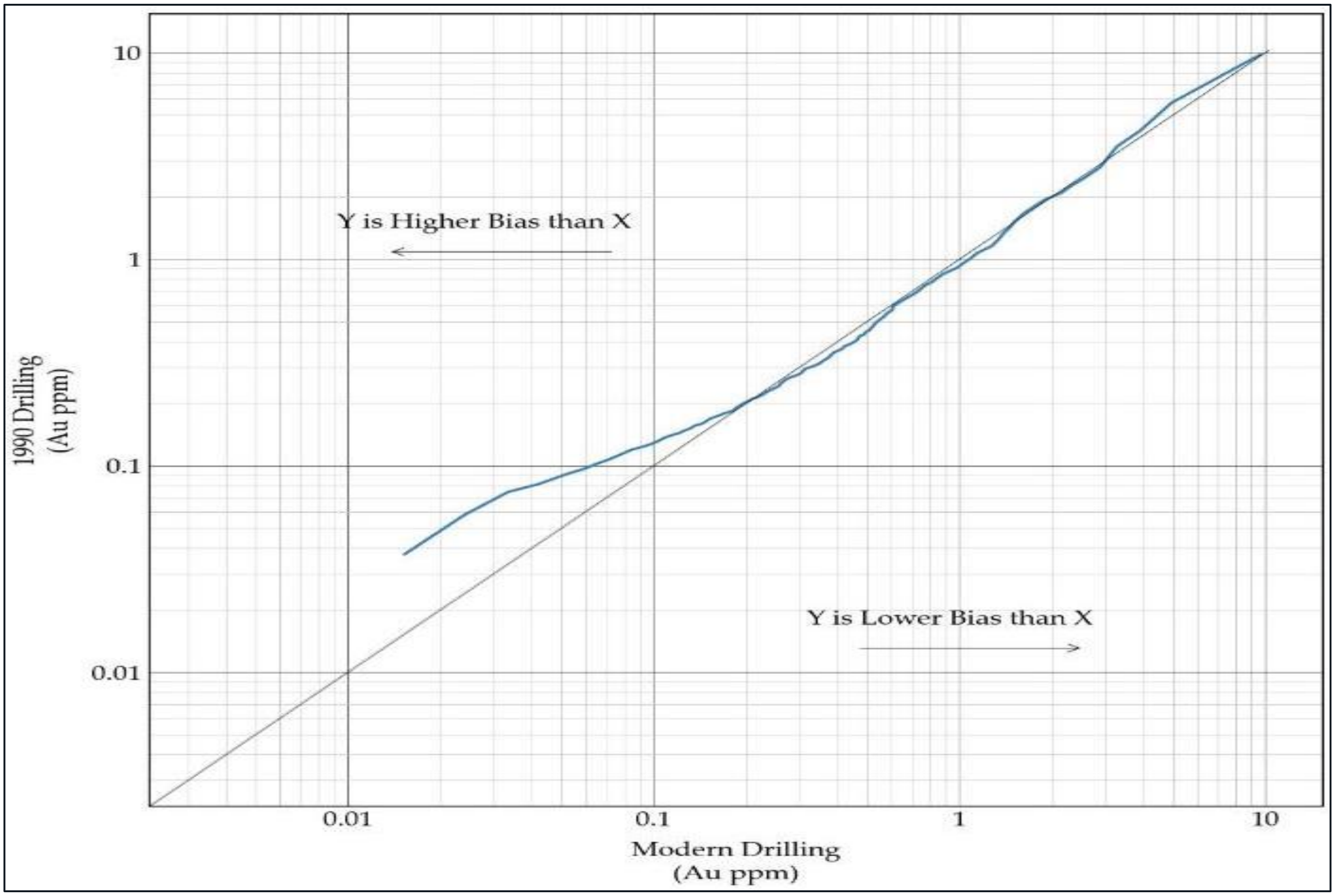
Figure 12-2 shows the cumulative histograms of AGR (1990s) and post-2000 composites within 30 m of each other. The AGR data compares favorably to the post-2000 data with the main discrepancies being in the low-grade portion of the dataset below 0.2 g/t. The means of both datasets are nearly the same, while the median of the AGR data is slightly lower. Figure 12-3 compares the composite data distributions of the AGR (1990s) and post-2000 data within 30 m of each other using a quantile to quantile (QQ) plot. The QQ plots are a graphical tool for comparing two distributions by plotting the matching quantiles from two distributions. A systematic departure above or below the 45°line implies high or low bias. Figure 12-3 shows that the AGR (1990s) data distribution compares favorably to the post-2000 data distribution.

Figure 12-2: Cumulative Histograms of AGR Composites and Post-2000 Composites within 30 m of Each Other



Source: APEX, 2023.

Figure 12-3: Quantile to Quantile Plot of the Distributions of the AGR Composites and Post-2000 Composites within 30 m of Each Other

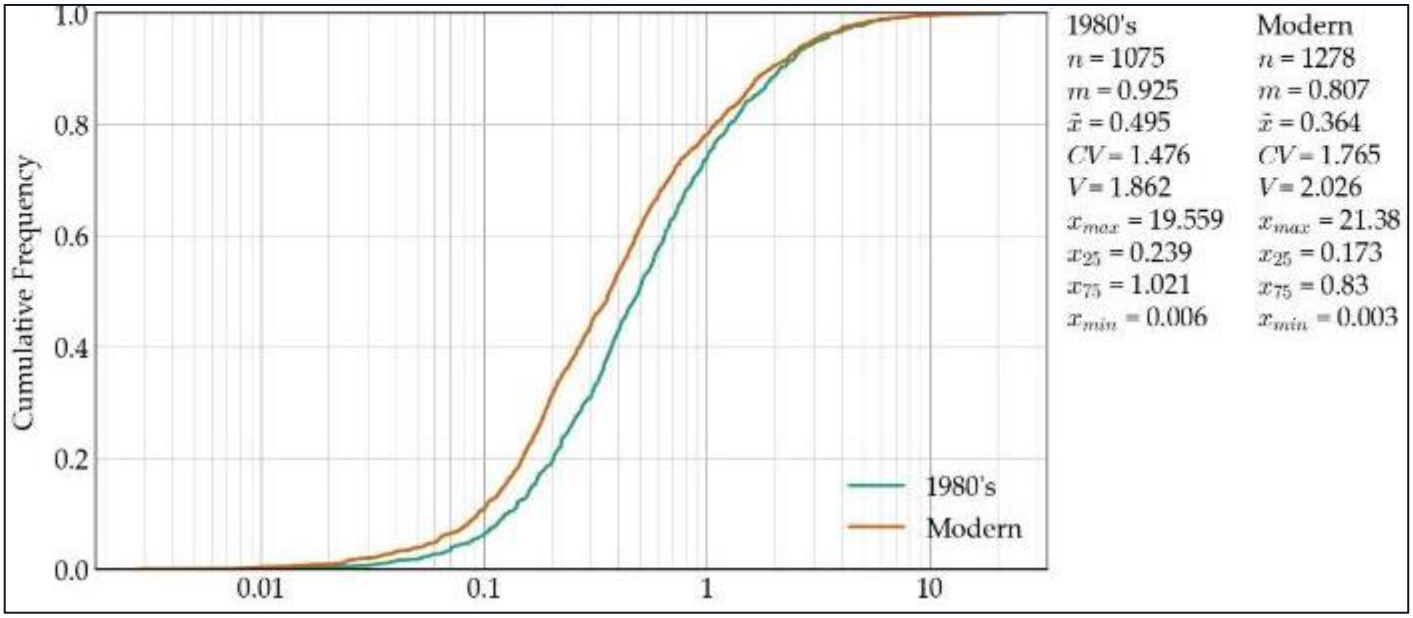


Source: APEX, 2023.

Figure 12-4 shows the cumulative histograms of the FMC (1980s) and post-2000 composites withing 30 m of each other. The FMC data shows a systematic departure from the post-2000 data with the main discrepancies being below 0.8 g/t. The mean and median of the FMC (1980s) dataset is higher than the post-2000 data.

Figure 12-5 compares the composite data distributions of the 1980s FMC and post-2000 data within 30 m of each other in a QQ plot and shows systematic departure above the 45° line for the FMC data. The systematic departure above the 45° line indicates that the FMC (1980s) data has a high bias compared to nearby post-2000 data. This supports the high bias difference in the means between the two datasets as shown in Figure 12-4.

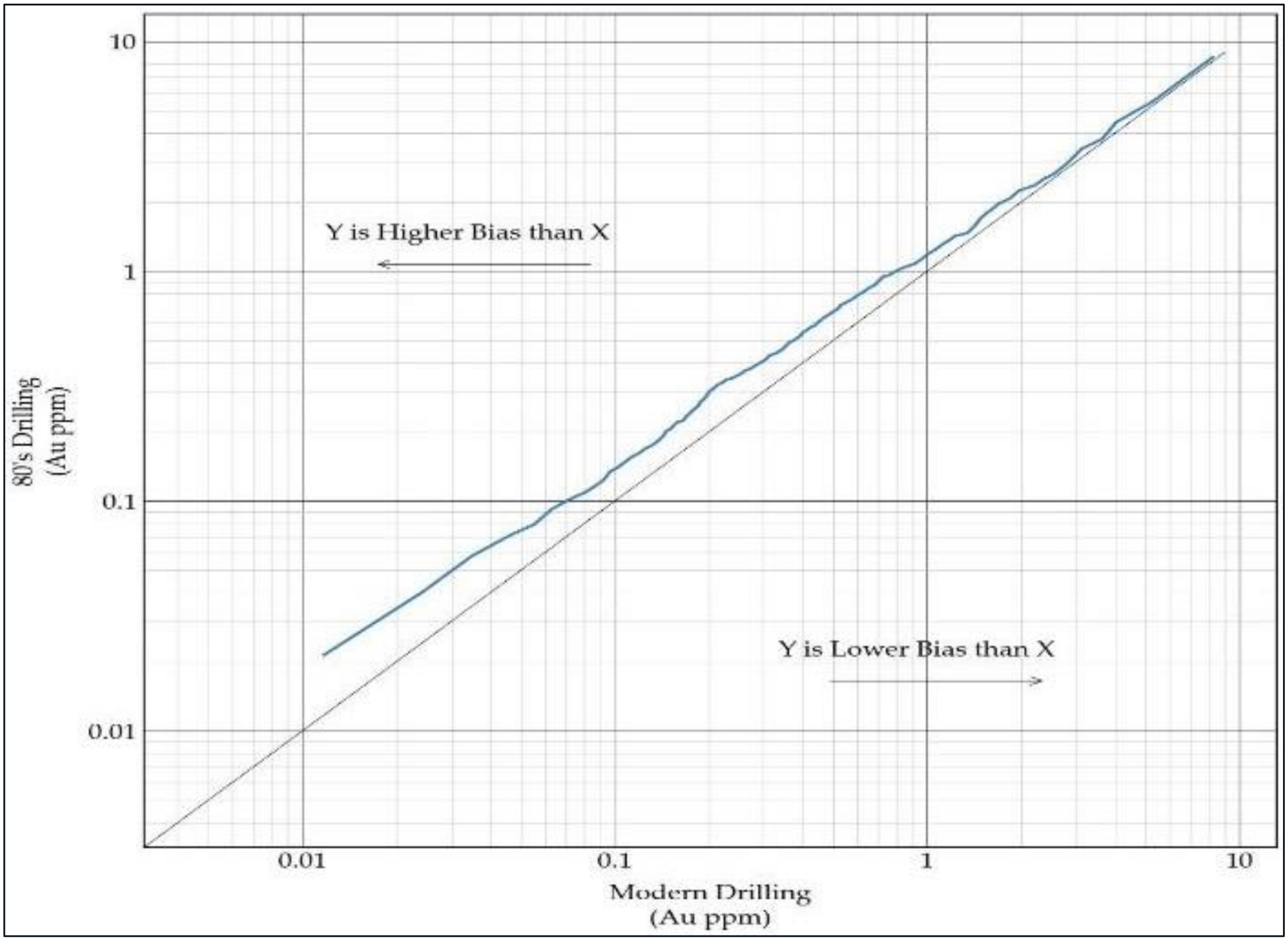
Figure 12-4: Cumulative Histograms of FMC Composites and Post-2000 Composites Within 30 metres of Each Other



Source: APEX, 2023.

It is not clear whether the bias is related to drilling methodology differences such as dry versus wet RC drilling, or if it is related to laboratory and assaying methodology differences.

Figure 12-5: Quantile to Quantile Plot of the Distributions of the FMC Composites and Post-2000 Composites within 30 Metres of Each Other



Source: APEX, 2023.

12.2.2 Recommendations

Mr. Dufresne considers the pre-2000 drill hole data to be well-documented, in good condition, and suitable for ongoing resource estimation studies. The inclusion of the AGR 1990s data should present no risk in the MRE based upon the review. The inclusion of the 1980s FMC data does, however, increase the risk of a slightly biased estimate in areas that rely on the 1980s FMC data; accordingly, the MRE in section 14 has adjusted the classification to a lower confidence level in areas that significantly rely on the 1980s FMC data.

Mr. Dufresne considers the current Lemhi drill hole database to be in good condition and suitable for ongoing resource estimation studies. As discussed in Sections 25 and 26, recommendations for conducting modern drilling in areas of the MRE that rely on significant numbers of historical 1980s FMC drill holes have been made in order to enable higher confidence in the database and the MRE.

12.3 Drill Hole Database Verification

APEX personnel compiled a DHDB containing the historical and 2020 drilling databases and incorporated the new 2021-2022 drilling by Freeman. The database includes collar, downhole survey, assay, geology, structural, and geotechnical data. The 2023 DHDB was validated 100% by APEX personnel and the validation work consisted of:

- Updates to historical collar metadata including company, drill dates, assay certificates, downhole survey types etc.
- Incorporating additional historical geological drill logs as needed
- Normalizing historical geology logs to the current geological mode
- Compiling and validating the current geological logs for incorporation into the database
- Rectifying any problems with the survey and collar files.

Once the re-construction of the DHDB was complete, spot checks of ~10% of the DHDB collars and assays confirmed it was in good condition and suitable for ongoing resource estimation studies. The DHDB contained a total of 514 holes. Of these, 501 are within the current property boundaries. Available drill holes, collar data, assay data, and drill logs are displayed in Table 12-1. A total of 506 drill holes has complete collar, assay, and drill log data. A further 11 holes have assay data with no collar information and are not included in the drillhole counts presented in the table below.

Table 12-1: Summary of Available Drill Hole Data

Company	Year	Total Drill Holes with Collars		Assay Data		Drill Log	
		RC	DDH	RC	DDH	RC	DDH
FMC Gold Corporation	1985	12	-	12	-	12	-
FMC Gold Corporation	1986	74	3	74	3	74	3
FMC Gold Corporation	1987	83	-	83	-	83	-
FMC Gold Corporation	1989	16	1	16	1	16	1
American Gold Resources	1993	39	2	39	2	39	2
American Gold Resources	1994	20	3	20	-	20	-
American Gold Resources	1995	96	4	95	-	95	-
Lemhi Gold Trust	2012	15	40	15	40	15	40
Freeman Gold Corp	2020	-	35	-	35	-	35
Freeman Gold Corp	2021	-	5	-	5	-	5
Freeman Gold Corp	2022	13	53	13	53	13	53
Total		**368**	**146**	**367**	**139**	**367**	**139**

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

A number of metallurgical test programs have been completed on the Lemhi Gold Project since 1994. A summary of the test programs is presented in Table 13-1.

Table 13-1: Summary of Metallurgical Test Programs

Year	Laboratory	Description
1994	Kappes Cassiday, Reno	Phase 1 - column leach, bottle roll tests on 7 composites
1995	Kappes Cassiday, Reno	Phase 2 - column leach, bottle roll tests on 1 composite
1995	Kappes Cassiday, Reno	Phase 3 - column leach, bottle roll tests on 2 composites
2021	SGS, Vancouver	11 samples tested in two phases; included gravity, bottle roll, flotation, comminution Additional phase of variability testing - 26 samples Solid/Liquid separation
2023	Base Met, Kamloops	Comminution on five samples Gravity and leach testing on two master composites CN detox and dewatering testing

13.2 Early Metallurgical Testwork – KCA

During the mid-1990s, Kappes Cassiday and Associates (KCA) conducted cyanide leach testing in a phased program, on behalf of American Gold Resources Corporation. The first test phase was conducted using three PQ drill core samples (DCC94 1-3) and four rotary percussive drill cuttings representing different lithology domains. The second test phase was conducted on a composite of DCC94-1 and DCC94-3 from Phase 1. The third test phase was conducted on three new drill core composites. These reports do not provide adequate detail on the origin of the samples.

The initial results indicated that a fine crush size was necessary to achieve gold extractions of greater than 70%. After Phase 1, column tests were only performed with crush sizes of minus 8 mesh. Gold extraction from the bottle roll tests averaged 94.4%. Results are summarized in Table 13-2. These historical results are difficult to relate to the current project so have not been used to estimate metallurgical performance, they are provided only for reference.

Table 13-2: KCA Leach Testing Summary

Phase	Sample ID	Test Type	Feed Size	Leach days	Calc. Feed Au g/t	Au Rec. %
1	DCC 94-1	column	-1/2"	40	0.93	53.3
	DCC 94-2	column	-1/2"	40	0.50	50.0
	DCC 94-3	column	-1/2"	40	1.00	50.0
	DCC 94-1	bottle roll	-150µm	2	0.90	96.6
	DCC 94-2	bottle roll	-150µm	2	0.37	83.3
	DCC 94-3	bottle roll	-150µm	2	1.15	94.6
	Phyllite	bottle roll	-150µm	3	1.96	95.2
	Quartzite	bottle roll	-150µm	3	1.74	94.6
	Quartz Vein	bottle roll	-150µm	3	4.14	97.7
	Mixed	bottle roll	-150µm	3	1.34	96.3
2	Composite	column	-8 mesh	40	1.12	78.1
	Composite	bottle roll	-150µm	2	1.40	93.3
3	Sample 1	bottle roll	-150µm	2	5.85	94.1
	Sample 2	bottle roll	-150µm	2	1.03	95.2
	Sample 3	bottle roll	-150µm	2	2.83	97.9

13.3 2021 Laboratory Testing - SGS

The SGS Mineral Services Inc. (SGS) laboratory in Vancouver, BC. began conducting a test work program on Lemhi material in January 2021. The program was divided into three phases:

- Phase 1: 2012 assay rejects – gravity plus leach testing on grade composites
- Phase 2: 2020 PQ drill core – comminution, gravity, and leach testing on master composites
- Phase 3: 2020 assay rejects – gravity plus leach testing on variability and lithology composites.

The samples and composite assemblies appear to have been selected to provide spatial coverage of the deposit and depth ranges that fall within the proposed pit shell. The samples represent material originating from near surface to depths of 200 m, the majority of the intervals originated from depths above 150 m below surface.

The initial ten variability samples originated from the 2012 drill program and were identified by drill hole and interval depths in feet. The subsequent 26 variability samples originated from the 2020 drill program and were identified by drill hole and interval depth in meters.

In total, 36 variability samples were prepared in the program, and identified by drill hole number and depth interval in feet. The variability samples did not all receive the same testing. For the brevity, results of these samples are discussed as averages, however in select cases, results from individual samples are discussed.

13.3.1 Head Assay Data - SGS

Head cuts were extracted from each sample and subjected to standard fire assaying for gold, sulphur assays by LECO, and a multi-element ICP scan. Screen metallic gold assays were also conducted on the composites, as well as whole rock analyses and carbon analyses. The screened metallic assays on Phase 1 composites, conducted with 500 g head samples, did not measure elevated gold contents in the +100 µm fractions. The screened metallic assays on the Phase 2 and 3 composites were conducted using 1 kg feed charges and did measure some increase in gold assay in the coarse fraction, most noticeably in the QUAR1 composite.

The gold content in the composites ranged from 0.26 g/t to 3.29 g/t while gold contents in the 26 assayed variability samples ranged between 0.03 to 7.22 g/t. Silver contents were generally below detection limits of 2 g/t, however a few samples measured elevated contents with 7 g/t being the highest value. The composites generally contained low levels of sulphur and copper, averaging 0.13% and 0.05 % respectively. The variability samples averaged 0.06% sulphur and 0.02 % copper. Selected samples contained elevated copper contents, as high as 0.22% in variability sample DH17 (44-49). Gold, sulphur, and copper grades are presented graphically in Figure 13-1.

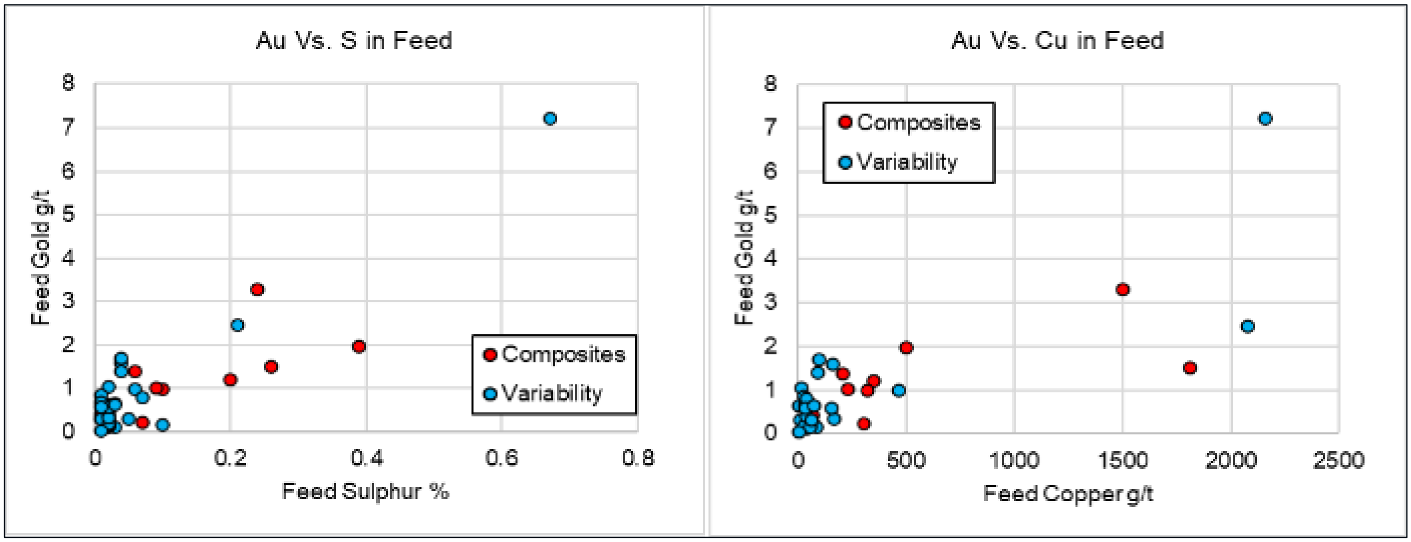
Organic carbon levels, which can affect leaching, were quite low and generally below detection limits. Zinc levels were generally below 40 g/t.

Table 13-3: Head Assay Data – SGS Metallurgical Samples

Phase	Sample ID	Au g/t	Ag g/t	S %	Cu %	TOC %
1	HG Comp 1	1.96	<2	0.39	0.05	0.16
	HG Comp 2	3.29	4	0.24	0.15	<0.05
	HG Comp 3	1.50	7	0.26	0.18	<0.05
	MG Comp 1	1.20	2	0.20	0.04	<0.05
	MG Comp 2	0.98	<2	0.10	0.03	<0.05
	LG Comp 1	1.01	<2	0.09	0.02	0.07
2	Comp 1B	0.42	<2	0.01	0.01	<0.05
	Comp 2B	0.23	<2	0.07	0.03	<0.05
3	Comp PHYL 1	0.66	<2	0.03	0.01	0.12
	Comp QUAR 1	1.38	<2	0.06	0.02	0.09
	Comp SILT 1	0.26	<2	0.02	0.01	<0.05
	Variability Averages	0.91	<2	0.06	0.02	-

Note: Screened metallic gold assays are reported for the composites.

Figure 13-1: SGA Head Assay Data – Au, S, and Cu



Source: Ausenco, 2023.

13.3.2 Comminution Testing – SGS

A series of comminution tests were completed on the composites, results are presented in Table 13-4. The drop weight test results indicate that the material is relatively soft with respect to breakage in a SAG mill. The Bond ball mill work index results suggest that the material is of medium hardness with respect to grinding in a ball mill.

Table 13-4: Comminution Data - SGS

Sample ID	JK DWT		cWi	Abrasion	Rod Mill Wi	Ball Mill Wi
	A x b	SCSE kWh/t	Avg. kWh/t	Index (g)	kWh/t	kWh/t
Comp 1	312.3	5.14	4.4	0.156	11.6	13.4
Comp 2B	85.4	7.21	5.4	0.37	13.1	13.9
HG Comp 1	-	-	-	-	-	14.5
HG Comp 3	-	-	-	-	-	18.0
MG Comp 2	-	-	-	-	-	14.0
PHYL 1	-	-	-	-	-	13.8
QUAR 1	-	-	-	-	-	15.1
SILT 1	-	-	-	-	-	13.8

13.3.3 Gravity Concentration Testing – SGS

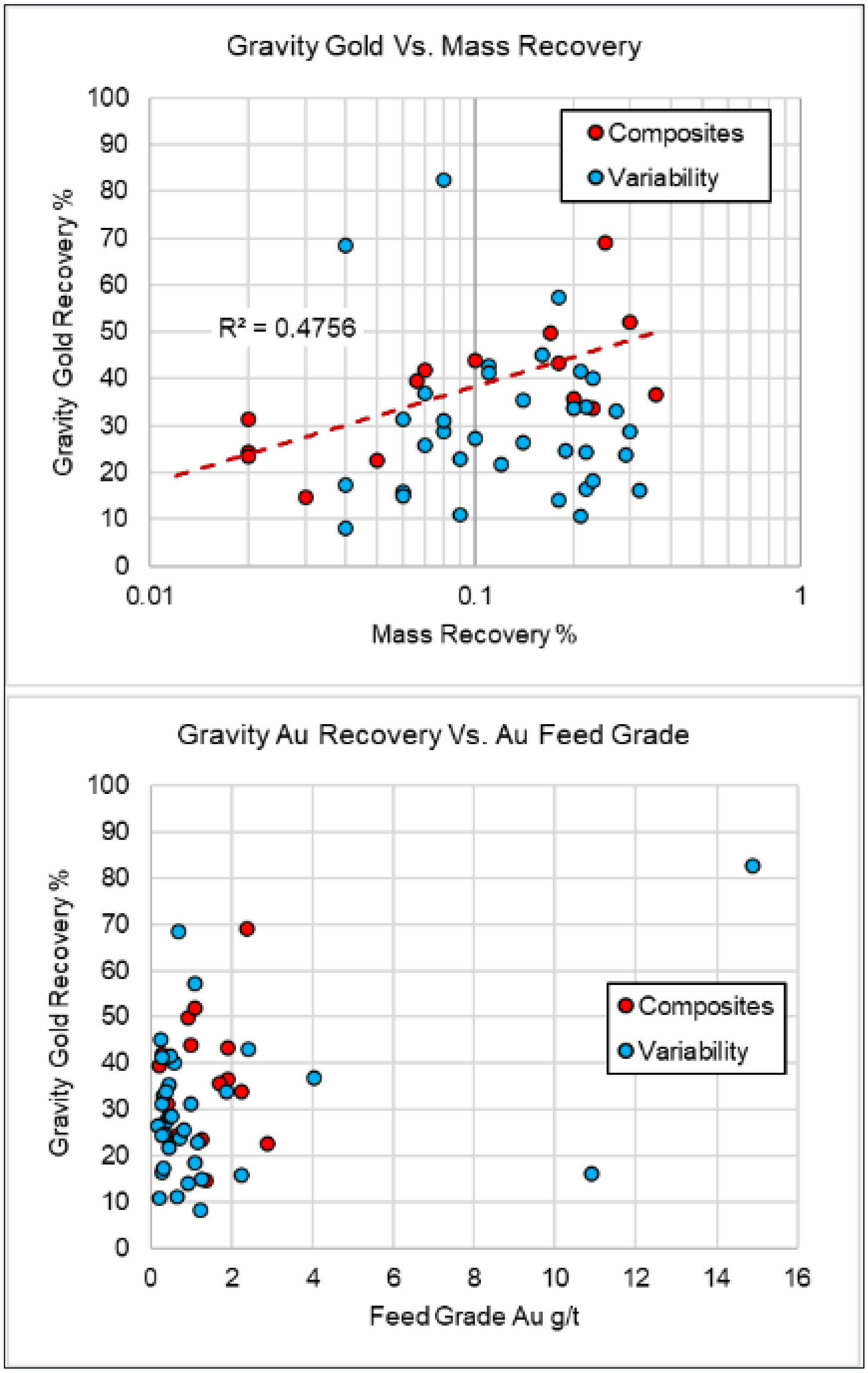
Gravity concentration tests were performed on ground samples using a laboratory Knelson concentrator followed by further upgrading of the Knelson concentrate on a Mozley table. Feed sizings ranged between 80 to 160 µm P_{80}. The feed samples masses ranged between 1 to 4 kg and final gravity concentrates ranged between 0.02 to 0.36% of the test feed mass. Gold recovery to the gravity concentrates of composites averaged 37%, ranging from 15% to 69%. Gold recovery to the gravity concentrates of 33 variability samples averaged 30%, ranging from 8 to 82%. Summarized results are presented in Table 13-5.

Recovery data is presented graphically in Figure 13-2. Variability gravity recovery does not appear to be related to mass recovery; however, a modest decreasing trend can be found in the composite data when plotted on a logarithmic scale, suggesting gravity recovery at plant scale mass recoveries might be less than 20%. There did not appear to be a relationship between gold feed grade and gravity recovery.

Table 13-5: Gravity Gold Recovery - SGS

Sample ID	Grind Size µm P_{80}	Test Feed kg	Calc. Head Au g/t	Gravity Concentrate	
				Mass %	Au Rec. %
HG Comp1	88	1	1.92	0.36	36.5
HG Comp1	134	1	2.26	0.23	33.8
HG Comp3	126	1	1.72	0.20	35.7
HG Comp3	96	1	1.92	0.18	43.3
MG Comp2	101	1	0.92	0.17	49.8
MG Comp2	125	1	1.01	0.10	43.8
LG Comp1	99	1	1.08	0.30	52.0
LG Comp1	126	1	2.38	0.25	68.9
MG Comp1	79	1	1.38	0.03	14.8
HG Comp2	161	1	2.89	0.05	22.6
Comp 1B	112	2	0.27	0.07	41.8
Comp 2B	110	2	0.21	0.07	39.6
Comp PHYL1	102	4	0.66	0.02	24.4
Comp QUAR1	109	4	1.28	0.02	23.5
Comp SILT1	111	4	0.41	0.02	31.3
Variability Averages	**106**	**2**	**1.53**	**0.15**	**30.1**

Figure 13-2: Gravity Gold Recovery Data



Source: Ausenco, 2023

13.3.4 Cyanide Leach Testing – SGS Composites

Bottle roll leach tests with sodium cyanide were conducted on both gravity tails and direct feed samples. The tests investigated the effect of feed particle size on leach performance. Coarse feed sizes, with top sizes ranging 6 mesh up to 19 mm, were leached for up to 21 days, while samples ground to 80 to 150 µm P_{80}were leached for 48 hours. Initial sodium cyanide concentrations varied from 1.5 to 2.5 g/t, and were maintained at 1 g/L through the leach period.

A summary of coarse bottle roll leach results is presented in Table 13-6.

Table 13-6: Coarse Feed Leach Performance – SGS Composites

Sample ID	Top Size mm	Time days	NaCN Solution g/L	NaCN Cons. kg/t	Calc feed Au g/t	Residue Au g/t	Au Extraction %
HG Comp1	3.36	11	2.5	2.34	2.17	0.36	83. 4
HG Comp3	3.36	11	2.5	3.25	1.58	0.63	60.0
MG Comp2	3.36	11	2.5	2.43	0.86	0.19	78. 0
LG Comp1	3.36	11	2.5	1.44	2.02	0.78	61. 4
Comp 1B	19	21	2.5	2.21	0.41	0.13	67. 2
	9.5	14	1.5	0.43	0.40	0.17	56. 7
	2.38	7	1.5	1.42	0.44	0.16	64. 3
Comp 2B	19	21	2.5	2.63	0.41	0.24	42. 3
	9.5	14	1.5	1.14	0.34	0.17	51. 4
	2.38	7	1.5	1.31	0.39	0.11	72. 1

Gold recoveries at the crushed feed sizings were generally low, ranging between 42 and 83% depending on top size and feed grade.

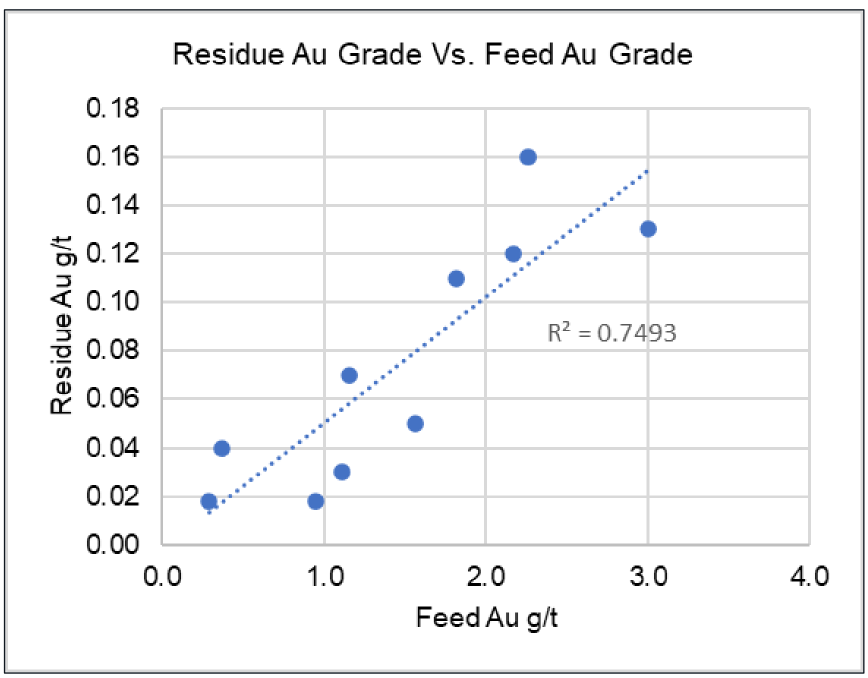
Gold extraction improved considerably following primary grinding to particle sizes ranging between 93 to 130µm P_{80}. Gold extraction averaged 95.8% on composite samples ground to these leach feed sizes, after leaching for 48 hours. Results are summarized in Table 13-7. Sodium cyanide consumptions averaged 0.76 kg/t.

Table 13-7: Ground Feed Direct Leach Performance – SGS Composites

Sample ID	Grind Size µm P_{80}	Time hours	NaCN Sol'n g/L	NaCN Cons. kg/t	Calc feed Au g/t	Residue Au g/t	Au Extraction %
HG Comp1	129	48	2.5	1.53	2.26	0.16	92.9
HG Comp1	97	48	2.5	1.56	2.17	0.12	94.5
HG Comp3	130	48	2.5	0.71	3.00	0.13	95.7
HG Comp3	98	48	2.5	1.13	1.82	0.11	94.0
MG Comp2	121	48	2.5	0.43	1.11	0.03	97.3
MG Comp2	95	48	2.5	0.50	0.95	< 0.02	99.0
LG Comp1	126	48	2.5	0.63	1.16	0.07	94.0
LG Comp1	93	48	2.5	0.60	1.56	0.05	96.8
Comp 1B	111	48	1.5	0.26	0.37	0.04	97.3
Comp 2B	111	48	1.5	0.20	0.29	<0.02	96.5

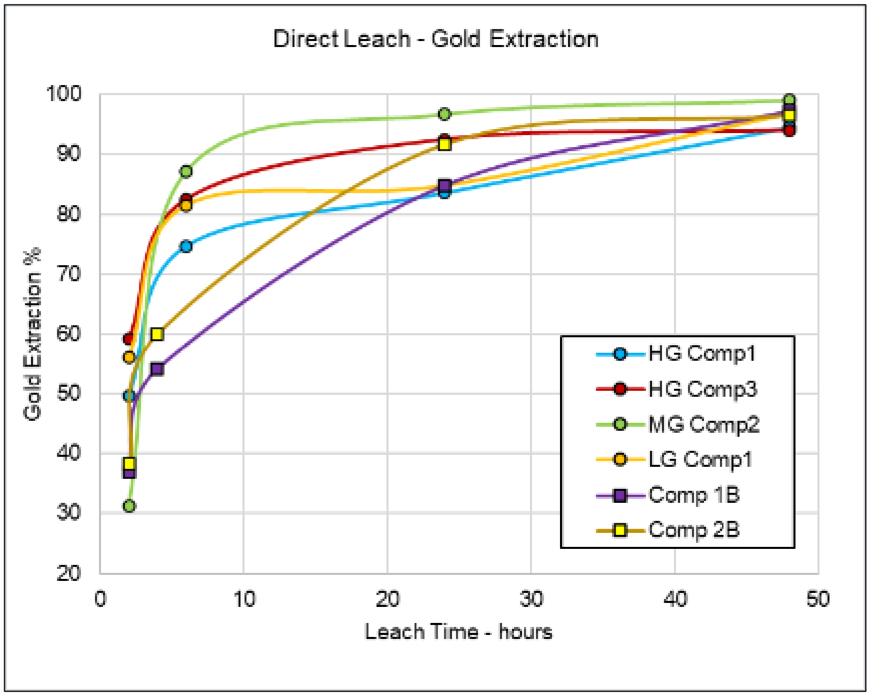
Residue gold contents decreased with finer primary grind sizes, however the increase in gold extraction was not always consistent. Residue gold contents tended to relate well with feed grade gold contents for this data set, as shown graphically in Figure 13-3. Leach kinetics for tests conducted at primary grind sizes near 100µm P_{80} are presented graphically in Figure 13-4. On average, 7% additional gold extraction occurred within the 24 - 48 hour leach period.

Figure 13-3: Direct Leach Residue Grades – SGS Composites



Source: Ausenco, 2023.

Figure 13-4: Direct Leach Gold Kinetics – SGS Composites at 100 µm P_{80}



Source: Ausenco, 2023.

A series of bottle roll leach tests were conducted on gravity tails products, which included both the Knelson and Mozley tails. A compilation of results on the composites is presented in Table 13-8.

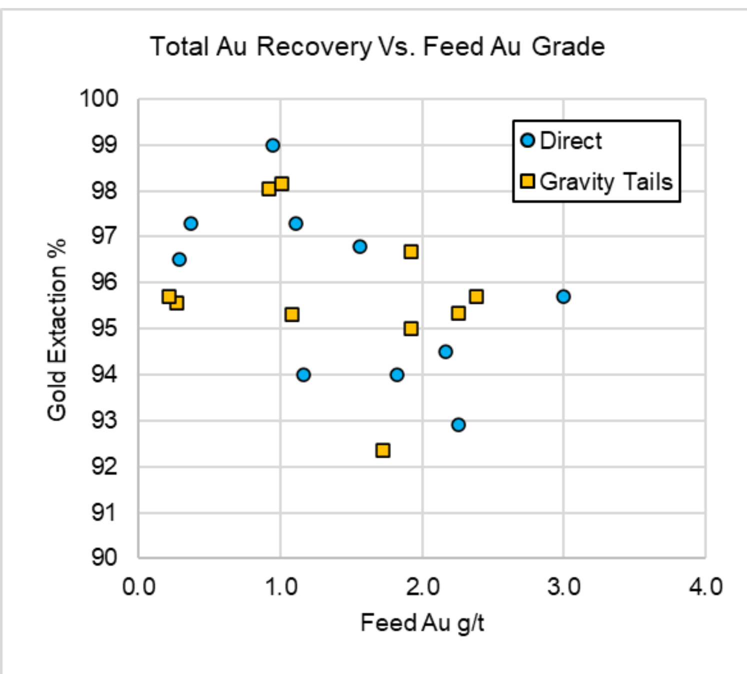
Table 13-8: Gravity Plus Leach and Direct Leach Results – SGS Composites

Sample	Leach Feed	Grind Size	Feed	Residue	Gravity	Leach	Overall
		µm P_{80}	Au g/t	Au g/t	Rec.%	Ext.%	Ext.%
HG Comp1	Direct	97	2.17	0.12	-	94.5	94.5
	Gravity Tail	88	1.92	0.05	36.5	95.9	96.7
	Direct	129	2.26	0.16	-	92.9	92.9
	Gravity Tail	134	2.26	0.09	33.8	94.0	95.4
HG Comp3	Direct	98	1.82	0.11	-	94.0	94.0
	Gravity Tail	96	1.92	0.08	43.3	92.7	95.0
	Direct	130	3.00	0.13	-	95.7	95.7
	Gravity Tail	126	1.72	0.12	35.7	89.2	92.3
MG Comp2	Direct	95	0.95	0.019	-	99.0	99.0
	Gravity Tail	101	0.92	0.019	49.8	98.1	98.1
	Direct	121	1.11	0.03	-	97.3	97.3

Sample	Leach Feed	Grind Size	Feed	Residue	Gravity	Leach	Overall
		µm P_{80}	Au g/t	Au g/t	Rec.%	Ext.%	Ext.%
	Gravity Tail	125	1.01	0.019	43.8	98.3	98.2
LG Comp1	Direct	93	1.56	0.05	-	96.8	96.8
	Gravity Tail	99	1.08	0.04	52	92.4	95.3
	Direct	126	1.16	0.07	-	94.0	94.0
	Gravity Tail	126	2.38	0.07	68.9	90.6	95.7
Comp 1B	Direct	111	0.37	0.04	-	97.3	97.3
	Gravity Tail	112	0.27	0.019	41.8	93.8	95.6
Comp 2B	Direct	111	0.29	0.019	-	96.5	96.5
	Gravity Tail	110	0.21	0.019	39.6	94.2	95.7

A recovery factor of 98% was applied to the gravity concentrate to reflect some losses that could occur across the intense cyanide leach circuit. Both sets of total circuit gold extractions average 95.8% when this factor is included. The results are presented graphically in Figure 13-5.

Figure 13-5: Total Circuit Extractions – SGS Composites



Source: Ausenco, 2023.

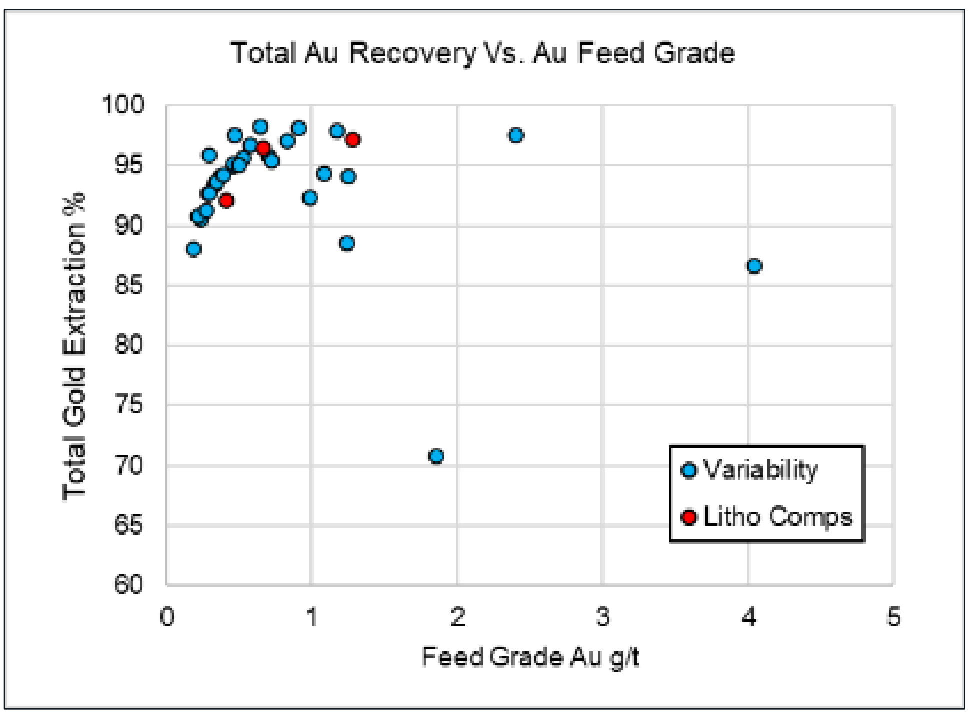
Bottle roll leach tests were conducted on the gravity tails of 31 variability samples, as well as three lithology composites. Results are summarized in Table 13-9. An extraction factor of 98% has been applied to the gravity concentrate to obtain the total circuit recoveries.

Table 13-9: Gravity Plus Leach Extractions – SGS Variability Samples

Sample	Grind Size µm P_{80}	Time hours	NaCN Solution g/L	NaCN Cons. kg/t	Calc Feed Au g/t	Residue Au g/t	Gravity Recovery Au %	Leach Extraction Au %	Total Extraction Au %
COMP PHYL1	102	36	1.5	1.75	0.66	0.02	24.4	96	96.5
COMP QUAR1	109	36	1.5	1.34	1.28	0.03	23.5	96.9	97.2
COMP SILT	111	36	1.5	1.41	0.41	0.02	31.3	89.4	92.1
Variability Averages (31 tests)	105	42	1.5	0.41	0.79	0.06	28.4	91.2	93.3

Results are presented graphically in Figure 13-6. Variability sample DH27 (68-72) measured an anomalous low total circuit gold extraction of 70.8% which could be related to its elevated copper content of 0.21%. Variability sample DH17 (388-394) also measured a low overall gold extraction of 86.6%, considering the high gold content of 4 g/t measured in the feed. A flotation test was conducted on an alternate gravity tail generated from DH17 (388-394) and was found to contain 0.28% Cu. Sodium cyanide consumptions were high in these 2 leach tests, averaging 2.6 kg/t, however cyanide levels were maintained at 500 ppm throughout the 48-hour leach period. If these two samples are excluded, the average gold extraction from the remaining variability samples becomes 94.3%.

Figure 13-6: Total Circuit Extractions – SGS Variability Samples



Source: Ausenco, 2023.

13.3.5 Flotation Testing – SGS Samples

Three flotation tests were conducted in the test program, two on gravity tails of variability samples and one direct feed composite. The two variability samples were selected for flotation testing since the residues following cyanide leaching contained elevated gold contents. The rougher flotation portion of each test was conducted at natural pH and used moderate dosages of PAX for a collector. The two cleaner flotation tests indicated that 62-65% of the gold and 74-89% of the copper in the ground feeds could be recovered to low-mass concentrates. A rougher flotation test on Comp 2B indicated that 93% of the feed gold could be recovered to a rougher concentrate that contained 7.8% of the feed mass. Results are summarized in Table 13-10.

Table 13-10: Flotation Test Data - SGS

Sample ID	Product	Grind Size	Flotation Feed Grade			Recovery to Concentrate %				Concentrate
		µm P_{80}	Au g/t	Cu %	S %	Mass	Au	Cu	S	
DH17 (383-388)	Gravity Tails	95	1.89	0.28	0.26	1.5	65.4	88.8	59.4	1st Cleaner
DH27 (68-72)	Gravity Tails	102	1.11	0.13	0.16	0.3	62.0	74.0	19.0	2nd Cleaner
Comp 2B	Direct Feed	79	0.26	-	0.027	7.8	92.6	-	80.8	Rougher

The flotation tailings from the two cleaner tests on variability sample material were subsequently bottle roll leached with sodium cyanide. In the test on DH17 rougher tails, 91.4% of the remaining gold was extracted to the combined carbon and final leach solution after 24 hours. In the test on DH27 combined rougher and cleaner tails, 92.9% of the remaining gold was extracted to the final leach solution after 36 hours.

It is uncertain whether including flotation in the process flowsheet is justified, however it could be used as a means to improve metallurgical performance on materials that contain elevated copper contents.

13.4 Recent Metallurgical Testing – Base Met Laboratory

Two master composites were provided for metallurgical testing and an additional 5 drill core intervals were provided for comminution testing.

13.4.1 Sample Composition

Assay rejects from four 2020 drill holes were provided to assemble Master Composite 1. Half drill cores from six recent drill holes (2022) were provided to assemble Master Composite 2. Quartzite was the major lithology represented by both composites. An additional five half core intervals from 2022 drill holes were provided for comminution testing, representing material that would be mined within the first 4 years of operation.

13.4.2 Feed Characterization

Head assay data on the metallurgical composites is presented in Table 13-11.

Table 13-11: Head Assay Data – Base Met Composites

Sample	Head Assays						
	Au g/t	Ag g/t	Cu g/t	S %	C %	TOC %	Hg g/t
MC-1	1.94	0.7	286	0.06	0.49	0.02	<1.0
MC-2	1.18	0.9	194	0.11	0.71	0.01	<1.0

The samples appeared to be similar in composition to the Medium Grade and Quartzite composites tested by SGS.

Mineral composition analyses were completed on the two composites using QEMSCAN. Composition data and sulphur deportment data are reported in Table 13-12 and Table 13-13.

Table 13-12: Mineral Composition Data – Base Met Composites

Mineral	Content - %	
	MC-1	MC-2
Pyrite	0.06	0.06
Chalcopyrite	0.07	0.04
Other Sulphides	0.02	0.01
Quartz	76.4	62.8
K-Feldspar	4.0	5.7
Sericite/Muscovite	13.6	21.9
Calcite	0.35	0.69
Other Carbonates	3.5	5.2
Other Minerals	2.0	3.7

Table 13-13: Sulphur Deportment Data – Base Met Composites

Mineral	% S Distribution	
	MC-1	MC-2
Pyrite	45.9	61.1
Chalcopyrite	19.2	9.23
Tetrahedrite	0.19	0.00
Other Sulphides	8.08	2.16
Barite	25.6	26.9
Other	1.02	0.63

Quartz was the dominant mineral in both composites. The composites contained low levels of sulphide minerals, between 0.1 to 0.15% in total. Pyrite was the primary sulphide mineral, followed by chalcopyrite. Barite accounted for 26% of the sulphur in the composites.

13.4.3 Comminution Testing – Base Met

Drop weight SMC tests and Bond ball mill work index tests were completed on samples of available drill core. Results are presented in Table 13-14 with results similar to the SGS comminution test results. The samples would be considered soft with respect to breakage in a SAG mill and of medium hardness with respect to ball mill grinding.

Table 13-14: Comminution Data – Base Met

Sample ID	SMC			Ball Mill Wi
	A x b	dWi kWh/t	SCSE kWh/t	kWh/t
FG22-009C (44-49m)	-	-	-	14.8
FG22-009C (67-71m)	92.4	2.9	7.02	14.1
FG22-010C (62-67m)	80.6	3.3	7.38	14.8
FG22-017C (22-35m)	134.7	2.0	6.22	14.6
FG22-025C (95-98m)	-	-	-	13.8
MC-1	97.5	2.7	6.89	-

13.4.4 Gravity Concentration Testing – Base Met

An E-GRG test was conducted on 20 kg portion of sample MC-1. The staged recovery test is conducted at three consecutively finer grind sizes using a laboratory Knelson concentrator. The calculated feed grade of the test charge was 1.5 g/t gold, and the cumulative gold recovery was 57.9% to a total concentrate containing 1.2% of the feed mass.

Gravity concentration tests were conducted on 4 kg portions of samples MC-1 and MC-2, to provide gravity tails for subsequent leach testing. The test charges were ground to a nominal sizing of 150 µm P_{80} and processed through a laboratory Knelson concentrator. The Knelson concentrate was then further upgraded on a Mozley table to obtain a high-grade, low-mass concentrate. 27-29% of the feed gold was recovered to the final gravity concentrates, which contained 0.015 to 0.030% of the feed mass.

Gravity test results are summarized in Table 13-15.

Table 13-15: E-GRG Results – Base Met

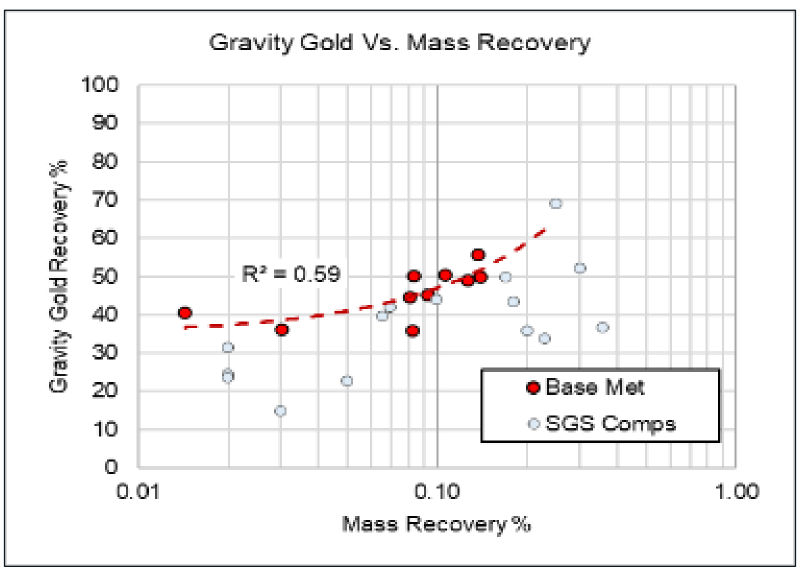
Sample ID	Test Type	Calc'd Feed Au g/t	Feed Sizing µm P_{80}	Product	Gravity Concentrate		
					Mass %	Au g/t	Au Dis''n %
MC-1	E-GRG	1.50	1181	Knelson Con 1	0.43	62.6	18.1
			284	Knelson Con 2	0.37	96.8	23.5
			139	Knelson Con 3	0.37	66.7	16.3
			-	Cum.Concentrate	1.17	74.6	57.9

Additional gravity concentration tests were conducted on 1 and 4 kg portions of MC-1 and MC-2 composites, to provide gravity tails for subsequent leach testing. The test charges were ground to the target size distribution and processed through a laboratory Knelson concentrator. The Knelson concentrate was then further upgraded on a Mozley table to obtain a high-grade, low-mass concentrate. Results are summarized in Table 13-16 and graphically in Figure 13-17. Between 36-56% of the feed gold was recovered to the gravity concentrates, which contained 0.014 to 0.14% of the feed mass. The gravity recoveries were generally better than measured on the SGS composites with both data sets appearing to follow a similar upper recovery limit.

Table 13-16: Gravity Recovery Data – Base Met

Sample ID	Grind Size	Test Feed	Calc.Head	Gravity Concentrate	
	µm P_{80}	kg	Au g/t	Mass %	Au Rec.%
MC-1	109	1	1.54	0.083	50.1
	134	1	1.41	0.083	35.8
	150	1	1.26	0.127	49.0
	173	1	1.88	0.106	50.4
	150	4	1.30	0.014	40.4
MC-2	115	1	1.33	0.140	50.0
	124	1	1.36	0.081	44.7
	153	1	1.76	0.137	55.6
	170	1	1.62	0.093	45.3
	150	4	1.63	0.030	36.2

Figure 13-17: Gravity Recovery – Base Met



Source: Ausenco, 2023.

13.4.5 Cyanide Leach Testing – Base Met

A series of bottle roll leach test were conducted on both direct feed and gravity tails for MC-1 and MC-2 samples. All tests were conducted using a NaCN solution strength of 2 Cg/L. Leach times were typically 24 hours, with 36-hour leach times were also tested for the larger grind size. Primary grind sizes ranged from 109 to 173 µm P_{80}.

Table 13-17: Cyanide Leach Data – Base Met

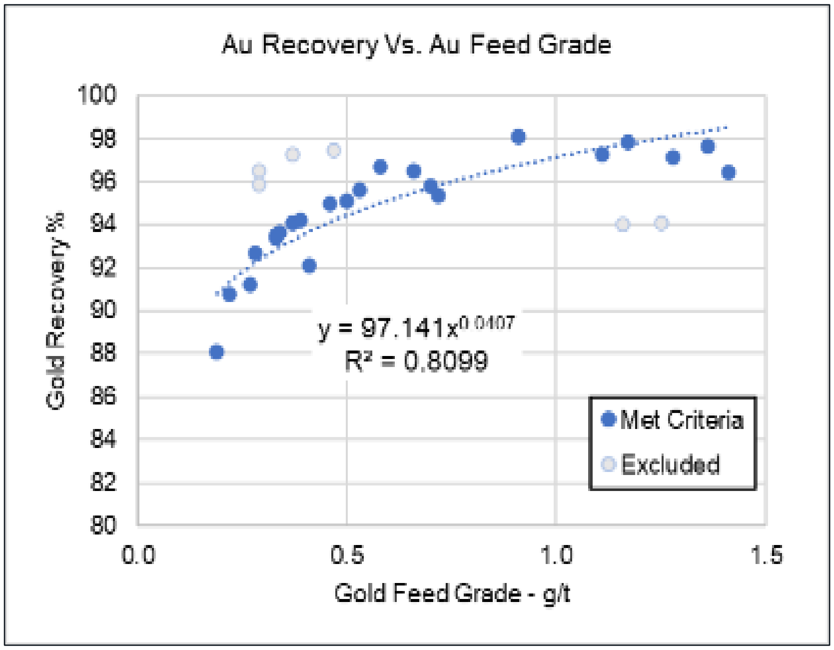
Sample	Grind Size µm P_{80}	Time hours	NaCN Cons. kg/t	Feed Au g/t	Residue Au g/t	Gravity Rec. %	Leach Ext. %	Overall Ext. %
MC-1	109	24	0.78	1.54	0.04	50.1	94.8	96.4
	134	24	0.77	1.41	0.04	35.8	95.6	96.5
	150	24	0.58	1.26	0.05	49.0	93.0	95.5
	173	24	0.67	1.88	0.05	50.4	94.6	96.3
	150	24	0.63	1.29	0.06	40.6	92.2	94.5
	150	36	0.69	1.30	0.05	40.3	94.2	95.7
MC-2	115	24	0.39	1.33	0.03	50.0	96.2	97.1
	124	24	0.43	1.36	0.02	44.7	97.3	97.6
	153	24	0.40	1.76	0.04	55.6	94.9	96.6
	170	24	0.38	1.62	0.04	45.3	96.0	96.9
	150	24	0.43	1.60	0.06	36.8	94.1	95.5
	150	36	0.48	1.61	0.04	36.6	96.1	96.8

The results indicate that there is a modest decrease in final residue gold contents at a nominal grind size of 125 µm P_{80} compared to 150 µm P_{80}, however the finest grind sizes tested did not show any further improvement in performance. It was not clear if the 36 hours of leach time applied improved recovery over the 24-hour extractions due to the small variances in residue grades on the tests conducted at 150 µm P_{80}.

13.5 Gold Recovery Estimate

Test data that met the criteria of appropriate grind size range (90-130µm P_{80}) and head grade range (below 1.5 g/t) was used in the development of a recovery model. The data set included composites and variability samples tested by SGS, as well as single tests on MC-1 and MC-2 from the Base Met testing. Direct leach results were used where available Consistent with the above discussion, a 98% extraction factor has been applied to gravity concentrates for gravity plus leach tests. Six samples were removed from the data set for the recovery equation as the results appeared to be inconsistent for the measured feed grades and are presented graphically in Figure 13-18.

Figure 13-18: Gold Recovery Model



Source: Ausenco, 2023

13.5.1 Cyanide Destruction

A 10 kg sample of Master Composite 2 was subjected to the developed cyanide leach process to provide slurry for subsequent cyanide destruction testing. The sample was ground to a feed sizing of 150 µm P_{80} and leached for 24 hours at 45% solids with 0.5 g/t NaCN and 10 g/L carbon. A series of six continuous CN destruction tests using the SO_2/air process were conducted on the leached slurry, results are presented in Table 13-18.

Table 13-18 Cyanide Destruction Test Data

Test	Retention Time (minutes)	pH	Product Solution Chemistry (mg/L)			Reagent Addition								
						g/g CN_{WAD}			g/L Feed Slurry			kg/t Solids		
			CN_{Total}	CN_{WAD}	Cu	SO_2 Equiv	Lime (CaO)	Cu	SO_2 Equiv	Lime (CaO)	Cu	SO_2 Equiv	Lime (CaO)	Cu
Feed	-	-	363	305	24.4	-	-	-	-	-	-	-	-	-
C1	60	8.2	0.7	0.5	0.09	5.0	6.4	0.16	0.91	1.95	0.05	1.44	3.09	0.08
C2	60	8.2	1.7	1.6	1.09	5.0	16.9	0.16	1.52	5.16	0.05	2.41	8.18	0.08
C3	60	8.2	1.9	1.7	1.22	5.0	6.6	0.33	1.52	2.00	0.10	2.41	3.18	0.16
C4	30	8.1	1.0	0.8	0.38	5.0	10.4	0.16	1.53	3.17	0.05	2.43	5.04	0.08
C5	30	8.1	1.0	0.9	0.50	3.0	6.2	0.16	0.91	1.90	0.05	1.45	3.01	0.08
C6	**30**	8.1	**0.9**	**0.8**	**0.19**	**3.0**	**2.0**	**0.05**	0.92	0.62	0.02	1.45	0.98	0.02

Notes: SO_2 added as sodium metabisulphite (SMBS), copper added as copper sulphate.

The final test provided acceptable discharge chemistry and the most optimized reagent dosages of this test series. WAD cyanide levels were effectively reduced from 305 mg/L to less than 1 mg/L in 30 minutes of slurry retention time. The SO_2 requirement of 3.0 g/g CN_{WAD} was typical of air/SO_2 process using sodium metabisulphite (SMBS), possibly somewhat low.

A sub-sample of the CN Detox tails was submitted for ABA analysis, which indicated that the material was not acid generating and measured a net neutralization potential (NNP) of 35 kg/$CaCO_3$ per tonne.

13.6 Solid Liquid Separation

A series of flocculant scoping and static settling tests were conducted by SGS on a composite of leach test products, as well as three lithology composite residues. The testing identified BASF Magnafloc 10 as suitable for the dewatering composite, while BASF Magnafloc 155 was selected for the three lithology composites. A summary of the static dewatering test results is presented in Table 13-19.

Table 13-19 SGS Static Settling Test Data

Sample	Size	Pulp pH	Flocculant	U/F Solids Density	Thickener Unit Area
	µm K80		g/t	% w/w	m2/(t/day)
Comp LS 1	122	10.6	12	60	0.07
Comp PHYL-L1	98	8.6	25	62	0.07
Comp QUAR-L1	104	8.6	20	68	0.06
Comp SILT-L1	105	8.6	25	64	0.08

Vacuum filtration tests were completed on the thickened underflow of the Comp LS1 material. Results from three test runs that produced the lowest moisture cakes are presented in Table 13-20.

Table 13-20 SGS Vacuum Filter Test Data

Sample	Feed solids	Vacuum level	Form/Dry Time Ratio	Cake Thickness	Throughput dry solids	Cake Moisture
	% w/w	inch Hg		mm	kg/m2-hr	% w/w
Comp LS 1	63	20	0.5	20	299	15.7
			0.2	15	172	13.3
			0.3	25	147	14.6

A series of solid-liquid separation tests were completed by Base Met Laboratory on sub-samples of the combined CN detox test products. Initial flocculant scoping tests and static settling tests determined that a flocculant AN913SH along with lime addition to pH 9 provided the best settling conditions evaluated. These conditions were then referenced for a series of dynamic thickening tests using a 100mm laboratory thickener. It was determined that a coagulant (SNF DB45 SH) was required to achieve the lowest overflow turbidity levels. Results are summarized in Table 13-21.

Table 13-21 Base Met Dynamic Thickening Test Data

Test	Density % w/w		Flocculant	Coagulant	pH	Rise Rate	Loading Rate	Turbidity
	Feed	U/F	g/t	g/t		m/hr	t/m2/hr	mg/L
D1-A	15	55.4	40	-	8.2	3.1	0.5	352
D1-B		56.9	40	-	8.2	4.4	0.7	Max
D1-C		26.8	60	-	8.2	4.4	0.7	553
D1-D		50.0	40	-	9.0	4.4	0.7	911
D1-E		45.8	40	20	9.0	4.4	0.7	27
D1-F		62.3	20	10	9.0	4.3	0.7	135

13.7 Deleterious Elements

There appears to be low levels of copper present in some samples that could affect sodium cyanide consumption levels. Two variability samples had elevated copper levels in the feed which appeared to affect direct cyanide leach recoveries, further testing is required to confirm if this is due to fine gold inclusions in chalcopyrite, or simply higher levels of NaCN were required.

While there are no other known deleterious elements that could have significant impact on potential economic extraction, further chemical characterization of leach solutions should be completed.

13.8 Comments on Mineral Processing and Metallurgical Testing

The samples tested in these metallurgical test programs suggest that gold present in the Lemhi material is amenable to recovery by conventional cyanide leaching techniques. The material is generally soft with respect to SAG milling and has a moderate ball mill work index. A portion of the gold in the samples was consistently recovered by gravity techniques, however it did not appear that the inclusion of a gravity circuit significantly improved overall recovery. Gold extractions by cyanide leaching appeared to plateau consistently in the range of 92-96% within 36 hours, following grinding to approximately 130µm P80. Solid-liquid separation results suggest that the ground material can be dewatered at unit rates that are typical for this processing flowsheet.

14 MINERAL RESOURCE ESTIMATE

14.1 Introduction

The 2023 Lemhi Gold Project Mineral Resource Estimate (2023 Lemhi MRE) herein is based upon historical drilling and drilling conducted on the Lemhi Property by Freeman between 2020 and 2022 and supersedes the prior 2021 maiden mineral resource estimate for the Lemhi Gold Project (2021 Lemhi MRE). Previous historical mineral resource estimates are discussed in Section 6 of this technical report and are all considered historical in nature and should not be relied upon.

This technical report section details an updated MRE completed for the Lemhi Gold Project by Mr. Warren Black, M.Sc., P.Geo. and Mr. Tyler Acorn, M.Sc. of APEX under the direct supervision of Mr. Michael Dufresne, M. Sc., P.Geol., P.Geo., the QP who takes responsibility for Section 14. Mr. Dufresne has visited the property on several occasions with the most recent visit February 18, 2022.

The workflow implemented for the calculation of the 2023 Lemhi MRE was completed using Micromine commercial resource modelling and mine planning software (v. 22. 0), Resource Modelling Solutions Platform (RMSP; v. 1.10.2), and Deswik CAD (v2022.2). Supplementary data analysis was completed using the Anaconda Python distribution and a custom Python package developed by Mr. Warren Black, M.Sc., P. Geo. and Mr. Tyler Acorn, M.Sc., both of APEX.

The drillhole database was validated by APEX personnel under the supervision of Mr. Dufresne as summarized in Section 12.3 Mr. Dufresne accepts the current Lemhi Gold Project drillhole database as reliable and suitable for use in ongoing mineral resource estimation.

Mineral resource modelling was conducted in the UTM coordinate system relative to the North American Datum (NAD) 1983, and Idaho State Plane Central FIPS 1102 (EPSG:6448) The mineral resource block model utilized a selective mining unit (SMU) block size of 2.5 m (X) by 2.5 m (Y) by 2.5 m (Z) to honour the mineralization wireframes. The percentage of the volume of each block below the top of bedrock surface and within each mineralization domain was calculated using the 3-D geological models and a 3-D topographic surface model. The gold grades were estimated for each block using ordinary kriging with locally varying anisotropy (LVA) to ensure grade continuity in various directions is reproduced in the block model. The MRE is reported as undiluted within a series of optimized pit shells. Details regarding the methodology used to calculate the MRE are documented in this technical report section.

Definitions used in this section are consistent with those adopted by CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29, 2019, and "Definition Standards for Mineral Resources and Mineral Reserves" dated May 10, 2014, and prescribed by the Canadian Securities Administrators' NI 43-101 and Form 43-101F1, Standards of Disclosure for Mineral Projects. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

14.2 Drillhole Data Description

Data from Freeman's 2020 – 2022 drilling program was captured and validated on-site during the drill program by APEX personnel. At the conclusion of the 2022 program, APEX personnel compiled the results with the previously validated historical data (Dufresne 2021), discussed in Section 12. The historical data has not changed since the 2021 MRE was completed. In the opinion of Mr. Dufresne, the current Lemhi Gold Project drillhole database is deemed to be in good condition and Mr. Dufresne accepts the database and considers it suitable to use in ongoing resource estimation studies.

In total, 442 drillholes intersect the estimation domains, summarized in Table 14-1. Within the estimation domains there were 22,138 m of drilling of which 63 m (0.3% of the total) is unsampled intervals, assumed to be waste, and assigned a nominal waste value, half the detection limit of modern assay methods (0.0025 g/t Au). Any sample intervals that have explicit documentation that drilling did not return enough material to allow for analysis are classified as insufficient recovery (IR) and were left blank. Samples with unknown detection limits and/or assay methodologies and in the database as zero were assigned a nominal waste value of 0.0025 ppm g/t Au.

Table 14-1: 2023 Lemhi Gold Project Drillhole Summary

Zone	Number of Drillholes	Total Metres Inside Domain
Main	433	22,068.5
Beauty	9	70

14.2.1 Data Verification

APEX personnel validated the mineral resource database by checking for inconsistencies in analytical units, duplicate entries, interval, length, or distance values less than or equal to zero, blank or zero-value assay results, out-of-sequence intervals, intervals, or distances greater than the reported drillhole length, inappropriate collar locations, survey and missing interval and coordinate fields. A small number of errors were identified and corrected in the database. A detailed discussion on the verification of historical (pre-2000) and modern (post-2011) drill hole data is provided in Section 11 and 12 of this technical report. As discussed in Section 12, recommendations are provided in Section 12 and 26 for conducting modern drilling in areas of the MRE that significantly rely on the historical FMC 1980s drilling to enable higher confidence in the database and the MRE. Mr. Dufresne considers the drillhole database suitable for mineral resource estimation.

14.3 Grade Estimation Domain Interpretation

Grade estimation domain wireframes were created using implicit modelling using the grade estimation domain coding. It was an iterative process utilizing many geological inputs. Several modelling geologists intricately familiar with the deposit provided input and review through various stages of grade estimation domain modelling and the estimation domain coding is adjusted as needed. This peer-review process is repeated until the final grade estimation domains are created. The critical inputs used to define the boundaries and orientation of the grade estimation domains are gold assays and, to a lesser extent, drillhole logging of quartz vein abundance.

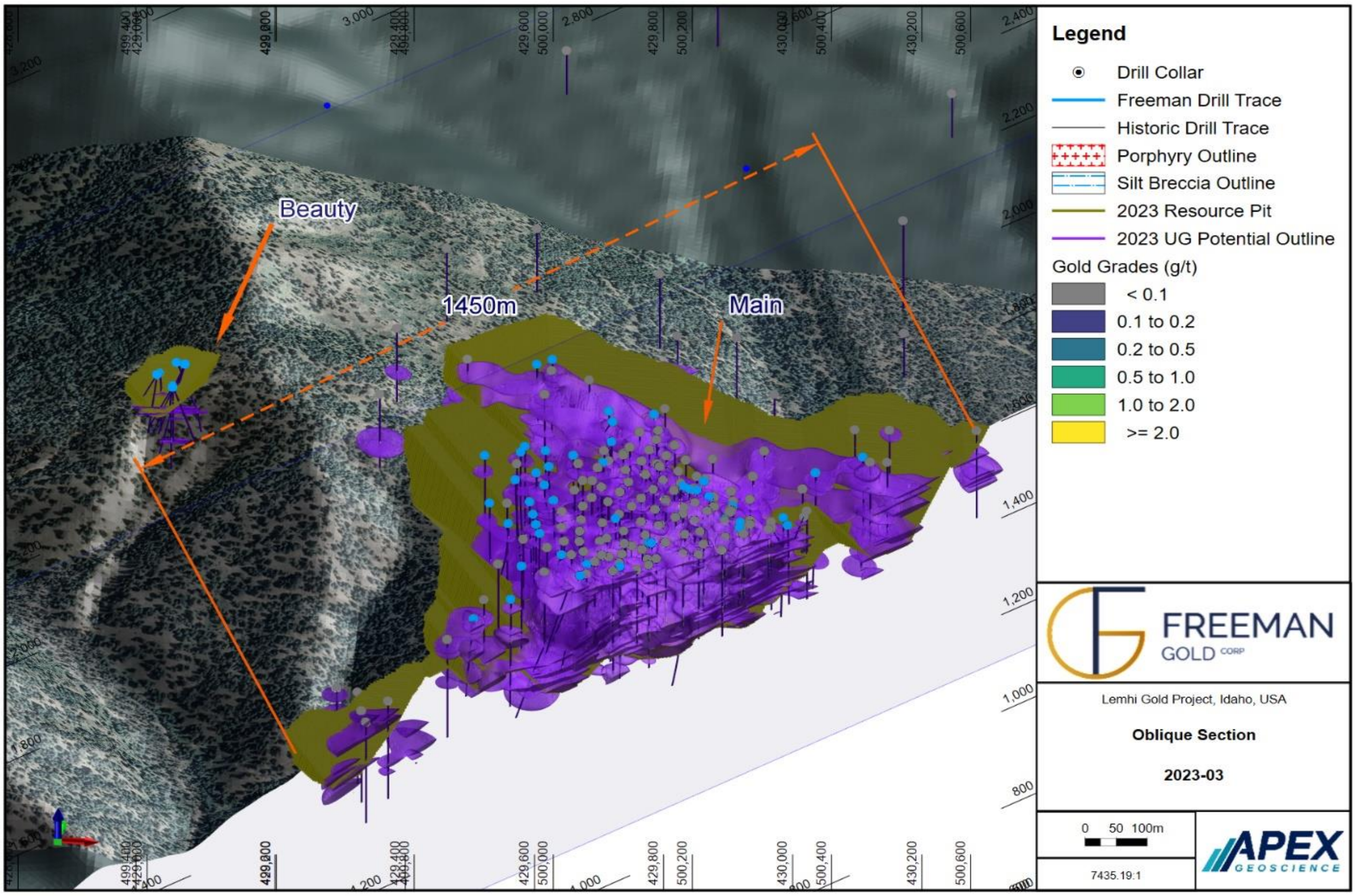
Mineralization at the Lemhi Gold Project is primarily represented by two dominant styles of gold mineralization. The primary mineralization occurs as a halo around the granodiorite intrusion, concentrated on the bottom side, with secondary mineralization along faults and shallow dipping foliation. It appears that both styles of mineralization generally occur in zones of stacked parallel sub-horizontal sheets. The Beauty Zone is ~700m west from the nearest modeled intrusion and is primarily controlled by a structurally complex fault zone. The mineralization is summarized by each resource estimation domain in Table 14-2. An orthogonal slice view of the estimation domains is shown in Figure 14-1.

Modelling geologists assign mineralized intervals to a specific grade estimation domain code to create the grade estimation domains using the logging features described above, fault models, commodity assays, and drill core photos. The primary goal is to ensure a single grade estimation domain connects similar style mineralization and honours structural and geological controls on their orientation and spatial continuity. Intervals that are not mineralized are categorized as waste.

Table 14-2: Lemhi Gold Project Grade Estimation Domain Descriptions

Grade Estimation Domains	Description
Main	Linear stacked gently dipping bodies that halo around the intrusion. Distal from the intrusion, smaller more abundant gently dipping bodies following foliation and faults. Two large structures control NE high-grade and NW high-grade zoner.
Beauty	Structurally controlled by complex fault zone

Figure 14-1: Orthogonal Slice View of the 2023 Lemhi MRE Grade Estimation Domains.



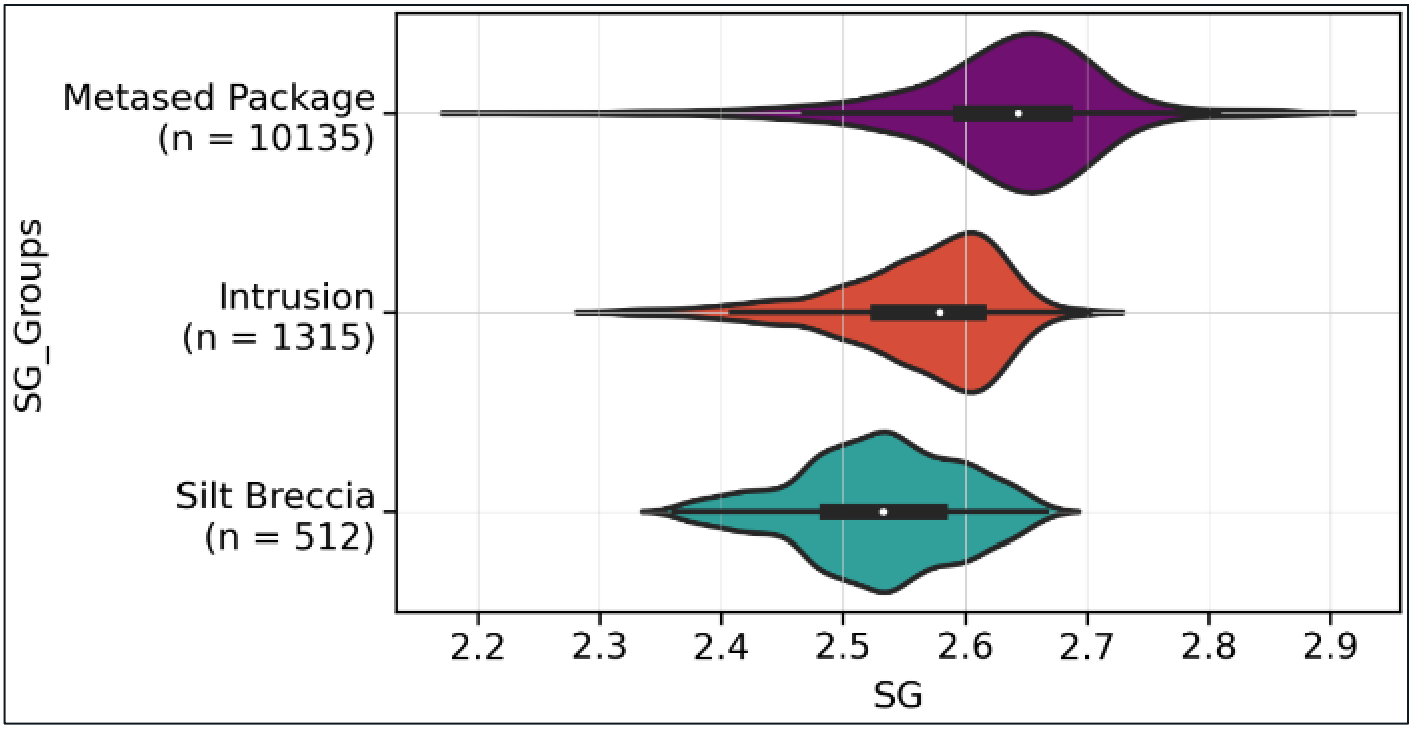
Source: APEX, 2023.

14.4 Exploratory Data Analysis (EDA)

14.4.1 Bulk Density

A total of 14,208 bulk density samples with measurement were available from the Lemhi Gold Project drillhole database. APEX personnel performed exploratory data analysis (EDA) of the bulk density samples available. Three main geologic units showed significant variation in density, Figure 14-2. The median SG value for each geologic unit was used for assigning density for material in the MRE. The EDA resulted in a change in the SG used in the MRE from 2.62 g/cm^3 in the 2021 MRE (Dufresne 2021) for mineralized material and unmineralized material to 2.64 g/cm^3 for metasedimentary package material. 2.58 g/cm^3 for intrusion material, and 2.53 g/cm^3 for silt breccia material, Table 14-3.

Figure 14-2: Constrained Bulk Density for Name from Drillholes.



Source: APEX, 2023.

Table 14-3: Geologic Domain Density Value

Geologic Unit	Median Density Value (g/cm^3)
MetaSed Package	2.64
Intrusion	2.58
Silt Breccia	2.53

14.4.2 Raw Analytical Data

Wireframe constrained assays were back coded in the assay database with rock codes that were derived from intersections of the mineralization solids and drillholes. The basic statistics of mineralization wireframe constrained assays are presented in Table 14-4.

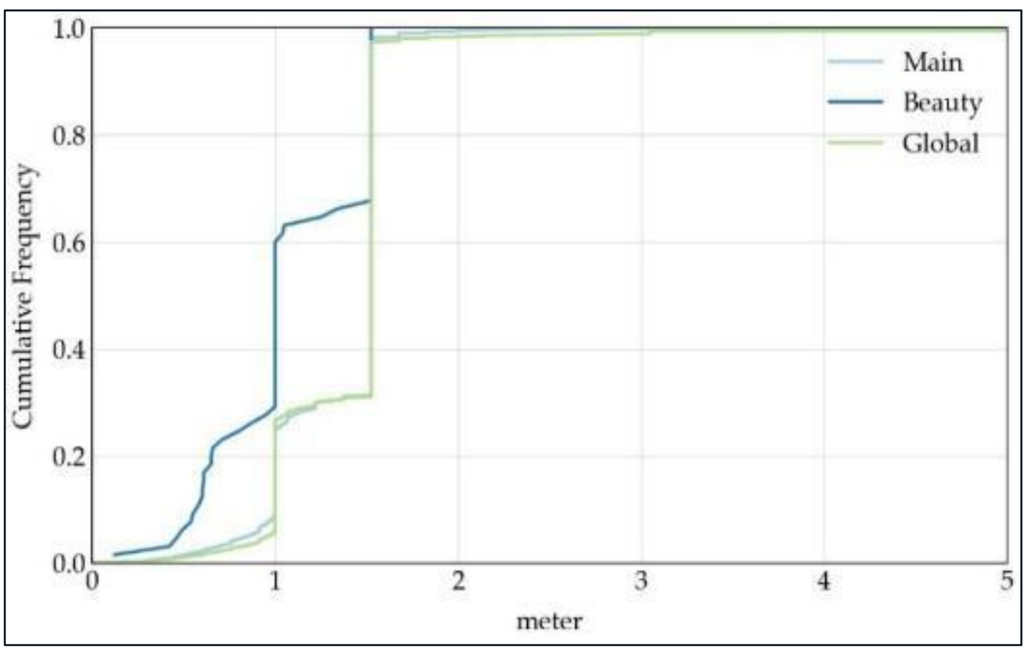
Table 14-4: Raw Gold (g/t) Assay Statistics for the 2023 Lemhi Gold Project Mineral Resource Area

Description	Global	Main	Beauty
Count	64,004	16,226	64
Mean	0.260	0.864	10.435
Median	0.018	0.338	0.351
Standard Deviation	2.143	2.193	55.476
Variance	4.594	4.808	3,077.532
Coefficient of Variation	8.245	2.539	5.316
Minimum	0.003	0.003	0.003
25 Percentile	0.003	0.161	0.089
50 Percentile	0.018	0.338	0.351
75 Percentile	0.120	0.789	0.790
Maximum	441.000	81.100	441.000

14.4.3 Compositing Methodology

Downhole assay sample length shows that sample interval lengths predominantly range from 1.0 to 1.5 m, shown in Figure 14-3 and Table 14-5. A composite length of 2.5 m was selected as the majority of sample interval lengths are equal to, or less than that length.

Figure 14-3: Distribution of Raw Interval Lengths Within the Estimation Domains



Source: APEX, 2023.

Table 14-5: Statistics of the Raw Interval Lengths Within the Estimation Domains.

Description	Global	Main	Beauty
Count	64,004	16,226	64
Mean	1.42	1.36	1.09
Median	1.52	1.52	1.00
Standard Deviation	1.02	0.33	0.37
Variance	1.03	0.11	0.14
Coefficient of Variation	0.72	0.25	0.35
Minimum	0.00	0.00	0.12
25 Percentile	1.00	1.01	0.93
50 Percentile	1.52	1.52	1.00
75 Percentile	1.52	1.52	1.52
Maximum	74.29	10.67	1.52

The length-weighted compositing process starts from the drillhole collar and ends at the bottom of the hole. The final composite intervals along the drillhole, however, cannot cross contacts between estimation domains; therefore, composites extending downhole are truncated when one of these contacts is intersected. A new composite begins at these contacts and continues to extend downhole until the maximum composite interval length is reached, or another truncating contact is intersected.

A balanced compositing method was chosen for the 2023 Lemhi MRE . The balanced compositing approach utilizes the same compositing steps as above but with a variable composite length. The combined length of all samples within each contiguous unit, that lie between two boundary contacts in a single drillhole, is used to determine what composite length to use for that contiguous unit. The composite length is chosen for each contiguous unit to provide a uniform length over that contiguous unit that is closest to the target composite length.

The goal is to achieve a uniform length closest to the target composite length for each contiguous unit. For example, if the length of the contiguous unit is 2.3 m and the target composite length is 1 m, then the choice would be made between 2.3 m/3 = 0.767 m vs 2.3 m/2 = 1.15 m lengths. As 1.15 m is closer to the desired composite length than 0.767 m, the contiguous unit would be split into two composites with lengths of 1.15 m, 1.15 m.

The balanced compositing method provides a similar volume of support over the estimation domain while minimizing the number of short composites and their potential influence in the grade interpolation process. To further reduce the influence of residual (orphan) composites, a minimum compositing length of 1.25 m was enforced.

14.4.4 Grade Capping

To ensure metal grades are not overestimated by including outlier values during estimation, composites are capped to a specified maximum value. Probability plots illustrating each composite's values are used to identify outlier values that appear greater than expected relative to each estimation domain's commodity distribution. Composites identified as potential outliers on the log-probability plots are evaluated in three dimensions (3-D) to determine if they are part of a high-grade trend or not. If identified, outliers are deemed part of a high-grade trend that still requires a grade capping level, the grade capping level used on them may not be as aggressive as the grade capping level used to control isolated high-grade outliers.

Grade capping was completed by assessing the composites within each domain. Table 14-6 indicates the grade capping levels determined using the log-probability plots. Visual inspection of the potential outliers revealed they have no spatial continuity with each other. Therefore, the grade capping levels for commodity as detailed in Table 14-6 are applied to all composites used to calculate the MRE.

Table 14-6: Au Grade Capping Levels Applied to Composites Before Estimation

Mineral Resource Area	Grade Capping Domain	Au Capping Level (g/t)	No. of Composites	No. of Capped Composites
Lemhi	Main	17.3	8935	10
	Beauty	50	27	1

14.4.5 Declustering

It is typical to collect data in a manner that preferentially samples high-value areas over low-value areas. This preferential sampling is an acceptable practice; however, it produces closely spaced data that are likely statistically redundant, which results in under-represented sparse data compared to the over-represented closer-spaced data. Therefore, it is desirable to have spatially representative (i.e., declustered) statistics for global mineral resource assessment and to check estimated grade models. Declustering techniques calculate a weight for each datum that results in sparse data having a higher weight than closely spaced data. The calculated declustering weights allow spatially repetitive summary statistics to be calculated, such as a declustered mean.

Cell declustering is performed globally on all composites within the grade estimation domains, which calculates a declustering weight for each composite. Cell declustering works by discretizing a 3-D volume into cells that are the same size. The sum of the weights of all the composites within the cell must equal one; therefore, the weight assigned to each composite is proportional to the number of composites within each cell. For example, if there are four composites within a cell, they are all assigned a declustering weight of 0.25.

As a rule of thumb, the cell size used to calculate declustering weights will ideally contain one composite per cell in the sparsely sampled areas. Visual evaluation of the sparsely sampled areas in a 3-D visualization software gives a rough idea of this size. Additionally, a high-resolution block model populated with the distance to each block's nearest composite can help guide the declustering of the cell size. The 90-percentile of the distance block model, with a cell size much lower than the final declustering cell size, approximates the optimal cell size. Finally, plotting a series of declustered means for a range of declustering cell sizes will help determine the optimal cell size. The optimal cell size will likely be when the declustered mean in the plot is locally low or high at a cell size that is very close to the two potential cell sizes that were determined from the visual review and calculated 90-percentile distance. Preferential sampling in high-grade zones results in a declustered mean that is likely within a local minimum. In contrast, preferential sampling in low-grade zones results in a declustered mean that is expected within a local maximum.

Calculated declustering weights for the grade estimation domain were constructed. Visual evaluation of the sparsely sampled areas in Micromine suggests similar cell sizes as the 90th-percentiles from the distance block model for each grade estimation domain. Plots comprised of a series of declustered means for a range of declustering cell sizes were utilized to inform the final cell sizes. Table 14-7 details the cell size used, which was very close to the size indicated by the visual evaluation and distance block model.

Table 14-7: Cell Size Used to Calculate Declustering Weights

Mineral Resource Area	Cell Declustering Size (m)
Main	35
Beauty	18

14.4.6 Final Composite Statistics

Summary statistics for the declustered and capped composites contained within the interpreted grade estimation domains, are presented in Table 14-8. The commodity assays within the grade estimation domain generally exhibit coherent individual statistical populations.

Table 14-8: Composite Au (g/t) Statistics for the Lemhi Gold Project Mineral Resource Area

Description	Global	Main	Beauty
Count	8,962	8,935	27
Mean	0.71	0.68	3.17
Median	0.35	0.35	0.53
Standard Deviation	1.49	1.11	9.46
Variance	2.22	1.22	89.53
Coefficient of Variation	2.11	1.63	2.98
Minimum	0.00	0.00	0.02
25th Percentile	0.21	0.21	0.24
50 th Percentile	0.35	0.35	0.53
75 th Percentile	0.71	0.71	1.39
Maximum	50.00	17.30	50.00

Note: Statistics consider declustering weights and capping.

14.5 Variography and Grade Community

Experimental semi-variograms for each domain are calculated along the major, minor, and vertical principal directions of continuity that are defined by three Euler angles. Euler angles describe the orientation of anisotropy as a series of rotations (using a left-hand rule) that are as follows:

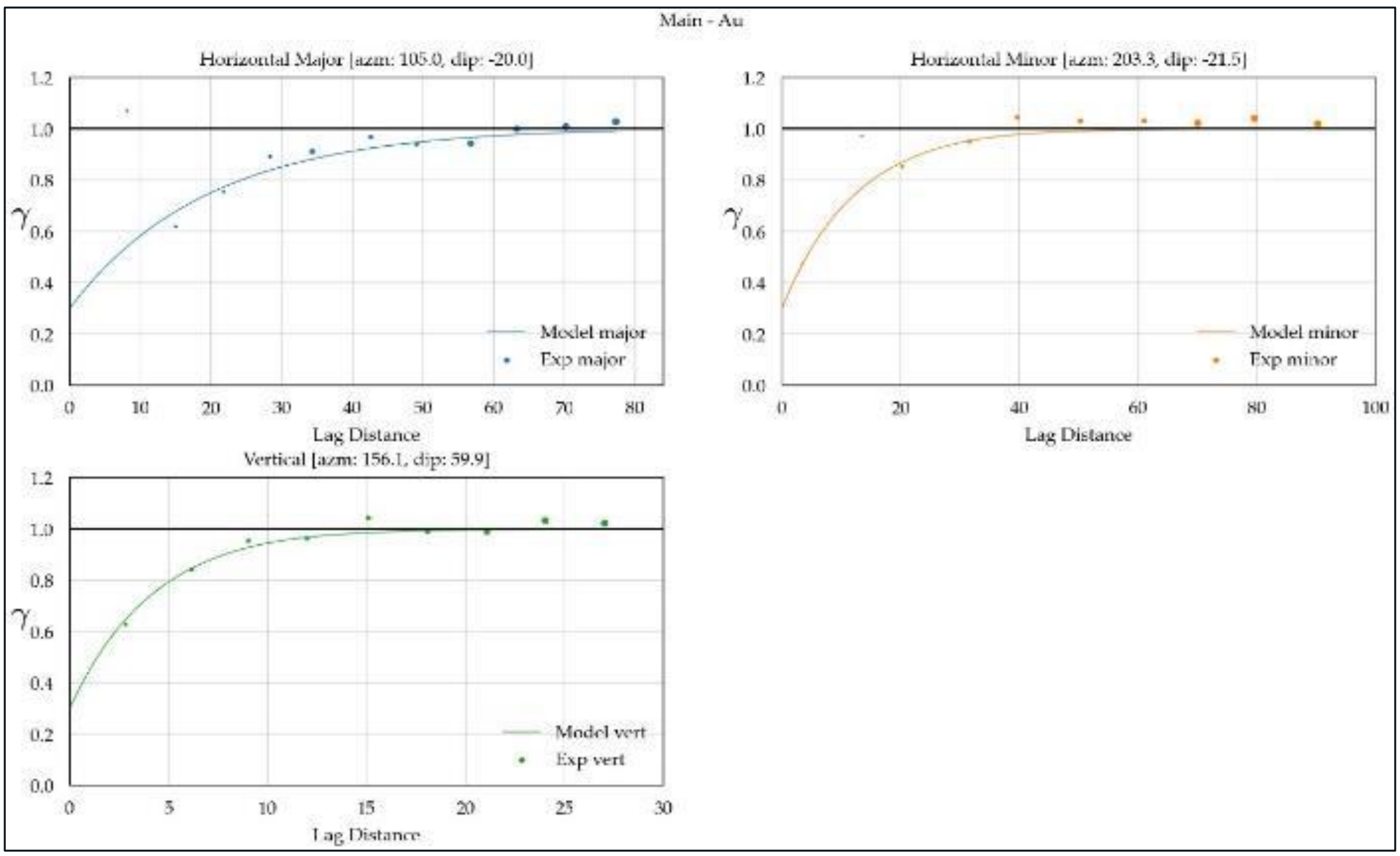
- Angle 1: A rotation the Z-axis (azimuth) with positive angles being clockwise rotation and negative representing counterclockwise rotation

- Angle 2: A rotation about the X-axis (dip) with positive angles being counterclockwise rotation and negative representing clockwise rotation
- Angle 3: A rotation about the Y-axis (tilt) with positive angles being clockwise rotation and negative representing counterclockwise rotation.

APEX personnel calculated standardized experimental correlograms using composites from the main domain area. The orientation of the primary geological controls on mineralization informed the principal directions of continuity upon which the variograms were calculated. Figure 14-4 illustrates the gold variogram modeled using composites from the Main Zone domain. Table 14-9 details the variogram parameters used for kriging within each grade estimation domain.

During grade estimation, the standardized variogram model is scaled to the variance of the composites within each individual grade estimation domain. The scaled nugget effect and covariance contributions for each variogram structure are used as input parameters for ordinary kriging. The ranges used for each of the mineralized zones are not changed from the standardized variogram model. LVA is used during grade estimation to define the orientation of the variogram on a per-block basis, which is explained in more detail in Section 14.7.

Figure 14-4: Main Au Variogram



Source: APEX, 2023.

Table 14-9: Au Variogram Parameters

Domain	Ang 1	Ang 2	Ang 3	Sill	C0	Structure 1					Structure 2				
						Type	C1	Ranges (m)			Type	C2	Ranges (m)		
								Major	Minor	Vert			Major	Minor	Vert
Main	Au	105	-20	23	0.610	exp	1.54	55	35	12	sph	0.10	75	40	12
Beauty	Au	105	-20	23	18.730	exp	70.24	50	40	20	sph	4.68	75	40	25

Note: Abbreviations are as follows:
1. C0 – nugget effect;
2. C1 – covariance contribution of first structure;
3. C2 – covariance contribution of second structure;
4. Vert – vertical;
5. sph – spherical variogram;
6. exp – exponential variogram.

14.6 Block Model

14.6.1 Block Model Parameters

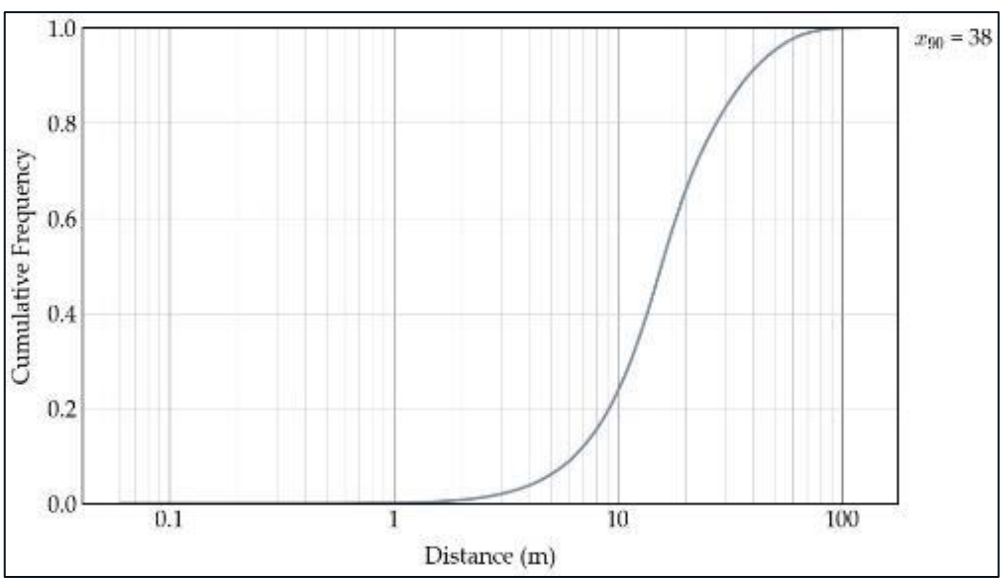
The block model used for the calculation of the 2023 Lemhi MRE fully encapsulates the estimation domains used for resource estimation described in Section 14. 3. When determining block model parameters, data spacing is the primary consideration. Additionally, the volume of the 3-D estimation domain wireframes needs to be adequately captured and potential mining equipment parameters need to be considered.

The data spacing of irregularly spaced drilling can be approximated by calculating the 90th percentile of a high-resolution block model of the distance from each block's centroid to the nearest sample. Estimation errors are introduced when kriging is used to estimate a grade for blocks with a size larger than 25% of the data spacing. As illustrated in Figure 14-5, the 90th percentile is 38 m. A block size of 2.5 m by 2.5 m by 2.5 m was selected, which is less than 25% of the approximated data spacing. A 5 m block model was evaluated; however, it did not adequately capture smaller scale features in the estimation domain, which were modelled by the 2.5 m block model. Table 14-10 details the grid definition used for the block model.

A block factor (BF) that represents the percentage of each block's volume that lies within the LG and HG lodes is calculated and used to:

- flag the dominant domain, by volume, for each block
- calculate the percentage of mineralized material and waste for each block.

Figure 14-5: Cumulative Frequency Plot Illustrating the Distance from Each Block's Centroid to the Nearest Composite Sample in Metres



Source: APEX, 2023.

Table 14-10: 2023 Lemhi MRE Block Model Definition

Direction	Origin*	No. of Blocks	Block Size (m)
X	499,192	774	2.5
Y	428,885	686	2.5
Z	1,130	268	2.5
Rotation	No rotation		

Notes: *Origin for a block model in RMSP represents the coordinates of the centroid of the block with minimum X, Y, and Z.

14.6.2 Volumetric Checks

A comparison of wireframe volume versus block model volume is performed to ensure there is no considerable over- or understating of tonnages (Table 14-11). The calculated BF for each block is used to scale its volume when calculating the total volume of the block model.

Table 14-11: Wireframe Versus Block Model Volume Comparison

Domain	Wireframe Volume (m^3)	Block Model Volume with Block Factor (m^3)	Volume Difference (%)
Main	24,195,534	24,191,379	0.02%
Beauty	80,660	71,014	12.72%

14.7 Grade Estimation Methodology

14.7.1 Grade Estimation of Mineralized Material

Ordinary kriging (OK) was used to estimate commodity grades for the 2023 Lemhi MRE block model. Only blocks that intersect the mineralization domain were estimated for commodity grades.

Estimation of blocks is completed with LVA, which uses different rotation angles to define the principal directions of the variogram model and search ellipsoid on a per-block basis. Blocks within the grade estimation domain are assigned rotation angles using a trend surface wireframe. This method allows structural complexities to be reproduced in the estimated block model. Variogram and search ranges are defined by the variogram model described in Section 14.5.

The boundaries between the estimation domains and country rock are treated as hard boundaries, meaning data from outside the domain cannot be used to inform the grade estimate inside the domain.

The correct volume-variance relationship was enforced by restricting the maximum number of conditioning data (composites) within ellipsoid sectors, the maximum number of composites per drillhole and the maximum number of conditioning data per search ellipsoid sector used. These restrictions are implemented to ensure the grade estimation models are not over smoothed and to limit the effect of high-grade samples, which would lead to inaccurate estimation of global tonnage and grade. The parameters used to enforce the right volume-variance relationship cause local conditional bias, however, ensure the global estimate of grade and tonnages is more accurately estimated.

To ensure that all blocks within the grade estimation domains are estimated and the correct volume-variance relationship is achieved, a three-pass method was used for each domain. Each pass uses the same variogram model, as modelled and detailed in Section 14.5, however, different search ellipsoid configurations are used, as illustrated in Table 14-12.

Different search ellipsoid configurations are used to control the smoothing inherit in kriging and manage influence of high-grade samples to achieve the correct volume-variance relationship. The three passes are normally not required since the blocks estimated during those passes are distant from composites, however, due to structural complexities and the limitation of search ellipses not being able to look along the trend of the folds, they were utilized in this case.

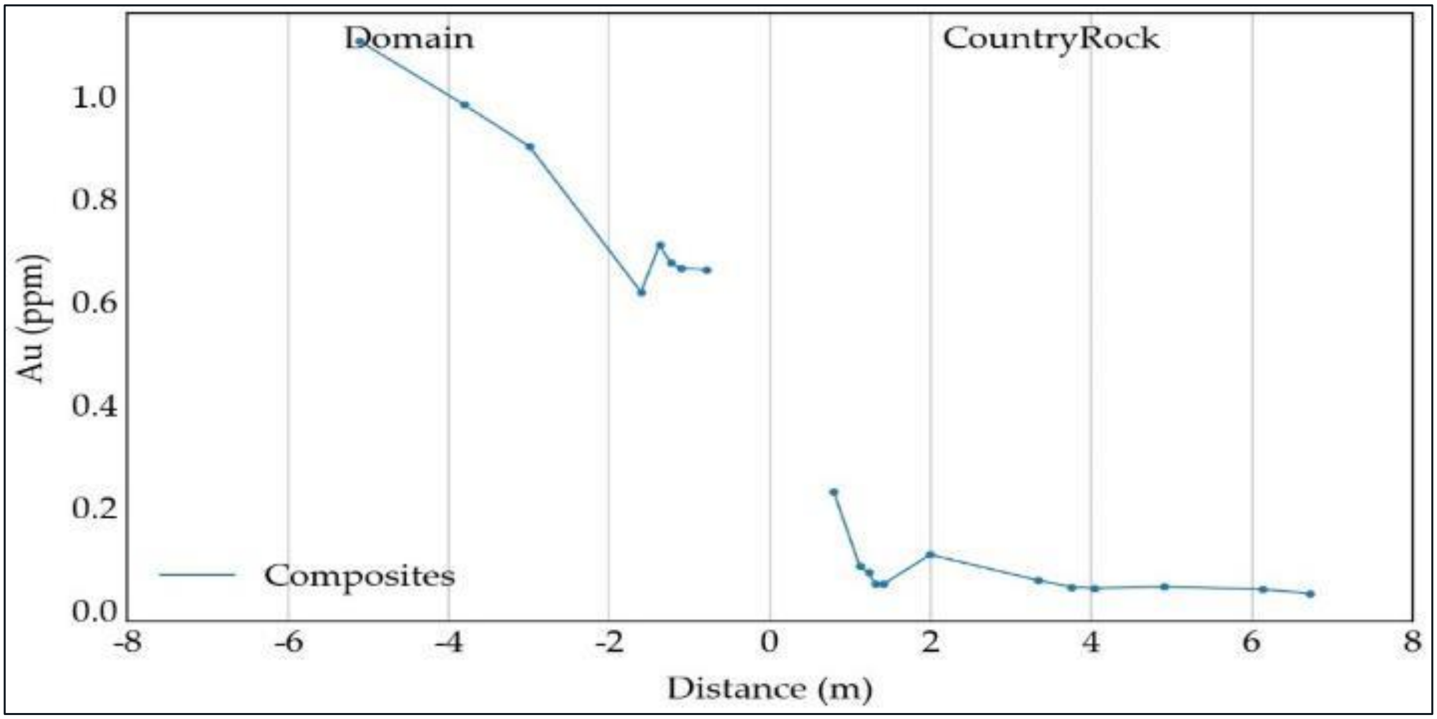
Table 14-12: 2023 Lemhi MRE Block Model Gold Interpolation Parameters

Estimation Domains	Pass	Max Search Ranges (m)			No. of Ellipse Sectors	Min No. of Comps	Max No. of Comps	Max No. of Comps per DH
		Major	Minor	Vertical				
Main	1	30	15	5	1	1	30	2
	2	75	40	10	1	1	30	3
	3	150	80	24	1	1	30	4
Beauty	1	30	15	5	1	1	20	2
	2	50	25	10	1	1	20	3
	3	100	80	15	1	1	20	2

14.8 Grade Estimation of Waste Material

The open pit optimization for evaluating reasonable prospects for future economic extraction relies on a whole block grade, therefore blocks that contain more than or equal to 0.8% waste by volume are diluted by estimating a waste gold value that is volume-weight averaged with the estimated gold grade. It is desired that the behavior of gold at the boundary between the estimation domain and waste beyond its boundary is reproduced. The nature of gold mineralization at the mineralized/waste contact is evaluated and used to determine a window to flag composites that are used to condition a waste gold estimate for blocks containing waste material. As illustrated in Figure 14-6, gold behaves in a statistically semi-soft manner, where the grade of the composite centroids flagged within an estimation domain transitions from mineralized to waste over a short window. Composites within a window of 2.5 m into waste and 2. 5 m into the estimation domain are used to estimate a waste gold value.

Figure 14-6: Contact Analysis of Gold Grade at the Boundary Between the 2023 Lemhi MRE Estimation Domains and Waste



Source: APEX, 2023.

14.9 Model Validation

14.9.1 Statistical Validation

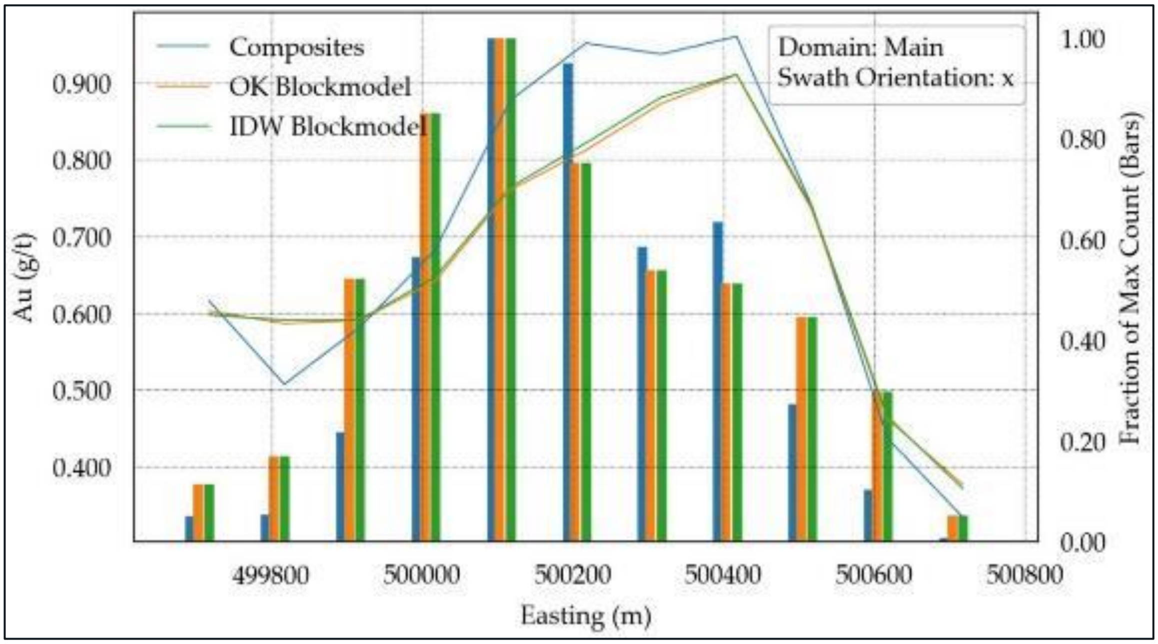
APEX personnel performed three varying statistical validation methods to ensure the estimated block model honours the input drillhole data. Swath plots are used to check that the block model honours directional trends, and volume-variance analysis is used to check that the proper quantity of minerals above varying cut-off grades is being estimated.

14.9.1.1 Direction Trend Analysis Validation

Swath plots verify that the estimated block model honours directional trends and identifies potential areas of over- or under-estimation of grade. The swath plots are generated by calculating the average metal grades of composites, and the OK estimated blocks. The block model evaluated comprises both the main and main-hg domains, that way, the entire zone can be evaluated overall. Examples of the swath plots used to validate the mineral resource estimate are illustrated in Figure 14-7 to Figure 14-9.

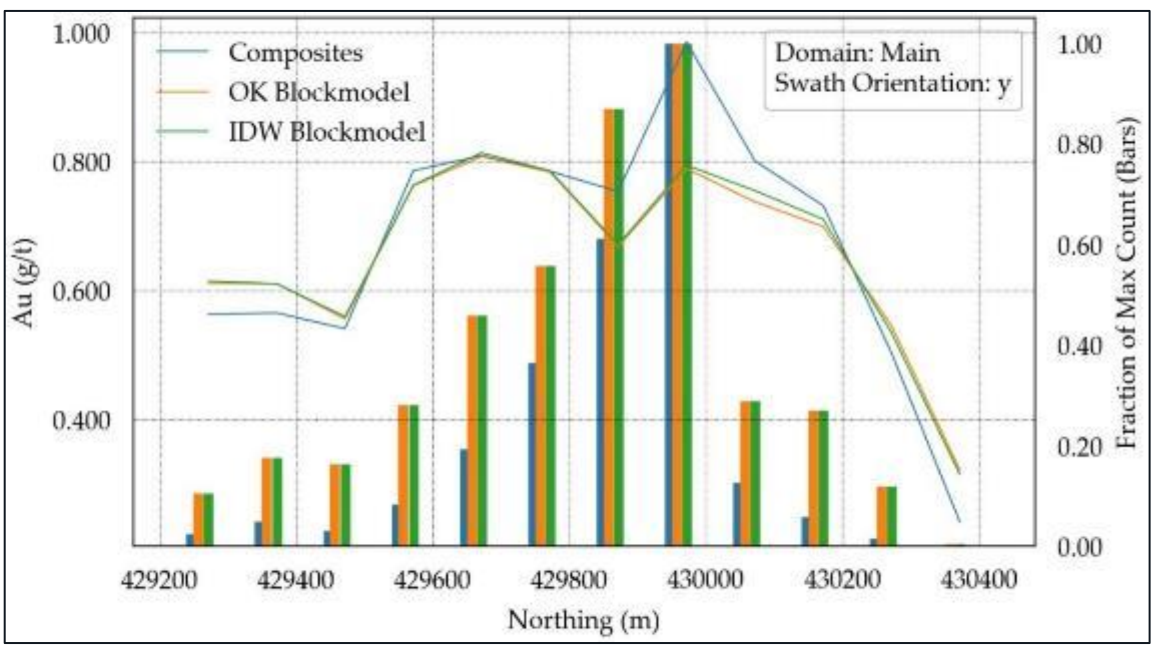
Overall, the block model compares well with the composites. There is some observed local over- and under-estimation. Due to the limited number of conditioning data available for the grade estimation in those areas, this result is expected.

Figure 14-7: 2023 Lemhi MRE Easting Au Swath Plot for the Main Zone Domain



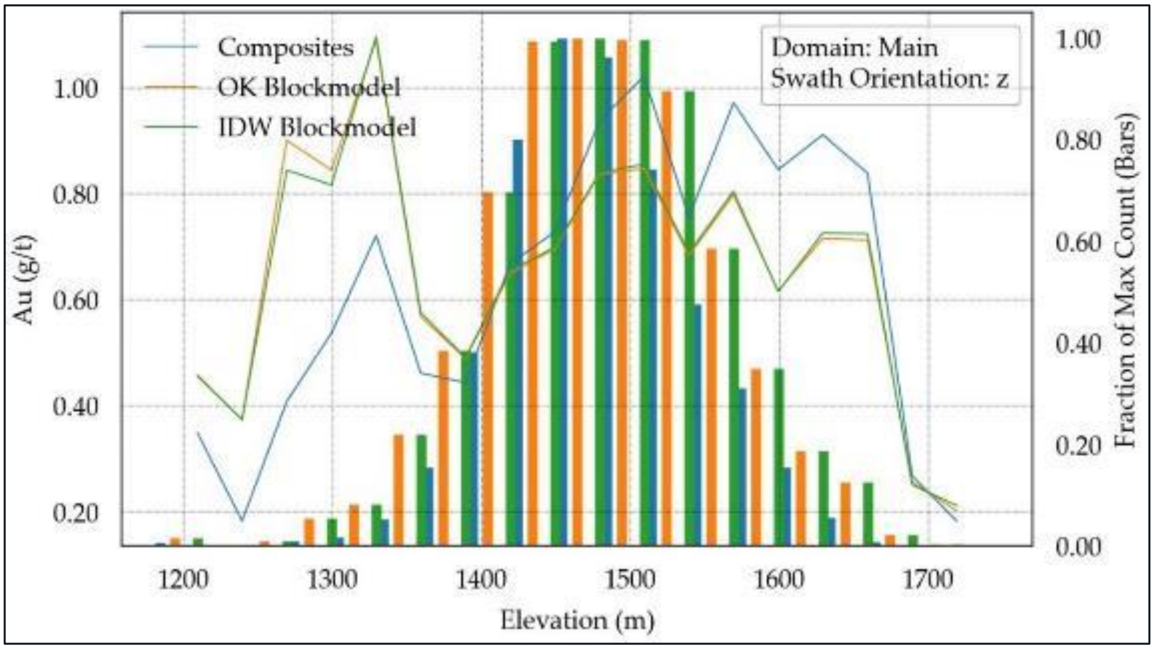
Source: APEX, 2023.

Figure 14-8: 2023 Lemhi MRE Northing Au Swath Plot for the Main Zone Domain



Source: APEX, 2023.

Figure 14-9: 2023 Lemhi MRE Elevation Au Swath Plot for the Main Zone Domain



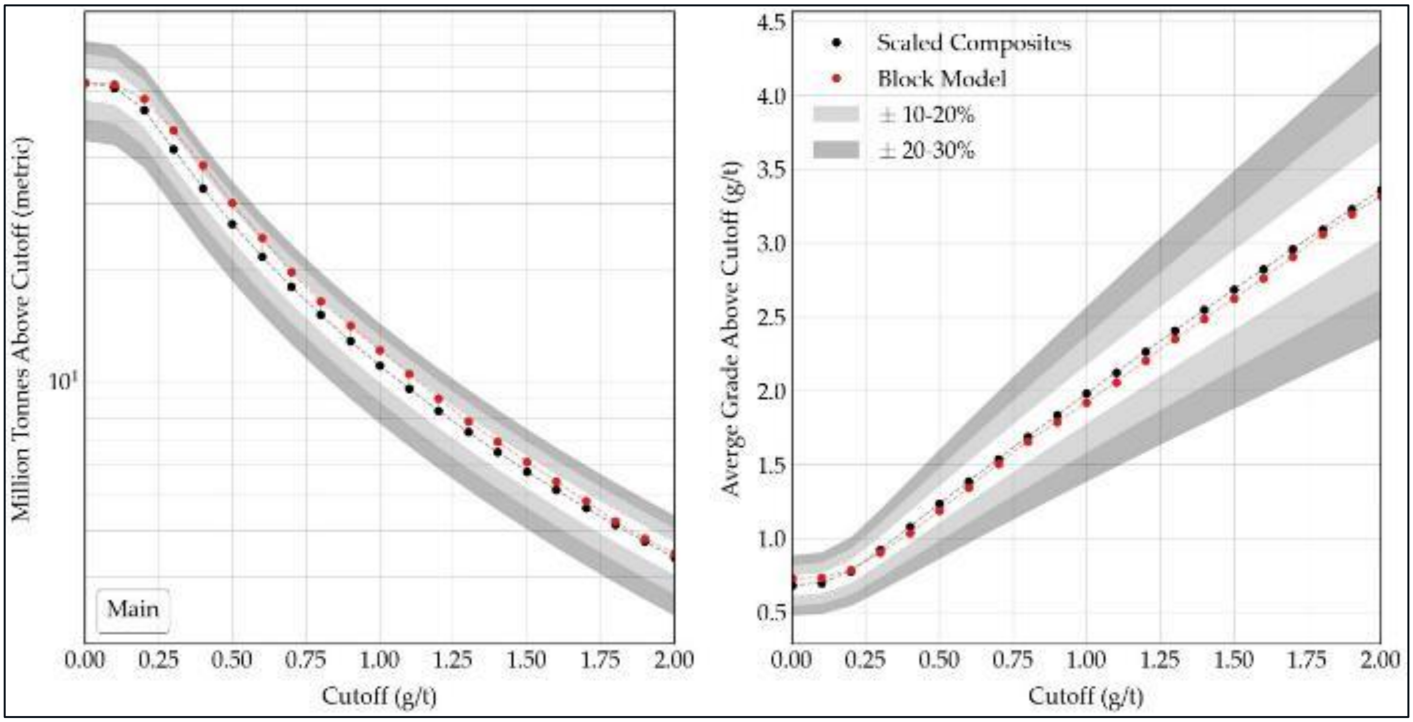
Source: APEX, 2023.

14.9.1.2 Volume-Variance Analysis Validation

Smoothing is an intrinsic property of kriging, and as described in Section 14. 7 volume-variance corrections are used to help reduce its effects. To verify that the correct level of smoothing is achieved, theoretical histograms that indicate each estimated metal's anticipated variance and distribution at the selected block model size are calculated. The scaled composite histograms are used to calculate expected tonnages and expected grades above a series of cut-off grades. Comparing the curves of the expected versus estimated values helps ensure the correct volume of mineral resource above varying cut-offs is being estimated.

Overall, the estimated grades within each domain illustrate the desired amount of smoothing. The gold estimated within the Main Zone domain, the primary host of the metal, achieved the desired amount of smoothing, see Figure 14-10. Gold estimation demonstrates adequate smoothing at the desired cut-off, additional modifications to the search strategy would introduce excessive bias.

Figure 14-10: Volume-variance Analysis for Main Zone Grade Estimation Domain

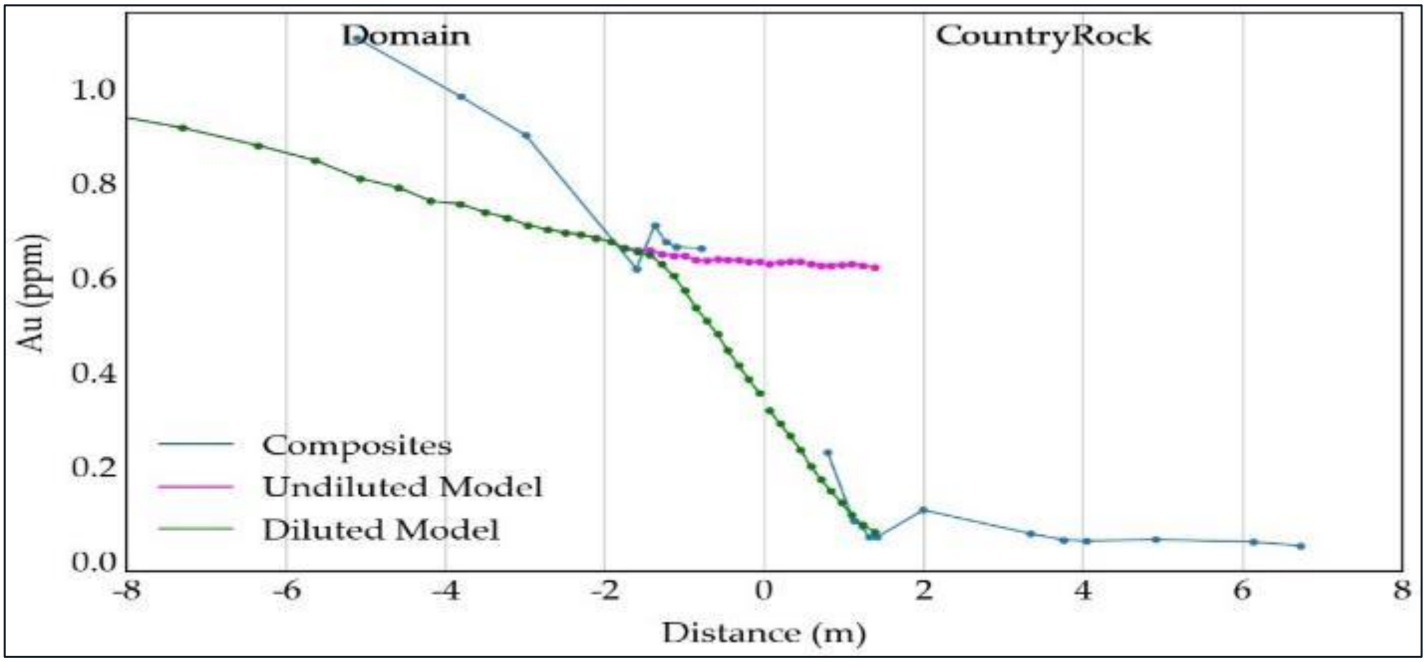


Source: APEX, 2023.

14.9.1.3 Contact Analysis Validation

As described in Section 14. 7, blocks within the 2023 Lemhi MRE block model that contain more than or equal to 0.8% waste by volume are diluted using the estimated waste gold and mineralized gold values. Ideally, the nature of gold mineralization at the mineralization/waste contact observed in the composites is reproduced in the block model. A contact analysis plot checking contact profile reproduction is illustrated in Figure 14-11. The mineralization/waste contact profile is adequately reproduced with some under-estimation into mineralized material.

Figure 14-11: Contact Analysis for the mineralized material to Waste Boundary

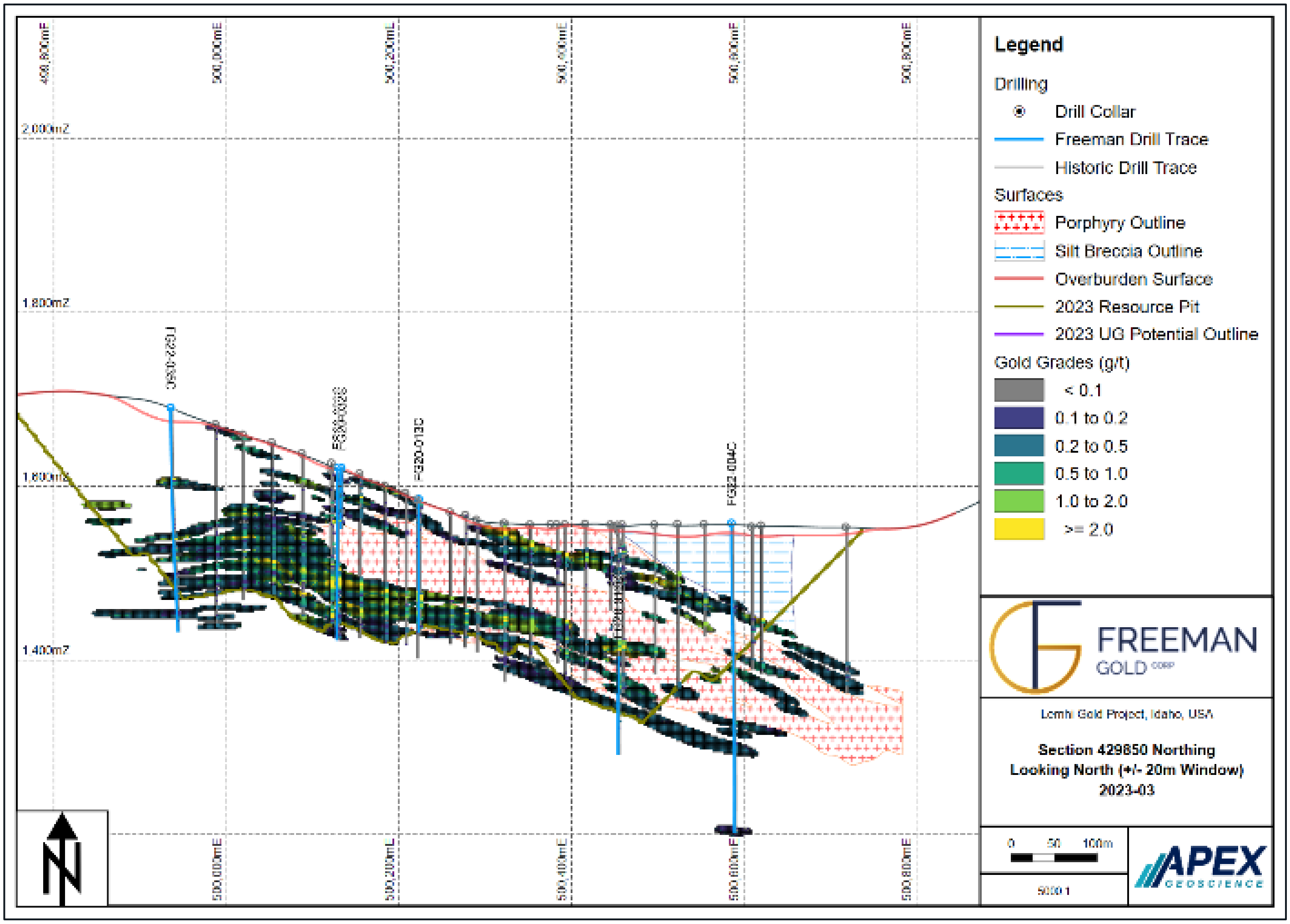


Source: APEX, 2023.

14.9.2 Visual Validation

APEX personnel visually reviewed the estimated block model grades in cross-sectional views comparing the estimated block model grades to the input composited drillhole assays and the modelled mineralization trends. The block model compares very well to the input compositing data. Local high- and low-grade zones within the mineral resource areas are reproduced as desired, and the LVA adequately maintains variable mineralization orientations. Figure 14-12 illustrates the grade estimation blocks used for the MRE.

Figure 14-12: **East-West Cross-section at 429850 Northing (Looking North) Illustrating Estimated Gold Grades and the Constraining Open Pit Shell Outline (Brown)**



Source: APEX, 2023.

14.10 Mineral Resource Clarification

14.10.1 Classification Definitions

The 2023 Lemhi MRE discussed in this technical report has been classified in accordance with guidelines established by the CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29, 2019, and CIM "Definition Standards for Mineral Resources and Mineral Reserves" dated May 14, 2014.

A measured mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with confidence sufficient to allow the application of modifying factors to support detailed mine planning and final evaluation of the economic viability of the deposit. Geological evidence is derived from detailed and reliable exploration, sampling and testing and is sufficient to confirm geological and grade or quality continuity between points of observation. A measured mineral resource has a higher level of confidence than that applying to either an indicated mineral resource or an inferred mineral resource. It may be converted to a proven mineral reserve or to a probable mineral reserve.

An indicated mineral resource is that part of a mineral resource for which quantity, grade or quality, densities, shape, and physical characteristics are estimated with sufficient confidence to allow the application of modifying factors in sufficient detail to support mine planning and evaluation of the economic viability of the deposit. Geological evidence is derived from adequately detailed and reliable exploration, sampling and testing and is sufficient to assume geological and grade or quality continuity between points of observation. An indicated mineral resource has a lower level of confidence than that applying to a measured mineral resource and may only be converted to a probable mineral reserve.

An inferred mineral resource is that part of a mineral resource for which quantity and grade or quality are estimated on the basis of limited geological evidence and sampling. Geological evidence is sufficient to imply but not verify geological and grade or quality continuity. An inferred mineral resource has a lower level of confidence than that applying to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

14.10.2 Classification Methodology

The 2023 Lemhi MRE is classified as measured, inferred, and indicated according to the CIM definition standards. The classification of the indicated and inferred mineral resources is based on geological confidence, data quality, and grade continuity of the data. The most relevant factors used in the classification process were the following:

- Density of conditioning data
- Level of confidence in drilling results and collar locations
- Level of confidence in the geological interpretation
- Continuity of mineralization
- Level of confidence in the assigned densities.

Mineral resource classification was determined using a multiple-pass strategy that consists of a sequence of runs that flag each block with the run number a block first meets a set of search restrictions. With each subsequent pass, the search restrictions decrease, representing a decrease in confidence and classification from the previous run. For each run, a search ellipsoid is centered on each block and orientated in the same way described in Section 14. 7. This process is completed separately from grade estimation.

Table 14-13 details the range of the search ellipsoids and the number of composites that must be found within the ellipse for a block to be flagged with that run number. The runs are executed in sequence from run 1 to run 2 to run 3. Table 14-14 details special data constraints utilized for each sequential run. As discussed in Section 12 and 14.2.1, grades that are influenced by FMC drilling have an increased uncertainty in the results, which is accounted for in determining the classification confidence level of the MRE. Classification is then determined by relating the run number that each block is flagged as to measured (run 1), indicated (run 2) or inferred (run 3). The measure pass (run 1) search strategy only considers non-FMC drilling. However, some blocks that met the run 1 search criteria considered composites from FMC drilling during estimation. Therefore, APEX identified all these blocks and downgraded them to Indicated. Indicated pass (run 2) search strategy was limited to non-FMC drilling; however, the metal estimation of indicated blocks were allowed to consider composites from FMC drillholes. The inferred pass (run 3) had no additional data constraints. Figure 14-13 illustrates the classification model used for the MRE.

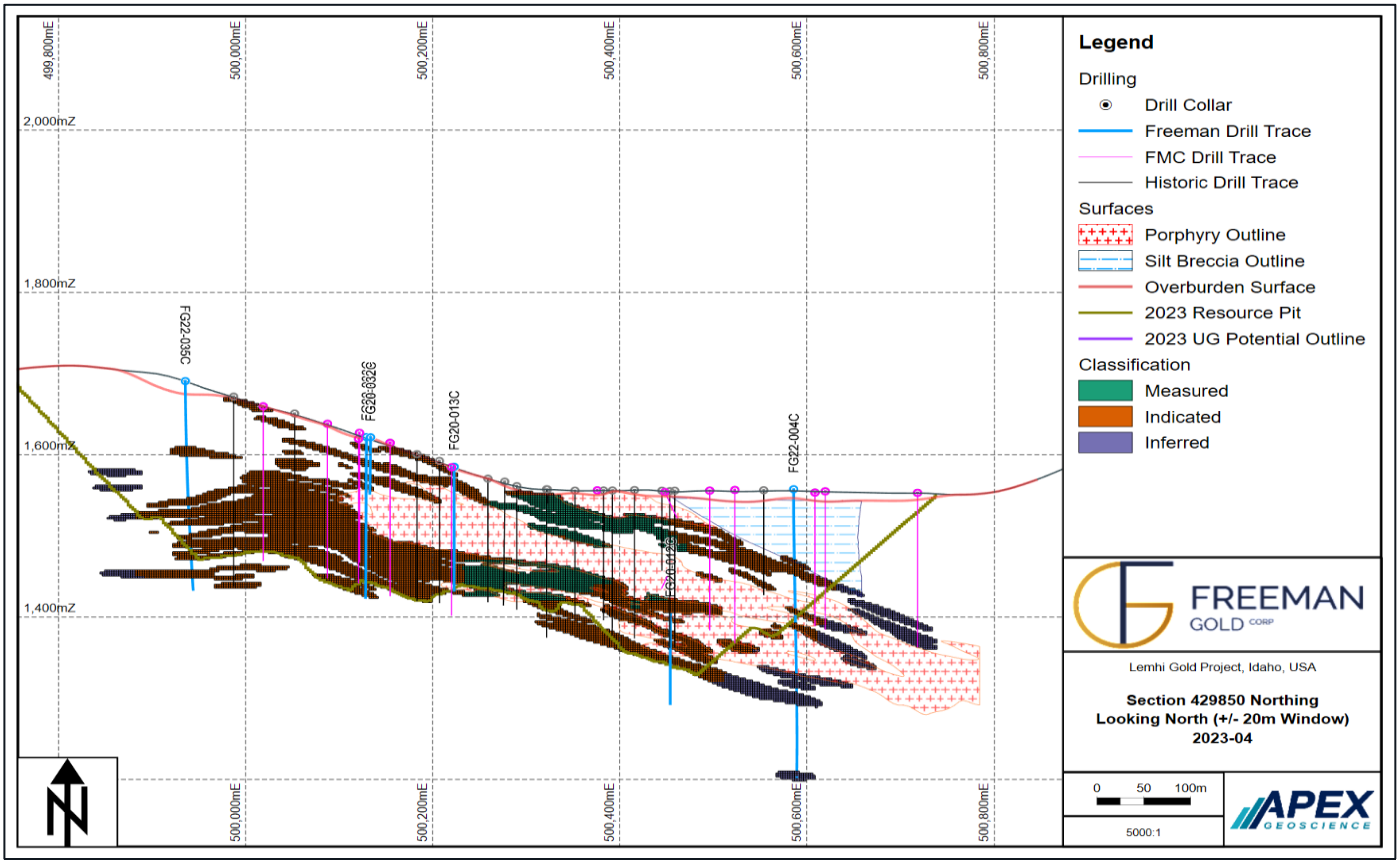
Table 14-13: Search Restrictions Applied During Each Run of the Multiple-pass Classification Strategy

Mineral Resource Area	Pass	Classification	Minimum No. of Drillholes	Ranges (m)		
				Major	Minor	Vertical
Main	1	Measured	5	50	35	15
Main	2	Indicated	3	75	60	25
Main, Beauty	3	Inferred	2	120	120	60

Table 14-14: Special Data Restrictions Applied to Each Classification Strategy

Domain	Classification	Special Data Constraints
Main	Measured	Data search does not consider FMC drilling, Blocks influenced by FMC drilling that meet search restrictions are downgraded to Indicated
Main	Indicated	Data search does not consider FMC drilling
Main, Beauty	Inferred	No Special Data Constraints

Figure 14-13: East-West Cross-section at 429850 Northing (looking North) Illustrating the Resource Classification Model Resource Constraining Pit Shell (Brown Line) for the 2023 Lemhi MRE



Source: APEX, 2023

14.11 Evaluation of Reasonable Prospects for Eventual Economic Extraction

14.11.1 Open Pit Parameters

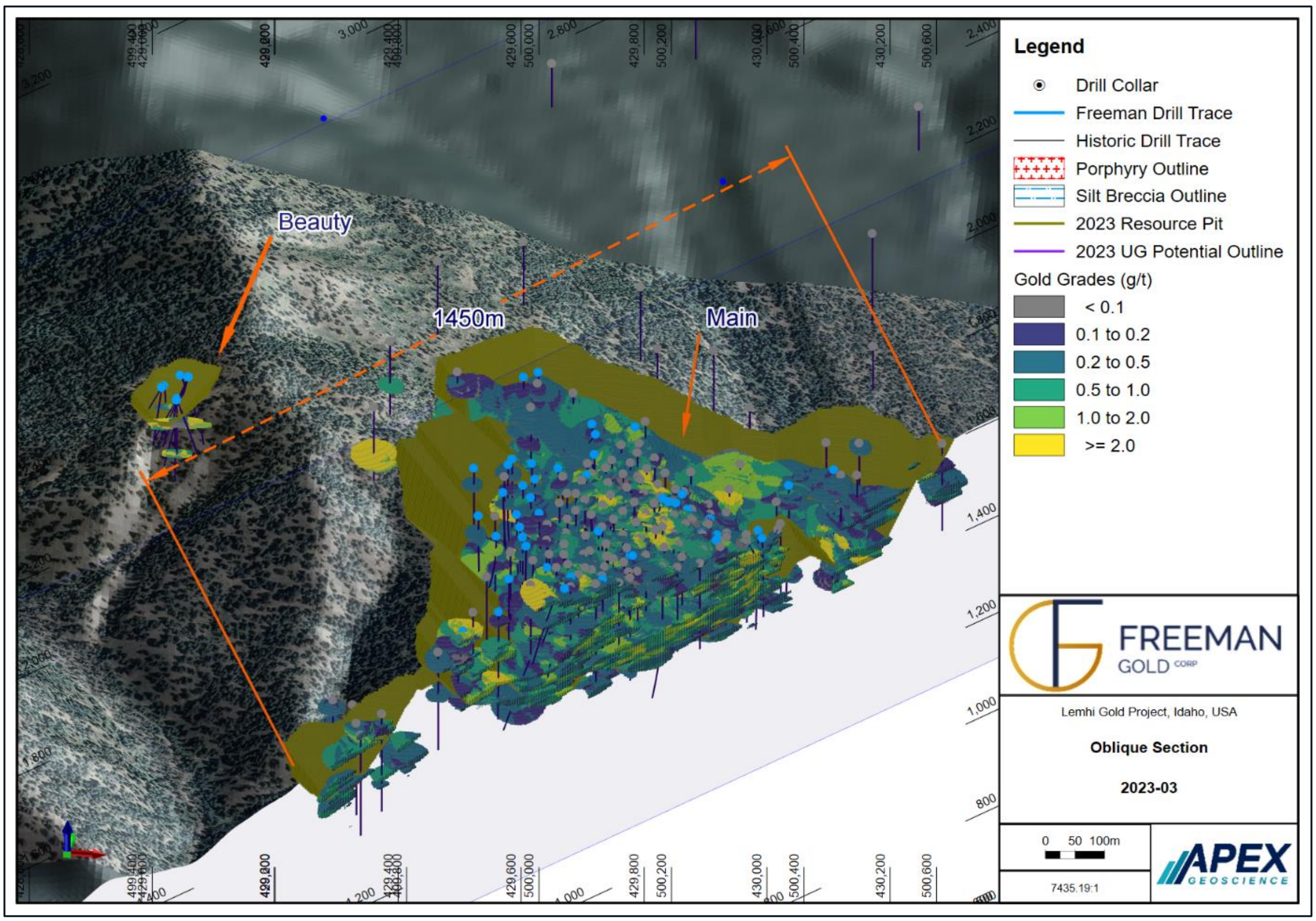
To demonstrate that the Lemhi Gold Property has the potential for future economic extraction, the Mineral Resource block model was subjected to several pit optimization scenarios to determine the prospect for eventual economic extraction. Pit optimization was performed with Deswik Pseudoflow.

The authors consider the parameters presented in Table 14-15 appropriate to evaluate the reasonable prospect for potential future economic extraction at the Lemhi Gold Project for the purpose of providing an MRE. The resulting pit shell is used to constrain the MRE stated in this report. Figure 14-14 illustrates the 2023 Lemhi MRE block model and the open pit shells used to constrain the MRE.

Table 14-15: Parameters Used for Resource Constraining Pit

Parameters	Unit	Value
Gold price	US$/oz	1750
NSR royalty	%	1.0
Exchange rate	US$/C$	0.77
Gold recovery VAT/HL	%	97 / 75
Mining cost – waste	US$/t mined	2.00
Mining cost – mineralized	US$/t mined	2.10
Processing cost VAT/HL	US$/t milled	8.00 / 2.40
General and administration cost	US$/t milled	2.00
Pit slope	degrees	50

Figure 14-14: 3-D Slice View of the 2023 Lemhi MRE Block Model and Resource Pit Shell



Source: APEX, 2023.

14.11.2 Out-of-Pit Mineral Resource Parameters

The CIM guidelines for mineral resources and mineral reserves require that a mineral resource be that part of a mineral deposit with reasonable prospects for eventual economic extraction. For the 2023 Lemhi underground MRE, the shrinkage stoping method was selected.

The calculated cut-off of 1.50 g/t Au was selected in reporting the underground mineral resource in the 2023 resource estimates. To isolate small areas of the resource that would not reasonably be minable in an open stope mining method, the underground mineral resources below the resource open pit are constrained by wireframe solids that encapsulate contiguous 2.5 m x 2.5 m x 2.5 m underground blocks that are above the 1.50 g/t Au cut-off with a volume greater than 1,400 m^3 and only in areas that showed continuity of mineralized grade.

14.11.3 Mineral Resource Estimate

The 2023 Lemhi MRE is reported in accordance with the CSA NI 43-101 rules for disclosure and has been estimated using the CIM “Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines” dated November 29, 2019, and CIM “Definition Standards for Mineral Resources and Mineral Reserves” dated May 10, 2014. The effective date of the mineral resource is January 17, 2023.

Mineral resource modelling was conducted in the UTM coordinate system relative to the North American Datum (NAD) 1983, and Idaho State Plane Central FIPS 1102 (EPSG:6448) The mineral resource block model utilized a SMU block size of 2.5 m (X) by 2.5 m (Y) by 2.5 m (Z) to honour the mineralization wireframes. The percentage of the volume of each block below the top of bedrock surface and within each mineralization domain was calculated using the 3-D geological models and a 3-D topographic surface model. The Au grades were estimated for each block using ordinary kriging with LVA to ensure grade continuity in various directions is reproduced in the block model. The MRE is reported as undiluted within a series of optimized pit shells. Details regarding the methodology used to calculate the MRE are documented in this technical report section.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, market, or other relevant issues. The quantity and grade of reported inferred resource is uncertain in nature and there has not been sufficient work to define the inferred mineral resource as an indicated or measured mineral resource.

The cut-off of 0.35 g/t Au was selected in reporting the in-pit constrained mineral resource and the cut-off of 1.5 g/t Au was selected for reporting the out-of-pit constrained mineral resource in the 2023 Mineral Resource Estimate using the 2.5 m (X) x 2.5 m (Y) x 2.5 m (Z) block size model, see Table 14-16.

Table 14-16: 2023 Lemhi Gold Project Mineral Resource Estimate (1-8)

Au Cut-off (g/t)	Zone	RPEEE Scenario	Classification	Tonnes	Au (oz)	Au Grade (g/t)	Au Grade (oz/st)
0.35	Main & Beauty	Open Pit	Measured	4,469,000	168,800	1.15	0.033
0.35	Main & Beauty	Open Pit	Indicated	25,553,000	819,300	0.98	0.029
0.35	**Main & Beauty**	**Open Pit**	**M&I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35	**Main & Beauty**	**Open Pit**	**Inferred**	**7,338,000**	**234,700**	**1.01**	**0.029**
1.5	Main & Beauty	Under Ground	Inferred	296,000	21,300	2.27	0.066
0.35/1. 5	Main & Beauty	Combined	Measured	4,469,000	168,800	1.15	0.033
0.35/1. 5	Main & Beauty	Combined	Indicated	25,553,000	819,300	0.98	0.029
0.35/1. 5	**Main & Beauty**	**Combined**	**M&I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35/1. 5	**Main & Beauty**	**Combined**	**Inferred**	**7,634,000**	**256,000**	**1.04**	**0.030**

Notes:
1. Contained tonnes and ounces may not sum due to rounding.
2. Mr. Michael Dufresne, P. Geol., P. Geo. of APEX Geoscience Ltd., who is deemed a qualified person as defined by NI 43-101 is responsible for the completion of the updated mineral resource estimation.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
5. The inferred mineral resource in this estimate has a lower level of confidence than that applied to an Indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could potentially be upgraded to an indicated mineral resource with continued exploration.
6. The mineral resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
7. The constraining pit optimization parameters assumed US$1,750/oz Au sale price, NSR Royalty of 1%, US$2. 10/t mineralized and US$2. 00/t waste material mining cost, 50° pit slopes, a VAT process cost of US$8. 00/t, HL process cost of US$2. 40/t and a general and administration (G&A) cost of US$2.00/t.
8. The effective date of the mineral resources Estimate is March 15, 2023.

14.11.4 Mineral Resource Sensitivity

Mineral resources can be sensitive to the selection of the reporting cut-off grade. For sensitivity analyses, other cut-off grades are presented for review. Mineral resources at various cut-off grades are presented for the in-pit and out-of-pit constrained mineral resources in Table 14-17 and Table 14-18, respectively.

Table 14-17: Sensitivities of In-pit-Constrained Mineral Resource Estimate with the Current Resource Highlighted

Classification	Au Cut-off (g/t)	Tonnes	Au (oz)	Au Grade (g/t)	Au Grade (ozt/st)
Measured	0	6,159,000	182,100	0.88	0.026
	0.15	6,040,000	181,700	0.90	0.026
	0.2	5,740,000	180,000	0.94	0.028
	0.3	4,849,000	172,800	1.08	0.032
	0.35	4,469,000	168,800	1.15	0.033
	0.4	4,119,000	164,600	1.22	0.035
	0.5	3,480,000	155,400	1.36	0.040
Indicated	0	35,771,000	898,600	0.76	0.022
	0.15	34,891,000	895,400	0.77	0.023
	0.2	32,856,000	883,800	0.81	0.024
	0.3	27,917,000	844,000	0.92	0.027
	0.35	25,553,000	819,300	0.98	0.029
	0.4	23,174,000	790,700	1.04	0.030
	0.5	18,913,000	729,100	1.18	0.035
Inferred	0	10,915,000	249,500	0.70	0.020
	0.15	10,692,000	248,700	0.71	0.021
	0.2	9,862,000	244,000	0.76	0.022
	0.3	8,118,000	230,000	0.88	0.026
	0.35	7,300,000	221,400	0.94	0.027
	0.4	6,564,000	212,600	1.00	0.029
	0.5	5,053,000	190,700	1.17	0.034

Table 14-18: Sensitivities of Out-of-pit Constrained Mineral Resource Estimate

Classification	Au Cut-off (g/t)	Tonnes	Au (oz)	Au Grade (g/t)	Au Grade (ozt/st)
Inferred	1	620,000	34,200	1.74	0.051
	1.5	296,000	21,300	2.27	0.066
	2	136,000	12,500	2.95	0.086
	2.5	83,000	8,700	3.41	0.100

14.12 Risks, Uncertainty, and Opportunity in the Mineral Resource Estimate

The Lemhi Property carries risks inherent in utilizing significant amounts of historical drilling. Specific risks center on the poor reproducibility of assay results from the 2012 LGT core twinning program as compared with historical RC hole results. Confirmation drilling completed in 2012 by LGT included twin holes of historic drill holes with both core and RC drilling methods. The results from the LGT twin holes indicate that 2012 core drilling returned a number of erratic and a few lower grade intersections for a number of holes versus historical RC drilling within the same mineralized zones. Historically these variances were also observed in comparisons between historical core holes and historical RC holes whereby the core holes returned lower overall assays for a particular interval.

Additionally, assaying both halves of split core has indicated that gold values can also vary significantly within a particular core interval, this is further confirmed by the duplicate analyses received to date in the 2020 Freeman Phase 1 drilling program. LGT's duplicate sampling using the 2012 pulps and rejects showed significant variances between fire assay and metallic screen assay results of as much as 300%. LGT duplicate sampling has also indicated variances of between 200% and 400%. Brewer (2019) concludes that while these variances are not the norm, they do indicate that the Lemhi Gold Deposit exhibits some significant nugget effects. The 2020 drill program has identified a significant number of occurrences of visible gold in several core holes, a further indication of potential nugget effects.

The issue of poor assay value reproducibility is poorly understood and requires further investigation. The discrepancy can, at least in large part, be explained by the indications of potential nugget effect in this deposit, along with the uncertainty of accurately "twinning" unsurveyed historical drill holes and, the inherent grade variance within a deposit that does have some mineralization related to quartz veining.

As reviewed in Section 12, the inclusion of 1980s FMC drill holes increases the risk of a slightly biased estimate in areas that rely on the 1980s FMC data. To this end, it is recommended that further infill drilling be completed in areas that significantly rely on the 1980s FMC data to increase the confidence level in those areas.

The authors are not aware of any other significant material risks to the MRE other than the risks that are inherent to mineral exploration and development in general. The authors of this report are not aware of any specific environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that might materially affect the results of this mineral resource estimate and there appears to be no obvious impediments to developing the MRE at the Lemhi Gold Project.

There are areas of the inferred block model that although the drilling was of sufficient density for an improved classification to indicated or better, the areas were dominated by 1980ss drillholes which in the authors review shows some systematic bias to high values which is related to increased risk. Some pointed infill drilling in these areas dominated by 1980s drillholes might improve the confidence in the data and lower the risk allowing for a higher classification.

In addition, the rock and soil sampling along with some limited exploration drilling outside of the main conceptual pit area, shows areas with a number of good results but with little follow-up exploration drilling. With further work, including drilling, there are opportunities to increase the mineral resources on the project.

15 MINERAL RESERVE ESTIMATES

This section is not relevant to this technical report.

16 MINING METHODS

16.1 Introduction

The deposit is amenable to open pit mining practices. Open pit mine designs, mine production schedules and mine capital and operating costs have been developed for the Lemhi deposit at a scoping level of engineering. The mineral resources form the basis of the mine planning.

Mine planning is based on conventional drill/blast/load/haul open pit mining methods suited for the project location and local site requirements. The open pit activities are designed for two years of construction followed by twelve years of operations. The subset of mineral resources contained within the designed open pits are summarized in Table 16-1, with a 0.25 g/t gold cut-off, and form the basis of the mine plan and production schedule for the life of mine (LOM).

Table 16-1: PEA Mine Plan Production Summary

Parameter	Value
PEA mill feed (LOM)	31,128 kt
Mill feed gold grade	0.88 g/t
Waste overburden and rock	121,903 kt
Waste to resource ratio	3.9

Notes:
1. The PEA Mine Plan and Mill Feed estimates are a subset of the March 15, 2023 Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a scoping level for the Lemhi deposit.
2. PEA Mine Plan and mill feed estimates are mined tonnes and grade; the reference point is the primary crusher.
3. Mill feed tonnages and grades include open pit mining method modifying factors, such as dilution and recovery.
4. Cut-off grade of 0.25 g/t assumes $1,700/oz. Au; 99. 95% payable gold; $4/oz off-site costs (refining, transport and insurance); a 1. 0% NSR royalty; and a 92% metallurgical recovery for gold.
5. The cut-off grade covers processing costs of $9. 20/t, administrative (G&A) costs of $1. 10/t, and low-grade stockpile Rehandle costs of $1. 00/t.
6. Estimates have been rounded and may result in summation differences.

The economic pit limits are determined using the Pseudoflow implementation of the Lerchs-Grossman algorithm. Ultimate pit limits are split up into six phases or pushbacks to target higher economic margin material earlier in the mine life. Upper benches will be accessed via internal cut ramps on topography, or via ramps left behind on phased pit walls. In-pit ramps will access material below the pit rim.

Pit designs are configured on 5 m bench heights, with minimum 8 m wide berms placed every four benches, or quadruple benching. Slopes of 25° are applied in the thin overburden layer above the deposit bedrock. Since there has been no geotechnical test work or analysis completed on the bedrock, the applied bench face and inter-ramp angles, 70-75° and 50-55° respectively, are scoping level assumptions based only on the rock type and overall depth of the open pit.

Resource from the open pit will report to a ROM pad and primary crusher directly northeast of the pit rim. The mill will be fed with material from the pits at an average rate of 2.5 Mt/a (6.8 kt/d), increasing to 3.0 Mt/a (8.2 kt/d) after four years of operation. Resources mined in excess of mill feed targets will be stored in a low-grade stockpile directly south of the ROM pad, and east of the open pit. This stockpile is planned to be completely reclaimed to the mill at the end of the mine life.

Waste rock will be placed in one of two facilities, each planned as a comingled facility with processed tailings. The north facility sits directly adjacent and uphill from the open pit, with its most northern point lying 1.2 km from the pit rim. The south facility sits 0.6 km southeast and downhill of the open pit, with its most southern point lying 2.0 km from the pit rim. The waste rock from the open pit has not been tested or analyzed for potential acid generation (PAG).

Topsoil and overburden encountered at the top of the pits will be placed in a dedicated stockpiles directly south of the open pit and kept salvageable for closure at the end of the mine life.

Mining operations will be based on 365 operating days per year with two 12-hour shifts per day. Owner managed operations are planned, utilizing a diesel-powered mining fleet.

Cost estimates for mining are based on benchmarking to other similar sized operations in western United States, mining 12-16 Mt/a. These operations typically include RC drills for bench-scale grade control drilling, DTH drills with 140 mm bit size for production drilling, emulsion based on blasting agents targeting 0.3 kg/t powder factors, 12 m^3 bucket size diesel hydraulic excavators and 14 m^3 bucket sized wheel loaders for production loading, and 91 t payload rigid-frame haul trucks for production hauling, plus ancillary and service equipment to support the mining operations.

In-pit dewatering systems will be established for the pit. All surface water and precipitation in the pits will be gravity drained, or directed via submersible pumps, to ex-pit settling ponds directly outside the pit limits.

The mine equipment fleet is planned to be purchased via a lease financing arrangement, with down payments occurring when the equipment is commissioned, and lease payments deferred for one year after the equipment is operational.

Maintenance on mine equipment will be performed in the field with major repairs and planned interval maintenance in the shops located near the process facilities.

16.2 Key Design Criteria

The following mine planning design inputs were used:

- The topography is based on a LiDAR survey of the region
- Re-blocked resource block models on 4 m spacing in all three dimensions
- Resource block model contains diluted mineralized gold grades, bulk densities, and resource classifications
- Measured, indicated, and inferred class mineral resource estimates included in-pit optimizations and mill feed estimates
- No geographical restrictions have been applied to the open pit footprints.

16.2.1 Net Smelter Price and Cut-off Grade

Net Smelter Price (NSP) is used for mine planning, in place of the market price for gold, to consider all off-site costs and determine revenue potential at the mine gate. The NSP calculation uses the inputs shown in Table 16-2, below.

Table 16-2: Net Smelter Price

Item	Unit
Gold Price	$1,700/oz
Payable Gold	99.95%
Gold Off-site Costs (Refining, Transport, Insurance)	$4.00/oz
Royalty	1.0%
Net Smelter Price	**$54/g**
	C$1,678/oz

The economic cut-off grade is chosen as the gold grade required to pay for processing costs, general and administration costs, and low-grade stockpile reclaim costs. The cut-off grade calculation uses the inputs shown in Table 16-3, below.

Table 16-3: Economic Cut-off Grade

Item	Unit
Net Smelter Price	$54/g
Process Recovery at Cut-off	92%
Process Costs	$9.20/t
G&A and Site Costs	$1.10/t
Stockpile Rehandle Costs	$1.00/t
Economic Cut-off Grade	0.23 g/t
Chosen Project Economic Cut-off Grade	**0.25 g/t**

16.2.2 Mining Loss & Dilution

The mineral resources are based on a 2.5 x 2.5 x 2.5 m resource model sub-block sizes. For mine planning, these blocks have been re-blocked to an open pit mining unit size of 4 x 4 x 4 m, which accounts for planned open pit mine operating conditions. This re-blocking to 4 m block spacing introduces ~12% dilution the original sub-block resource model, when measured at a 0.40 g/t gold cut-off grade.

This approach to calculating dilution and loss is considered appropriate for the current mine plan. The calculated 4 m re-blocked mill feed gold grades are taken as representative of the diluted ROM material that the operator will be able to achieve when pursuing the throughputs targeted in this mine plan.

Additional mining operational losses have not been directly accounted for and are assumed to be considered indirectly within the dilution measurements, which is considered appropriate for scoping level estimates of mill feed tonnes and grade.

16.2.3 Pit Slopes

No open pit geotechnical work has been done on the Lemhi deposit. Scoping level assumptions have been made for the pit configuration and overall pit slope angles based on rock types and overall pit depth from surface. Open pit slope assumptions described below are reasonable for scoping level engineering on the project.

Pit designs are configured on 5 m bench heights, with minimum 8 m wide berms placed every four benches, or quadruple benching. Two zones are included for bedrock based on depth of the pit along different azimuths, with unique bench face angles, and subsequent inter-ramp angles in these zones. These slope criteria are summarized in Table 16-4.

Table 16-4: Pit Slope Design Inputs

Domain	Azimuth Range (°)	Bench Face Angle (°)	Inter-ramp Angle (°)	Bench Height (m)	Calc Berm Width (m)	Overall Angle (°) (for pit optimization)
Overburden	All	25	25	20	0	25
South Bedrock	15 – 265	75	55	20	8.6	50
North Bedrock	265–-15	70	50	20	9.5	45

16.3 Pit Optimization

The economic pit limits are determined using the Pseudoflow implementation of the Lerchs-Grossman algorithm. This algorithm uses the gold grades and bulk density for each block of the 3D block model and evaluates the costs and revenues of the blocks within potential pit shells. The routine uses input economic and engineering parameters and expands downwards and outwards until the last increment is at break-even economics.

Additional cases are included in the analysis to evaluate the sensitivities of open pit mined resources to waste mining ratio and high-grade/low-grade areas of the deposits. In this study, the various cases, or pit shells, are generated by varying the input gold price and comparing the resultant waste and mill feed tonnages and gold grades for each pit shell.

By varying the economic parameters while keeping inputs for metallurgical recoveries and pit slopes constant, various generated pit cases are evaluated to determine where incremental pit shells produce marginal or negative economic returns. This drop-off is due to increasing waste mining ratios, decreasing gold grades, increased mining costs associated with the larger or deeper pit shells, and the value of discounting costs before revenues. The economic margins from the expanded cases are evaluated on a relative basis to provide payback on capital and produce a return for the project. At some point, further expansion does not provide significant added value. A pit limit can then be chosen that has suitable economic return for the deposit.

For each pit shell, an undiscounted cashflow (UCF) is generated based on the shell contents and the economic parameters listed in Table 16-5. The UCF for each case is compared to reinforce the selected point at which increased pit expansions do not increase the project value. Note that the economics are only applied for comparative purposes to assist in the selection of an optimum pit shell for further mine planning; they do not reflect the actual financial results of the mine plan.

The chosen pit shell is then used as the basis for more detailed design and economic modelling.

Price inputs for the Pseudoflow runs are listed Table 16-2 above and operating cost assumptions are provided in Table 16-5. The input gold price varies from US$200/oz to US$2,750/oz.

Table 16-5: Operating Cost Inputs into Pseudoflow Shell Runs

Item	Unit
Pit rim mill feed mining cost	$3.00/t
Pit rim waste mining cost	$2.35/t, pit rim of 1,600 masl
Incremental haulage cost	$0.015 per every 4 m bench below pit rim
Processing cost	$9.20/t
Process Recovery	96.9%
General/Administration cost	$1.10/t

16.3.1 Ultimate Pit Limit

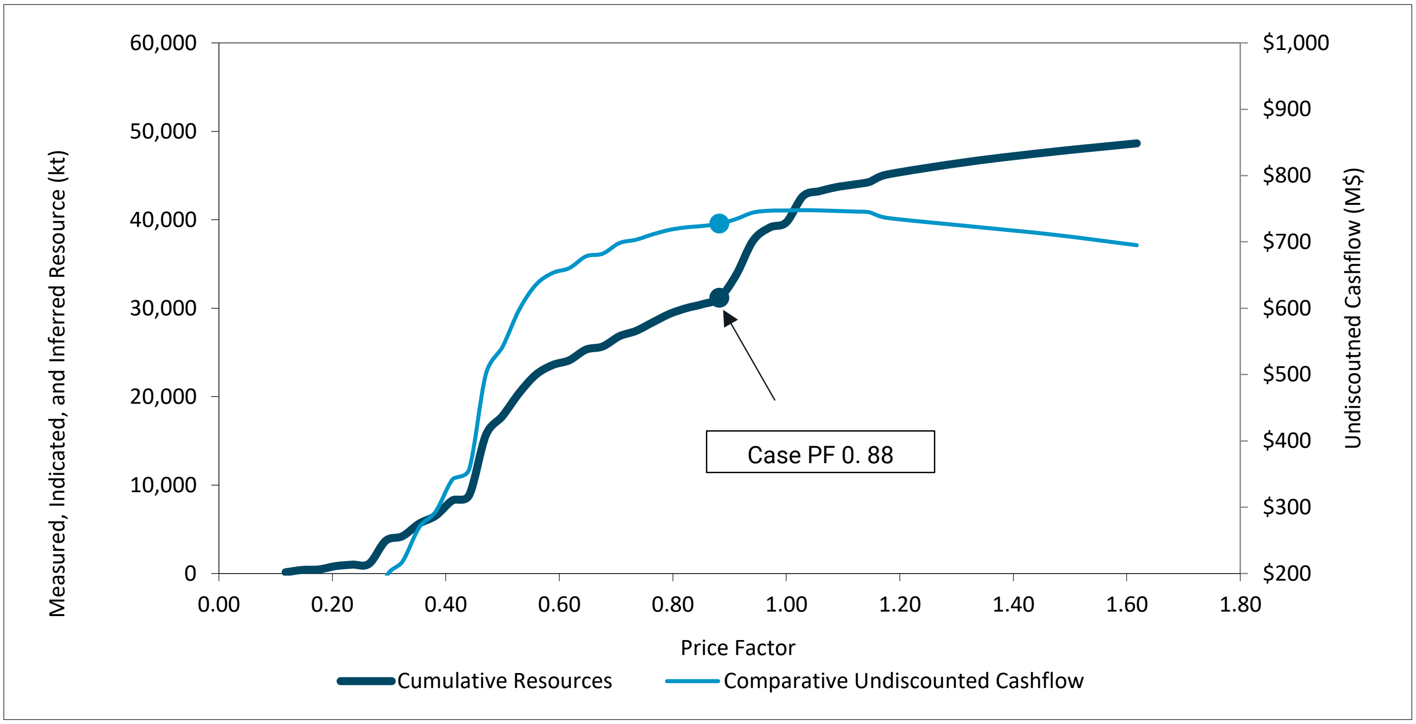
Figure 16-1 shows the contents of the generated Pseudoflow pit shells for the Lemhi deposit. An inflection point can be seen in the curve of cumulative resources and UCF by pit case. This point indicates price factor (PF) Case 0.88 as a point at which larger pit shells will not produce significant increases to project value.

The pit shell generated from Case PF0.88 is selected as the ultimate pit limits for Lemhi and is used for further mine planning as a target for detailed open pit designs with berms and ramps.

16.4 Pit Designs

Contents of the designed open pits are presented in Table 16-6. The contents for each designed pit phase are presented graphically in Figure 16-2.

Figure 16-1: Lemhi Pseudoflow Pit Shell Resource Contents by Case



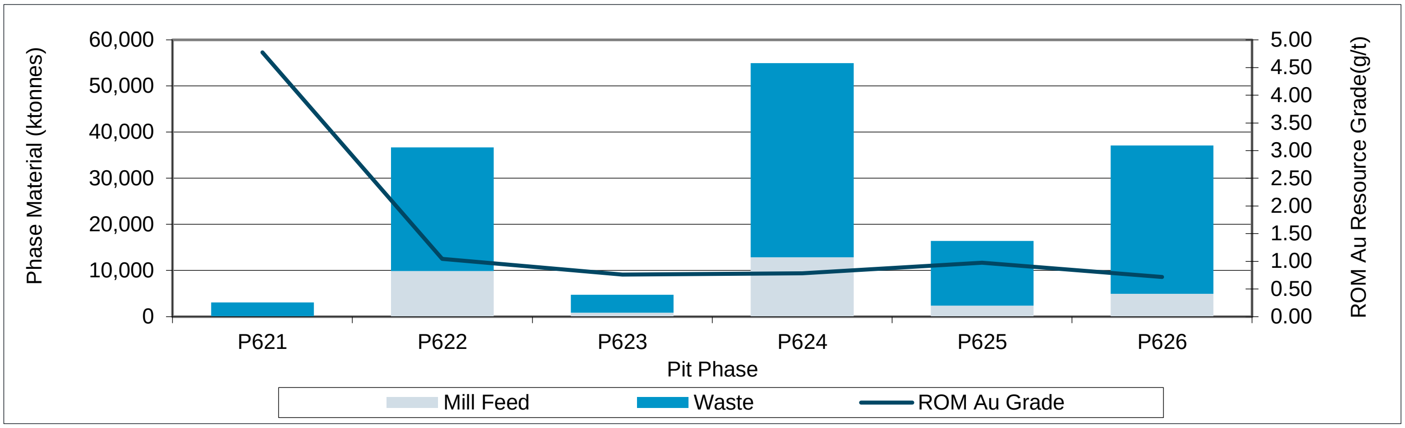
Source: Moose Mountain, 2023.

Table 16-6: Contents of Designed Pit Phases

Pit Phase	Pit Name	Mill Feed (Mt)	Diluted Gold Grade (g/t Au)	Au Metal (koz)	Waste (Mt)	W:O Ratio (t/t)
Beauty Zone	P621	0.1	4.77	13	3.0	35.7
Starter Phase	P622	9.9	1.04	332	26.8	2.7
SW Starter Phase	P623	0.9	0.76	21	3.9	4.5
West Pushback	P624	12.9	0.79	325	42.1	3.3
South Pushback	P625	2.4	0.97	76	14.0	5.8
Final Pushback	P626	5.0	0.72	115	32.1	6.5
Grand Total		**31.1**	**0.88**	**881**	**121. 9**	**3.9**

Notes:
1. The PEA Mine Plan and Mill Feed estimates are a subset of the March 15, 2023 Mineral Resource estimates and are based on open pit mine engineering and technical information developed at a scoping level for the Lemhi deposit.
2. PEA mine plan and mill feed estimates are mined tonnes and grade, the reference point is the primary crusher.
3. Mill Feed tonnages and grades include open pit mining method modifying factors, such as dilution and recovery.
4. Cut-off grade of 0.25 g/t assumes $1,700/oz. Au; 99. 95% payable gold; $4/oz off-site costs (refining, transport, and insurance); a 1.0% NSR royalty; and a 92% metallurgical recovery for gold.
5. The cut-off grade covers processing costs of $9.20/t, administrative (G&A) costs of $1.10/t, and low-grade stockpile Rehandle costs of $1.00/t.
6. Estimates have been rounded and may result in summation differences.

Figure 16-2: Designed Phase Pit Contents



Source: Moose Mountain, 2023.

16.4.1 In-Pit Haul Roads

In-pit haul roads are designed 25 m wide to facilitate two-way travel for 90 t payload rigid-frame haul trucks. Haul road grades are limited to a maximum of 10%. Access ramps are not designed for the last bench (5 m) of the pit bottom, on the assumption that the bottom ramp segment will be removed using some form of retreat mining. The bottom five ramped benches (25 m) of the pit use one-way haul roads of 19 m width and 12% grade since bench volumes and traffic flow are reduced.

16.4.2 Pit Phases

Ultimate pit limits are generally split up into phases or pushbacks to target higher economic margin material earlier in the mine life. Minimum pushback distances of 50 m are honoured. The Beauty Zone pit is mined as a standalone single pit phase. The main Lemhi deposit pit is split into five phases with the higher-grade, lower strip ratio early pit phases mined ahead of lower grade, higher strip ratio pushbacks to the ultimate pit limit. Targets for the initial pit phases use Case PF 0.41 Case PF 0.56 and Case PF0.65 of the optimization runs described in Section 16.3.1.

16.4.3 Pit Designs

The pit designs are shown in Figure 16-3 (final pit phase) and Figure 16-4. Original topography contour polylines are shown on 5 m vertical intervals. Sections through the deposit showing the re-blocked resource model grades are illustrated in Figure 16-5 to Figure 16-9.

16.4.3.1 Beauty Zone Phase, P621

This phase targets the high-grade mineralization of the Beauty Zone. The upper benches of this phase will be accessed via ex-pit ramps to the 1,790 masl on the south side of the pit, wrapping around the hill side to the main deposit area. One-way in-pit haul ramp access is planned from the pit exit at 1,720 masl to the bottom of the target pit shell at the 1,665 masl elevation. This haul ramp significantly shallows the overall pit angles and increases the strip ratio required to access the resource.

16.4.3.2 Starter Phase, P622

This phase targets the higher-grade, lower strip ratio portion of the deposit outlined by the Case PF 0.41 pit shell described in Section 16.3.1. The upper benches of this phase will be accessed via in-pit cut ramps up to 1,680 masl developed during the construction period of the project. Pit ramps are left behind in the highwall for access to future west highwall pushbacks. These ramps run from the 1,650 masl elevation in the west, down to the pit exit at the 1,580 masl elevation in the north. In-pit ramping is also incorporated from the pit exit, running counterclockwise down to a switchback at the 1,470 masl and clockwise to the pit bottom on the 1,440 masl elevation.

16.4.3.3 SW Starter Phase, P623

This phase targets two standalone southwest portions of the main deposit. The upper benches of this phase will be accessed via in-pit cut ramps up to 1,680 masl developed during the construction period of the project. One-way in-pit haul ramp access is planned from the upper pit exit at 1,650 masl to the bottom of the upper target pit shell at the 1,630 masl elevation. One-way in-pit haul ramp access is planned from the lower pit exit at 1,575 masl to the bottom of the lower target pit shell at the 1,520 masl elevation.

16.4.3.4 West Pushback, P624

This phase targets deeper, higher waste mining ratio mineralization below and west of the P622 pit, outlined by the Case PF 0.56 pit shell described in Section 16.3.1. The pit highwall is pushed to the final limits in the west. The upper benches of this phase will be accessed via in-pit cut ramps developed up to 1,720 masl. Benches between 1,650 masl and the pit exit at 1,605 masl will utilize in-pit ramps left behind in the P622 walls. In-pit ramping is also incorporated from the pit exit, running clockwise down to a switchback at 1,540 masl, then counterclockwise down to another switchback at 1,470 masl, and finally clockwise to several pit bottom on the 1,420 masl elevation.

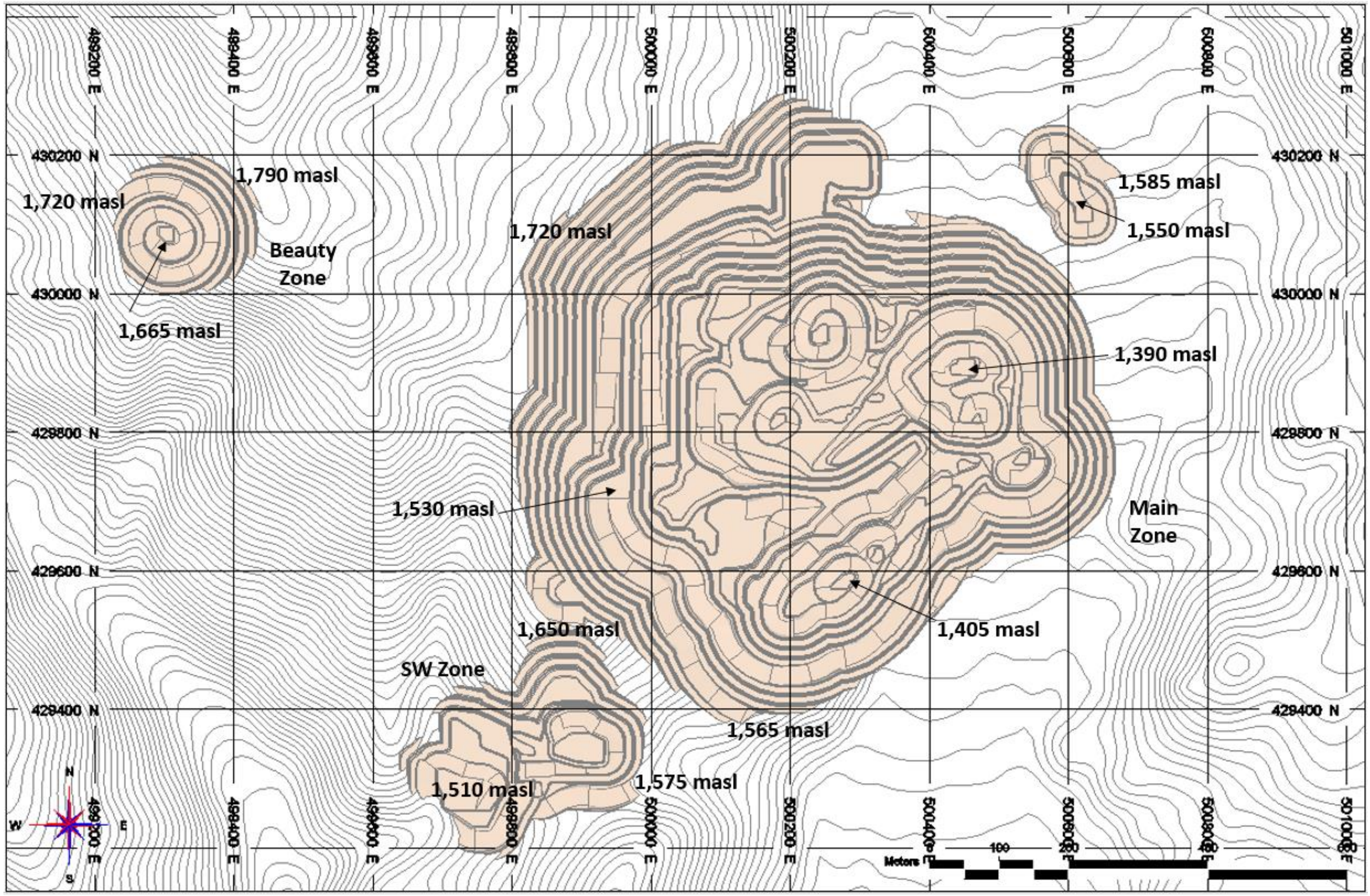
16.4.3.5 South Pushback, P625

This phase targets deeper, higher waste mining ratio mineralization south of the P622/P624 pits, outlined by the Case PF 0.65 pit shell described in Section 16.3.1. The pit highwall is pushed to the south with room for further southwest and southeast pushbacks to the final pit limits. In-pit ramping is incorporated from the pit exit at 1,545 masl, running clockwise down to a switchback at 1,480 masl, then counterclockwise down to another switchback at 1,470 masl, and finally clockwise to two separate pit bottoms on the 1,420 masl and 1,400 masl elevations.

16.4.3.6 Final Pushback, P626

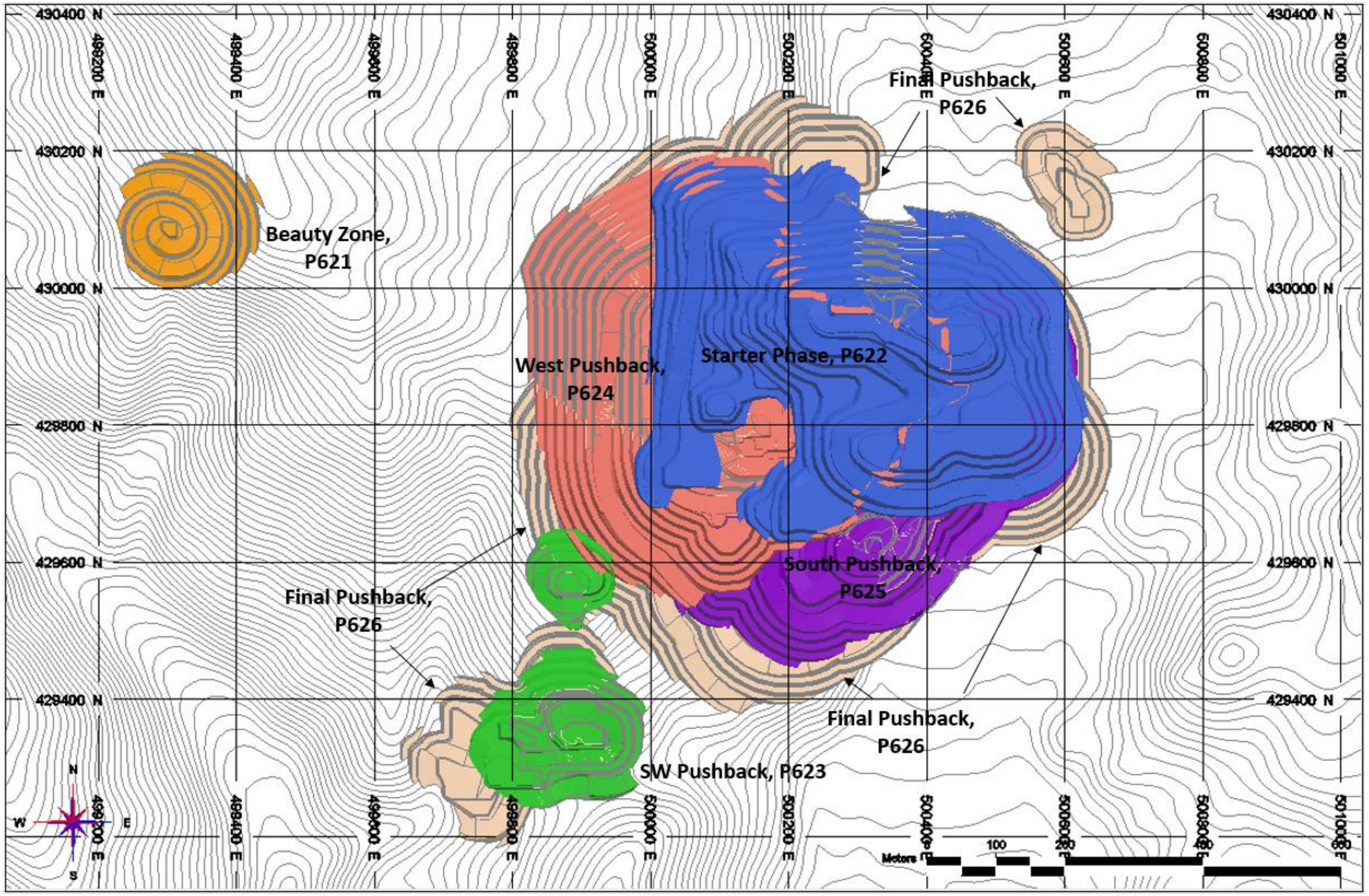
This final pit phase targets several pit bottoms west, south, and north of the initial pit phases. A standalone pit to the northeast is developed off a one-way ramp from the pit exit at 1,585 masl, to the pit bottom at 1,550 masl. The P623 SW starter phase is extended north and west to a new pit bottom at 1,510 masl. The remaining pit is pushed out to the north, west and south, utilizing existing ex-pit and previous phase in-pit ramps located between the 1,710 masl and the pit exit at 1,565 masl. In-pit ramping is also incorporated from the pit exit, running clockwise down to a switchback at 1,530 masl, then counterclockwise down several pit bottoms on the 1,420 masl, 1,1410 masl, 1,405 masl, and 1,390 masl elevations.

Figure 16-3: Pit Design, P626



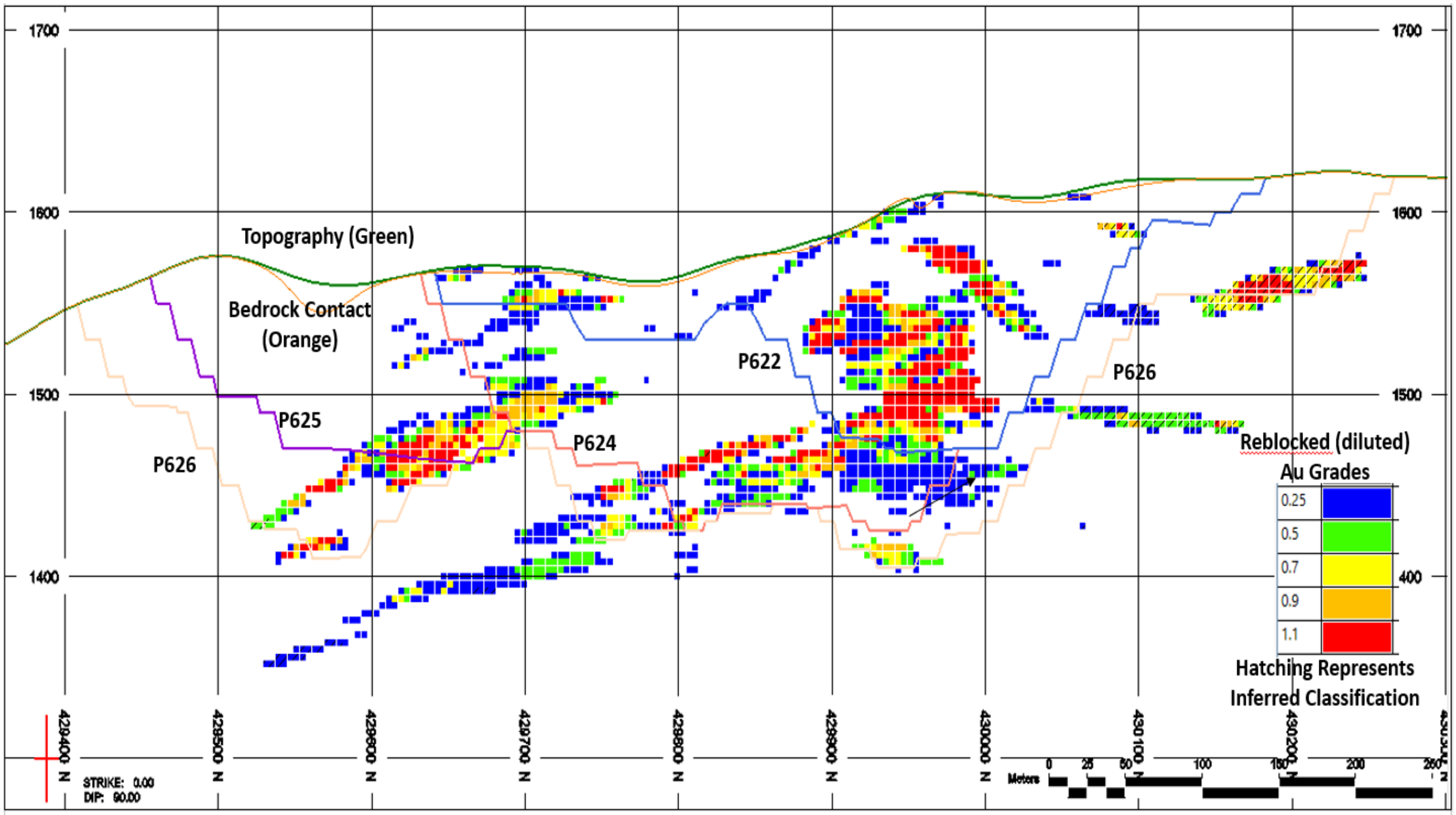
Source: Moose Mountain, 2023.

Figure 16-4: Phased Pit Designs



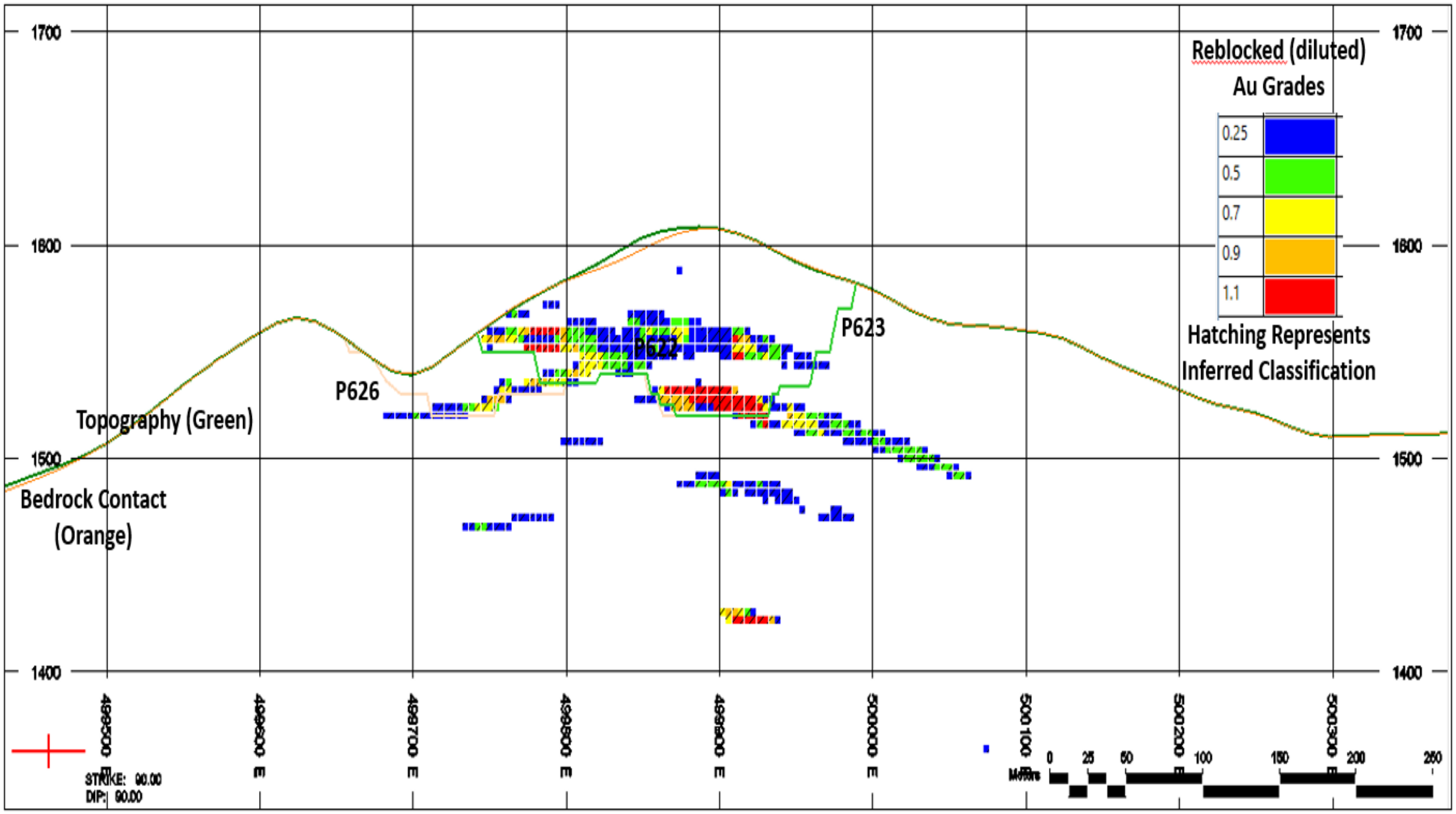
Source: Moose Mountain, 2023.

Figure 16-5: Pit Designs, NS Section 500,242E



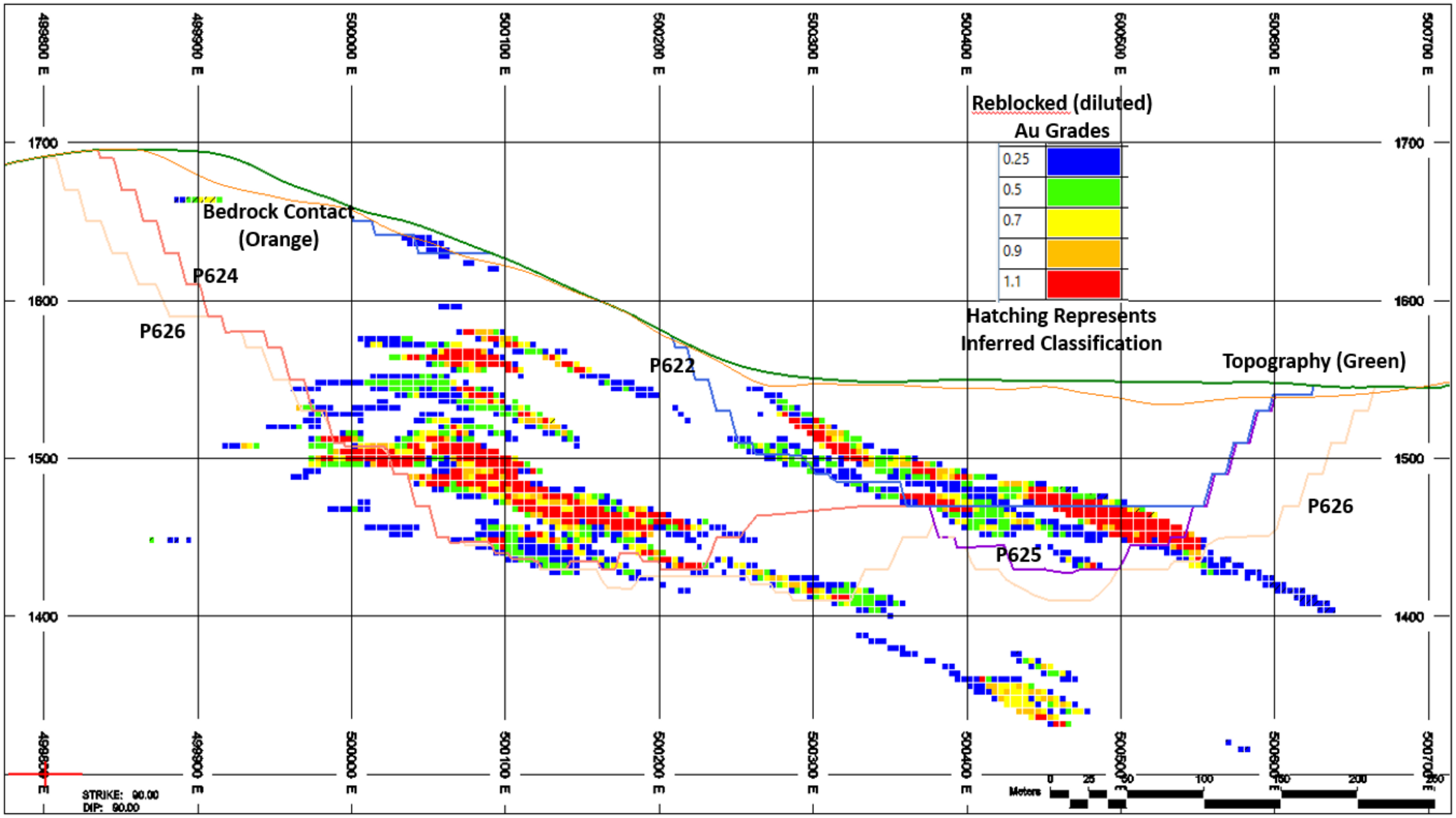
Source: Moose Mountain, 2023.

Figure 16-6: Pit Designs, EW Section 429,350E



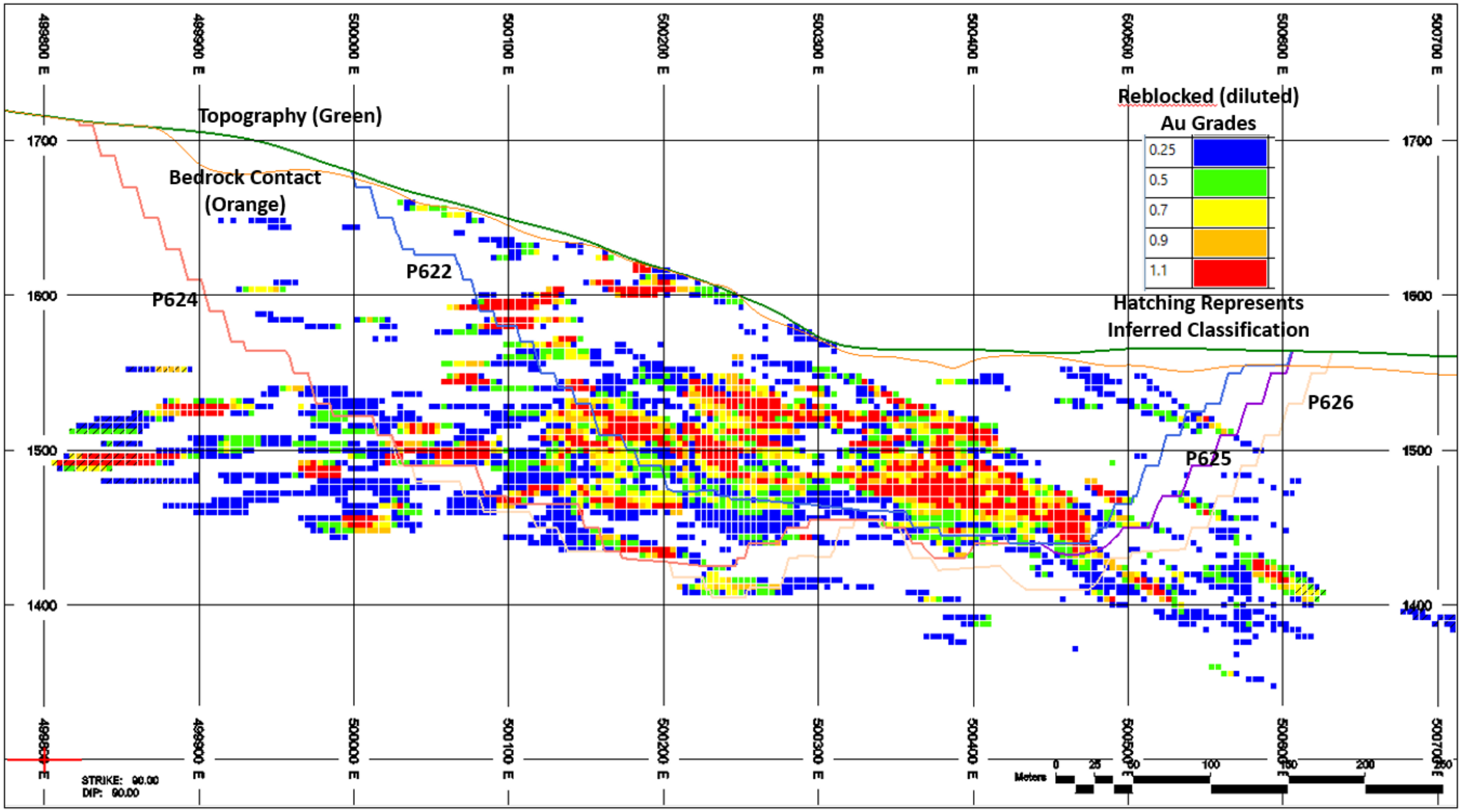
Source: Moose Mountain, 2023.

Figure 16-7: Pit Designs, EW Section 429,785E



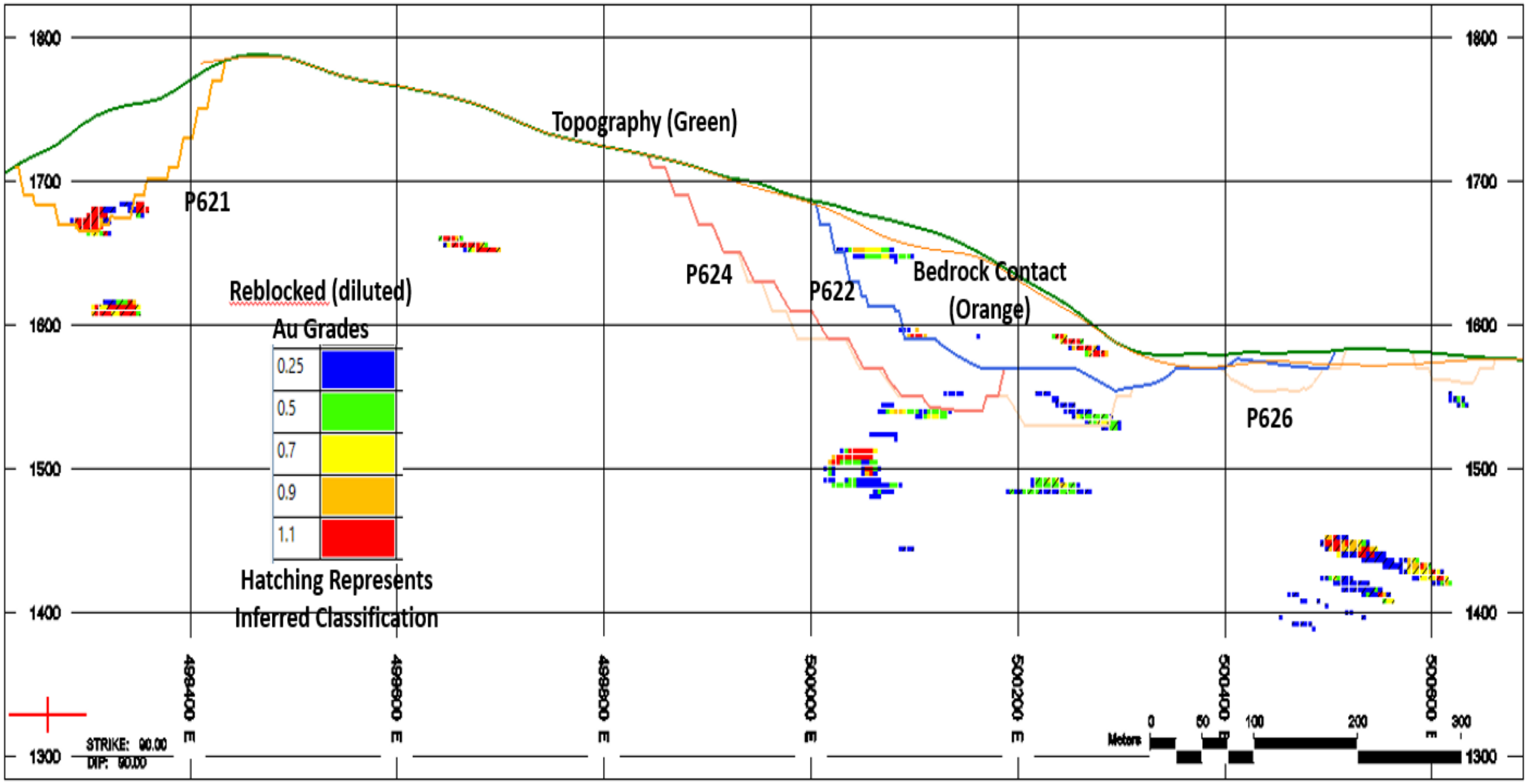
Source: Moose Mountain, 2023.

Figure 16-8: Pit Designs, EW Section 429,940E



Source: Moose Mountain, 2023.

Figure 16-9: Pit Designs, EW Section 430,085E



Source: Moose Mountain, 2023.

16.5 Low-Grade Storage Facilities

When resources are mined from the pit, they will either be delivered to the crusher, the run-of-mine (ROM) stockpile located next to the crusher, or the low-grade stockpiles.

The crusher and ROM stockpiles are located 0.3 km northeast of the pit limits.

Cut-off grade optimisation on the mine production schedule sends resource between 0.25 g/t and 0.35 g/t Au to a low-grade stockpile located directly southeast of the ROM pad and east of the open pit. These stockpiled resources are planned to be re-handled back to the crusher before the pits are exhausted.

Preliminary designs for these facilities are completed assuming:

- Bottom-up construction/top down reclamation.
- 30° overall slopes.
- Storage density of 2. 00 t/m^3
- Average height of 25 m from topography to crest.

The low-grade stockpiles are shown in the project layout drawings in Figure 18-1.

16.6 Waste Rock Storage Facilities

Waste rock will be placed in one of two facilities, each planned as a comingled facility with processed tailings. The north facility sits directly adjacent and uphill from the open pit, with its most northern point lying 1.2 km from the pit rim. The south facility sits 0.6 km southeast and downhill of the open pit, with its most southern point lying 2.0 km from the pit rim.

Design criteria for the waste rock storage facilities are described in Section 18.4.

The waste rock from the open pit has not been tested or analyzed for PAG. It is assumed that the waste rock from both deposits is net acid neutralising and there has been no consideration for segregation of different rock types in the planned storage facilities. Further test work and analysis is recommended to better classify waste materials according to acid generating potential, and to confirm that a blending strategy is the preferred method handling any potentially acid generating waste rock.

Backfilling of the open pit was examined as an opportunity, but space is too limited as the pit continually expands deeper in all directions until the final pit phase.

Topsoil and overburden encountered at the top of the pits will be placed in a dedicated stockpiles directly south of the open pit and kept salvageable for closure at the end of the mine life.

The waste storage facilities are shown in the project layout drawings in Figure 18-1.

16.7 Ex-Pit Haul Roads

Mine haul roads external to the open pits are planned to haul resource and waste materials from the open pits to the scheduled destinations.

Design criteria for the ex-pit haul roads is described in Section 18.

The ex-pit haul road layouts are shown in the project layout drawing in Figure 18-1.

16.7.1 Production Schedule

Production requirements by scheduled period, mine operating considerations, product prices, recoveries, destination capacities, equipment performance, haul cycle times, and operating costs are used to determine the optimal production schedule from the phased pit contents.

The overall production schedule is included as Table 16-8. The open pit mine production schedule for all the deposits is included as Figure 16-10 and shows the production tonnage and grade forecast; Figure 16-11 provides an illustration of the projected material mined and waste mining ratio.

The production schedule is based on the following parameters:

- The mineral resource and associated waste material quantities are split by pit phase and bench quantities.
- The operations are scheduled in annual periods.
- An annual mill feed rate of 2,500 kt/a (6.8 kt/d) is targeted.
- This is increased to 3,000 kt/a (8.2 kt/d) in Year 5 of the Project.
- Mill throughput ramp-up is assumed to occur in the construction phase, such that the first year of mill operations is at the target mill throughput. Low grade resources are planned to be stockpiled well in advance of the mill ramp-up period.
- Within a given pit phase, each bench is fully mined before progressing to the next bench.
- Pit phases are mined in sequence, where the second pit phases do not mine below the first pit phases.
- Pit phase vertical progression in mineralized area is limited to no more than 36 m in each year, or 9 benches; average annual phase progression is 28 m.
- Pre-stripping done in the construction period, Years -2 and -1, is done to open the pits sufficiently to supply mill feed at the target throughput rate in Year 1 of the Project.
- Resource tonnes released in excess of the mill capacity are stockpiled, including those mined in the construction phase.
- Low-grade resource is stockpiled and re-handled to the primary crushers later in the mine life.

- Shovel and haul truck operating hour estimates are run as part of the mine schedule. Haul cycle times are simulated from all pit benches to all destinations. Total pit production is balanced on calculated hauler operating hour requirements. This strategy is used to avoid large spikes and dips in the number of haulers in the LOM schedule but leads to some variations in total tonnes mined in each period. Cycle time simulations should be refined in future engineering studies.

16.7.2 Mining Sequence

The pit operations will run for two years of construction and twelve years of mill operations. The general mine sequence through the various pit phases is illustrated in Table 16-7.

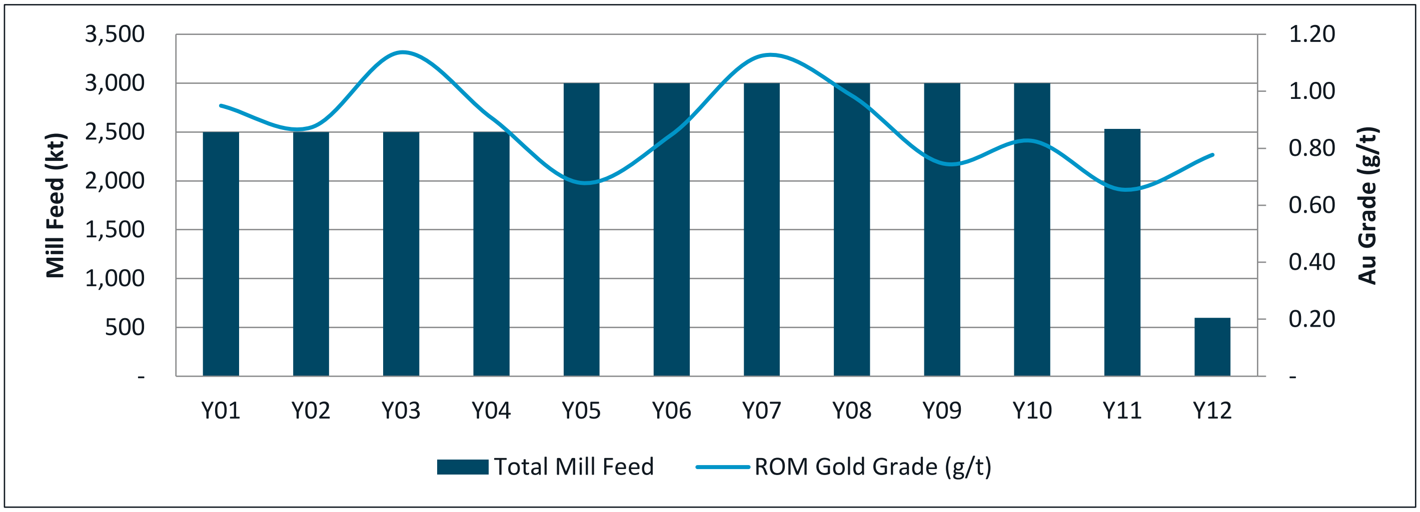
Table 16-7: Pit Phase Sequence

Phases Mined	Y-2	Y-1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y10	Y11	Y12
Beauty Zone, P621														
Starter Phase, P622														
SW Starter, P623														
West Pushback, P624														
South Pushback, P625														
Final Pushback, P626														

Table 16-8: Mine Production Schedule

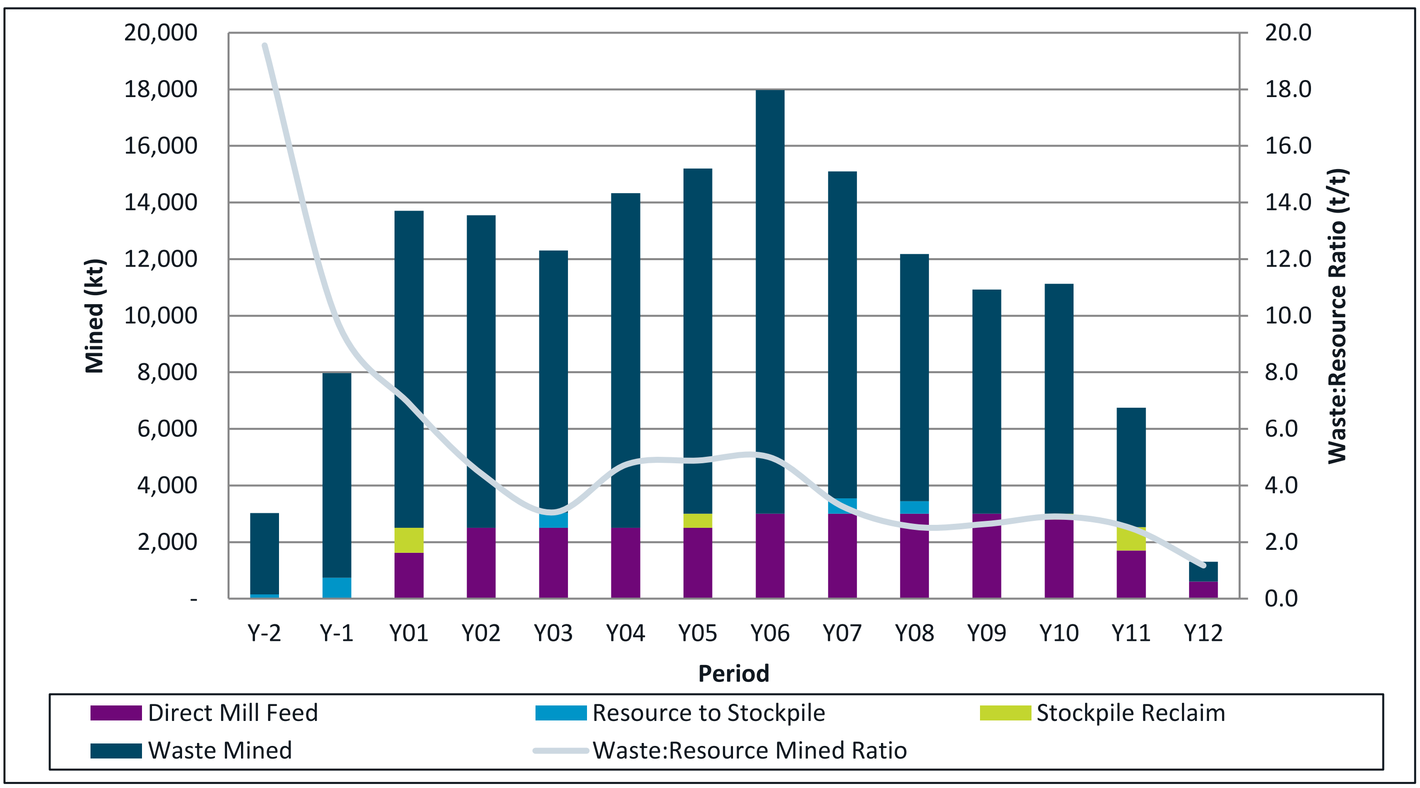
Total Mine Production	Year	LOM	Y -2	Y -1	Y 1	Y 2	Y 3	Y 4	Y 5	Y 6	Y 7	Y 8	Y 9	Y10	Y11	Y12
Mill feed	kt	31,128	0	0	2,500	2,500	2,500	2,500	3,000	3,000	3,000	3,000	3,000	3,000	2,531	598
ROM Au	g/t	0.88	0.00	0.00	0.95	0.87	1. 14	0.91	0.68	0.85	1. 12	0.98	0.75	0.83	0.65	0.78
Mill feed gold	koz	881	0	0	76	70	91	73	65	82	108	95	72	80	53	15
Resource mined	kt	31,128	147	734	1,625	2,500	3,040	2,500	2,500	3,000	3,542	3,444	3,000	2,800	1,697	598
ROM Au	g/t	0.88	0.82	0.89	0.98	0.87	0.99	0.91	0.75	0.85	1. 00	0.90	0.75	0.86	0.83	0.78
To stockpile	kt	2,408	147	734	0	0	541	0	0	0	542	444	0	0	0	0
ROM Au	g/t	0.51	0.82	0.89	0.00	0.00	0.30	0.00	0.00	0.00	0.30	0.30	0.00	0.00	0.00	0.00
Stockpile retrieval to mill	kt	2,408	0	0	875	0	0	0	500	0	0	0	0	200	833	0
ROM Au	g/t	0.51	0.00	0.00	0.89	0.00	0.00	0.00	0.30	0.00	0.00	0.00	0.00	0.30	0.30	0.00
Waste mined	kt	121,903	2,881	7,236	11,203	11,051	9,264	11,833	12,202	14,971	11,554	8,740	7,922	8,126	4,221	702
Total mined from pits	kt	153,031	3,028	7,970	12,828	13,551	12,304	14,333	14,701	17,971	15,096	12,184	10,922	10,926	5,918	1,300
Total moved	kt	155,439	3,028	7,970	13,703	13,551	12,304	14,333	15,201	17,971	15,096	12,184	10,922	11,126	6,752	1,300

Figure 16-10: Mine Production Schedule, Mill Feed Tonnes and Grade (All Deposits)



Source: Moose Mountain, 2023.

Figure 16-11: Mine Production Schedule, Material Mined and Waste Mining Ratio (All Deposits)



Source: Moose Mountain, 2023.

16.8 Operations

Owner operated and managed open pit mine operations are planned to be typical of similar operations in western United States.

Grade control drilling is carried out to better delineate the resource in upcoming benches. A grade control system is planned to provide field control for the loading equipment to selectively mine resource-grade material separately from the waste.

In-situ rock is drilled and blasted to create suitable fragmentation for efficient loading and hauling of both resource and waste rock. Loading in resource zones will be completed with a hydraulic excavator, and in waste zones with a hydraulic excavator and a wheel loader, depending on grade control requirements. Resource and waste rock will be hauled out of the pit and to scheduled destinations with off-highway rigid-frame haul trucks.

Mine pit services will include:

- haul road maintenance
- pit floor and ramp maintenance
- mobile fuel and lube services
- ditching
- dewatering
- secondary blasting and rock breaking
- snow removal
- lighting
- transporting personnel and operating supplies.

Mining operations are based on 365 operating days per year with two 12-hour shifts per day. An allowance of 12 days of no production has been built into the mine schedule to allow for adverse weather conditions.

16.9 Mining Equipment

The following mine equipment descriptions are based on typical fleet contingents utilized in other North American open pit mine operations. It should be expected that equipment specifications and fleet sizes will be altered with further project engineering and optimization.

Grade control drilling will be carried out with diesel hydraulic truck mounted RC drills. Production drilling will be carried out with 140 mm diesel driven DTH drills.

Reliable mining equipment commonly found in the open pit mining industry has been selected for the loading and hauling fleet. Hydraulic excavators (12.0 m^3 bucket) are proposed based on their ability to minimize losses and dilution for the grade control operations. Front-end wheel loaders (14.0 m^3 bucket) are proposed based on their ability to load the haulers in three to four passes, and their ability to load the crusher when required. Rigid-frame haulers (91 t payload) are proposed to be flexible enough to use on the smaller pit benches and in selective mining scenarios but are not so small that the fleet size is excessive. Articulated haul trucks (40 t payload) are included as a support hauler for overburden, as well as accessing smaller pit mining areas such as pit bottoms of initial bench access when bench diving.

Graders will be used to maintain the haul routes for the haul trucks and other equipment within the pits and on all routes to the various waste storage locations and the crusher. Articulated trucks that are outfitted with a water tank (35,000 L) and gravel spreader are included for haul road maintenance. Track dozers (325 kW) are included to handle waste rock to the various construction and waste storage locations and to support the in-pit activities. Front-end wheel loaders (4.5 m^3 bucket) and hydraulic excavators (3.8 m^3 and 3.0 m^3 bucket) are included as pit support, grade control support, and general back-up loaders for the main fleet. Custom fuel/lube trucks are included for mobile fuel/lube support. Various small mobile equipment pieces are proposed to handle all other pit service and mobile equipment maintenance functions.

Pits will be dewatered via gravity drainage out of horizontal drilled holes in the pit walls, or with conventional dewatering equipment: submersible pumps placed in-pit bottom sumps, and/or vertical pumping wells established along the pit perimeter. A nominal amount of pumping has been assumed for this pit, based on other regional open pits, but it is recommended to conduct additional hydrogeologic test work and analysis to further refine this estimate in future mine planning. Pit water will be pumped to collection ponds adjacent to the pits, where it will be managed as per the overall site water management plan.

Mine fleet maintenance activities are generally performed in the maintenance facilities located near the plant site.

Primary mining equipment requirements are summarized in Table 16-9. The equipment classes, as well as number of units, are preliminary scoping level estimates, and modifications in future studies should be anticipated.

Table 16-9: Primary Mining Fleet Requirements

	Start-Up	Peak (Y04-Y09)
Drilling		
Diesel DTH/RC tracked drill, 140 mm (5. 5”) holes	3	5
Loading		
Wheel loader, 14. 0 m^3 bucket	1	1
Hydraulic excavator, 12. 0 m^3 bucket	1	2
Hauling		
Rigid-frame haul truck, 91 t payload	4	12
Articulated haul trucks, 41 t payload	2	2

16.10 Risks

The project is at a scoping level of engineering. There has been limited geotechnical, hydrogeological, and geochemical information and data collected across the project. Further field work, lab work, and modelling are required to advance engineering to the next stages of pre-feasibility or feasibility. It can be anticipated that further field drilling and advancement of the project engineering will materially alter the existing mine plan, reducing the plan’s risk and identifying and exploiting the potential opportunities that arise.

Risks to the preliminary economic assessment (PEA) defined mill feed quantities, gold grades, associated waste rock quantities, and the estimated costs to exploit include changes to the following factors and assumptions:

- Metal Prices
  - Decreases in metal prices may increase the economic cut-off grade, or reduce the size of the open pit, with either outcome reducing the size of the resource base to include into the mine plan.
- Interpretations of mineralization geometry and continuity in mineralization zones
  - Decreases in the resource base could significantly alter the mine plan.
- Geotechnical and hydrogeological assumptions
  - Geotechnical sampling, testwork, and analysis may show a required shallowing of pit slope angles, which likely would in turn increase the overall LOM stripping ratio to access the resource.
  - Hydrogeological sampling, testwork, and analysis may identify needs for a more onerous (costly) pit water management and pit slope depressurization solution.
- Geochemical assumptions for mined resource and waste materials
  - Geochemical sampling, testwork, and analysis, specifically in the open pit waste rock, may identify a more onerous (costly) PAG management solution.
- Ability of the mining operation to meet the annual production rate and anticipated grade control standards and recoveries
  - Reduced selectivity with the mining fleet, reduced mining or milling recoveries, or increased mining dilution would result in an increased cost of achieving the planned PEA metal production.
- Operating cost assumptions and cost creep
- Ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate
- Ability to access capital for project financing.

17 RECOVERY METHODS

17.1 Overview

The process plant design incorporates a staged expansion approach allowing the throughput to be expanded. The process flowsheet for the Lemhi Gold Project was selected based on the preliminary metallurgical testing, discussed in Section 13, and Ausenco's process design expertise, and flowsheet trade-off study and was tailored to support the ramp-up of the plant throughput in Phase 2 starting from fifth year of operation. The unit operations selected are standard technologies used in gold processing plants.

The staged expansion of the process plant over the mine life is presented below:

- Phase 1 (Years 1 to 4) – The process plant is operated at a throughput of 2.5 Mt/a.
- Phase 2 (Years 5+) – The pre-leach thickener is added, and grind size is increased to 130 µm to process material at throughput of 3.0 Mt/a.

17.2 Process Design Criteria

Along with the design parameters listed in section 17.1, additional design criteria of the process plant are listed in Table 17-1.

Table 17-1: Process Design Criteria, Phase 1

Parameter	Units	Value
Plant throughput, Years 1 to 4	Mt/a	2.5
Life of mine	y	11.2
Gold head grade – average (LOM)	g/t	0.88
Gold head grade – maximum	g/t	1.15
Crushing plant availability	%	75
Mill availability	%	92
Bond crusher work index (cWi), design	kWh/t	5.4
Bond ball mill work index (bWi), design	kWh/t	14.8
Bond abrasion index (Ai), design	-	0.26
ROM specific gravity (SG)	-	2.58
Comminution circuit		
Crushing plant capacity, design	t/h	381
Crushing circuit product size, P_{80}	mm	58
Grinding circuit capacity, design	t/h	310
Grinding circuit configuration	-	SAB

Parameter	Units	Value
Grinding circuit product size, P_{80}	µm	110
Classification cyclones O/F pulp density	%w/w	44
Leach and adsorption circuit		
Leach system	type	Leach – CIL
Leach + CIL residence time	h	36
Leach circuit recovery	%	96
Leach + CIL tanks	#	1+6
Desorption		
Carbon batch size	t	6
Type of stripping system	-	Pressure Zadra
Cyanide destruct circuit		
Detox residence time	min	90
Detox WAD cyanide discharge target	mg/L CN_{WAD}	<5.0

Source: Ausenco, 2023

17.3 Process Plant Description

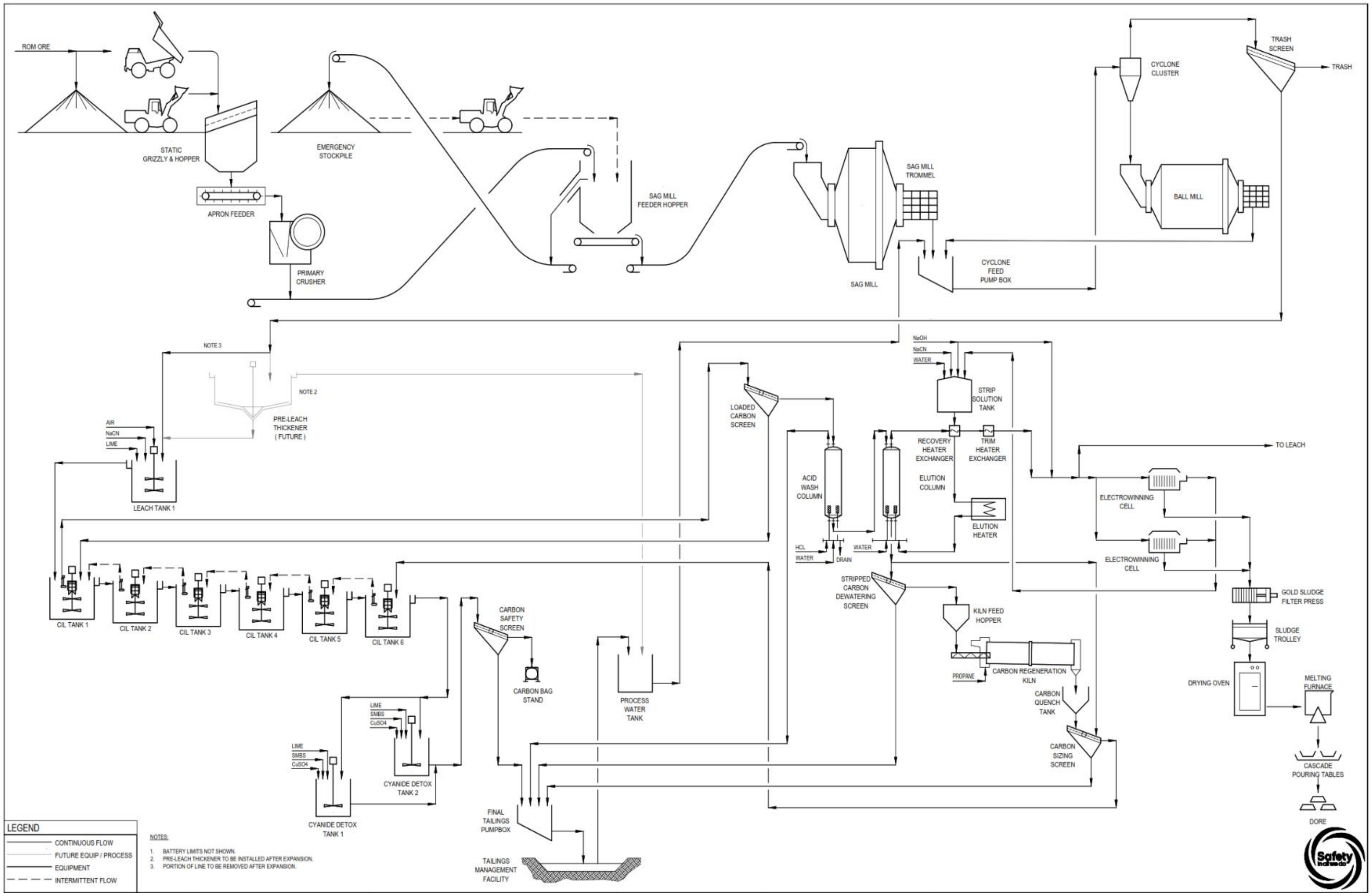
Process design is comprised of the following circuits:

- primary crushing of ROM material
- semi-autogenous grinding (SAG) mill followed by ball mill with cyclone classification
- leach and carbon-in-leach adsorption, a pre-leach thickener will be added for the throughput expansion
- acid washing and elution of loaded carbon
- electrowinning and smelting to produce doré
- carbon regeneration
- cyanide destruction
- tailings disposal.

17.3.1 Process Flowsheet

An overall process flow diagram (PFD) showing the unit operations in the process plant is presented in Figure 17-1.

Figure 17-1: Process Flowsheet



Source: Ausenco 2023.

17.3.2 Crushing Circuit

Run-of-mine (ROM) material is hauled from the mine and stockpiled on the ROM pad with one day storage capacity or directly tipped into to the hopper equipped with a static grizzly. Material from the hopper is discharged by gravity to an apron feeder and fed into the primary jaw crusher where it is crushed to product size (P_{80}) of 56 mm. The primary crushing plant has an operating availability of 75%. The jaw crusher discharge is then transported to the SAG mill feeder hopper via the overland conveying system. Material will be transferred into the mill via SAG mill feed conveyor. The hopper will be designed as an overflow bin such that overflow material from the hopper will be transported to an emergency stockpile by a conveyor should hopper capacity be exceeded. The crushing plant and associated materials handling equipment is sized at the outset for the Phase 2 throughput.

- Major equipment in this area includes:
- ROM hopper with static grizzly
- Primary jaw crusher (160 kW)
- Primary crusher conveyor
- SAG mill feed hopper
- emergency stockpile and feed conveyor
- SAG mill feed conveyor.

17.3.3 Grinding Circuit

An overland conveyor will deliver the crushed material to the grinding circuit consisting of a SAG mill followed by ball mill in closed configuration with hydro cyclones. The circuit is sized based on a grinding circuit feed size (F_{80}) of 58 mm and a circuit product size (P_{80}) of 110 µm. SAG mill slurry will discharge onto a rubber-lined trommel screen with trommel oversize discharging to a bunker for regular collection and disposal. The trommel undersize will combine with the ball mill discharge in the cyclone pump box where the slurry will be diluted to the desired pulp density with process water and pumped to the cyclone cluster. Overflow from the cyclones at 44% solids w/w will report to a trash screen followed by the leach circuit. Cyclone underflow will return to the ball mill directly for further size reduction. In the Phase 2, the ball charges of both the SAG and ball mill will be increased, and the target circuit product size (P_{80}) will be increased to 130 µm to accommodate for the throughput expansion.

Major equipment in this area includes:

- SAG mill (2 MW)
- ball mill (4 MW)
- cyclone cluster
- cyclone overflow trash screen.

17.3.4 Leach and Adsorption

Hydrocyclone overflow gravitates to the leach and carbon-in-leach (CIL) area via a trash screen. The trash screen will remove any debris or trash from the slurry before leaching. Trash screen undersize slurry will be fed into the leach/adsorption circuit consisting of one leach tank and six CIL adsorption tanks providing total residence time of 36 hours. Air is sparged to the tanks to maintain adequate dissolved oxygen levels for leaching. Hydrated lime is added to adjust the operating pH to the desired set point of 10.5-11 and cyanide solution is added to the first leach tank.

Regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIL circuit and is advanced counter-currently using carbon advance/transfer pumps from a downstream to an upstream tank. Slurry from the last CIL tank gravitates to the cyanide detoxification tanks. Each CIL tank has a mechanically swept carbon retention screen to retain the carbon while allowing the slurry to flow by gravity to the downstream tank. Loaded carbon is transferred from the first CIL tank to the loaded carbon screen followed by the carbon elution circuit using a recessed impeller pump. The leach-adsorption circuit is sized for the expansion throughput in Phase 1.

For the expansion, a pre-leach thickener will be added to dewater trash screen undersize slurry to 50% solids to maintain 36 hours residence time through the leaching circuit.

Major equipment in this area includes:

- one mechanically agitated leach tank
- six mechanically agitated CIL tanks with interstage screens.

17.3.5 Cyanide Detoxification and Tailings Disposal

CIL tailings exiting the last CIL tank pass through a carbon safety screen and are pumped into two cyanide detoxification tanks in parallel. Carbon retained on the safety screen is removed into bulk bags. Cyanide detoxification will take place using the SO_2/air process. In this process copper sulphate is used as a catalyst. Hydrated lime is used to maintain the pH of the reaction. The cyanide detoxification makes use of two tanks in parallel that have each been sized for a total residence time of 90 minutes. The cyanide destruction tanks are equipped with oxygen addition points and agitators to ensure that the oxygen and reagents are thoroughly mixed with the tailings slurry. The tailings slurry discharges into final tailings pumpbox and pumped to the North Co-placement Storage Facility (CPSF). A filter plant will be added in the second year of operation to produce filtered tailings for placement in the CPSFs. The details of the CPSFs are described in section 18.4. The cyanide detoxification circuit and associated material handling equipment is sized at the outset for the expansion throughput in Phase 2.

Major equipment in this area includes:

- carbon safety screen
- two mechanically agitated detoxification tanks
- tailings pressure filter (installed in Year 2).

17.3.6 Carbon Acid Wash and Elution.

Loaded carbon slurry is pumped from the first CIL tank to the loaded carbon screen. Screen undersize is pumped back to the first CIL tank, while screen oversize discharges to the acid wash column. Loaded carbon will be washed with a weak hydrochloric acid solution at 2 BV/h rinse rate to remove impurities and residual leaching reagents that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon. Entrained water will drain from the column and the column will refill with the hydrochloric acid solution from the bottom up. Once the column is filled with acid, it will be left to soak, after which the spent acid will be rinsed from the carbon and discarded to the final tailings pump box.

The acid-washed carbon will be hydraulically transferred to the elution column for gold stripping via a pressure Zadra system. Hot elution solution consisting of a mixture of water, sodium hydroxide, and sodium cyanide is passed through the carbon bed to desorb the gold and other adsorbed species from the carbon surface. Pregnant solution from the elution column is transferred to electrowinning. Electrowinning barren solution is then recirculated through the elution column via a heater. A heat exchanger preheats the barren eluate by recovering some heat from the pregnant solution. When an elution cycle is complete, the circuit is ready to initiate a new acid wash and elution cycle. The acid wash, elution and carbon regeneration circuits are sized at the outset for the expansion throughput in Phase 2.

Major equipment in this area includes:

- loaded carbon screen
- acid wash column
- elution column
- recovery heat exchanger
- elution heater.

17.3.7 Carbon Regeneration

The stripped carbon is dewatered by a screen over a feed hopper that feeds an electric rotary kiln via a screw feeder. The kiln is operated at 750°C in an atmosphere of superheated steam to restore the activity of the carbon. Carbon discharging from the kiln will be quenched in water and pumped over a carbon sizing screen to remove undersized carbon fragments. As carbon will be lost by attrition, fresh carbon is added to the circuit as needed in the carbon quench tank. Carbon sizing screen oversize reports to the last CIL tank while undersize slurry is discharge into final tailings pumpbox.

Major equipment in this area includes:

- stripped carbon dewatering screen
- regeneration kiln
- carbon sizing screen.

17.3.8 Electrowinning and Gold Room

Gold is recovered from the elution pregnant solution by electrowinning process. The pregnant solution is pumped through electrowinning cells fitted with stainless steel mesh cathodes. An electrical current is applied across the cells, causing gold to deposit on the surface of the cathodes. Expected electrowinning plating time is 16 hours. Barren solution is recirculated to the elution columns with a periodic bleed to the leach circuit in order to prevent the build-up of impurities. The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and gravitates to the sludge hopper. The sludge is filtered, oven dried, mixed with fluxes, and smelted in a single pot furnace to produce gold doré. The electrowinning and smelting process takes place within a secure and supervised gold room.

17.4 Reagents Handling and Storage

The reagent handling system will include unloading and storage facilities, mixing tanks, stock tanks, transfer pumps, and feeding equipment. Each set of compatible reagents mixing and storage systems will be located within containment areas to prevent incompatible reagents from mixing. Appropriate ventilation, fire and safety protection, eyewash stations, and safety data sheet (SDS) stations will be located throughout the facilities. Sumps and sump pumps will be provided for spillage control.

Table 17-2: Reagents Handling & Storage

Reagents	Preparation Method	Use
Sodium Cyanide	Received in bulk bags; mixed with raw water and transferred to a storage tank; dosed to elution and cyanide leaching circuits	Leaching reagent
Lime	Received as powder in bulk bags; mixed with raw water and transferred to a storage tank and dosed to cyanide leaching and cyanide destruction circuits	For pH control
Copper Sulphate	Received as powder in bulk bags; mixed with raw water and transferred to a storage tank and dosed to the cyanide destruction circuit.	Catalyst in the detoxification reaction
Hydrochloric acid	Received in intermediate bulk containers (IBC) totes as concentrate solution at nominally 33% HCl by volume; dosed to acid wash circuit	Acid wash reagent
Activated Carbon	Received on site as a granulated solid in bulk bags	Adsorption reagent
Sodium Hydroxide	Received in IBC totes as concentrated solution at nominally 50% NaOH by volume; dosed to elution circuit and eluate tanks	Elution and electrowinning circuit
SMBS	Received as a powder in bulk bags; mixed with raw water and transferred to storage tank; dosed to cyanide detoxification circuit	Cyanide destruction agent
Flocculant	Received as a powder in bulk bags; mixed with raw water and transferred to storage tank; dosed to pre-leach thickener	Thickening aid

Source: Ausenco, 2023.

Reagent consumptions are based on testwork results and standard industry practices. A summary of the nominal estimated reagent and consumable rates are presented on an annual basis in Table 17-3.

Table 17-3: Major Reagents Consumptions Summary

Reagents	Unit	Phase 1	Phase 2
Sodium Cyanide	t/a	1,750	2,100
Lime	t/a	3,750	4,500
SMBS	t/a	4,250	5,100
Copper Sulphate	t/a	200	240
Activated Carbon	t/a	100	120
Jaw Crusher Liner	Sets/a	4	4
SAG Mill Liners	Set/a	1	1
SAG Mill Media	t/a	456	643
Ball Mill Liners	Sets/a	1	1
Ball Mill Media	t/a	1,566	1,822

Source: Ausenco, 2023.

17.5 Services – Water, Air, Power

17.5.1 Water

17.5.1.1 Freshwater

Fresh water will be provided to a freshwater storage tank, where it will be further pumped for various application points, including reagent preparation, gland seal, elution circuit, and general mill make-up water supply. Approximately 450,000 m^3/a of fresh water will be required for make-up to the process plant.

17.5.1.2 Potable Water

Potable water is produced by an on-site potable water plant which processes water from the freshwater tank and makes it fit for consumption and human use. Potable water is stored in a tank for distribution to the processing plant.

17.5.1.3 Process Water

Process water will be made up of tailings reclaim water, contact water and freshwater make-up. After a filter plant is added in the second year of operation, process water will be made up of filtrate and wash water from tailings filter and freshwater make-up. Process water will be stored in a process water tank and pumped to various circuits in the process plant.

17.5.1.4 Fire water

Fire water for the process plant is sourced from the freshwater tank. A dedicated pump skid consisting of an electrical pump, jockey pump, and diesel pump will supply water from the fire water reserve volume to a fire water reticulation system that services the plant. The fresh water tank level will maintain a minimum level of water for use by the fire water system.

17.5.1.5 Gland Seal Water

Gland seal water is taken from the fresh water tank and pumped to various pumps throughout the processing plant, including sump pumps.

17.5.2 Air

High-pressure air will be produced by compressors to meet plant requirements. The high-pressure air supply will be dried and used to satisfy both plant air and instrument air demand. Dried air will be distributed via the air receivers located throughout the plant. Low pressure air will be supplied to leaching and detoxification circuits as a source of oxygen.

17.5.3 Power

The total power requirements for the process plant are 78,704 MWh/a in Phase 1 and 93,895 MWh/a in Phase 2 after throughput expansion. The average power demand for the process plant and estimated power consumption for each area is given in Table 17-4.

Table 17-4: Power Requirements

Area	Average Demand (kW)		Power Consumption (MWh/a)	
	Phase 1	Phase 2	Phase 1	Phase 2
Crushing	294	383	2,833	3,687
Grinding	4,810	5,879	46,349	56,650
Leach-CIL	691	691	6,656	6,656
Detox	279	279	2,689	2,689
Water Services	243	291	2,338	2,805
Elution/Goldroom	1,708	2,050	16,460	19,752
Reagents	143	172	1,379	1,655
Thickener	-	8	-	81
Total	**8,168**	**9,744**	**78,704**	**93,895**

Source: Ausenco, 2023.

18 PROJECT INFRASTRUCTURE

18.1 Overview

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

- Mine facilities include administration offices, truck shop and wash bay, and mine workshop.
- Common facilities, including an entrance/exit gatehouse, a security/medical office. Overall site administration building, potable water and fire water distribution systems, compressed air, power distribution facilities, diesel reception, and communications area.
- Process facilities housed in the process plant, including grinding and classification; leach and carbon-in-leach (CIL) adsorption; acid washing, electrowinning, and smelting; carbon regeneration; cyanide destruction; assay laboratory; process plant workshop; and warehouse.
- Other infrastructure includes on-site camp, waste management facilities, and two co-placement storage facilities (CPSFs).

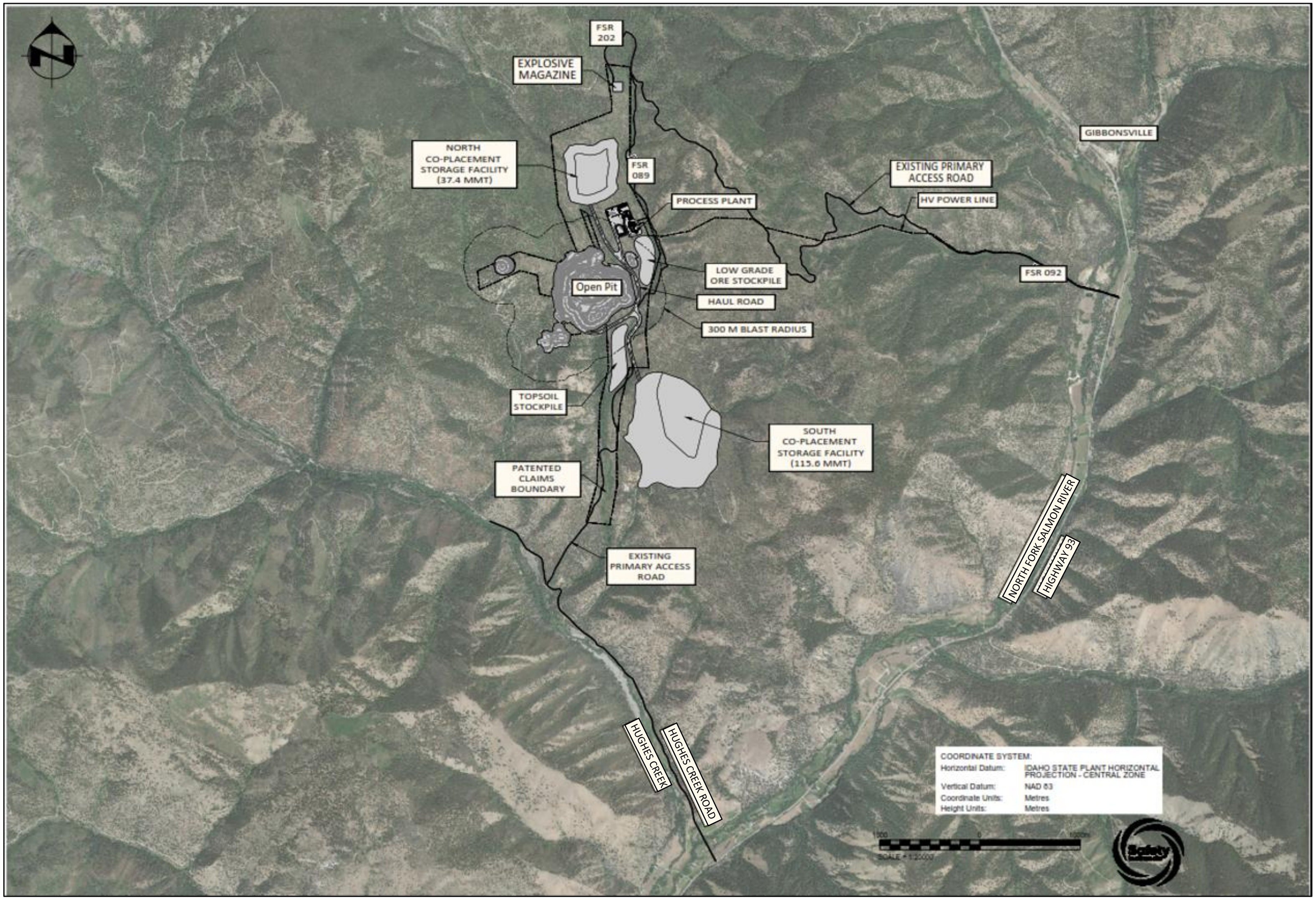
The mine and process facilities will be serviced with potable water, fire water, compressed air, power, diesel, communication utilities, and sanitary systems.

Site selection and location for project infrastructure was guided by the following considerations:

- The facilities described above must be located on the Lemhi patent land to the greatest extent possible.
- Locating the two CPSFs close to the open pit to reduce haul distance.
- Locating primary crushing close to the Lemhi deposits to reduce haul distance.
- Utilize the natural high ground for the run-of-mine (ROM) pad as much as possible.
- Separate heavy mine vehicle traffic from non-mining, light vehicle traffic.
- Locate the process plant near an existing primary access road.
- Locate the process plant in an area safe from flooding.
- Place mining, administration, and process plant staff offices close together to limit walking distances between them.

The Lemhi site layout is shown in Figure 18-1.

Figure 18-1: Lemhi Infrastructure Layout Plan



Source: Ausenco, 2023.

18.2 Off-Site Infrastructure

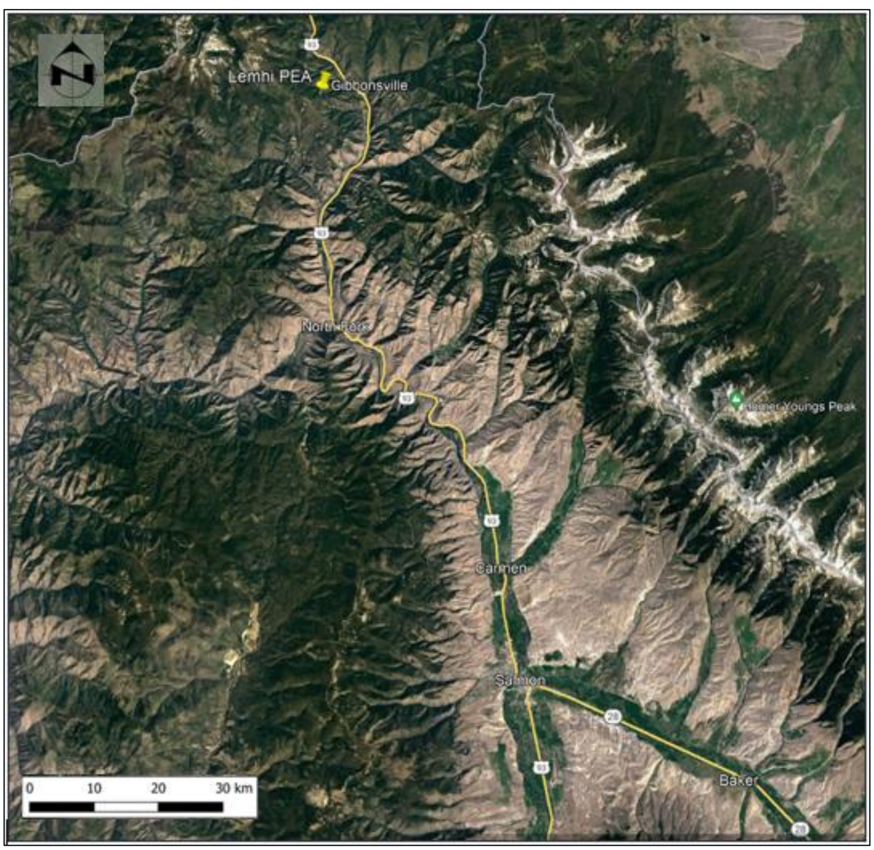
18.2.1 Site Access

The Lemhi Project site is accessible via multiple routes. The primary access is through Salmon, Idaho, via paved and gravel roads. From US Highway 93, driving north for 34 km leads to North Fork, then continuing 7.4 km to Hughes Creek Road (USFS Road 091). Following this road west for 3.2 km and then north along Ditch Creek Road for 3.1 km leads to a two-track road which leads northwest to the Lemhi Gold Property. Both Hughes Creek and Ditch Creek roads are well-maintained gravel roads managed by USFS and/or Lemhi County, offering dependable access.

Another route is through Granite Mountain Road (USFS Road 092), found 7.5 km north of Hughes Creek Road from Highway 93. This route follows Votler Creek westward, encircles Granite Mountain's south side, and descends into the Little Ditch Creek drainage, intersecting with Ditch Creek Road near the northern end of the Lemhi Gold Property, 8 km from Highway 93. While suitable for summer access with heavy equipment and supplies, the road's current condition prevents winter travel due to high altitude and insufficient berms.

Figure 18-2 shows the project location and proximity to the city of Salmon.

Figure 18-2: Lemhi Project Location



Source: Google Earth, 2022.

To access the Lemhi Project site and process plant, routing will be upgraded as the access is through mountainous terrain that features some switchbacks and sharp turns. As part of upgrading activities, some of these switchbacks and turns will be improved to meet the transportation needs of the site. The proposed access route avoids both residential areas in the region and the project's 300 m blast radius included in the project's open pit mine design. Figure 18-2 shows the access route from Highway 93N. The blue line represents USFS Road 092, and the orange line represents USFS Road 089.

On-site roads will be required to provide access to the plant, truck shop, administration building, and explosives magazine. These roads will be designed and constructed to allow two-way, light vehicle and, in some areas, mine truck traffic. All internal mine roads will be all-season, gravel-paved road.

18.2.2 Water Supply

The wells on site will provide potable water for the site, as well as water for the building facilities and the process plant.

18.2.3 Power Supply and Distribution

The project will be grid-powered all year-round. In Phase 1 (2.5 Mt/a), the maximum power demand is 12.8 MW with an average operating load of 9.0 MW. In Phase 2 (3.0 Mt/a), the maximum power demand is 15.3 MW with an average operating load of 10.7 MW. In addition to the process power requirements, these values include a 10% allowance for auxiliary power needs, such as for offices and workshops.

Site power will be supplied from the local grid via a 5 km power line that will be constructed for the project. A 35.5 kV power line passes near the project site through the settlement of North Fork to provide the maximum power demand of 15.3 MW in Phase 2. The power supply is sourced from a dedicated power plant at a rate of US$0.04/kWh.

All electrical rooms will be adequately rated for the environment and outfitted with heating and ventilation, lighting, small power transformers, distribution boards, and uninterrupted power supply (UPS) systems. To reduce installation time, the electrical rooms will be prefabricated modular buildings, installed on structural framework above ground level for bottom entry of cables. The electrical rooms will be located as close as practical to the electrical loads thereby minimizing voltage drop concerns and reducing cable cost, and they will have medium-voltage / low-voltage motor control centres (MCCs) and variable frequency drives (VFDs) to power the process plant loads.

18.3 On-site Infrastructure

18.3.1 Site Preparation

The site access road will be connected to the on-site road to provide access to the project site. The typical method of clearing, topsoil removal, and excavation will be employed. The preliminary site development will include drains, safety bunds, and backfilling with granular material and aggregates for road structure. Site civil work includes design for the following infrastructure:

- Roads for light vehicles and heavy equipment
- Access roads

- Topsoil stockpile area
- Mine facility platforms and process facility platforms
- ROM stockpile area
- Co-placement storage facility area
- Water management facilities, ditches, and drainage channels.

18.3.2 Buildings

Three types of buildings have been incorporated into the project design: modular, fabric buildings, and pre-engineering. The buildings for each area are listed in Table 18-1.

Table 18-1: On-site Building Description

Building Description	Building Construction	Length (m)	Width (m)	Height (m)	Area (m^2)	Volume (m^3)
Mineralized material storage and reclaim	Fabric building	49.0	42.0	19.0	2,058	39,102
Mill building	Pre-engineered building	36.6	24.5	24.0	897	21,521
Process plant office	Modular building	12.2	6.7	8.2	82	670
Water and air services	Pre-engineered building	11.3	24.5	12.0	276	3,313
Main administration building	Modular building	18.5	18.5	3.4	342	1,164
Gatehouse and truck scale	Modular building	6.3	3.8	3.4	24	81
Mining office	Modular building	18.0	11.0	2.41	198	477
Mine office and changerooms	Modular building	22.0	18.5	3.4	407	1,384
Plant office and changerooms	Modular building	18.5	14.7	3.4	272	925
CIL/reagent electrical room	Modular building	23.2	6.0	4.2	139	585
Grinding area electrical room	Modular building	23.2	6.0	4.2	139	585
Assay laboratory	Modular building	24.5	13.7	3.4	336	1,141
Truck shop	Fabric building	44.0	40.0	14.0	1,760	24,640
Truck shop warehouse	Fabric building	37.0	16.0	14.0	592	8,288
Truck wash building	Fabric building	12.5	10.0	13.0	125	1,625

18.3.2.1 Accommodation

No camp accommodation is planned for the project because workers will reside in, and commute from nearby communities, namely Salmon, Idaho.

18.3.2.2 Gate House and Truck Scale

The gate house is a security trailer office with a lockable gate and communications to the main site. The truck scale is located adjacent to the main access road by the guard house.

18.3.2.3 Main Administration Building

The main administration building is a modular, multiple level, building comprised of a change/lunch facility, offices, meeting rooms, washrooms, desks, fire protection, and alarm systems. The offices will have space for relevant employees. There will be 20 processing plant offices and 21 general and administrative offices.

18.3.3 Fuel

Fuel will be delivered to the mine site via tanker trucks. The fuel storage system consists of several above ground tanks, including diesel tank, gasoline tank, and various propane tanks. The diesel and gasoline tanks are insulated and heated to prevent fuel gelling. The tanks will be contained in a lined containment berm to assure no fuel can leak into the environment.

18.3.4 On-Site Roads

The project site has unpaved roads connecting the access road to the gatehouse. In addition to the existing roads on site, new roads will be constructed linking the guard house, the administration building, the process plant, the explosive storage buildings, the primary and waste crusher, and the CPFS.

18.3.5 Stockpiles

The barren stripping material from the open pit mine will be sent to either the waste rock storage facilities or the CPSF dam for construction, while the mineralized material will be sent either to crushing plant directly or to the main stockpile areas. The stockpile will have a 1-day storage capacity. All mill feed is currently envisioned to be hauled from the pit rim by 91 t payload rigid-frame haul trucks.

18.3.6 Mining Infrastructure

18.3.6.1 Truck Shop/Wash

The truck shop/wash is a pre-engineered building with a concrete floor, overhead crane, overhead doors as well as fire protection and alarm systems. There will be a total of four maintenance bays, two assigned to preventive maintenance, one to corrective maintenance, and one for multiple purposes. Additionally, a single welding bay and single truck wash bay will be located at the front of the truck workshop building.

18.3.6.2 Ex-Pit Haul Roads

Mine haul roads external to the open pits are designed to haul mineralized and waste materials from the open pits to the scheduled destinations. The mine haul roads are designed with the following key inputs:

- 25 m wide ex-pit haul roads that incorporate a dual-lane running width and berms on both edges of the haul road
- sized to handle 90-tonne payload rigid-frame haul trucks
- 8% maximum grade.

The ex-pit haul road layouts are shown in the site layout drawing in Figure 18-1.

18.3.6.3 Mine offices

The mine office is a modular building for open pit operations. The building is equipped with fire protection and an alarm system.

18.3.7 Process Plant Infrastructure

18.3.7.1 Plant Warehouse/Shop

The plant warehouse/shop is a pre-engineered building with concrete floor, overhead doors, fire protection, and alarm systems. This building will be used for general storage for equipment spares for the process plant, maintaining and storing light vehicles assigned to the plant, and repairing and maintaining process plant equipment as necessary. This building is equipped with fire protection and an alarm system.

Process Plant Control Room

The process plant control room is a modular office attached to the process plant and contains dual operator stations. This building is equipped with fire protection and an alarm system.

18.3.7.2 Assay Laboratory

The assay laboratory is a one-story modular building comprised of storage area, office, scale room, atomic absorption room, wet lab, and met labs. This building is equipped with fire protection and an alarm system. The laboratory requires bottled nitrogen and hoods with ventilation.

18.4 Co-placement Storage Facility (CPSF)

A desktop siting and waste material deposition trade-off study was carried out to evaluate potential sites and disposal technologies for tailings and waste rock. Based on the life of mine (LOM) production schedule there is not sufficient storage capacity within Freeman Gold's Patented Claim Boundary to store LOM waste materials; therefore, a permit from United States Department of Agriculture – Forest Service (USFS) to deposit waste materials on Salmon-Challis National Forest is required, for which approval could take up to two years.

A two-year CPSF was developed that contains both tailings and waste rock due to area constraints within the Patented Land and the balance of the LOM waste material on National Forest Lands. The waste rock production schedule allows for an initial traditional rock shell slurry tailings storage facility along with placing any excess waste rock within the CPSF.

The remainder of the LOM waste rock and tailings sites are outside the Patented Area to due area constraints. Traditional rock shell slurry tailings facility and waste rock storage facility were evaluated; however due to permitting, an approval timeline to start developing these facilities, there is insufficient time to construct a slurry tailings facility. Therefore, a second CPSF will be developed that combines filtered tailings and waste rock. The waste materials will be placed in 2 m – 5 m lifts.

The tailings and waste rock will be permanently stored in two CPSFs: the initial facility will be located north of the process plant (North CPSF) and the second CPSF will be located south of the open pit (South CPSF) and shown in Figure 18-1 and Figure 18-3.

The primary design objectives for the CPSFs are the secure containment of tailings and waste rock to protection of regional groundwater and surface water during mine operations and in the long term (post-closure).

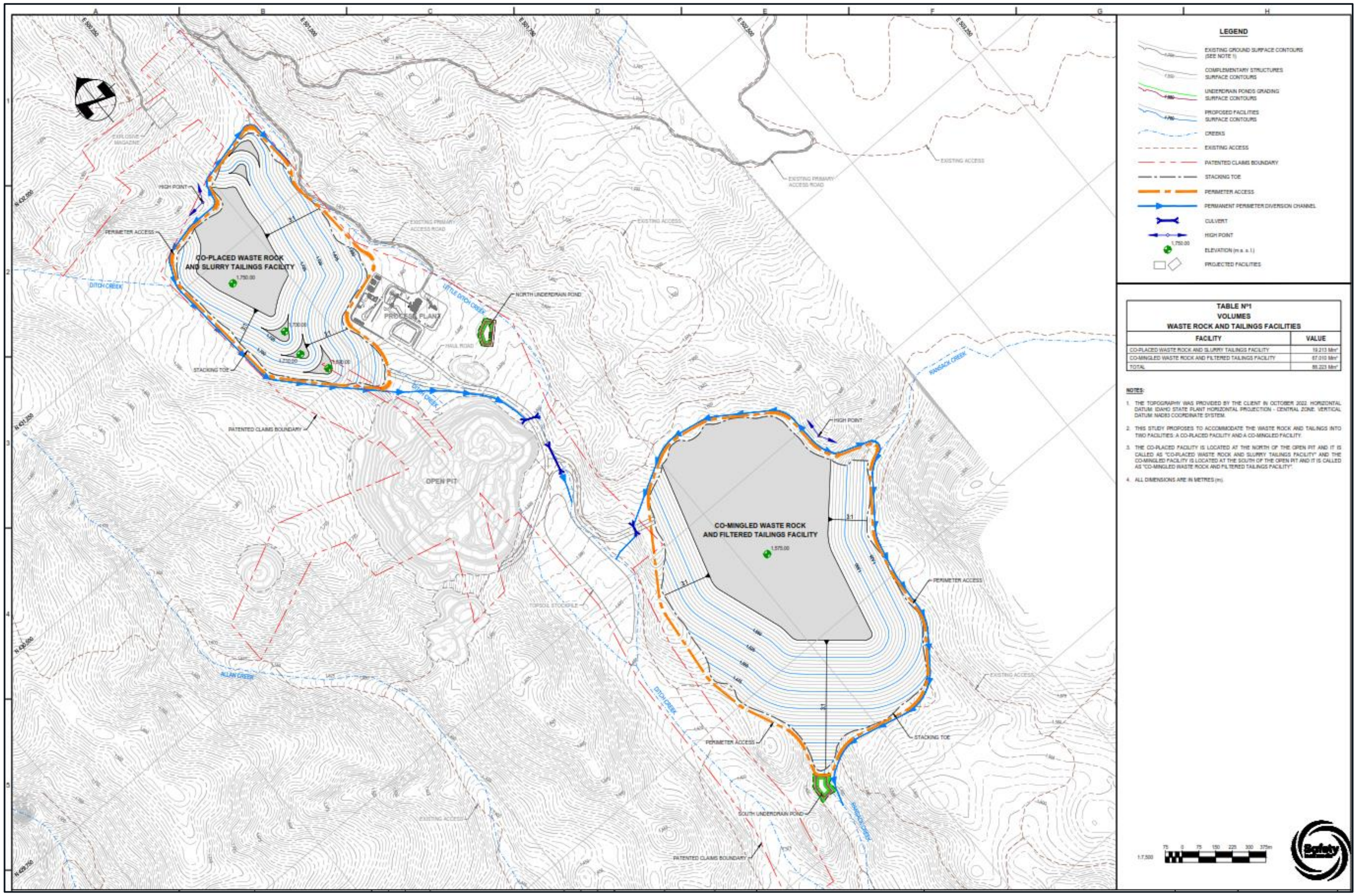
The design of the CPSF and associated water management facilities accounts for the following:

- Staged development of the facilities over the life of the project.
- Flexibility to accommodate operational variability in the waste rock and tailings (plant shutdowns, deposit variability, and placement during variable climate conditions).
- Control, collection, and removal of contract water from the facility during operations for reuse as process water to the maximum practical extent.

The design criteria for the CPSFs considered the following requirements for waste materials:

- North co-placement storage facility
- Tailings slurry storage requirement: 5 Mt (3.5 Mm^3)
- Waste rock storage requirement: 32.4 Mt (16 Mm^3)
- South co-placement storage facility
- Tailings filtered storage requirement: 26.1 Mt (15.8 Mm^3)
- Waste rock storage requirement: 89.5 Mt (44.8 Mm^3)
- Slurry tailings density: dry density of 1.45 t/m^3
- Filtered tailings density: dry density of 1.65 t/m^3
- Waste rock density: dry density of 2.0 t/m^3
- Limiting watershed disturbance
- Limiting impacts to wildlife and fisheries resources.
- The CPSFs also include the following:
- North co-placement storage facility
- East embankment
- Interior co-placement of slurry tailings and waste rock
- South co-placement storage facility
- Exterior rock shell
- Interior co-placement of filter tailings and waste rock.

Figure 18-3 Co-disposal Storage Facility Layout



Source: Ausenco, 2023.

18.4.1 Co-placement Storage Facility Hazard Classification

The design standards for the CPSFs are based on the relevant state guidelines for construction of mining tailings and waste rock storage facilities. The following regulations and guidelines were used to determine the dam hazard classification and suggested minimum target levels for some design criteria, such as the inflow design flood (IDF) and earthquake design ground motion (EDGM): Idaho's Department of Water Resources regulation IDAPA 37.03.05.

Based on the simplified dam breach analysis and expected area of inundation downstream of the North CPSF, the consequence of a dam failure is "high" and for the south CPSF, the consequence is "significant" based on IDAPA regulations. Therefore, the facilities were designed in accordance with the recommended parameters in this guideline.

18.4.2 Facility Design

The CPSF footprints will be logged and cleared for foundation preparation and embankment construction for the north CPSF and the footprint for direct stacking in the south CPSF in phases. Basin preparation will include the removal of soft overburden material from low points within the topography. Soft overburden materials will be removed beneath the north CPSF embankment foundation prior to fill placement and the footprint of the south CPSF. The focus of material removal is expected to be within low lying points. A foundation drainage network will be developed within the base of the north CPSF embankment and the footprint of the south CPSF using selective placement of waste rock and dual wall high density polyethylene (HDPE) pipe wrapped in a non-woven geotextile fabric.

18.4.3 North CPSF

The starter east embankment will be constructed of waste rock during the pre-production period using downstream raise methodology that provides the most stable configuration of the embankment raise methods. Then a filter zone and low permeability soil zone will be constructed on the upstream face of the starter embankment overlaid with a liner low density polyethylene (LLDPE) geomembrane. Subsequent embankment raises will utilize thick layers of waste rock and filter and low permeability material using upstream embankment raises.

The waste rock and filter and low permeability materials will be transported to the CPSF using haul trucks where it will be spread and compacted with dozers and compactors into 1-m lifts. The east embankment will be constructed with overall 3:1 (H:V) interior slopes and 3:1 (H:V) exterior slopes based on stability analyses. The construction of upstream raises will continue in the same manner until the end of the South CPSF.

18.4.4 South CPSF

The south CPSF will be constructed with filtered tailings and waste rock using co-placement and co-mingling methodologies with a waste rock exterior shell to prevent erosion. Both filtered tailings and waste rock will be transported to the CPSF using haul trucks where it will be spread and compacted with dozers and compactors into 2 m to 5 m lifts. The exterior slopes will be constructed with overall 3:1 (H:V) based on stability analyses. The construction will continue in the same manner until the end of the south CPSF life.

18.4.5 Stability Analysis

A section through the highest portions of the north CPSF embankments and the highest portions of the remaining south CPSF were selected as the critical section. Stability of facilities were assessed using limit-equilibrium modelling software slope/W, (Geostudio, 2018). Analyses were undertaken for both static and pseudo-static (earthquake loading) conditions with the calculated factors of safety (FOS) higher than the minimum required values in accordance with IDAPA of 1. 5 FOS for static and 1. 0 FOS for pseudo-static. The two facilities are designed to withstand potential dynamic displacement without release of tailings during the maximum design earthquake event. The facility stability analyses exceeded both static and pseudo-static IDAPA guidelines.

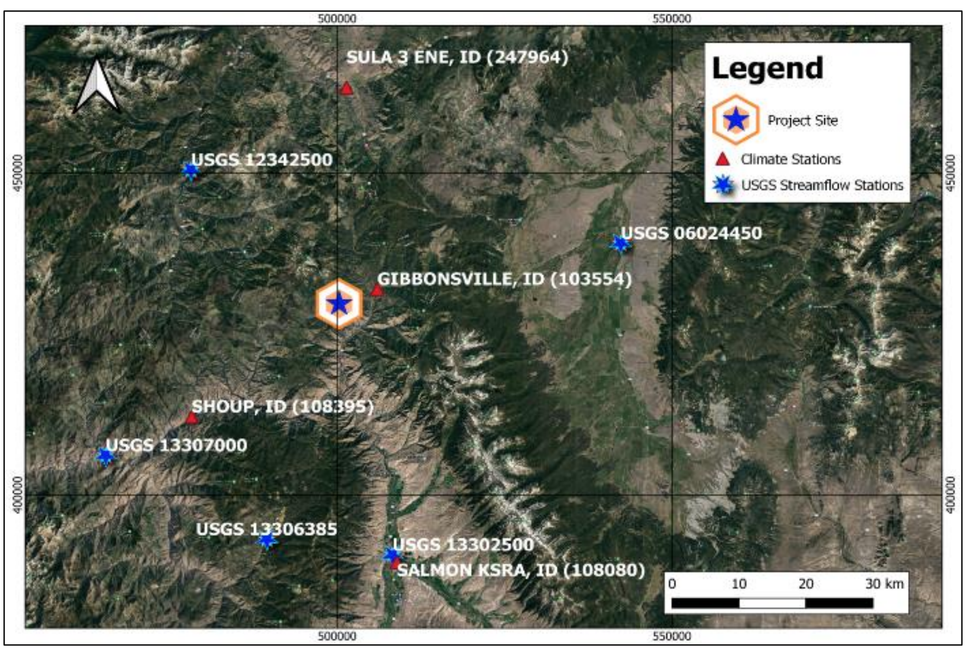
18.4.6 Monitoring

Instrumentation and monitoring will be required to CPSFs' performances and must be incorporated in the next phase of the study. Vibrating wire piezometers (VWPs) will be installed to monitor pore pressure within the CPSFs along with slope inclinometers and survey monuments will be installed in the permanent embankments and slopes to monitor slope movement and deformation.

18.5 Site Water Management

The climate stations close to the project site and with sufficient minimum data history are Gibbonsville, Shoup, Salmon KSRA and Sula 3 ENE (Figure 18-4). Table 18-2 summarizes their geographical location relative to the site and their data history period. Evaporation data was obtained from Western Regional Climate Centers (WRCC). Standard daily pan evaporation is measured using a four-foot diameter Class A evaporation pan.

Figure 18-4: Climate and Hydrometric Stations Near the Project Site



Source: Ausenco, 2022.

Table 18-2: Climate Stations Near the Lemhi Site

Station Name	Station ID	Distance to site (km)	Elevation (m)	Latitude	Longitude	First Year	Last Year
Gibbonsville	103554	5.8	1386	506069	431987	1895	2011
Shoup	108395	29	1038	478267	412189	1966	2011
Salmon KSRA	108080	42	1208	508915	389561	1967	2016
Sula 3 ENE	247964	33	1351	505974	431996	1955	2012

IDF curves were obtained from the Precipitation Frequency Atlas of the Western United States. Table 18-3 summarizes extreme storm events of the various return. Precipitation depths of these extreme events are shown below:

Table 18-3: Extreme Storm Events at Lemhi Project Site

Storm Events	Precipitation (cm)	Precipitation Intensity (cm/h)
2-year 6-hour	2.2	0.4
2-year 24-hour	3.5	0.2
100-year 6-hour	4.7	0.8
100-year 24-hour	7.5	0.3

Water management facilities were designed using the 100-year 24-hour event as the design storm.

18.5.1 Rainfall-Runoff Modelling

To estimate design flows throughout the water management system, flooding from the design event was routed along the alignments using the rational method. This method was selected due to the small drainage areas and the uniform soil and cover characteristics of the site. USGS Water stations in the vicinity of the project were prorated to estimate the design flows for the Lemhi Ditch Creek diversion channel. Several stations from the USGS water stations in the vicinity of the project site were examined, and four were selected for the regional analysis. The selected stations were chosen based on the similarity of topographic and hydrologic features, proximity to the project site and duration of available historical data. The USGS (ID:13306385) station was chosen to perform frequency analysis, and its peak flow rates were prorated based on drainage area ratio.

18.5.2 Water Management Structures

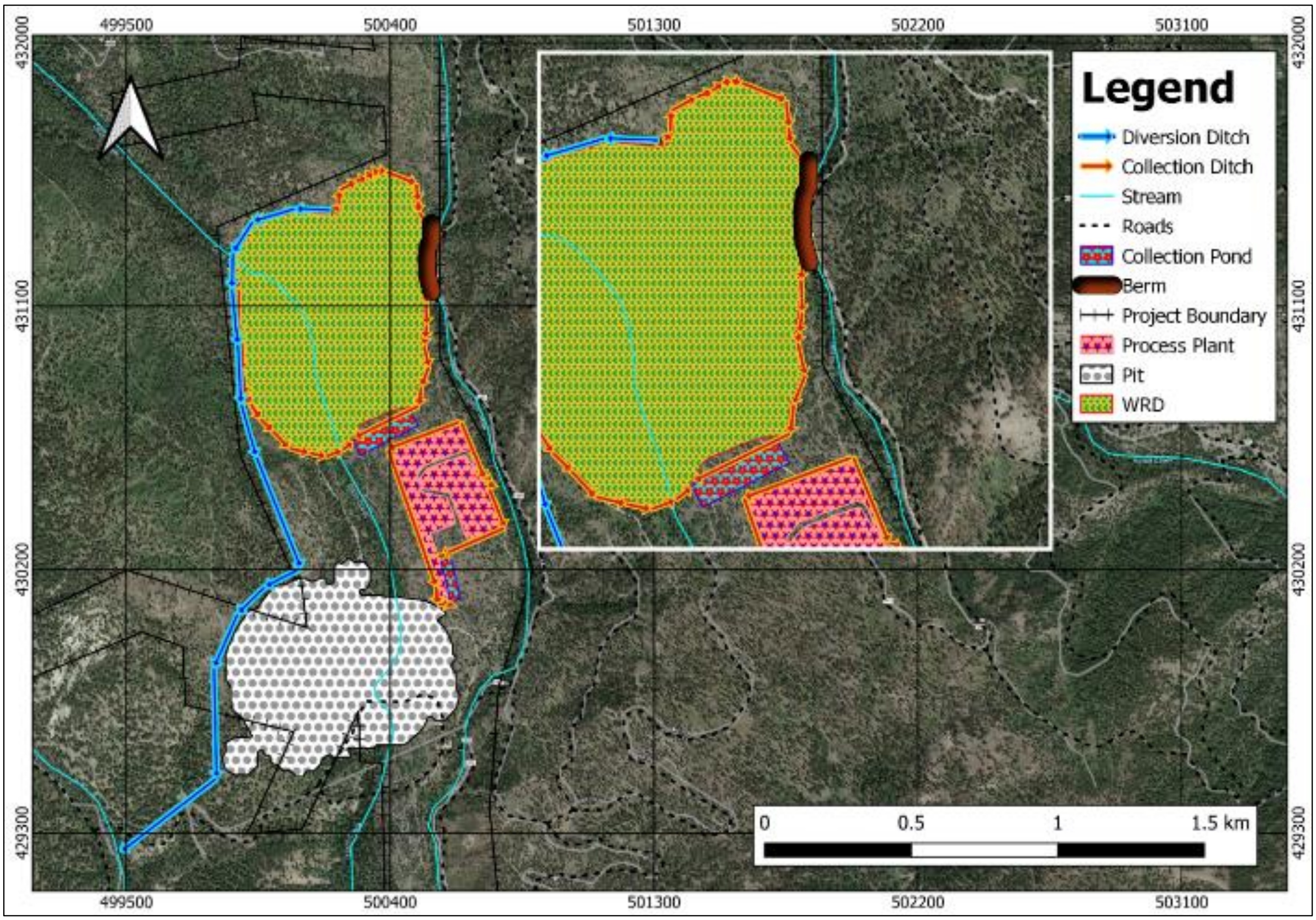
This section summarizes a list of proposed water management structures for the Lemhi mine site. The major structures are as follows:

- Diversion ditches – diversion ditches are required to divert clean runoff away from the facilities and to minimize the amount of contact runoff to be collected and managed. The design criterion for the diversion ditches was the conveyance of 1:100-year peak flow without overflow.
- Collection ditches – collection ditches collect contact runoff from the waste rock dump (WRD) and processing plant Area. The design criterion for collection ditches was the conveyance of 1:100-year peak flow without overflow.

- Collection ponds – collection ponds were proposed to store contact runoff from the collection ditches. The collection pond's design criteria were to store 1:100-year 24-hour flood with a minimum freeboard of 0.5 m. The stored contact water should be treated and released to the environment or reused for processing purposes.
- Berm – due to the close proximity of the WRD and the adjacent watercourse, Little Ditch Creek, a berm is proposed to prevent non-contacts water from migrating into the WRD. It is envisioned that this berm will be located outside of the watercourse and will serve to maintain water within the watercourse in the area of the WRD.

Figure 18-5 indicates the location of mine water management facilities relative to Lemhi pit and WRD.

Figure 18-5: Location of Mine Water Management Facilities



Source: Ausenco, 2022

18.5.2.1 Conceptual Design and Quantity Estimates

Ditches and ponds were sized using estimated peak flow rates and flood volumes from the rational Method and frequency analysis results. The collection ditches were designed to be 2.5H:1V trapezoid channels with an overall depth of 0.5 m and bottom widths ranging from 0.5 m to 3 m. The diversion ditch is a riprap lined 3H:1V trapezoid channel with an overall depth of 1.6 m and bottom width of 3 m. Collection ponds were sized with a depth of 2.5 m.

18.5.3 Site-Wide Water Balance

A preliminary site-wide water balance analysis was performed for the Lemhi Project. In this analysis, a comparison between water requirements and available water from the collection system was made to identify the site-wide water balance. This analysis has been made for the site's average, wet, and dry climate conditions. The following water components were considered in this calculation:

- Surface runoff from precipitation on WRD, process plant area and pits
- Evaporation from ponds and pits
- Process water requirement
- Tailing storage facility reclaim capacity.

As shown in Table 18-4, there is a net annual water deficit of 18 m^3/h and 33.7 m^3/h for average climate scenarios. If the water quality of collection ponds can satisfy environmental discharge requirements, the freshet excess flow can be discharged to the environment. Otherwise, flow should either be treated or pumped to TSF for storage.

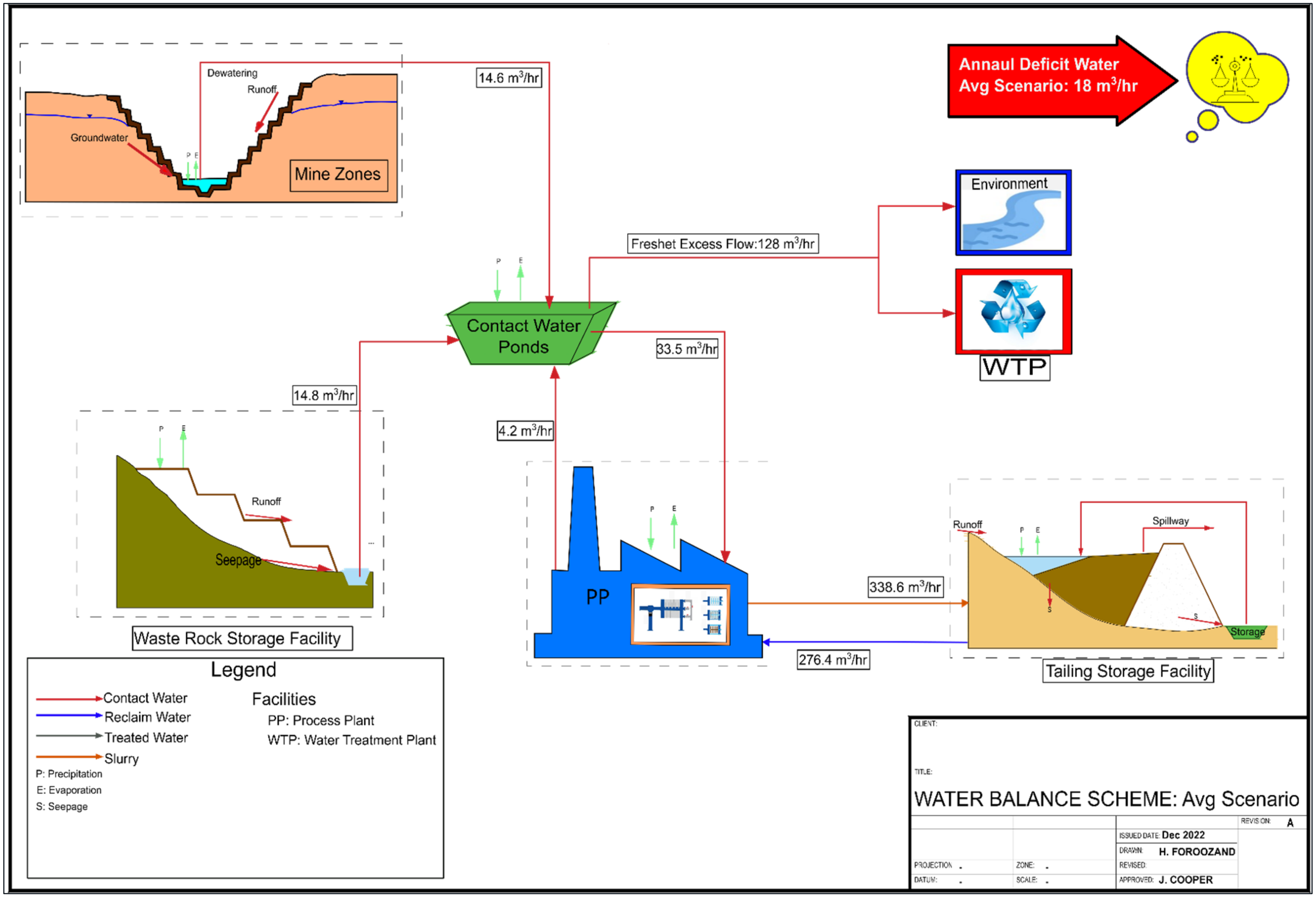
Figure 18-6 illustrates the flow diagram across the site for average climate scenarios. Note that the existing water in the final product is not shown in these figures.

Table 18-4: Site-Wide Water Balance (m^3/h)–- Average Condition

Water Component (m^3/h)	Jan	Feb	Mar	Apr	May	Jun	Jul	Aug	Sep	Oct	Nov	Dec	Annual
Process Plant water demand	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6	338.6
Raw Water Make-up	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5	51.5
Precipitation Contact Water on Pits													
Pit Precipitation	0.0	0.0	14.9	81.8	27.9	16.7	1.3	4.1	13.0	10.0	5.3	0.0	14.6
TSF Reclaim Water													
Reclaim Water from Tailings Storage Facility	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4	276.4
Contact Water from Net Precipitation and Evaporation													
Process Plant Area	0.0	0.0	3.7	20.4	7. 0	6.0	2.6	3.0	3.3	2.5	1.3	0.0	4.2
Waste Rock Dump	0.0	0.0	13.3	72.9	24.9	21.3	9.4	10.8	11.6	8.	4.8	0.0	14.8
Pond Direct Precipitation	0.0	0.0	0.7	3.9	1.3	1.2	0.5	0.6	0.6	0.5	0.3	0.0	0.8
Pond Evaporation	0.0	0.0	0.0	0.0	0.0	2.	3.8	3.3	0.0	0.0	0.0	0.0	0.8
Water Deficits/Excess (-/+)	-51.5	-51.5	-18.9	127.6	9.7	-9.4	-41.5	-36.3	-22.9	-29.6	-39.8	-51.5	-18.0

Note: The Pit dewatering values are calculated based on precipitation only. Groundwater input must be added in the next phase.

Figure 18-6: Annual Average Water Balance – Average Condition



Source: Ausenco, 2022.

19 MARKET STUDIES AND CONTRACTS

19.1 Introduction

It was assumed in this PEA that the Lemhi Gold Project will produce gold in the form of doré bars. The market for doré is well-established and accessible to new producers. The doré bars will be refined in a certified North American refinery—there are many in the United States and Canada—and the gold will be sold on the spot market.

19.2 Market Studies

Freeman Gold has not completed any formal marketing studies with regards to gold production that will result from the mining and processing of mineralized material from the project into gold doré bars. Gold 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 is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with multiple buyers and sellers active at any given time.

19.3 Commodity Price Projections

The economic analysis for the project was performed assuming a base case gold price of US$1,750/oz. This price assumption is supported by consensus forecasts from numerous financial institutions. The QP have reviewed these studies and analyses and the results support the assumptions in the technical report. As of October 13, 2023, the trailing two-year gold price was US$1,847/oz and the trailing three-year gold price was US$1,840/oz.

19.4 Contracts

Freeman Gold plans to contract out the transportation, security, insurance, and refining of doré gold bars. Freeman Gold may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in the United States and in countries throughout the world. For the PEA, a cost of US$4.30/oz Au was assumed for treatment and refining.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This section provides an overview of the setting of the Lemhi Gold Project. It outlines existing biological and physical baseline conditions, proposed new baseline studies to support future permitting applications, existing permits, and future regulatory and permitting requirements including required management plans for water, site environmental monitoring, and waste disposal. In addition, this section also discusses socio-economic baseline conditions, the status of community consultation and engagement, and conceptual mine closure and reclamation planning for the project. Recommendations are also provided if the decision is made to progress the project through the pre-feasibility study, feasibility study, environmental assessment and permitting phases.

20.2 Environmental and Social Setting

The Project is in Lemhi County in east-central Idaho, within the Salmon River Mountains, a part of the Bitterroot Range which forms the Idaho-Montana border. The property is 40 km north of the town of Salmon, ID and 6 km west of Gibbonsville, ID. The approximate centre of the property in Universal Transverse Mercator (UTM) NAD83 Idaho State Plane coordinates is Easting 500,275, Northing 429,900.

The project consists of ten patented mining claims (placer and lode), one patented mill site claim and 333 unpatented mining claims, covering an area of 2,727 ha of mineral rights The unpatented mining claims are located within the Salmon-Challis National Forest, which is federal land administered by the United States Department of Agriculture – Forest Service (USFS).

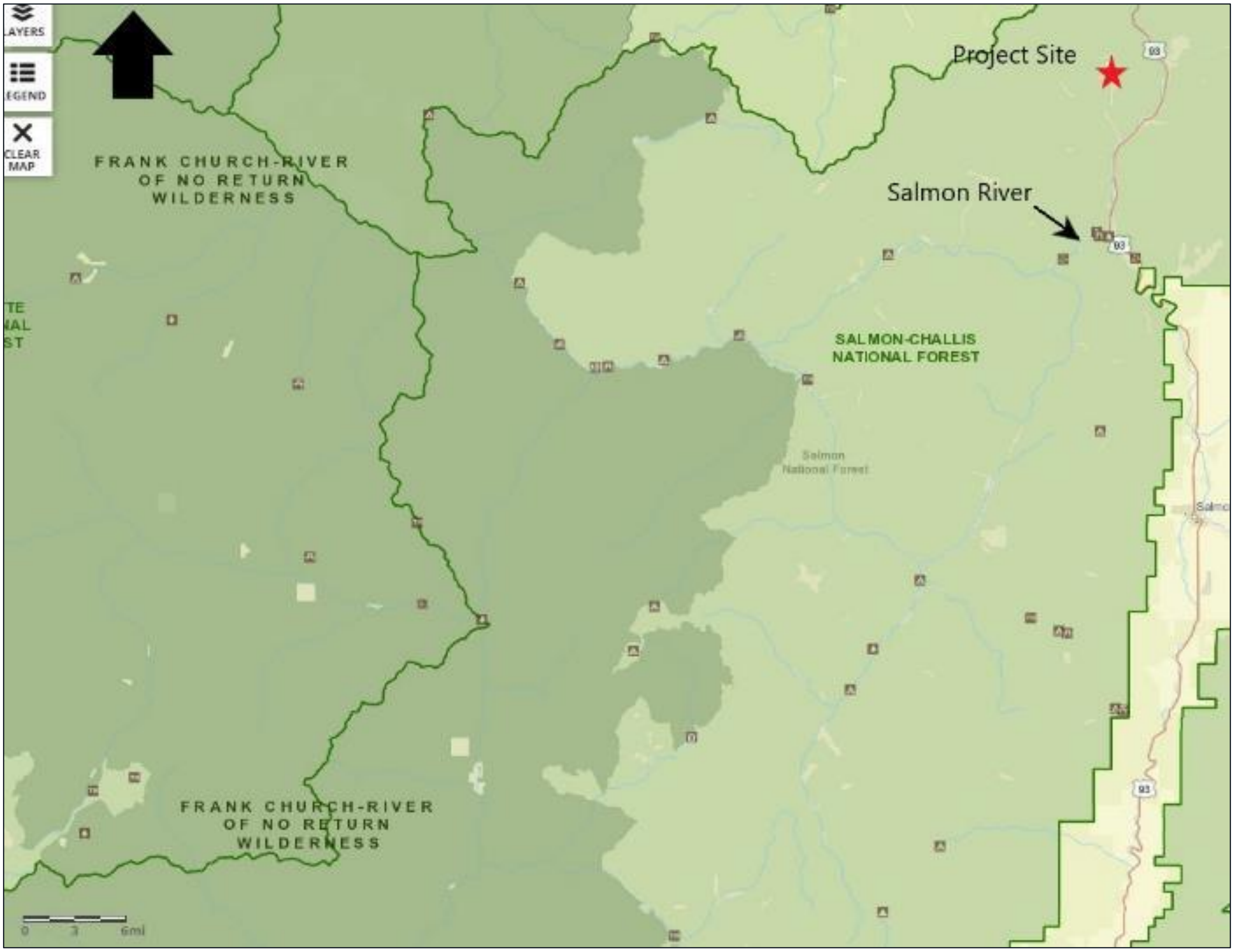
Information on the project climate and physiographic setting is included in Section 5.

There are environmentally sensitive areas located downstream of the project, including:

- The main stem of the Salmon River is located 15 km south and downstream of the project. This river hosts four federally listed fish species and provides designated critical habitat for Snake River sockeye and Snake River spring/summer chinook. It is reported to be a key area for the survival and recovery of federally listed salmon, steelhead, and bull trout. In terms of habitat, this reach is a migratory route for salmon, as well as rearing habitat for many other fish species. The segment of the main stem from the mouth of the North Fork of the Salmon River downstream to Long Tam Bar is federally designated as a National Wild and Scenic River.
- The Frank Church River of No Return Wilderness Area is a protected wilderness area located 40 km southwest of the project. This wilderness area encompasses 9,580 km^2 and protects several mountain ranges, wildlife fauna, and the Salmon River.

The location of these environmentally sensitive areas relative to the project site are shown in Figure 20-1.

Figure 20-1: Environmental Setting



Source: USFS, 2023.

The environmental baseline studies available for review include:

- Ditch Creek Baseline Fish Population and Habitat Surveys (Karen Kuzis Consulting 1995)
- ARD Potential of Humbug Project Rocks (Hart Crowser 1995)
- 1995 Baseline Monitoring Report Surface Water and Groundwater (Hart Crowser 1996)
- 1996 Baseline Monitoring Data Technical Memorandum (Hart Crowser 1996a)
- Jurisdictional Wetland Determination for Humbug Gold Project (Selkirk Environmental 1996)
- Final Terrestrial Vegetation Report (HDR Engineering 2012)

- Draft Wetland Delineation Report (HDR Engineering 2012a).

A summary of the available environmental, social and community studies and factors potentially affecting the project are provided in the following sections. It should be noted that much of the data collected for baseline studies is not recent. New baseline studies that document existing or recent conditions will be required to support baseline development and impact assessment. In assessing the utilization of older baseline data, direct discussions with state and federal regulators will be required. To support the development of this section, a desktop review of publicly available sources was conducted to supplement the information contained in the historical environmental baseline studies.

20.2.1 Hydrology and Climate

The project area is located within the Hughes Creek Basin. Hughes Creek is a tributary to the North Fork Salmon River, which subsequently flows into the Salmon River 15 km south of the project area. Project infrastructure is located within four sub-basins: Ditch Creek, Little Ditch Creek, Ransack Creek, and Humbug Creek. Most of the proposed mine facilities are within the Ditch Creek sub-basin. The location of site infrastructure is shown on Figure 18-1. The locations of the main watercourses in the vicinity of the project site are shown in Figure 20-2.

The hydrological regime of the project region is a snowmelt-dominated streamflow regime. The general area is characterized by high flows in the late spring due to snow melt and low flows during the winter months. Flows decrease through the drier summer months, with some rebound in discharges during late fall.

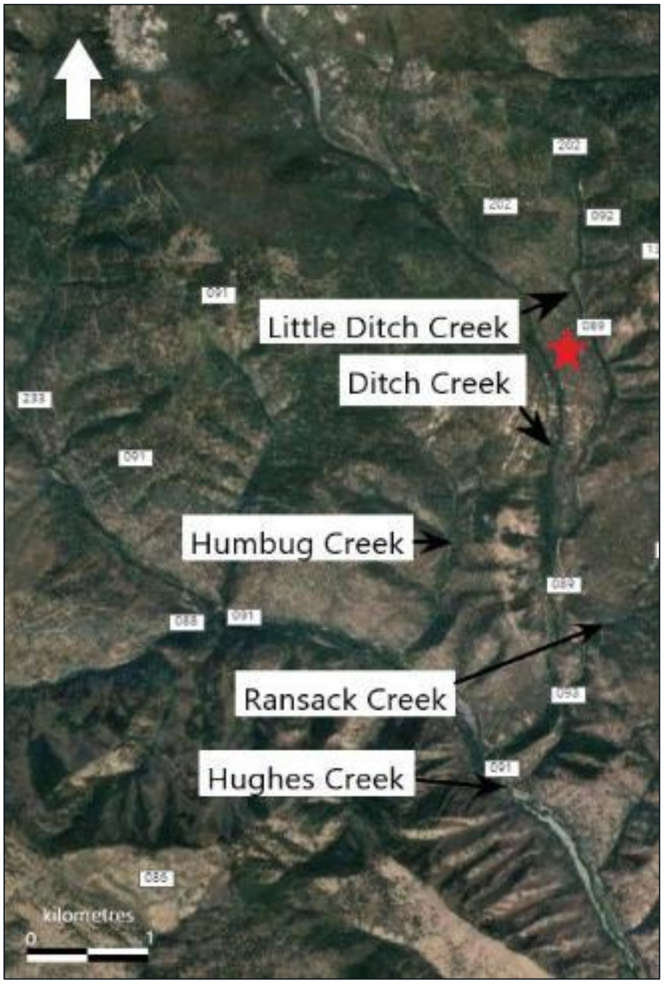
Hart Crowser conducted hydrometric monitoring in 1995-1996 (Hart Crowser 1996, Hart Crowser 1996a) with 11 hydrometric monitoring stations. The monitoring stations were located on the two streams crossing the conceptual mine site (Ditch Creek and Little Ditch Creek) and on the downstream receiving water body (Hughes Creek). The monitoring locations were located upstream, midstream, and downstream of the planned operations area and they were concentrated at confluence points for the streams. Stream flow was gauged five times each year (March, May, June, August, October). Staff gauge readings were made weekly between March and November to track shorter term and seasonable variability in flow conditions.

The data collected indicated that the Ditch Creek system accounts for 25-30% of the total flow in Hughes Creek. Changes in stream flow rates between gauging locations in Ditch Creek suggest that surface water/groundwater interaction varies seasonally and with location in the basin. During periods of relatively high flow, Ditch Creek flow decreased as it entered the project site indicating a losing stream condition with surface water entering the groundwater flow system. Evidence of losing stream conditions was not observed during low flow periods (Hart Crowser 1996).

Stream flowrates exhibited seasonal trends. Peak flows occurred at all locations during snow melt in the late spring months. Relatively high flowrates continued into June but declined to base flow levels through July and August. Base flows were observed to be relatively consistent through October, and then display a modest increase in November at the end of the monitoring period (Hart Crowser 1996).

Based on the available environmental studies reviewed, there were no indications to date of completed site-specific meteorological investigations. Section 18.9.1 describes data from meteorological stations close to the project site.

Figure 20-2: Watercourses in The Vicinity of Project Site



Source: Google, 2023.

Longer-term monitoring of the project study area will be required as the project advances through FS, EIS, and permitting to further characterize the hydrological conditions and develop a water balance model and long-term LOM water management plan. Site-specific meteorological data will also be required. Section 26.10 provides recommendations for the meteorological and hydrological studies that will support the advancement of the project through the PFS stage.

20.2.2 Surface Water Quality

Surface water quality monitoring programs were conducted in 1995-1996 by Hart Crowser (Hart Crowser 1996, Hart Crowser 1996a). The 1995-1996 monitoring programs collected preliminary baseline water quality data from the 11 monitoring stations described in Section 20.2.1. Seasonal water quality samples were collected in March, May, June, August, and October.

The monitoring programs indicated that surface water quality at the sampling locations in Ditch Creek, Little Ditch Creek, and Hughes Creek are pH neutral, with soft water hardness. Background concentrations for dissolved metals in these streams did not exceed criteria for protection of aquatic life or protection for human health as defined in IDAPA 58.01.02– Water Quality Standards. Consistency was noted in surface water quality data collected in the three streams.

Long-term water quality monitoring efforts should focus on areas potentially affected by proposed mine infrastructure (refer to Section 18) and should support the future requirements of an Environmental Impact Statement (EIS). Section 26.10 provides recommendations for the surface water quality studies that will support the advancement of the project through the PFS stage.

20.2.3 Hydrogeology and Groundwater Quality

Hart Crowser established a network of nine groundwater monitoring wells (Hart Crowser 1996, Hart Crowser 1996a) within the Ditch Creek drainage basin. The wells are located within or adjacent to the proposed mine site and were completed in two hydrogeologic regimes: unconsolidated alluvium deposits filling the valley floor and water-bearing intervals within the underlying bedrock system. Groundwater monitoring events occurred in 1995 and 1996.

A conceptual hydrogeologic model was developed for the site which includes the following key hydrostratigraphic units (Hart Crowser 1996):

- Unconsolidated deposits host the uppermost water-bearing zone present beneath the proposed mine site. Drilling indicated that these deposits consist of sand and gravel with lesser amounts of silt. Thicknesses range from 6 m to 15 m in the area between Ditch Creek and Little Ditch Creek.
- Groundwater characteristics in bedrock are variable as compared to the overlying unconsolidated deposits. Observed groundwater conditions in drill holes were variable with higher permeability horizons in fractured zones within larger intervals of relatively impervious bedrock formations. Drilling identified water-bearing horizons concentrated in two different intervals, between elevations 1,524 m to 1,554 m and 1,463 m to 1,494 m.

Water level monitoring indicated that groundwater flows towards to the axis of Ditch Creek valley and exits the basin towards the south. Consistency between groundwater elevation measured in bedrock and alluvium wells suggests continuity between the two units (Hart Crowser 1996). Depth to water below ground surface was measured at 30 m in upland areas and 3 m – 7.5 m in the valley.

Hydraulic conductivities of the two water-bearing systems were estimated using in-situ response testing methods. The alluvium hydraulic conductivity estimates were consistent with sand and gravel deposits containing a finer grained silt matrix. In the bedrock system, the hydraulic conductivity was controlled by fracture density and the interconnection between fracture sets, both of which can vary greatly (Hart Crowser 1996).

Groundwater quality data indicated that groundwater in unconsolidated alluvium differs from samples collected in the bedrock system. Groundwater samples in the bedrock system had a relatively high total dissolved solids and major cation concentrations compared to samples from the alluvial system. Piper/Stiff diagrams indicated that carbonate/bicarbonate is a major constituent in both systems, but that chloride, sodium, and potassium are significant constituents only in the bedrock system. The pH values averaged 7.5 and 10.1, in the alluvium and bedrock systems, respectively (Hart Crowser 1996, Hart Crowser 1996a). Background concentrations in total chromium, lead, iron, and manganese were consistently elevated in the alluvium system relative to IDAPA 58.01.11 – Ground Water Quality Rule. In the bedrock system, total iron was consistently elevated relative to IDAPA 58.01.11.

As the project advances through the FS, EIS, and permitting stages, groundwater monitoring and sampling data will be required to adequately support the EIS and to support the development of an integrated numerical 3D groundwater model and a long-term LOM water balance and water management plan. Section 26.10 provides recommendations for hydrogeological studies that will support the advancement of the project through the PFS stage.

20.2.4 Fish and Fish Habitat

The main waterbodies that may potentially be impacted by the project include Ditch Creek, Little Ditch Creek, Ransack Creek, Hughes Creek, and North Fork Salmon River. A summary of fish species observed in theses waterbodies based on publicly available data from the Idaho Fish and Game department is presented in Table 20-1.

Table 20-1: List of Observed Fish Species in The Project Area

Mainstem Streams	Common Name	Scientific Name
Ditch Creek	Cutthroat trout	Oncorhynchus clarkia
	Westslope Cutthroat trout	Oncorhynchus clarkia lewisi
	Rainbow trout	Oncorhynchus mykiss
	Sculpin (general)	Cottus sp.
Hughes Creek	Brook trout	Salvelinus fontinalis
	Bull trout	Salvelinus confluentus
	Chinook salmon	Oncorhynchus tshawytscha
	Cutthroat trout	Oncorhynchus clarkii
	Dace (general)	Rhinichthys sp.
	Mountain whitefish	Prosopium williamsoni
	Rainbow trout	Oncorhynchus mykiss
	Sculpin (general)	Cottus sp.
	Steelhead	Oncorhynchus mykiss
North Fork Salmon River	Brook trout	Salvelinus fontinalis
	Bull trout	Salvelinus confluentus
	Chinook salmon	Oncorhynchus tshawytscha
	Cutthroat trout	Oncorhynchus clarkii
	Dace (general)	Rhinichthys sp.
	Mountain whitefish	Prosopium williamsoni
	Northern pikeminnow	Ptychocheilus oregonensis
	Pacific lamprey	Lampetra tridentata
	Sculpin (general)	Cottus sp.
	Steelhead	Oncorhynchus mykiss
	Sucker (general)	Catostomus sp.
	Whitefish (general)	Prosopium sp.

Source: Idaho Fish and Game, 2023.

Karen Kuzis Consulting completed a baseline survey in September 1995 for fish population and habitat in Ditch Creek (Karen Kuzis Consulting 1995). The survey included a 10 km long sampling reach within Hughes Creek and Ditch Creek, starting 1 km above the confluence of Hughes Creek and the North Fork Salmon River. Twelve randomly selected 100 m long sampling sites were chosen within Ditch Creek and Hughes Creek. Fish were captured at all 12 sampling sites; species included sculpin, rainbow trout, and cutthroat trout.

There were 21 macroinvertebrate orders and families represented in the Ditch Creek samples with individuals representing all the main functional feeding groups. The orders Ephemeroptera, Plecoptera, and Trichoptera were well represented. This indicated that Ditch Creek is relatively free of pollution and provides suitable habitat complexity since these groups are considered pollution sensitive and reflect the structural complexities of stream microhabitat. The presence of significant percentages of shredders, scrapers, and filters are indicative of low levels of silt and adequate inputs from the riparian zone (Karen Kuzis Consulting 1995).

20.2.4.1 Fish Habitat

The channel in Ditch Creek appeared to be highly mobile with numerous dry overflow channels. High gradient riffles were the dominant habitat type. The interstitial spaces among cobbles and small pocket pools in the riffles were utilized by the fish sampled (Karen Kuzis Consulting 1995).

In October 2010, the U. S. Fish and Wildlife Service issued a Final Rule on designated critical habitat for Bull trout in the coterminous United States. Ditch Creek and Hughes Creek were designated as critical habitat as part of the Salmon River Basin Critical Habitat Unit (National Archives and Records Administration 2010). The Endangered Species Act prohibits the “take” (harm, harass, kill) of fish and wildlife species classified as endangered or threatened, and prohibits the destruction or adverse modification of their designated critical habitat unless otherwise authorized.

To establish a better understanding of fish community and habitat baseline conditions within the project site and to support future permitting and approvals, further sampling and assessments are recommended. Section 26.10 provides recommendations for fish and fish habitat studies that will support the advancement of the project through the PFS stage. In the long term, the baseline program for fish and fish habitat for the project should be designed to support the requirements of an environmental impact assessment. Studies on fish community and fish habitat should include other aquatic resources such as benthic invertebrates and periphyton.

20.2.5 Soils Vegetation and Wildlife

20.2.5.1 Soils and Vegetation

HDR Engineering completed a vegetation baseline assessment in June 2012. The baseline assessment included six vegetation transects, each 3 m x 3 m, within the patented mineral claims. The assessment counted all plants in each of the vegetation stratums (tree, shrub, forb, grass) for each transect. The main project area is primarily located on rocky slopes. Soils in the area are generally gravelly silt loams, loams, and sandy loams. Upland forest throughout the project area primarily consisted of Ponderosa pine, Douglas fir, snowberry, heartleaf arnica, and elk sedge. Other associated species commonly included snowbrush ceanothus, serviceberry, birchleaf spriea, silvery lupine, Virginia strawberry, pussytoes, and common yarrow. Vegetation types were consistent throughout all the transects that were taken (HDR Engineering 2012).

The U. S. Fish and Wildlife Service Information for Planning and Consultation (IPaC) web-based tool indicates that Whitebark pine (threatened) is a listed plant species potentially affected by activities at the project location (USFWS, 2023). However, the 2012 vegetation baseline assessment noted that suitable habitat for whitebark pine does not exist within the project area and the species was not observed during the vegetation survey (HDR Engineering 2012).

The U. S. Forest Service lists 55 potential plant species of conservation concern within the Salmon-Challis National Forest (USFS 2023). However, none of these species were observed within the project area during the 2012 vegetation baseline assessment (HDR Engineering 2012).

20.2.5.2 Wetland Delineation

Wetland delineations in the project area were completed in 1995 (Selkirk Environmental 1996) and 2012 (HDR Engineering 2012a). The 2012 survey delineated and mapped 15 wetlands totalling 3 acres. These wetlands include 59.7 acres of palustrine forested, 3.3 acres of palustrine scrub-shrub and 0.06 acres of palustrine emergent marsh. These wetlands were described as follows:

- Palustrine Forested Wetland: the palustrine forested wetland community typically occurs within the riparian corridor of Ditch Creek and Little Ditch Creek and is the largest wetland type found in the project area. Dominant riparian vegetation along these drainages typically includes Engelmann spruce, mountain alder, Rocky Mountain maple, thimbleberry, redosier dogwood, horsetail, marsh marigold, and various sedges.
- Palustrine Scrub-shrub Wetland: the palustrine scrub-shrub wetland community typically occurs outside the main channels of the riparian corridor of Ditch Creek and Little Ditch Creek. Scrub-shrub wetlands are typically dominated by mountain alder, Rocky Mountain maple, thimbleberry, and redosier dogwood, with an understory of various grasses, sedges, and forbs.
- Palustrine Emergent Marsh: the palustrine emergent marsh community is the least common type found within the project area. Palustrine emergent marsh typically occurs in open areas inundated by hydrology. Common vegetation species include sedges, rushes, moss, and various forbs.

The National Wetland Inventory (NWI) identifies additional wetlands in the project area that were not included in the 1995 and 2012 wetland delineations (USFWS 2023a). These wetlands include Freshwater Emergent and Freshwater Forested/Shrub vegetation communities and occur in the vicinity of the proposed comingled waste and tailings south facility.

20.2.5.3 Wildlife and Wildlife Habitat

Based on the available environmental studies reviewed, there are no indications of wildlife or wildlife habitat investigations completed to date.

The U. S. Fish and Wildlife Service iPaC web-based tool indicates the following listed species are potentially affected by activities at the project location (USFWS 2023):

- Canada lynx (threatened)
- Grizzly bear (threatened)

- North American wolverine (proposed threatened)
- Monarch butterfly (candidate for listing).

The iPaC web-based tool indicates that there are bald eagles and/or golden eagles in the project area (USFWS 2023). Bald and golden eagles are protected under the Bald and Golden Eagle Protection Act and the Migratory Bird Treaty Act. Any person or organization who plans or conducts activities that may result in impacts to bald or golden eagles, or their habitats, should follow appropriate regulations and consider implementing appropriate conservation measures.

Additional surveys will need to be completed related to the areas of terrain/soils, vegetation/ecosystem, and wildlife/wildlife habitat for the mine infrastructure presented in Section 18. Section 26.8 provides recommendations for soils, vegetation and wildlife studies that will support the advancement of the project through the PFS stage.

20.2.6 Geochemistry

Hart Crowser completed an initial acid rock drainage (ARD) characterization of the project area as part of the baseline environmental work. No previous ARD work had been completed. In June 1995, 46 samples of core and drill cuttings were collected as part of an initial ARD/ML characterization program. Rock samples represented by mineralized material and waste rock and the major rock types in the vicinity of the deposit.

The initial conclusions based on the preliminary ARD testwork were (Hart Crowser 1995):

- Waste rock samples were not considered acid producing.
- Apart from one sample, mineralized material samples had neutralization potential to acid potential ratio (NP/AP ratio greater than 3. 0) and were not considered acid producing.
- One sample of mineralized material had an NP/AP ratio equal to 1.5 and its acid generation potential was considered uncertain.

It is noted that the methodologies used in the analyses of mine rock have improved since 1995, when these studies were originally conducted. The initial ARD characterization program did not cover the entire range of disturbed area and material types that could potentially be affected by the project. ARD characterization beyond this initial assessment should continue in accordance with currently accepted methodologies. Additional sample selection and analyses have been recommended in Sections 26.8 and 26.10 to help support and advance the project through the PFS stage.

20.2.7 Socio-Economic, Cultural Baseline Studies and Community Engagement

20.2.7.1 Land Use and Cultural Heritage

Baseline socio-economic and cultural baseline studies have not yet been completed for the Lemhi Gold project. Class I (literature search), Class II (field reconnaissance), and Class III (Intensive Cultural Resources Inventory) cultural resource surveys have also not been completed. Cultural resource surveys will be required at the appropriate time as the project advances into the feasibility and permitting phases and the full extent of the disturbed footprint of the project has been identified.

20.2.7.2 Government

Freeman Gold will need to engage and collaborate with federal, state, regional, and municipal government agencies and representatives as required with respect to topics such as land and resource management, protected areas, official community plans, environmental and social baseline studies, and effects assessments. Freeman Gold will be required to participate in a project-specific working group at the early stages of the NEPA process which will include representatives from various government groups. Freeman Gold will be required to consult with the working group on project-related developments during the NEPA process.

20.2.7.3 Community and Tribal Consultation

Based on the available information, there are no indications to date of community or tribal consultation completed by Freeman Gold.

Environmental review of the project plan under NEPA will include public scoping to obtain input from the local community and tribal members and to develop alternatives to the proposed action. Furthermore, federal actions for environmental justice defined in Executive Orders 12898, 13985, and 14008 will be incorporated in the NEPA review. At this time, and based on currently available information, environmental justice issues are not an anticipated concern.

The NEPA review will also include government-to-government consultation between USFS and the Nez Perce and Shoshone-Bannock Tribes. During this consultation, a determination will be made if traditional cultural properties, cultural landscapes, sacred sites, or tribal resource collection areas exist in the areas that would be affected by the project. Tribal consultation for projects in the region as conducted by other parties has identified instream water quality and fisheries as priority issues. In consideration of the existing regulatory framework protecting water quality and aquatic resources and the associated permit compliance requirements that will apply to the project, there will be a requirement to address these issues and avoid impacts to the greatest extent possible.

20.2.7.4 Community Engagement

Based on the available information, there are no indications to date of community engagement completed by Freeman Gold.

20.3 Water Management and Waste Disposal Facilities

As described in Section 18, the preliminary design contemplates modifications and/or infill to sections of Ditch Creek, Little Ditch Creek, and Ransack Creek. The current proposal is for a diversion channel to collect water from the upstream reaches of Ditch Creek and divert flows it to Humbug Creek, which is also within the Hughes Creek watershed. This diversion will affect an 5.3 km long section of Ditch Creek between the point of diversion and the confluence with Hughes Creek. Since Ditch Creek is considered fish-bearing and critical habitat for Bull trout, this diversion will impact fish habitat along the affected reach. Humbug Creek was not surveyed during previous baseline studies; however, a desktop review of publicly available information does not indicate this waterbody is fish-bearing and it is not considered critical habitat for Bull trout. As part of the overall site water management strategy, a review of reasonable and prudent alternatives will need to be conducted as a means of minimizing or eliminating adverse impacts to fish and fish habitat.

The preliminary mine plan contemplates the establishment of two CPSFs: the North CPSF to be located within Freeman's Gold patented claim boundary, and a South CPSF to be located on federal land within the Salmon-Challis National Forest (administered by the USFS). As such, permitting and approval for the mine will be subject to the National Environmental Policy Act (NEPA) review process. An overview of the NEPA review process is described in Section 20.4.2.

20.4 Permitting

This section summarizes the existing permits in place for the project and the federal and state legislation and associated permits, licenses and approvals that will apply or potentially will apply to the construction and operations of the project, as currently proposed.

20.4.1 Existing Permits

Freeman Gold received approval of a Plan of Operations application to the USFS on May 23, 2022. The Plan of Operations (POO-2021-081646) approved drilling on 28 new pads off patented claims. It also allows testing of four high priority exploration targets and 22 resource expansion and infill drill holes within the northwest, southwest, and southern margins of the Lemhi Gold Deposit.

Freeman Gold received a Permit to Appropriate Water (Permit No. 75-15005) from the Idaho Department of Water Resources on May 23, 2022. The permit allows for water rights for both mining and domestic water use in four sections within the patented mining claims. It allows for groundwater use in Township 26N, Range 21E sections 28, 29, 32, 33 of 15.3 L/s and a maximum of 24,500 L/day for domestic use.

20.4.2 National Environmental Policy Act (NEPA) Review Process

Some project infrastructure is located on federal National Forest lands administered by the USFS. As such, permitting and approval for the mine will be subject to the NEPA review process and the requirements stipulated in a Record of Decision (ROD) for an Environmental Impact Statement (EIS) prepared by the USFS as the lead agency. The major phases of the NEPA review process are scoping (inviting review and comment on the Project Plan by the public and other interested parties to define the scope of issues to be addressed in the EIS), preparation of a draft EIS and public comment period, preparation of a final EIS, and issuance of a ROD.

The USFS regulations at 36 CFR Part 228 Subpart A require a mine be operated in accordance with an approved plan. As part of the NEPA review process, a Project Plan would be submitted to USFS describing the proposed methods of construction, operation, closure, and reclamation. Key components of the Project Plan include descriptions of mining and processing, water and waste rock management plans, and the closure and reclamation plan. The project plan will also describe the best management practices and environmental design features for protection of air, surface and groundwater, terrestrial and aquatic habitat, wetlands and riparian areas, and soils. Section 20.5 lists several plans that are anticipated to be required during the EIS/permitting process.

The NEPA process will require a thorough series of environmental baseline studies and an EIS that provides a complete property description, identification, and analysis of environmental impacts (both positive and negative) of the Project Plan. The EIS would also include the development of environmental design features to reduce or eliminate negative impacts for the proposed action.

A list of baseline studies (not necessarily complete) that will be required to support project permitting and EIS preparation are listed below:

- Air quality and meteorology
- Aquatic resources and aquatic habitat
- Cultural resources
- Geochemistry
- Geological resources
- Geotechnical hazards
- Groundwater hydrology
- Hazardous Materials
- Noise
- Scenic resources and reclamation cover materials
- Surface water hydrology
- Timber resources
- Vegetation, botanical resources, and non-native plants
- Wetlands and riparian resources
- Wildlife resources and wildlife habitat
- Access and transportation
- Climate change
- Environmental justice
- Land use and land management
- Public health and safety
- Recreation
- Social and economic conditions
- Special designations
- Tribal rights and interests.

The USFS decision would be made and recorded at the same time the final EIS is published. USFS would issue a ROD as the final step in the EIS process. The ROD by USFS to approve the Project Plan is the primary authorization allowing Freeman Gold to proceed with development of the Project. The ROD would describe the proposed action, the decision, the environmentally preferred alternative, the approved alternative, and mitigation and monitoring requirements. USFS would publish the ROD on the USFS website and notify the interested parties. The ROD would define any modifications that are required to be made to the Project Plan and approved by USFS prior to Freeman Gold beginning project activities. The ROD would also define the requirements for supporting plans as components of the Project Plan. These supporting plans will describe the environmental design features, monitoring programs, and mitigation measures developed for the Project. Finally, the ROD would define the permits that must be obtained from Federal, State of Idaho, and Lemhi County agencies prior to Freeman Gold beginning project activities. The Project Plan must be updated as necessary to incorporate the terms of the ROD prior to final approval.

20.4.3 Anticipated Federal, State, and County Approvals and Authorizations

Table 20-2 presents a preliminary list of the key federal, state, and county authorizations, licenses, and permits that will be required to develop the project.

Table 20-2: Preliminary List of Federal, State, and County Permits Likely Required for The Lemhi Gold Project

Legislation	Issuing Agency	Authorization	Purpose
Federal			
National Environmental Policy Act (NEPA)	U. S. Department of Agriculture – Forest Service (USFS) – Lead Agency	EIS Record of Decision (ROD)	Minimize or avoid adverse environmental, heritage, health, social, and economic effects and incorporate environmental factors and Indigenous and stakeholder consultation into decision making.
National Forest Roads and Trails Act	USFS	Road Use Permit	Authorizes use of a Forest Service Road.
National Forest Management Act		Timber Sale Permit and Contract	Authorizes cutting and removal of trees of merchantable size from National Forest.
Endangered Species Act	U. S. Fish and Wildlife Service (USFWS)	Biological Opinion (BO)	Protect and recover imperilled species and the ecosystems upon which they depend.
Clean Water Act	U. S. Army Corps of Engineers (USACE)	Section 404 Dredge and Fill Permit	Authorizes placement of fill material (including mine tailings and waste rock) into "Waters of the United States".
Communications Act	Federal Communications Commission (FCC)	Permit	Authorization is required for use of radio equipment on site.
Safe Explosives Act	Bureau of Alcohol, Tobacco, Firearms and Explosives (BATFE)	Permit	Explosives authorizations are required during construction and operations. Permits are required for transport, storage, and use of explosives.
Federal Mine Safety and Health Act	Mine Safety and Health Administration (MSHA)	Mine Identification Number	An MSHA Mine ID is required for each U. S. mine site before any operations may begin.
State of Idaho			
Clean Water Act	Idaho Department of Environmental Quality (IDEQ)	Idaho Pollutant Discharge Elimination System (IPDES) Permit	An IPDES Permit is required for point source discharges from the mining operation to "Waters of the United States".
		Construction General Permit (CGP)	Authorizes stormwater discharges associated with construction activity.
		Multi-Sector General Permit (MSGP)	Authorizes stormwater discharges associated with industrial activity.
		Section 401 Certification	A federal agency may not issue a permit or license to conduct any activity that may result in any discharge into "Waters of the United States" unless a Section 401 water quality certification is issued by the State.
Clean Air Act		Air Quality Permit to Construct and Operate	Authorizes discharge of airborne emissions to the environment.
Groundwater Quality Protection Act		Point of Compliance (POC) Permit	Outlines monitoring, sampling, and reporting requirements that meet groundwater quality standards.
Idaho Code, § 39-118A		Cyanidation Permit	Authorizes operation, closure, and reclamation of processing facilities that utilize cyanidation.
Safe Drinking Water Act		Drinking Water System License	Authorizes operation of a drinking water system on site.
		Wastewater Treatment System Permit	Authorizes wastewater treatment and disposal on site.
Idaho Solid Waste Facilities Act		Solid Waste Permit	Authorizes construction and operation of solid waste landfill on site.
Idaho Code §42-2	Idaho Department of Water Resources (IDWR)	Consumptive & Non-Consumptive Water Rights	Authorizes diversion, storage, and use of water on site.
Idaho Dam Safety Act		Dam Safety Approval (if needed)	Authorizes construction of water retaining dams and mine tailings impoundments.
Idaho Stream Channel Protection Act		Stream Channel Alteration Permit	Authorizes modification, alteration, or relocation of any stream channel within or below the mean high-water mark.
National Historic Preservation Act	Idaho State Historic Preservation Office (SHPO)	Section 106 Project Review	Approval of a historic/cultural resources assessment by the SHPO.
Idaho Code § 39-1 and § 39-36	Idaho Department of Health and Welfare (ISHW)	Septic System Approval	Authorizes operation of an on-site septic system.
Idaho Code §47-15	Idaho Department of Lands (IDL)	Mine Operating Plan (PRO) Approval	All surface mines must submit and obtain approval of a comprehensive mine operating plan for mining activities on patented land.
		Mine Reclamation and Closure Plan (RCP) Approval	All surface mines must submit and obtain approval of a comprehensive reclamation plan for mining activities on patented land.
Idaho Code §40-312, 49-201, *49-1001, 49-1004, and 49-1005*	Idaho Transportation Department (ITD)	Special Use Permits	Authorizes vehicles or loads on highways which exceed the sizes and weights allowed by *Idaho Code* §49-1001, 49-1002, or 49-1010.
Lemhi County			
	Planning and Zoning Department	Conditional Use Permits	Authorizes conditional use permits, as required, in accordance with the Lemhi County Development Code.
	Building Department	Building Permits	Authorizes building according to building code regulations.
	Road Department	Annual Road Use Permits	Permit addresses standard operating procedures for the County maintained road route to be used, snow removal, dust suppression, and seasonal load limits.

20.4.3.1 Idaho Joint Review Process

The IDL is responsible for implementation of the Idaho Joint Review Process, which was established to coordinate and facilitate the overall mine permitting process in the state. It involves an interagency memorandum of understanding (MOU) between involved state and federal agencies and addresses a process to achieve pre-analysis coordination in approving/administering exploration permits, interagency agreement on plan completeness, alternatives considered, draft and final permits, bonding during mine plan analysis, and interagency coordination related to compliance, permit changes and reclamation/closure for major mining projects. In Idaho, the Joint Review Process was established as the basis for interagency agreement (state, federal, and local) on all permit review requirements. The focus of the process is concurrent analysis timelines. This would include, for example, in the case of the Lemhi Gold Project the NEPA process, IPDES permit, Section 404 permit, Section 401 Certification of these key permits, and the state Cyanidation permit. The Idaho Joint Review Process would play a key role in achieving two primary permitting goals: 1) increased communication and cooperation between the various involved governmental agencies, and 2) reduced conflict, delay, and costs in the permitting process.

20.4.3.2 Anticipated Major Federal Authorizations

Endangered Species Act

The purpose of the Endangered Species Act (ESA) is to protect and recover imperiled species and the ecosystems upon which they depend. Under ESA Section 7, federal agencies must consult with the USFWS or the National Oceanic and Atmospheric Administration (NOAA) Fisheries, depending on the species, when any action the agency carries out, funds, or authorizes (such as through a permit) may affect a listed endangered or threatened species. The ESA prohibits the “take” (harm, harass, kill) of fish and wildlife species classified as endangered or threatened, and prohibits the destruction or adverse modification of their designated critical habitat unless otherwise authorized. Federal agencies are required to “conserve endangered or threatened species, and to ensure that their actions are not likely to jeopardize the continued existence of any of these species or adversely modify their designated habitat” (ESA, 16 U. S. C. Section 1538a). Some adverse effect is allowable, with the issuance of an incidental take permit made pursuant to a Biological Opinion (BO) by the USFWS or NOAA. The BO must first determine that the “federal action” (issuance of federal permits in this case) would not jeopardize the existence of the species.

The following listed species are, or may be, in the vicinity of the project site. Consultation under ESA Section 7 is required on any federal action that may affect these species or their designated critical habitats (*):

- Bull trout (threatened)*
- Canada lynx (threatened)
- Grizzly bear (threatened)
- Whitebark pine (threatened)

One additional species, North American wolverine (currently listed as proposed threatened), may follow a Conferencing pathway during ESA consultation. An agency must “Conference” with the USFWS or NOAA Fisheries on any agency action that is likely to jeopardize the continued existence of any species proposed to be listed or to destroy or adversely modify critical habitat proposed to be designated for such species.

Clean Water Act

A Dredge and Fill Permit (or "Section 404 permit") is required under Section 404 of the Clean Water Act for the discharge or dredged or fill material placed into Waters of the United States. A 2009 U. S. Supreme Court decision found mine tailings to be "fill", and can, therefore, be placed into Waters of the United States with an approved Section 404 permit. Fill materials include tailings and waste/development rock. Other activities that may require a Section 404 permit are:

- road construction
- bridges
- construction of dams for water storage
- stream diversions
- other infrastructure (power transmission line)
- certain reclamation activities.

The final "Revised Definition of 'Waters of the United States'" rule was published in the Federal Register on January 18, 2023, and the rule took effect on March 20, 2023. However, the final rule is not currently operative in certain states, including Idaho, and for certain parties due to litigation. Instead, the pre-2015 regulatory regime is being implemented.

Based on the pre-2015 definition, watercourses through the project area and their adjacent wetlands are likely to be considered "Waters of the United States" and will be subject to the Section 404 permit process.

A complementary Section 401 water quality certification will be required from the State of Idaho. A federal agency may not issue a permit or license to conduct any activity that may result in any discharge into "Waters of the United States" unless a Section 401 water quality certification is issued by the State.

20.4.3.3 Anticipated Major State Authorizations

20.4.3.3.1 Wastewater Discharge Permits

In 2018, the U. S. Environmental Protection Agency (EPA) granted authority to IDEQ to administer and enforce the IPDES (formerly the NPDES) program. The IDEQ will administer the approved IPDES program regulating discharges of pollutants into waters of the United States under its jurisdiction.

An IPDES permit is required for point source discharges from the mining operation to "waters of the United States". Likely point discharges would include treated mine drainage and any other discernible or discrete point associated with mining and processing at the site. Additionally, the project would be subject to performance standards for new sources for its respective industrial source category. This means the project would have to demonstrate that it is applying the best available control technology to meet applicable water quality standards.

Permitting authority for stormwater permits transferred to IDEQ from EPA in 2021. Stormwater discharges associated with this industrial activity can be authorized under related permits, the Construction General Permit (CGP) during construction and the Multi-Sector General Permit (MSGP) during operations. Stormwater is defined as "rain or melting snow that does not immediately soak into the ground."

Where flows are from conveyances that are not contaminated by contact with overburden or other mine waste, a permit is not required. Hence, the water management scheme developed for the project endeavors to collect and convey clean water around the mining operation and discharge downstream the extent possible.

20.4.3.3.2 Air Quality Permit to Construct and Operate

This permit is required by IDEQ prior to construction. The IDEQ Air Permit to Construct assesses the air pollutant emissions from stationary sources, determines the allowable impacts to air quality and prescribes measures and controls to reduce and/or mitigate impacts.

20.4.3.3.3 Groundwater Point of Compliance (POC) Permit

The State of Idaho's policy is to protect groundwater while allowing mining activities to take place. To implement this policy, the "Ground Water Quality Rule" allows mine operators to request that IDEQ set points of compliance around the mining area, as opposed to within the mining area, which outline monitoring, sampling, and reporting requirements that meet ground water quality standards. This practice is designed to ensure that mining activities do not adversely impact groundwater and interconnected surface waters while enabling extraction to take place.

20.4.3.3.4 Cyanidation Permit

This permit is required by IDEQ and is applicable for cyanidation facilities, defined as "That portion of a new processing facility, or a material modification or a material expansion of that portion of an existing processing facility, that utilizes cyanidation and is intended to contain, treat, or dispose of cyanide containing materials including spent mineralized material, tailings, and process water." The project process flowsheet intends to produce gold doré on site and uses cyanide in its production.

20.4.3.3.5 Stream Channel Alteration Permit

This permit is required by IDWR for a modification, alteration, or relocation of any stream channel within or below the mean high-water mark. The preliminary design contemplates modifications to Ditch Creek, Little Ditch Creek, and Ransack Creek as part of the overall conceptual mine plan. This permit would be obtained in conjunction with any Section 404 permit obtained for the same purpose.

20.4.3.3.6 Dam Safety Approval (if needed)

The IDWR must approve construction of dams greater than 3 m high impounding a reservoir exceeding 61,674 m^3 in volume. The Application to Construct a Dam includes design plans and specifications for construction of the dam. Mine tailings impoundments greater than or equal to 9.1 m high are regulated by IDWR in the same manner. Design and construction requirements for mine tailings impoundment structures are described in IDAPA 37.03.05; water dams are described in IDAPA 37.03.06.

20.4.3.3.7 State Historic Preservation Office

The historic/cultural resources assessment will need approval by the SHPO.

20.5 Environmental Management and Monitoring Plans

As the project progresses through the PFS and EIS/permitting stages, environmental management and monitoring plans will be required to guide the development and operation of the project and mitigate and limit environmental impacts. These plans will be complementary to the engineered designs that will be required for the storage of tailings, waste rock, mineralized material, and conveyance/storage (refer to Section 18 of this report). A preliminary list of the plans that should be considered are provided below.

- Mine plan of operations
- Explosives management plan
- Hazardous materials management lan
- Waste management plan
- Spill prevention, control, and countermeasure (SPCC) plan
- Emergency response plan
- Fire prevention and response plan
- Wildlife management plan
- Greenhouse gas inventory management plan
- Public access control plan
- Waste rock management plan
- Geochemical characterization and monitoring plan
- Spill prevention and response plan
- Mine site traffic management plan
- Fugitive dust control plan
- Terrestrial and aquatic habitat management plan
- Water quality monitoring plan
- Reclamation and closure plan
- Revegetation plan
- Invasive plant management plan

- MSHA ground control plan
- MSHA Part 48 Training plan.

20.6 Other Potential Environmental Concerns

There are no active Superfund Sites in the vicinity of the Project area (USEPA 2023).

20.7 Conceptual Mine Closure and Reclamation Plan

All surface mines must submit and obtain approval of a comprehensive reclamation and closure plan for mining activities on patented land as administered by the Idaho Department of Lands. This includes detailed operating plans showing pits, mineral stockpiles, overburden piles, tailings facilities, haul roads, and all related facilities. A reclamation and closure plan must also address applicable best management practices and provide for financial assurance in the amount necessary to reclaim those mining activities.

The Mine Plan of Operations (MPO) submitted to USFS under the NEPA process must also include a reclamation and closure plan. In addition, a reclamation report with a Reclamation Cost Estimate (RCE) for the closure of the project is required. USFS will review the RCE, and the bond is determined prior to USFS issuing a decision on the MPO.

A key closure objective for the mine will be for effluent to meet applicable water quality objectives without ongoing treatment. The current conceptual closure and reclamation plan for the project includes the following measures:

- Partial backfilling of open pits with waste rock and flooding of the remaining open pit.
- The mineralized material stockpile will be reclaimed, once depleted.
- The surface infrastructure on the site will be decommissioned and removed from the site upon completion of mining.
- Explosives, explosives magazines, fuel, and storage facilities will be removed from the site.
- Concrete slabs and footings will be broken and placed appropriately to meet project closure and reclamation objectives.
- Process buildings, pipelines, conveyor systems, and equipment will be removed from site or appropriately landfilled in an approved facility.
- CWTF will be re-contoured for geotechnical stability, capped with a graded earthfill/rockfill cover to facilitate runoff and minimize infiltration, and revegetated.
- Compacted surfaces including laydowns, civil pads, and roads will be decompacted, re-contoured, capped with a graded earthfill/rockfill cover to facilitate runoff and minimize infiltration, and revegetated.
- Water treatment will be continued until water quality meets discharge criteria. Once water quality meets discharge criteria, water treatment will be stopped, diversions will be decommissioned, and the site will be allowed to discharge naturally.

- For mine roads, Freeman Gold will remove culverts and install cross-ditches for drainage. The mine site access road will not be deactivated as it will be required for access for continued reclamation activities and monitoring.

Closure planning will include engagement with stakeholders to determine post-mining land use objectives and necessary investigations required to achieve and monitor those objectives.

The estimated closure and reclamation costs are discussed in Section 21.2.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the Lemhi Gold Project. The calculations are based on an open pit mining operation, a processing plant to produce gold doré, off-site and on-site infrastructure, a tailings storage facility, and the owner's expenses and provisions. The Project anticipates a LOM for 11.2 years, with provisions in place for an expansion scheduled during the fifth year. Initially, the plant is projected to handle a throughput of 2.5 Mt/a, with this capacity set to increase to 3.0 Mt/a following the expansion.

21.2 Capital Costs Estimate

21.2.1 Capital Cost Summary

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with +50%/-30% accuracy based on guidelines from the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2023 US dollars based on Ausenco's in-house database of projects and advanced studies, as well as experience from similar operations.

The estimate includes open pit mining, processing, on-site infrastructure, tailings and waste rock facilities, off-site infrastructure, project indirect costs, project delivery, owner's costs, contingency, and expansion costs. The capital cost summary is presented in Table 21-1. The total initial capital cost for the Lemhi Gold Project is US$190.2 M; and LOM sustaining costs are US$101.2 M. Closure costs are estimated at US$29.9 M, with salvage credits of US$12.0 M.

Table 21-1: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost LOM (US$M)	Expansion Cost (US$M)	Total Capital Cost LOM (US$M)
1000	Mine	41.3	60.4	2.1	103.8
3000	Process Plant	67.0	1.7	3.5	72.2
4000	Tailings	10.2	37.9	-	48.1
5000	On-Site Infrastructure	18.5	0.2	-	18.7
6000	Off-Site Infrastructure	2.3	-	-	2.3
	Total Directs	**139.2**	**100.2**	**5.6**	**245.1**
7100	Field Indirects	6.4	-	0.3	6.6
7200	Project Delivery	11.8	-	0.4	12.2
7500	Spares + First Fills	2.9	1. 0	0.2	4.1
8000	Owner's Cost	3.7	-	-	3.7
	Total Indirects	**24.7**	**1.0**	**0.9**	**26.6**
9000	Contingency	26.2	-	1.1	27.3
	Project Total	**190.2**	**101.2**	**7.6**	**298.9**

Note: Totals may not sum due to rounding.

21.2.2 Basis of Capital Cost Estimate

All capital and operational costs were developed based on Ausenco's in-house database of costs and labour rates. The estimate is prepared in the base currency of United States dollars (currency: USD; symbol: US$). Where applicable, pricing has been converted to United States dollars based on exchange rate of 0.74 (US$/C$).

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

- Mine schedules
- Conceptual engineering design by Ausenco and MMTS
- Major mechanical equipment costs are based on vendor quotations, first principles, and Ausenco's database of historical projects
- Cost for concrete, steel, instrumentation, in-plant piping, and platework were factored by benchmarking against similar projects with equivalent technologies and unit operations
- Engineering design at a PEA level
- Topographical information was considered
- Data from similar recently completed studies and projects
- A growth and contingency allowance were included.

21.2.3 Mine Capital Costs (WBS 1000)

Mine capital costs have been derived from historic data collected by MMTS at other US and Canadian open pit mining operations, applied to the Lemhi mine plan and PEA production schedule.

Pre-production mine operating costs (i.e., all mine operating costs incurred before mill start-up) are capitalized and included in the capital cost estimate. Pre-production pit operating costs include drill and blast, load and haul, support, and general mine expense (GME) costs. All mine operations site development costs—such as clear and grub, topsoil stripping, haul road construction, stockpile preparation, pit dewatering, and explosive pad preparation—are capitalized.

The mine equipment mobile fleet is planned to be purchased either through financing or lease agreements with the vendors. Down payments and monthly lease payments are capitalized through the initial and sustaining periods of the project. Down payments are applied when the equipment is required for operations, but lease payments have been deferred until one year after the equipment is put into operations. All expansion and replacement fleet purchases made after Year 5 of the project are planned as capital (non-lease) purchases.

The following items are also capitalized:

- Explosives storage facilities and magazine
- Site GPS (global positioning system) and machine guidance systems
- Mine survey gear and supplies
- Radio communications systems
- Geology, grade control, and mine planning software licenses
- Maintenance tooling and supplies
- Mine rescue gear and safety supplies.

Table 21-2 summarizes the mine area capital cost estimates for the Lemhi PEA Project. The QP's opinion is these estimates are reasonable for the location and planned mine development and can be used for a PEA.

Table 21-2: Lemhi Mine Area Capital Cost Summary

WBS	Item	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)	Expansion Capital Cost (US$M)
1100	Pre-Strip	26.4	-	-
1200	Mine Infrastructure	2.2	-	-
1300	Mining Equipment	12.7	60.4	2.1
	Mining Total	**41.3**	**60.4**	**2.1**

Note: Totals may not sum due to rounding.

21.2.4 Process Capital Costs (WBS 3000)

Process plant costs are summarized in Table 21-3. Direct costs include all contractors' direct and indirect labour, permanent equipment, materials, freight, and mobile equipment associated with the physical construction of the areas.

Process equipment requirements are based on conceptual process flowsheets and process design criteria as defined in Section 17. Major mechanical equipment was sized based on the process design criteria to derive equipment lists. Mechanical equipment, electrical equipment, and building supply costs were based on recent and historical budget quotes from similar projects, adjusted to reflect the size of the project.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering designs were completed to a PEA study level of definition. Bulk material quantities were derived for earthworks and priced from other benchmark projects. All other quantities for electrical and instrumentation, concrete, steel, piping, cable, and platework were factored and priced.

Table 21-3: Summary of Process Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)	Expansion Capital Cost (US$M)
3100	Crushing	11.9	-	-
3200	Grinding	21.8	-	0.3
3300	Leach	9.9	-	-
3400	Elution/Gold room	8.0	-	-
3500	Detox	2.1	-	-
3600	Reagents	2.1	-	0.6
3700	Water Services	3.2	-	-
3800	Plant General	7.9	-	0.5
3900	Thickening	-	1.7	2.0
	Process Plant Total	**67.0**	**1.7**	**3.5**

Note: Totals may not sum due to rounding.

21.2.5 Tailings Capital Costs (WBS 4000)

The breakdown of the tailing facilities capital cost is presented below in Table 21-4. These facilities include tailings handling and tailings storage facility.

Table 21-4: Summary of Tailings Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
4100	Comingled facility	8.9	36.1
4200	Outside facility	0.1	0.0
4300	Pond	0.6	0.4
4400	Diversion channel	0.4	0.8
4500	Temporary Ditch for Construction	0.3	0.6
	Tailings Total	**10.3**	**37.9**

Note: Totals may not sum due to rounding.

21.2.6 On-Site Infrastructure Capital Costs (WBS 5000)

On-site infrastructure costs are summarized in Table 21-5. The costs were developed based on Ausenco's in-house database of costs and include the following:

- site preparation and bulk earthworks
- site power distribution
- fuel storage
- warehousing, office and workshops
- site services
- site water management services
- on-site roads
- assay/met lab.

Table 21-5: Summary of On-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
5110	Site preparation	6.4	0.2
5130	Site power distribution	5.9	-
5140	Site water distribution (piping)	0.2	-
5160	Fueling station	0.2	-
5180	Plant/haul roads	0.5	-
5210	On-site buildings (assay/met lab)	1.6	-
5220	Admin buildings and change rooms	0.4	-
5230	Security and gatehouse	0.0	-
5230	Workshops and warehouse	1.1	-
5240	Truck shop	2.1	-
	On-Site Infrastructure Total	**18.5**	**0.2**

Note: Totals may not sum due to rounding.

21.2.7 Off-Site Infrastructure Capital Costs (WBS 6000)

Off-site infrastructure costs are summarized in Table 21-6. The costs were developed based on Ausenco's in-house database of costs and labour rates. The off-site infrastructure includes upgrades to the main access road, water supply, pipeline, and high voltage power supply.

Table 21-6: Summary of Off-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)
6100	Main access road	0.8	-
6200	Power supply	1.5	-
	On-Site Infrastructure Subtotal	**2.3**	**-**

Note: Totals may not sum due to rounding.

21.2.8 Indirect Capital Costs (WBS 7000 and 8000)

Indirect capital costs are calculated as a percentage of the direct non-mining costs. Indirect costs are summarized in Table 21-9 and described in the following subsections.

Project indirect costs include the following:

- Temporary construction facilities and services
- Commissioning representatives and assistance
- On-site materials transportation and storage
- Spares (commissioning, initial, and insurance)
- Freight and logistics
- Engineering, procurement, and construction management services.

Table 21-7: Summary of Indirect Costs

WBS	WBS Description	Initial Capital Cost (US$M)	Sustaining Capital Cost (US$M)	Expansion Capital Cost (US$M)
7100	Field indirects	6.4	-	0.3
7200	Project delivery	11.8	-	0.4
7300	Spares + first fills	2.9	1.0	0.2
8000	Owner's costs	3.7	-	-
	Indirects Total	**24.7**	**1.0**	**0.9**

Note: Totals may not sum due to rounding.

21.2.8.1 Project Indirects (WBS 7100)

The project field indirect costs have been based on Ausenco's historical project costs of similar nature and have been included at a rate of 6.5% of the total direct non-mining cost or US$6.4 M.

21.2.8.2 Project Delivery (WBS 7200)

The project delivery cost has been calculated at 12% of total non-mining direct costs based on Ausenco's historical project costs of similar nature. This includes the following:

- Engineering, procurement, and construction management services (EPCM)
- Commissioning services.

Project delivery costs are estimated at US$11.8 M.

21.2.8.3 Spares and First Fills (WBS 7500)

The spares and first fills have been calculated at 3% of total non-mining direct costs based on Ausenco's historical project costs of similar nature and are estimated at US$2.9 M. An additional US$1.0 M (1% of total non-mining direct costs) allowance is allocated for operating spares in the first year of operation.

21.2.8.4 Owner's (Corporate) Capital Costs (WBS 8000)

The owner's costs are estimated as 3% of total direct costs and are calculated to be US$3.7 M. Owner's cost include the following:

- Project staffing and miscellaneous expenses
- Pre-production labour
- Home office project management
- Home office finance, legal, and insurance.

21.2.8.5 Closure Costs

The estimated total reclamation and closure costs, exclusive of taxes and contingency, for the Lemhi Project is US$29.9 M.

21.2.8.6 Salvage Value

Salvage value for the Lemhi Project is estimated at US$12.0 M. Salvage value was calculated as 15% of the total process and mining equipment direct costs.

21.2.9 Contingency (WBS 9000)

Contingency accounts for the difference in costs from 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, it is essential that the estimate include a provision to cover the risk from these uncertainties.

The estimate contingency does not accommodate the following:

- Abnormal weather conditions
- Changes to market conditions affecting the cost of labour or materials
- Changes of scope within the general production and operating parameters
- Effects of industrial disputations
- Financial modelling
- Technical engineering refinement
- Estimate inaccuracy.

The estimated contingency includes the following:

- 5% of total mining direct costs during construction
- 20% of total non-mining direct costs during construction
- 20% of total indirect costs during construction
- 10% of total owner's costs during construction.

The total estimated contingency for the project is US$26.2 M during construction and US$1 M during expansion.

21.2.10 Life-of-mine (LOM) Sustaining Capital

21.2.10.1 Overview

The LOM sustaining cost for the project is estimated at US$101.2 M, which includes US$60.4 M in mine sustaining costs and US$40.8 M in additional facilities costs.

21.2.10.2 Mining

Down payments and monthly lease payments for the mine equipment fleet purchased throughout the life-of-mine (LOM) are capitalized through the sustaining periods of the project. Down payments are applied when the equipment is required for operations, but lease payments have been deferred until one year after the equipment is put into operation. All expansion and replacement fleet purchases made after Year 5 of the project are planned as capital (non-lease) purchases.

A sustaining total of US$60.4M was estimated for mine equipment purchases.

21.2.10.3 Additional Facilities

The sustaining cost under additional process facilities includes the tailings storage facility expansion and process plant costs. The total estimated LOM sustaining costs of tailings storage facility and processing plant are US$37.9 M andUS$1.7 M, respectively. An additional US$1.0 M capital is allocated for spares in the first year of operation.

21.2.11 Exclusions

The following costs and scope are excluded from the capital cost estimate:

- land acquisitions
- taxes not listed in the financial analysis
- sales taxes
- scope changes and project schedule changes and the associated costs
- any facilities/structures not mentioned in the project summary description
- geotechnical unknowns/risks
- further testwork and drilling programs
- environmental approvals
- this study or any future project studies, including environmental impact studies
- operational readiness costs
- working capital
- any facilities/structures not mentioned in the project summary description.

21.3 Operating Costs

Operating costs for the project consist of those related to mining, processing of mineralized material, tailings disposal, maintenance, power, and general administration activities.

A summary of the operating costs is presented below in Table 21-9.

Table 21-8: Summary of Operating Costs

Cost Area	LOM Cost (US$M)	LOM Annual Cost (US$M/a)	LOM Unit Cost (US$/t milled)
Mining	355.8	31.7	11.43
Process	281.2	25.0	9.03
G&A	33.2	3.0	1.07
Total	**670.3**	**59.7**	**21.53**

Note: Totals may not sum due to rounding

21.3.1 Basis of Estimate

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q2 2023 US dollar without allowances for inflation.
- The annual power costs were calculated using a unit price of US$0.04/kWh.
- Plant crusher availability is assumed to be 75%, rest of the process plant availability assumed to be 92%.
- Reagent consumption rates are based on metallurgical testwork results and in-house benchmarks.
- Grinding media consumption rates are based on mineral material characteristics as described in Section 13.

21.3.2 Mine Operating Costs

Mine operating unit costs are based on benchmarking to other similar sized operations in western United States, mining 12-16 Mt/a. The unit costs, summarized in Table 21-9, are applied to the Lemhi PEA mine production schedule to estimate total dollars spent in each year of construction and operations. The mining unit rate is consistently applied over the LOM.

Table 21-9: Mine Operating Cost Summary

Cost Area	US$/t mined
Ore Mining	3.00
Waste Mining	2.35
Blended Mining	**2.51**
when applied to PEA production schedule over LOM	**$11.43/t milled**

The QP's opinion is that the estimates are reasonable for the location and planned mine operation activities and can be utilized for a PEA.

21.3.3 Process Operating Costs

The operating costs are estimated from benchmarks on available operational data for equivalent gold operations. The overall LOM processing operating cost is US$281.2 M and the overall LOM G&A cost is US$33.2 M. The average annual operating cost is shown on Table 21-10.

Table 21-10: Process Operating Cost Summary

Cost Area	Total LOM Cost (US$M)	LOM		Phase 1		Phase 2	
		Annual Cost (US$M/a)	Unit Cost (US$/t Milled)	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Reagents	132.6	11.8	4.26	10.6	4.26	12.8	4.26
Steel consumables	40.4	3.6	1.30	3.4	1.36	3.8	1.27
Labour	51.8	4.6	1.66	4.6	1.84	4.6	1.54
Power	41.0	3.7	1.32	3.3	1.32	3.9	1.31
Maintenance	10.0	0.9	0.32	0.9	0.38	0.9	0.33
Laboratory consumables	3.6	0.3	0.12	0.3	0.13	0.3	0.11
Light vehicles	1.8	0.2	0.06	0.2	0.06	0.2	0.06
Processing Subtotal	**281.2**	**25.0**	**9.03**	**23.3**	**9.36**	**26.5**	**8.88**
General & administration	33.2	3.0	1.07	3.0	1.18	3.0	0.99
Process and G&A Total	**314.4**	**28.0**	**10.10**	**26.3**	**10.54**	**29.5**	**9.87**

Note: Totals may not sum due to rounding.

21.3.3.1 Reagents

Reagent usage was estimated based on an interpretation of the available testwork as well as benchmarked usage from comparable operations. Reagent costs were based on internal data, which is developed from vendor quotations for other projects. The major reagent cost details for Phase 1 and Phase 2 are shown in Table 21-11.

Table 21-11: Process Operating Cost Summary

Reagents	Phase 1		Phase 2	
	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Cyanide	4.5	1.80	5.4	1.80
Lime	0.8	0.31	0.9	0.31
Carbon	0.4	0.15	0.4	0.15
SMBS	3.4	1.35	4.0	1.35
Copper Sulphate	0.6	0.23	0.7	0.23
Elution reagents	1.1	0.43	1.3	0.43
Reagents Total	**10.6**	**4.26**	**12.8**	**4.26**

Note: Totals may not sum due to rounding

21.3.3.2 Consumables

The consumables considered in this cost summary are crusher and mill liners, mill grinding media, and screening media. The usage was estimated from benchmarking databases of similar mineralization. The unit costs were based on a regression of internal data obtained from vendor quotations for similar projects. The details are shown in Table 21-12.

Table 21-12: Consumables Cost Summary

Consumable	Phase 1		Phase 2	
	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)	Annual Cost (US$M/a)	Unit Cost (US$/t Milled)
Mill media	1.9	0.76	2.3	0.78
Mill liners	1.1	0.46	1.1	0.38
Crusher consumables	0.3	0.14	0.3	0.11
Subtotal	**3.4**	**1.36**	**3.8**	**1.27**

Note: Totals may not sum due to rounding.

21.3.3.3 Labour

Processing production labour was developed using benchmarks from similar projects and includes operation departments such as management, metallurgy, mill operations, maintenance, and the assay lab. Labour includes all processing and maintenance labour costs.

Each position was defined and classified as salary and wages. Costs included taxes and benefits. A total of 57 persons is required for the process plant and the process maintenance shop. The estimate was based on providing a labour force to support continuous operation at 24 hours a day, 365 days a year. The total process plant labour operating cost including management is US$4.6 M/a which is constant in phase 1 and phase 2. The labour cost is US$1.84/t in Phase 1 and US$1.54 in Phase 2. Life of mine labour cost is US$1.66/t.

21.3.3.4 Power

The power cost is calculated from the estimated power draw determined from the preliminary mechanical equipment list plus an allowance for other auxiliary services. The total installed power for pre-expansion phase is estimated at 12.8 MW with an estimated annual consumption of 78,704 MWh (9.0 MW). Phase 2 annual power consumption is expected to increase to 93,895 MWh (10.7 MW). Electricity will be provided to the site at a unit cost of US$0.04/kWh. The unit power cost for the process plant is estimated at US$1.32/t plant feed for Phase 1 and US$1.31/t plant feed for Phase 2.

21.3.3.5 Maintenance Consumables

Annual maintenance cost was calculated based on a total installed mechanical capital cost by area using an average 3% factor. The factor was applied to mechanical equipment, platework and piping. The total LOM maintenance operating cost is US$10.0 M.

21.3.3.6 Laboratory Services

The operating cost estimate for laboratory activity was developed from an estimate based on a review of the flowsheet and benchmarking similar projects. The laboratory is designed to handle grade control samples, mill solids samples, water testing, concentrate quality assays, and other miscellaneous tests, as required. The annual costs are estimated at US$0.3 M.

21.3.4 General and Administrative Operating Costs

General and administrative costs are expenses not directly related to the operation of the process plant but required to support safe and effective operation of the facility and satisfy legislative requirements in some cases. These costs were developed using Ausenco's in-house data on existing operations, and include costs such as the following:

- Human resources, including training, recruiting, and community relations
- Site administration, maintenance, and security, including subscriptions, memberships, advertisement, office supplies, and garbage disposal
- Health and safety, including personal protective equipment, and first aid
- Environmental, including water sampling and tailings management facility operating costs
- It & telecommunications, including hardware and support services
- Contract services, including insurance, consulting, sanitation and cleaning, license fees, and legal fees
- The annual costs are estimated at US$3.0 M and shown in Table 21-10.

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
- Proposed mine production plan
- Projected mining and process recovery rates
- Assumptions regarding mining dilution and estimated future production
- Sustaining costs and proposed operating costs
- Assumptions regarding closure costs and closure requirements
- Assumptions regarding environmental, permitting, and social risks.

Additional risks to the forward-looking information include the following:

- changes to costs of production from what is assumed
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- accidents, labour disputes, and other risks of the mining industry
- geotechnical or hydrogeological considerations during mining 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
- changes to site access, use of water for mining purposes, and to time to obtain environment and other regulatory permits

- ability to maintain the social license to operate
- changes to interest rates
- changes to tax rates.

Important cautionary aspects of a preliminary economic analysis include its preliminary nature, that it includes inferred mineral resources considered too geologically speculative to have economic considerations applied to them that might enable them to be categorized as mineral reserves, and there is no certainty that the PEA will be realized.

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

22.2 Methodologies Used

The project was evaluated using a discounted cash flow analysis based on a 5% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs, operating costs, treatment costs, refining costs, taxes, and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections.

Cash flows were taken to occur at the midpoint of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and as such, actual post-tax results may differ from estimates. A sensitivity analysis was performed to assess the impact of variations in gold price, discount rate, operating costs, initial capital costs, mill recoveries, and mill head grades.

The capital and operating cost estimates are presented in Section 21 in Q2 2023 US dollars. The economic analysis was run based on a constant dollar value, with no inflation.

22.3 Financial Model Parameters

The economic analysis was performed assuming a gold price of US$1,750/oz, which was based on consensus analyst estimates. The forecasts 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:

- Construction will take 18 months.
- The project has a mine life of 11.2 years (last year is a partial year).
- The results are based on 100% ownership.
- The project will be capital cost funded with 100% equity (no financing cost assumed).
- All cash flows are discounted to the start of construction using a mid-period discounting convention.
- All metal products will be sold in the same year they are produced.

- Treatment and refining charges will be applied as described in Section 19.
- Project revenue will be derived from the sale of gold doré.
- No contractual arrangements for refining currently exist.

22.3.1 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 by Freeman Gold, assuming a blended corporate tax rate of 25% to reflect federal and Idaho state taxes.

At the base case gold price assumption, total tax payments are estimated to be US$127.2 M over the LOM.

22.3.2 Working Capital

An estimation of working capital has been incorporated into the economic analysis based on the following assumptions:

- Accounts Receivable 0 days
- Inventories 30 days
- Accounts Payable 30 days.

22.3.3 Closure Costs and Salvage Value

Closure costs and salvage value are applied at the end of the life-of mine (LOM). Closure costs were estimated to be US$29.9 M, and salvage value was estimated to be US$12.0 M.

22.3.4 Royalties

A 1.0% royalty has been assumed for the project, resulting in approximately US$14.9 million in royalty payments over life of mine. It has been assumed that the company has bought back 1.0% of a previously outstanding 2.0% NSR for approximately US$1.0 million prior to the start of project construction, resulting in the financial model carrying a 1.0% NSR. As the financial model is based on an asset level, the US$1.0 million outflow has not been incorporated in the financial model.

22.4 Economic Analysis

The pre-tax NPV discounted at 5% is US$297 M; the IRR is 26.9%; and the payback period is 3.3 years. On a post-tax basis, the NPV discounted at 5% is US$212.4 M; the IRR is 22.8%; and payback period is 3.6 years. A summary of project economics is summarized in

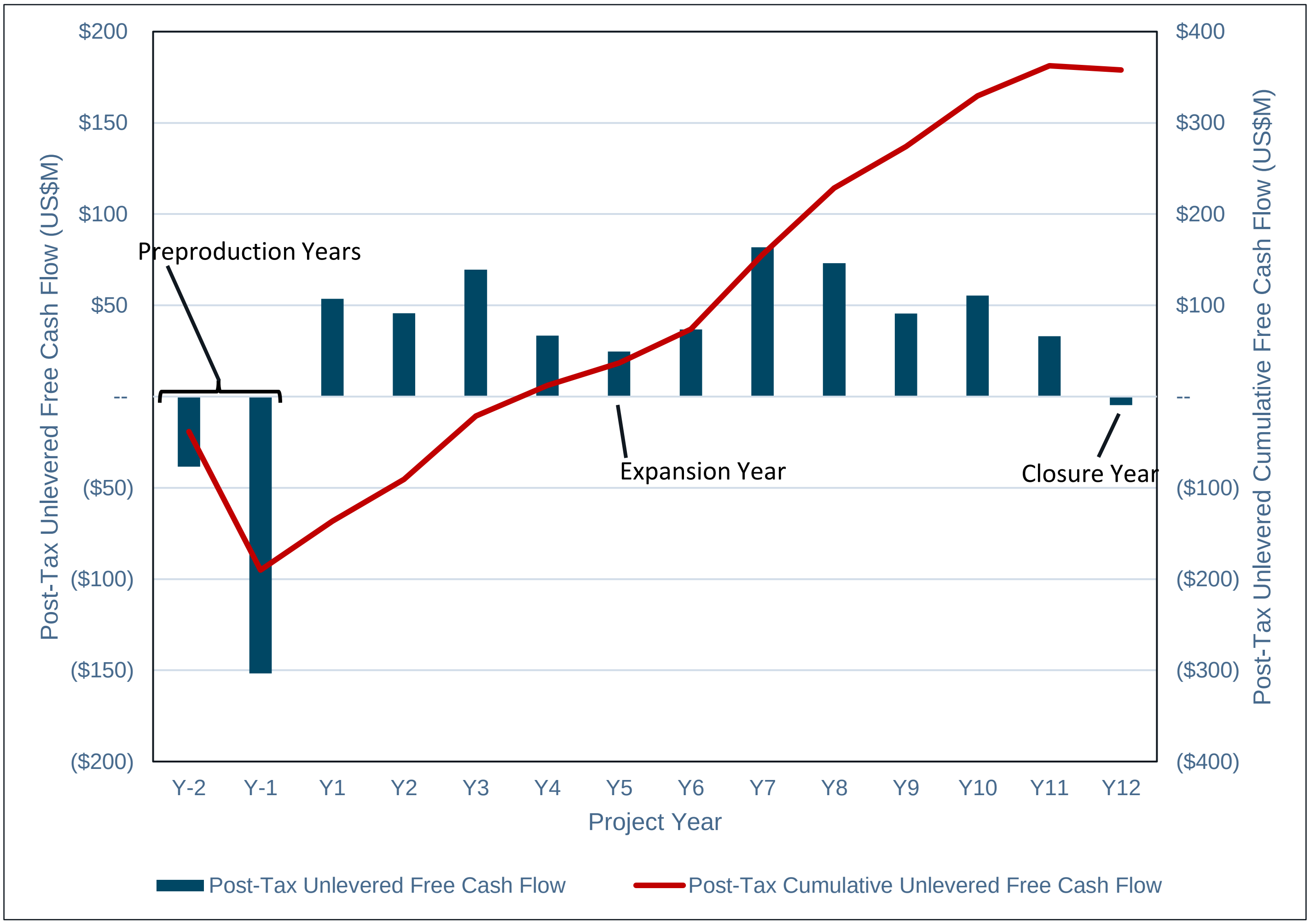
Table 22-1 and illustrated in Figure 22-1. The analysis was done on an annual cashflow basis; the cashflow output is shown Table 22-2.

Table 22-1: Economic Analysis Summary

General	Unit	LOM Total/Avg.
Gold Price	US$/oz	1,750
Mine Life	years	11.2
Total Waste Tonnes Mined	kt	121,903
Total Mill Feed Tonnes	kt	31,128
Strip Ratio	waste: mineralized rocks	3.9
Production	**Unit**	**LOM Total/Avg.**
Mill Head Grade	g/t	0.88
Mill Recovery Rate	%	96.7
Total Payable Mill Ounces Recovered	koz	851.9
Total Average Annual Payable Production	koz	75.9
Operating Costs	**Unit**	**LOM Total/Avg.**
Mining Cost (incl. rehandle)	US$/t mined	2.51
Mining Cost (incl. rehandle)	US$/t milled	11.43
Processing Cost	US$/t milled	9.03
General & Administrative Cost	US$/t milled	1.07
Total Operating Costs	US$/t milled	21.53
Treatment & Refining Cost	US$/oz	4.30
Net Smelter Royalty	%	1.0
Cash Costs[1]	US$/oz Au	809
All-In Sustaining Costs[2]	US$/oz Au	957
Capital Costs	**Unit**	**LOM Total/Avg.**
Initial Capital	US$M	190
Expansion Capital	US$M	8
Sustaining Capital	US$M	101
Closure Costs	US$M	30
Salvage Value	US$M	12
Financials – Pre-Tax	**Unit**	**LOM Total/Avg.**
Net Present Value (5%)	US$M	297
Internal Rate of Return	%	26.9
Payback	years	3.3
Financials – Post-Tax	**Unit**	**LOM Total/Avg.**
Net Present Value (5%)	US$M	212.4
Internal Rate of Return	%	22.8
Payback	years	3.6

Notes:
1. Cash costs consist of mining costs, processing costs, mine-level G&A and treatment and refining charges, and royalties.
2. All-in sustaining costs include cash costs plus expansion capital, sustaining capital, closure costs and salvage value.

Figure 22-1: Projected LOM Post-Tax Unlevered Free Cash Flow



Source: Ausenco, 2023.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV, IRR of the project, using the following variables: gold price, discount rate, operating costs, initial capital costs, mill recoveries, and mill head grades. Table 22-4 shows the post-tax sensitivity analysis results; pre-tax sensitivity results are shown in Table 22-3.

Table 22-2: Cash Flow Forecast on an Annual Basis

Dollar figures in Real 2023 US$M unless otherwise noted	Units	Total/Avg.	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Macro Assumptions																
Gold Price–- Flat	US$/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
Free Cash Flow Valuation																
Revenue	**US$M**	**1,491**	--	--	129	119	156	123	109	138	185	161	121	134	89	25
Operating Cost	US$M	(670)	--	--	(58)	(60)	(57)	(62)	(66)	(74)	(67)	(60)	(57)	(57)	(42)	(10)
Refining Charges	US$M	(4)	--	--	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)	(0)
Royalties	US$M	(15)	--	--	(1)	(1)	(2)	(1)	(1)	(1)	(2)	(2)	(1)	(1)	(1)	(0)
EBITDA	**US$M**	**802**	**--**	**--**	**69**	**57**	**97**	**60**	**42**	**63**	**116**	**99**	**62**	**75**	**46**	**15**
Initial Capital	US$M	(190)	(38)	(152)	--	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	US$M	(101)	--	--	(16)	(12)	(18)	(11)	(11)	(15)	(8)	(3)	(2)	(2)	(2)	(2)
Closure Capital	US$M	(30)	--	--	--	--	--	--	--	--	--	--	--	--	--	(30)
Salvage Value	US$M	12	--	--	--	--	--	--	--	--	--	--	--	--	--	12
Change in Working Capital	US$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**US$M**	**485**	**(38)**	**(152)**	**54**	**46**	**79**	**42**	**31**	**47**	**107**	**96**	**60**	**74**	**44**	**(5)**
Pre-Tax Cumulative Unlevered Free Cash Flow	US$M		*(38)*	*(190)*	*(137)*	*(91)*	*(12)*	*30*	*61*	*108*	*216*	*312*	*372*	*446*	*490*	*485*
Unlevered Cash Taxes	US$M	(127)	--	--	--	(0)	(10)	(8)	(6)	(10)	(25)	(23)	(15)	(18)	(11)	--
Post-Tax Unlevered Free Cash Flow	**US$M**	**358**	**(38)**	**(152)**	**54**	**46**	**70**	**33**	**25**	**37**	**82**	**73**	**46**	**55**	**33**	**(5)**
Post-Tax Cumulative Unlevered Free Cash Flow	US$M		*(38)*	*(190)*	*(137)*	*(91)*	*(21)*	*12*	*37*	*74*	*156*	*229*	*274*	*330*	*363*	*358*
Production																
Total Resource Mined	kt	31,128	147	734	1,625	2,500	3,040	2,500	2,500	3,000	3,542	3,444	3,000	2,800	1,697	598
Total Waste	kt	121,903	2,881	7,236	11,203	11,051	9,264	11,833	12,202	14,971	11,554	8,740	7,922	8,126	4,221	702
Strip Ratio	w:MR	3.92	19.55	9.85	6.89	4.42	3.05	4.73	4.88	4.99	3.26	2.54	2.64	2.90	2.49	1.17
Total Material Mined	kt	153,031	3,028	7,970	12,828	13,551	12,304	14,333	14,701	17,971	15,096	12,184	10,922	10,926	5,918	1,300
Mill Feed	kt	31,128	--	--	2,500	2,500	2,500	2,500	3,000	3,000	3,000	3,000	3,000	3,000	2,531	598
Mill Head Grade (Au)	g/t	0.88	--	--	0.95	0.87	1.14	0.91	0.68	0.85	1.12	0.98	0.75	0.83	0.65	0.78
Contained (Au)	koz	881	--	--	76	70	91	73	65	82	108	95	72	80	53	15
Mill Recovery (Au)	%	96.7	--	--	96.9	96.6	97.6	96.8	95.6	96.5	97.6	97.1	96.0	96.4	95.5	96.1
Gold Production	koz	852	--	--	74	68	89	71	63	79	106	92	69	77	51	14
Gold % Payable	%	99.95	--	--	99.95	99.95	99.95	99.95	99.95	99.95	99.95	99.95	99.95	99.95	99.95	99.95
Payable Gold	**koz**	**852**	**--**	**--**	**74**	**68**	**89**	**70**	**63**	**79**	**106**	**92**	**69**	**77**	**51**	**14**
Total Revenue	**US$M**	**1,491**	**--**	**--**	**129**	**119**	**156**	**123**	**109**	**138**	**185**	**161**	**121**	**134**	**89**	**25**
Operating Costs																
Total Operating Costs	**US$M**	**670**	**--**	**--**	**58**	**60**	**57**	**62**	**66**	**74**	**67**	**60**	**57**	**57**	**42**	**10**
Mine Operating Costs (incl. Rehandle)	US$M	356	--	--	32	33	31	35	37	44	38	31	28	28	16	3
Mill Processing	US$M	281	--	--	23	23	23	23	27	27	27	27	27	27	23	5
G&A Costs	US$M	33	--	--	3	3	3	3	3	3	3	3	3	3	3	1
Refining & Royalties																
Treatment and Refining Charges	**US$M**	**4**	**--**	**--**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**	**0**
Royalties	**US$M**	**15**	**--**	**--**	**1**	**1**	**2**	**1**	**1**	**1**	**2**	**2**	**1**	**1**	**1**	**0**
Cash Costs																
Cash Cost *	US$/oz Au	809	--	--	811	903	663	895	1,080	956	658	677	848	767	849	690
All-in Sustaining Cost (AISC) **	US$/oz Au	957	--	--	1,025	1,076	863	1,156	1,254	1,152	737	705	876	792	887	2,073
Capital Expenditures																
Initial Capital	**US$M**	**190**	**38**	**152**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Expansion Capital	**US$M**	**8**	**--**	**--**	**--**	**--**	**--**	**8**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Total Sustaining Capital	**US$M**	**101**	**--**	**--**	**16**	**12**	**18**	**11**	**11**	**15**	**8**	**3**	**2**	**2**	**2**	**2**
Closure Cost	**US$M**	**30**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**30**
Salvage Value	**US$M**	**12**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**12**

Notes:
1. Cash costs consist of mining costs, processing costs, mine-level G&A and treatment and refining charges, and royalties.
2. All-in sustaining costs include cash costs plus expansion capital, sustaining capital, closure costs and salvage value.

Table 22-3: Pre-Tax Sensitivity Analysis

	Pre-Tax NPV (US$M) Sensitivity to Discount Rate					
	Gold Price (US$/oz)					
Discount Rate		$1,450	$1,600	$1,750	$1,900	$2,050
	1.0%	204	322	440	558	676
	3.0%	155	259	362	465	568
	5.0%	115	206	297	388	478
	8.0%	68	143	219	295	370
	10.0%	42	110	177	245	313

	Pre-Tax IRR (%) Sensitivity to Discount Rate					
	Gold Price (US$/oz)					
Discount Rate		$1,450	$1,600	$1,750	$1,900	$2,050
	1.0%	14.4	20.9	26.9	32.6	38.0
	3.0%	14.4	20.9	26.9	32.6	38.0
	5.0%	14.4	20.9	26.9	32.6	38.0
	8.0%	14.4	20.9	26.9	32.6	38.0
	10.0%	14.4	20.9	26.9	32.6	38.0

	Pre-Tax NPV (US$M) Sensitivity to Operating Costs					
	Gold Price (US$/oz)					
Operating Costs		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	212	303	393	484	575
	(10.0%)	164	254	345	436	527
	--	115	206	297	388	478
	10.0%	67	158	249	339	430
	20.0%	19	110	200	291	382

	Pre-Tax IRR (%) Sensitivity to Operating Costs					
	Gold Price (US$/oz)					
Operating Costs		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	21.3	27.3	33.0	38.5	43.8
	(10.0%)	18.0	24.1	30.0	35.5	40.9
	--	14.4	20.9	26.9	32.6	38.0
	10.0%	10.7	17.5	23.7	29.5	35.1
	20.0%	6.7	13.9	20.4	26.4	32.1

	Pre-Tax NPV (US$M) Sensitivity to Initial Capital					
	Gold Price (US$/oz)					
Initial Capital		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	154	245	335	426	517
	(10.0%)	135	225	316	407	498
	--	115	206	297	388	478
	10.0%	96	187	278	368	459
	20.0%	77	168	258	349	440

	Pre-Tax IRR (%) Sensitivity to Initial Capital					
	Gold Price (US$/oz)					
Initial Capital		$1,450	$1,600	$1,750	$1,900	$2,050
	(20.0%)	19.6	27.0	34.0	40.7	47.2
	(10.0%)	16.8	23.7	30.1	36.2	42.2
	--	14.4	20.9	26.9	32.6	38.0
	10.0%	12.4	18.5	24.1	29.5	34.6
	20.0%	10.6	16.4	21.8	26.8	31.6

	Pre-Tax NPV (US$M) Sensitivity to Recovery Mill					
	Gold Price (US$/oz)					
Recovery Mill		$1,450	$1,600	$1,750	$1,900	$2,050
	(2.0%)	98	187	276	365	454
	(1.0%)	107	197	286	376	466
	--	115	206	297	388	478
	1.0%	124	216	307	399	491
	2.0%	133	225	318	411	503

	Pre-Tax IRR (%) Sensitivity to Recovery Mill					
	Gold Price (US$/oz)					
Recovery Mill		$1,450	$1,600	$1,750	$1,900	$2,050
	(2.0%)	13.1	19.6	25.5	31.1	36.6
	(1.0%)	13.8	20.2	26.2	31.9	37.3
	--	14.4	20.9	26.9	32.6	38.0
	1.0%	15.1	21.5	27.6	33.3	38.8
	2.0%	15.7	22.2	28.2	34.0	39.5

	Pre-Tax NPV (US$M) Sensitivity to Head Grade					
	Gold Price (US$/oz)					
Head Grade		$1,450	$1,600	$1,750	$1,900	$2,050
	(2.0%)	97	186	275	364	453
	(1.0%)	106	196	286	376	466
	--	115	206	297	388	478
	1.0%	125	216	308	400	491
	2.0%	134	226	319	412	504

	Pre-Tax IRR (%) Sensitivity to Head Grade					
	Gold Price (US$/oz)					
Head Grade		$1,450	$1,600	$1,750	$1,900	$2,050
	(2.0%)	13.0	19.5	25.5	31.1	36.5
	(1.0%)	13.7	20.2	26.2	31.8	37.3
	--	14.4	20.9	26.9	32.6	38.0
	1.0%	15.1	21.6	27.6	33.3	38.8
	2.0%	15.8	22.2	28.3	34.0	39.6

Table 22-4: Post-Tax Sensitivity Analysis

Post-Tax NPV (US$M) Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,450	$1,600	$1,750	$1,900	$2,050
1.0%	145	234	323	412	501
3.0%	106	185	263	340	418
5.0%	74	144	212	281	349
8.0%	36	95	152	209	266
10.0%	16	68	120	170	221

Post-Tax IRR (%) Sensitivity to Discount Rate — Gold Price (US$/oz)

Discount Rate	$1,450	$1,600	$1,750	$1,900	$2,050
1.0%	11.9	17.6	22.8	27.6	32.1
3.0%	11.9	17.6	22.8	27.6	32.1
5.0%	11.9	17.6	22.8	27.6	32.1
8.0%	11.9	17.6	22.8	27.6	32.1
10.0%	11.9	17.6	22.8	27.6	32.1

Post-Tax NPV (US$M) Sensitivity to Operating Costs — Gold Price (US$/oz)

Operating Costs	$1,450	$1,600	$1,750	$1,900	$2,050
(20.0%)	148	217	285	353	422
(10.0%)	111	180	249	317	385
--	74	144	212	281	349
10.0%	37	107	176	244	313
20.0%	(1)	70	139	208	276

Post-Tax IRR (%) Sensitivity to Operating Costs — Gold Price (US$/oz)

Operating Costs	$1,450	$1,600	$1,750	$1,900	$2,050
(20.0%)	18.0	23.2	27.9	32.5	36.8
(10.0%)	15.0	20.4	25.4	30.1	34.5
--	11.9	17.6	22.8	27.6	32.1
10.0%	8.5	14.6	20.1	25.1	29.7
20.0%	4.9	11.4	17.2	22.4	27.2

Post-Tax NPV (US$M) Sensitivity to Initial Capital — Gold Price (US$/oz)

Initial Capital	$1,450	$1,600	$1,750	$1,900	$2,050
(20.0%)	113	182	251	319	388
(10.0%)	94	163	232	300	368
--	74	144	212	281	349
10.0%	55	124	193	262	330
20.0%	36	105	174	242	311

Post-Tax IRR (%) Sensitivity to Initial Capital — Gold Price (US$/oz)

Initial Capital	$1,450	$1,600	$1,750	$1,900	$2,050
(20.0%)	17.1	23.8	29.8	35.4	40.7
(10.0%)	14.3	20.4	26.0	31.1	36.0
--	11.9	17.6	22.8	27.6	32.1
10.0%	9.8	15.2	20.1	24.6	28.9
20.0%	7.9	13.1	17.8	22.1	26.1

Post-Tax NPV (US$M) Sensitivity to Recovery Mill — Gold Price (US$/oz)

Recovery Mill	$1,450	$1,600	$1,750	$1,900	$2,050
(2.0%)	61	129	196	263	330
(1.0%)	68	136	204	272	340
--	74	144	212	281	349
1.0%	81	151	220	289	358
2.0%	88	158	228	298	368

Post-Tax IRR (%) Sensitivity to Recovery Mill — Gold Price (US$/oz)

Recovery Mill	$1,450	$1,600	$1,750	$1,900	$2,050
(2. 0%)	10.7	16.4	21.6	26.4	30.9
(1. 0%)	11.3	17.0	22.2	27.0	31.5
--	11.9	17.6	22.8	27.6	32.1
1. 0%	12.4	18.2	23.4	28.2	32.7
2. 0%	13.0	18.7	24.0	28.8	33.3

Post-Tax NPV (US$M) Sensitivity to Head Grade — Gold Price (US$/oz)

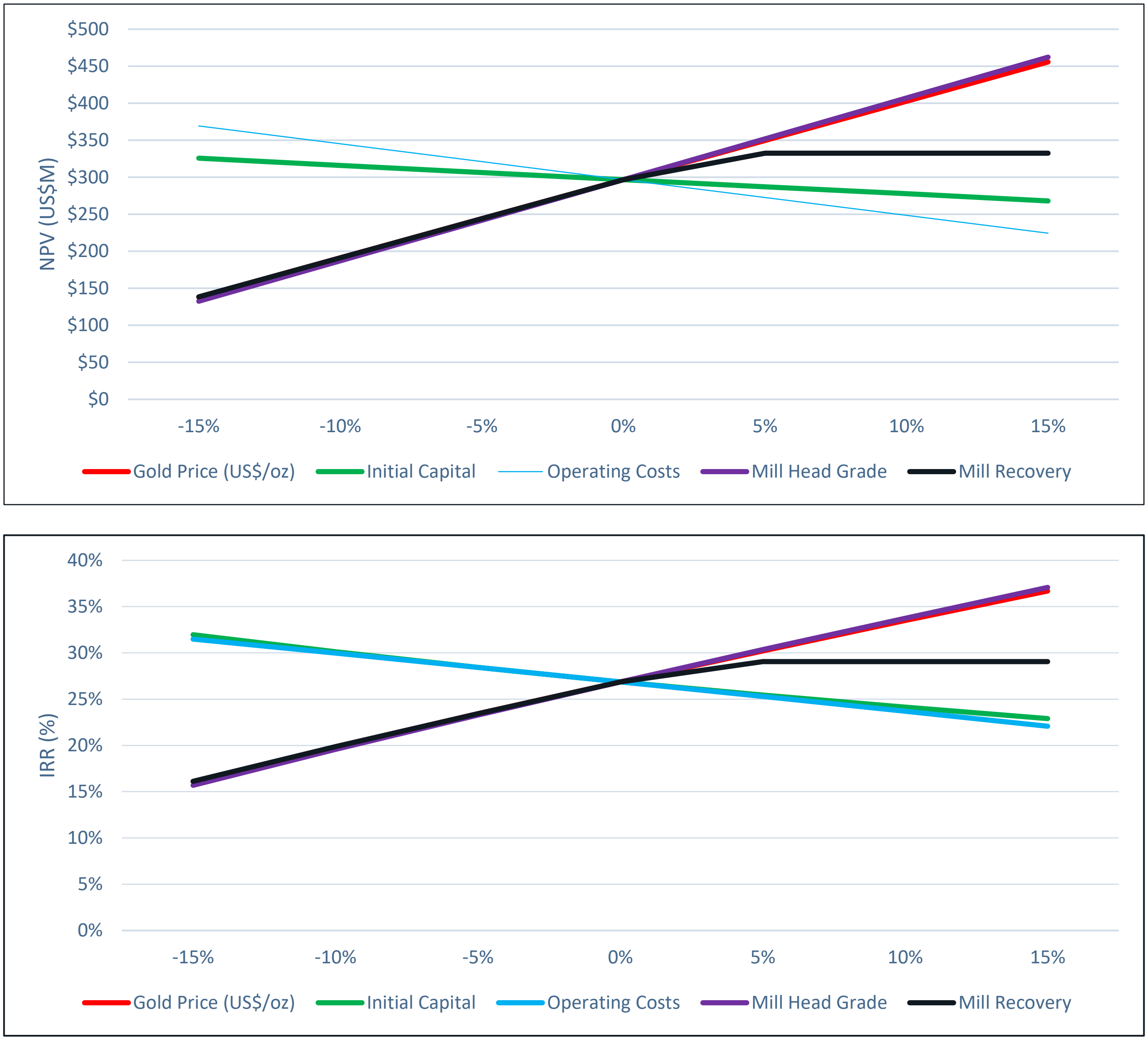
Head Grade	$1,450	$1,600	$1,750	$1,900	$2,050
(2.0%)	60	128	196	263	330
(1.0%)	67	136	204	272	339
--	74	144	212	281	349
1.0%	82	151	221	290	359
2.0%	89	159	229	299	369

Post-Tax IRR (%) Sensitivity to Head Grade — Gold Price (US$/oz)

Head Grade	$1,450	$1,600	$1,750	$1,900	$2,050
(2. 0%)	10.6	16.4	21.6	26.4	30.9
(1.0%)	11.2	17.0	22.2	27.0	31.5
--	11.9	17.6	22.8	27.6	32.1
1.0%	12.5	18.2	23.4	28.2	32.8
2.0%	13.1	18.8	24.0	28.8	33.4

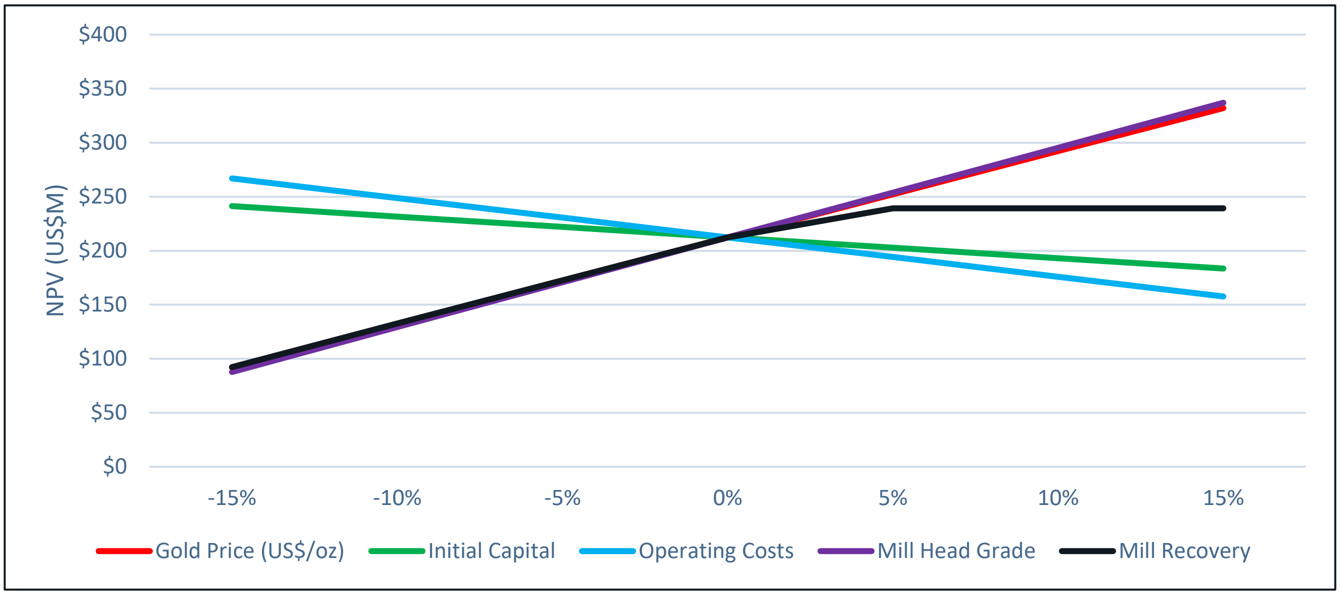
As presented in Figure 22-2 and Figure 22-3, the sensitivity analysis showed that the project is most sensitive to changes in gold price, mill head grade and mill recovery, and to a lesser extent, changes in operating costs and initial capital costs.

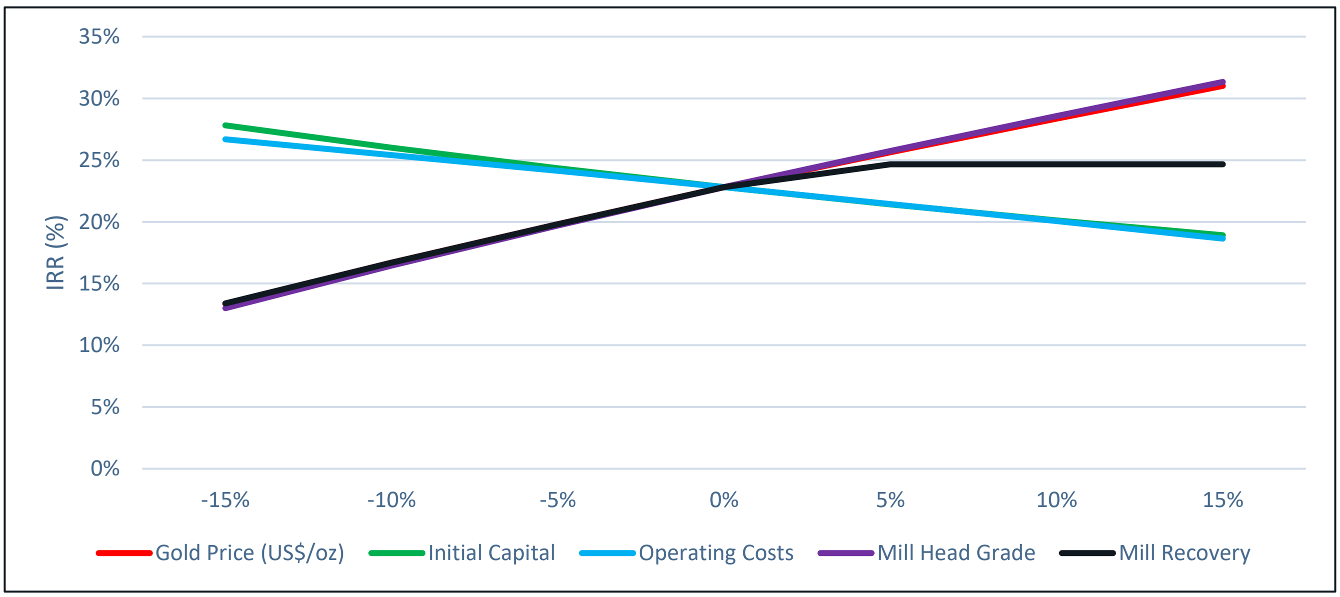
Figure 22-2: Pre-Tax NPV, IRR Sensitivity Results



Source: Ausenco, 2023.

Figure 22-3: Post-Tax NPV, IRR Sensitivity Results





Source: Ausenco, 2023.

23 ADJACENT PROPERTIES

This section is not relevant to this technical report.

24 OTHER RELEVANT DATA AND INFORMATION

This section is not relevant to this technical report.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The qualified persons (QPs) note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this technical report.

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

The Lemhi Gold Project, including 11 patented and 286 unpatented claims that are 100% owned along with 46 unpatented claims that are under option by Freeman Gold, is located in Lemhi County, Idaho (ID), USA, within the Salmon River Mountains, a part of the Bitterroot Range which forms the Idaho-Montana border. The property is 40 km north of the town of Salmon and 6 km west of Gibbonsville, ID, USA. The project is comprised of ten patented mining claims (placer and lode), one patented mill site claim, and 332 unpatented mining claims, totalling 2,727 ha of mineral rights and 249 ha of surface rights. Freeman Gold controls a 100% interest in all 11 patented claims and all 332 unpatented mining claims subject to certain cash payments over time and royalties either outright or through its wholly owned subsidiary company Lower 48 Resources (Idaho) LLC (Lower 48).

A total of 11 patented claims and 53 unpatented claims were purchased from Lemhi Gold Trust (LGT) by Lower 48 Resources (Idaho) LLC (Lower 48) through a closed auction bid process in November 2019. Freeman has also optioned 46 unpatented claims that are owned by BHLK2, LLC (BHLK). Freeman recently purchased outright the Moon #100 and Moon #101 unpatented mining claims (Moon Claims) from Vineyard Gulch Resources, LLC (Vineyard), located within the historical resource area. An additional 231 unpatented claims were staked by Freeman in 2020 and 2021. Freeman closed a transaction to acquire Lower 48 on April 16, 2020. BHLK retains a 2% net smelter return (NSR) royalty on production from the Lemhi Gold Project including the 11 patented claims, the 46 unpatented BHLK claims under option and most of the 284 Lower 48 unpatented claims through area of interest clauses in the agreements.

The patented mining claims originated as unpatented mining claims and were converted to private ownership through the Patent and Mineral Survey process. The patented claims on the Lemhi Gold Property were patented between 1890 and 1910. Corner survey monuments are intact (with several observed by the author) and the United States Forest Service (USFS) has placed markers delineating USFS land boundaries along the claim boundaries. In order to keep the claims in good standing annual real estate taxes must be paid to Lemhi County. If the annual taxes are paid the patented claims will remain in good standing in perpetuity.

The 332 unpatented Bureau of Land Management (BLM) federal lode mining claims are administered by the USFS. The claims are ultimately owned by two entities (Freeman/Lower 48 and BHLK):

- 46 unpatented claims staked by BHLK of Missoula, Montana in 2011 and 2017 and currently under option by Lower 48 (Freeman Gold)
- 53 claims staked by LGT in September 2019, purchased by Lower 48 in November, 2019

- 223 claims staked by Lower 48 in April 2020 and eight claims staked by Lower 48 in April 2021
- Two claims (the Moon Claims) purchased by Lower 48 from Vineyard in 2020.

Any portion of an unpatented claims which overlaps a patented claim is deemed invalid. The valid portion of all unpatented claims totals 2,479 ha.

BHLK obtained a 2% NSR royalty on all 11 patented mining claims and 74 surrounding unpatented mining claims through a deed of royalty upon LGT's purchase of the project in 2011. The deed of royalty describes a 2-mile area of interest and is still active today. The 74 unpatented mining claims were optioned by LGT from BHLK in 2011 and cover the area currently represented by BHLK's 46 unpatented mining claims. The 46 unpatented mining claims are under option and Freeman may earn a 100% interest in the claims with cash payments totalling US$1. 0 M over seven years, at which time the BHLK 2% NSR will extend over most of the unpatented claims through the active deed of royalty.

Freeman was recently granted a Permit to Appropriate Water (No. 75-15005), which allows for water rights for both potential future mining and domestic use in four sections within the company's patented mining claims. The permit allows the use of 0.54 m^3/s of water from ground water sources for future processing in a gold operation and 24600 L/day for domestic use. The permit was obtained from the Idaho Department of Water Rights (IDWR).

Freeman has also recently received an approval of a Plan of Operations (POO) application to the USDA-Forest Service (USFS), Salmon and Challis National Forests, North Fork Ranger District, submitted in September 2021. The plan was approved May 23, 2022, as POO-2021-081646 and allows for an expanded drill program with additional access on the unpatented BLM mineral claims. Freeman is currently permitted to draw water from a number of wells on the patented mineral claims for drilling.

25.3 Geology and Mineralization

The Lemhi Gold Project is located within the Cordilleran fold and thrust belt and more locally the Trans-Challis fault system. This broad 20-30 km-wide system of en-echelon northeast-trending structures extends from Idaho City, ID northeast to the Idaho-Montana border; over 270 km in strike length. It is one of many structures within the Idaho-Montana porphyry belt, a wide northeast-trending alignment of porphyry-related deposits, which parallels the contact between the Cordilleran fold and thrust belt and the Idaho batholith and corresponds to a zone of strike-slip faults, late graben faults and northeast-trending magnetic features.

Locally, the Lemhi Gold Property is largely underlain by Mesoproterozoic quartzites and phyllites with porphyritic dacite sills, dykes and flows of the Eocene Challis volcanics preserved in down-dropped fault blocks. Numerous faults crosscut the property forming grabens and half grabens. On the property, a large low-angle fault passes through Ditch Creek and is filled with Quaternary gravels covering part of the mineralization that comprises the Lemhi Gold Deposit. The mineralization on the property is hosted in structurally controlled quartz vein swarms and quartz flooded zones and occurs in close spatial association with low-angle faulting and several intrusive bodies.

Gold was discovered and mined from the area in the 1890s to mid-1900s. Modern exploration of the property area commenced in 1984. FMC Gold Company (FMC) conducted exploration over the current property area between 1984 and 1991. FMC completed geologic mapping; rock, soil, and vegetation sampling, geophysical surveys, and RC and core drilling over the property. FMC defined an area of strong gold mineralization along the western slope of Ditch Creek. AGR acquired the property in 1991 and conducted exploration over the area until 1996. The FMC and AGR drilling delineated a gold deposit: the Humbug Deposit (now known as the Lemhi Gold Deposit), on the patented claims (MS 784 A and B, 2512 and 1120) which comprise the current Lemhi Gold Property. The Lemhi Gold Deposit is 1000 m east-west by 1100 m north-south. A prominent west-northwest-trending zone of higher-grade mineralization and a northeast-trending zone of strong mineralization were identified within the deposit. The mineralization is interpreted to be structurally controlled by northwest and northeast high-angle faults that intersect a low-angle (possible thrust) fault. In the footwall of an intrusion and along its western terminus the gold mineralization is thick (30-70 m) and can occur in multiple stacked zones. In the hanging wall gold mineralization is considerably thinner and more erratic. In the core of the deposit, the low-grade envelope of mineralization is greater than 200 m thick.

25.4 Exploration, Drilling and Analytical Data Collection in Support of Mineral Resource Estimation

The 2020 surface exploration program conducted by Freeman consisted of the following activities:

- Soil orientation survey (conventional B-horizon, partial extraction leach techniques such as IL and MMI sampling)
- Prospecting, rock, and chip sampling
- Ground magnetic survey
- 3D Induced polarization survey
- Core drilling.

Until Freeman's 2020 program, no significant surface exploration had been conducted on the property since the late 1980s. During Freeman's 2020 exploration program, modern soil geochemical techniques utilizing partial extraction methods including MMI and IL were tested. The results of this soil orientation program will guide further exploration in under-explored areas with significant glacial or glacial-fluvial cover, such as areas west and north of the deposit.

In addition, the entire claim group was covered with a magnetic survey, and the core resource area was covered with a 3D IP survey.

Drilling completed on the property in 2012 by LGT and in 2020-2022 by Freeman has returned encouraging results in both infill and step-out drilling. All 55 LGT holes and most of the 106 Freeman core holes have intersected gold mineralization. The new geological interpretation resulting from the data obtained from the core drilling has also identified additional potential exploration targets, including:

- Deep feeder zones
- Down-dip mineralization to the south
- Extensions of known mineralization to the west and southwest associated with intrusions
- "Hidden" targets below the glacial cover immediately to the north of the known deposit.

Freeman's 2020 drilling program consisted of the completion of 7,149 m in 35 core holes of infill and steep-out drilling. As part of the drilling program, Freeman commissioned a series of metallurgical studies to characterize the amenability of the mineralized material to certain recovery processes. During 2021-2022, Freeman completed another drilling program, including core and RC drilling, totalling 15,351 m in 71 drill holes. The metallurgical studies along with the new drilling have assisted in delineation and improvement of the existing geological and mineralization model into a coherent 3D model allowing for the construction of a modern MRE, as presented within this technical report.

In 2023, APEX personnel validated and compiled an updated drill hole database (DHDB) to include the 2021-2022 Freeman drilling programs. The new validated 2023 Freeman DHDB was utilized in constructing the MRE in this technical report.

During the author's site visits, the locations of several historical collars on the property have been confirmed. Pulp re-assays of 2012 core drilling collected in 2019 (Dufresne, 2020) returned values which have close correlation with the original assays for the samples in question, confirming the validity of the 2012 assay results.

Based on the review of historical information, recent re-assay results and the current 2020-2022 program results, the authors consider the Lemhi Gold Property a property of significant merit that requires further exploration and delineation work.

25.5 Metallurgical Testwork

Metallurgical test data indicates that samples taken from the Lemhi deposit are amenable to conventional grinding and cyanide leach processes. The recovery estimates for the proposed flowsheet are suitable for use in a preliminary economic assessment, but more testwork is required for further engineering studies, such as a pre-feasibility study. The material appears to be quite soft with respect to breakage in a SAG mill, has moderate ball mill work index values, and can achieve gold extractions of 95% within 24 hours following grinding to 110 µm P_{80}. 40% of the feed gold was amenable to recovery by gravity methods, however it did not appear that the addition of gravity concentration improved overall circuit performance.

A recovery model was developed from the test results which provides a reasonable estimate of gold recovery as a function of gold content in the feed. The recovery equation is based on the metallurgical performance of 23 unique feed samples that represented a range of spatial locations and lithologies across the deposit. The samples were processed at a similar grind size and spanned a range of feed grades that were similar to the mine plan.

25.6 Mineral Resources Estimates

The Lemhi Project database contains a total of 506 drill holes with collar information, and assays covering 91,747 m of drilling with 64,299 drill hole sample intervals. The sample database contains a total of 62,670 samples assayed for gold. The Lemhi Project MRE utilized 442 drill holes of which 284 drill holes were completed between 1983 and 1995, and 158 drill holes were completed between 2012 and 2022. Inside the mineralized domains, there is a total of 16,290 samples analyzed for gold. Standard statistical treatments were conducted on the raw and composite samples resulting in a capping limit of 17.3 g/t gold Au applied to the composites for the Main Zone and 50 g/t Au for the Beauty Zone. The current DHDB was validated by APEX personnel and is deemed to be in good condition and suitable for use in ongoing MRE studies. Mr. Michael Dufresne, M.Sc., P. Geol., P. Geo, President of APEX, is an independent qualified person (QP) and is responsible for the database validation and MRE.

Mineral resource modelling was conducted in the UTM coordinate system relative to the North American Datum (NAD) 1983, and Idaho State Plane Central FIPS 1102 (EPSG:6448) The mineral resource block model utilized a SMU block size of 2.5 m (X) by 2.5 m (Y) by 2.5 m (Z) to honour the mineralization wireframes. The percentage of the volume of each block below the top of bedrock surface and within each mineralization domain was calculated using the 3-D geological models and a 3-D topographic surface model. The Au grades were estimated for each block using ordinary kriging with LVA to ensure grade continuity in various directions is reproduced in the block model. The MRE is reported as undiluted within a series of optimized pit shells. Details regarding the methodology used to calculate the MRE are documented in this technical report section.

Gold mineralization at the Lemhi Gold Project is primarily of two dominant styles. The primary mineralization occurs as a halo around the granodiorite intrusion, concentrated on the bottom side, with secondary mineralization along faults and shallow dipping foliation. It appears that both styles of mineralization generally occur in zones of stacked parallel sub-horizontal sheets. The Beauty zone is ~700 m west from the nearest modeled intrusion and is primarily controlled by a structurally complex fault zone.

A total of 14,208 bulk density samples are available from the Lemhi Property drillhole database. APEX personnel performed EDA of the bulk density sample data available. Three main geologic units showed significant variation in density. The median specific gravity (SG) value for each geological unit was used for assigning density for material in the MRE. The EDA resulted in a change in the SG used in the 2021 MRE from 2.62 g/cm^3 (Dufresne 2021) for mineralized material and unmineralized material to 2.64 g/cm^3 for metasedimentary rocks material, 2.58 g/cm^3 for intrusion material, and 2.53 g/cm^3 for silty breccia material.

All reported mineral resources occur either within a pit shell optimized using values of US$1,750 per ounce of gold or in shapes outside of the pit shell that display potential for underground stopes. The measured, indicated, and inferred mineral resources are undiluted and constrained within an optimized pit shell at a 0.35 g/t lower cut-off. Out-of-pit potential underground mineral resources utilized a 1.5 g/t Au lower cut-off and constrained with continuous shapes that yield a minimum of 1,400 m^3. The MRE comprises a combined measured and indicated mineral resource of 30.022 Mt at 1.00 g/t Au for 988,100 oz of gold, and an inferred mineral resource of 7.634 Mt at 1.04 g/t Au for 256,000 oz of gold, see Table 25-1. The MRE for the Main Zone covers a surface area of 1,320 m by 740 m and extends down to a depth of 240 m, and remains open on strike to the north, south and west as well as at depth. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Table 25-1: 2023 Lemhi Gold Project Mineral Resource Estimate (1-8).

Au Cut-off (g/t)	Zone	RPEEE Scenario	Classification	Tonnes	Au (oz)	Au Grade (g/t)	Au Grade (oz/st)
0.35	Main & Beauty	Open Pit	Measured	4,469,000	168,800	1.15	0.033
0.35	Main & Beauty	Open Pit	Indicated	25,553,000	819,300	0.98	0.029
0.35	**Main & Beauty**	**Open Pit**	**M&I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35	**Main & Beauty**	**Open Pit**	**Inferred**	**7,338,000**	**234,700**	**1.01**	**0.029**
1.5	Main & Beauty	Under Ground	Inferred	296,000	21,300	2.27	0.066
0.35/1. 5	Main & Beauty	Combined	Measured	4,469,000	168,800	1.15	0.033
0.35/1. 5	Main & Beauty	Combined	Indicated	25,553,000	819,300	0.98	0.029
0.35/1. 5	**Main & Beauty**	**Combined**	**M&I**	**30,022,000**	**988,100**	**1.00**	**0.029**
0.35/1. 5	**Main & Beauty**	**Combined**	**Inferred**	**7,634,000**	**256,000**	**1.04**	**0.030**

Notes:
1. Contained tonnes and ounces may not add due to rounding.
2. Mr. Michael Dufresne, P. Geo., P. Geo. of APEX Geoscience Ltd., who is deemed a qualified person as defined by NI 43-101 is responsible for the completion of the updated mineral resource estimation.
3. Mineral resources that are not mineral reserves do not have demonstrated economic viability.
4. The estimate of mineral resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues.
5. The inferred mineral resource in this estimate has a lower level of confidence than that applied to an indicated mineral resource and must not be converted to a mineral reserve. It is reasonably expected that the majority of the inferred mineral resource could potentially be upgraded to an indicated mineral resource with continued exploration.
6. The mineral resources were estimated in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), CIM Standards on Mineral Resources and Reserves, Definitions (2014) and Best Practices Guidelines (2019) prepared by the CIM Standing Committee on Reserve Definitions and adopted by the CIM Council.
7. The constraining pit optimization parameters assumed US$1750/oz Au sale price, NSR Royalty of 1%, US$2.10/t mineralized and US$2.00/t waste material mining cost, 50° pit slopes, a VAT process cost of US$8/t, HL process cost of US$2.40/t and a general and administration (G&A) cost of US$2/00/t.
8. The effective date of the mineral resources estimate is March 15, 2023.

The 2023 Lemhi MRE is classified according to the CIM "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" dated November 29th, 2019 and CIM "Definition Standards for Mineral Resources and Mineral Reserves" dated May 10th, 2014.

25.7 Mining Methods

A reasonable open pit mine plan, including pit designs, mine production schedules, and mine capital and operating costs have been developed for the Lemhi Project PEA.

Pit layouts and mine operations are typical of other regional open pit gold operations, and the unit operations within the developed mine operating plan are proven to be effective for these other operations.

The mine production schedule and estimated mine capital and operations costs are reasonable at a scoping level of engineering and support the cash flow model and financials developed for the PEA.

25.8 Recovery Methods

In its initial phase, the plant is designed to process material at a rate of 2.5 Mt/a (6,849 t/d), and this capacity will be elevated to 3.0 Mt/a (8,219 t/d) after the expansion in the fifth year with an average head grade of 0.88 g/t Au to produce doré. The process plant features the following:

- crushing of ROM material
- SAG mill with trommel screen followed by a ball mill with cyclone classification
- leach + carbon-in-leach (L/CIL) adsorption
- carbon desorption followed by electrowinning and smelting to produce doré
- cyanide destruction/wet tailings deposition.

The process plant flowsheet designs were based on testwork results, financial evaluations, and industry standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and LOM operating costs. The comminution and recovery processes are conventional and well-established in the mining industry with no significant elements of technological innovation.

25.9 Project Infrastructure

The Lemhi Project includes on-site infrastructure such as civil, structural and earthworks development, site facilities and buildings, on-site roads, water management systems, and site electrical power facilities. Off-site infrastructure includes site access roads, fresh water supply, and power supply. The site infrastructure will include:

- Mine facilities, including mining administration offices, a mine fleet truck shop and wash bays, and a mine workshop.
- Common facilities including an entrance/exit gatehouse, a security/medical office, overall site administration building, potable water and fire water distribution systems, compressed air, power generation and distribution facilities, diesel reception and combustion plants, communications area, and sanitation systems.
- Process facilities include crushing, grinding and classification, leaching, electrowinning, reagent mixing and distribution, assay laboratory, and process plant workshop and warehouse.
- Other infrastructure includes the CPSFs that are designed in accordance with State regulations for mine waste storage facilities.

The project is located in Lemhi County in east-central Idaho, within the Salmon River Mountains, a part of the Bitterroot Range which forms the Idaho-Montana border. The property is 40 km north of the town of Salmon, ID and 6 km west of Gibbonsville, ID.

The Lemhi Project site is accessible via multiple routes. The primary access is through Salmon, Idaho, involving paved and gravel roads. To access the Lemhi Project site and process plant, routing will be upgraded as the access is through mountainous terrain that features some switchbacks and sharp turns. As part of upgrading activities, some of these switchbacks and turns will be improved to meet the transportation needs of the site. The proposed access route avoids both residential areas in the region and the project's 300 m blast radius for the project's open pit mine design.

The fresh water will be supplied from the wells located on site. This water will be the source of potable water on site, used for the building facilities and the process plant.

Electrical power will be supplied from the local grid via a 5 km power that will be constructed for the project. The power line will be connected to a high voltage line that passes nearby the project site and distributed to different power requirements across the project site.

Fuel will be delivered to the mine site via tanker trucks. The fuel storage tanks are insulated and heated to prevent fuel gelling. The tanks will be contained in a lined containment berm to assure no fuel can leak into the environment.

The plant site will consist of infrastructure necessary to support the processing operations with all buildings and structures constructed to comply with all applicable codes and regulations.

Site selection and location for project infrastructure was guided by the following considerations:

- Locating the facilities described above on the Lemhi patent land to the greatest extent possible.
- Locating two CPSFs close to the open pit to reduce haul distance.
- Locating primary crushing close to the Lemhi deposits to reduce haul distance.
- Utilizing the natural high ground for the ROM pad as much as possible.
- Separating heavy mine vehicle traffic from non-mining, light vehicle traffic.
- Locating the process plant near an existing primary access road.
- Locating the process plant in an area safe from flooding.
- Placing mining, administration, and process plant staff offices close together to limit walking distances between them.

25.10 Co-Placement Storage Facility (CPSF)

Two waste materials are generated during the mining process: waste rock and tailings. Tailings and coarse waste rock material will be transported independently, but not mixed to form a single discharge stream, into co-placement storage facility (CPSF). For the Lemhi Project, two co-placement storage facilities will be constructed over the life of mine (LOM), the North CPSF and the South CPSF.

The North CPSF will be constructed first since it is within their patented claims boundary while Freeman Gold obtain permit to store waste materials on federal National Forest lands. The North CPSF has a storage capacity of 37. 4 Mt of tailings and waste rock. This facility will be a slurry tailings facility with upstream raises since there is sufficient waste rock to develop a starter embankment. The facility has storage capacity for over 2 years of tailings and waste rock.

The South CPSF will be commissioned in Year 3 after obtaining the permit to place waste materials on federal lands. There is insufficient time to construct an embankment for slurry tailings; therefore, the North CPSF will be a paddock style construction utilizing waste rock to create cells and the tailings will be filtered to a cake and placed in the cells. The south CPSF has a storage capacity of 115.6 Mt of tailings and waste rock. This facility will be a slurry tailings facility with upstream raises since there is sufficient waste rock to develop a starter embankment. The facility has storage capacity for over 2 years of tailings and waste rock.

For the North CPSF tailings will be conveyed to the site in a pipeline and decant water will be reclaimed from the back of the facility. For the South CPSF both tailings and waste rock will be transported by haul trucks.

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 not 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 Environmental, Permitting, and Social Considerations

A number of limited field and screening environmental baseline studies and reports were completed between 1995 and 2012. The programs involved the collection of baseline data within the proposed project footprint area and commenced the process of identifying potential environmental constraints and opportunities related to the proposed development of the project.

It should be noted that the field data collected for baseline studies is not recent. Updated and expanded baseline studies will be required to support continued baseline development and future impact assessment. New baseline data should be collected and analyzed in accordance with current applicable scientific standards and methodologies and historical baseline data reviewed from that perspective. In assessing the utility of using older baseline data, direct discussions should be conducted with state and federal regulators.

In addition, there have been no baseline studies completed to date on air quality, meteorology, noise, greenhouse gases and climate change, wildlife and wildlife habitat, and cultural resources. Ongoing and expanded baseline studies will be required to support the project through pre-feasibility, feasibility, and environmental impact statement/permitting stages of the project. The results of baseline studies and identification of environmental constraints and cultural resources can be used to minimize impact of the project on valued ecosystem components and to optimize the location and operation of project infrastructure. Baseline study recommendations for the purpose of advancing the project to the PFS stage are provided in Section 26.7.

As discussed in Section 20.2.4, based on a review of available literature, four listed species (threatened) are, or may be, in the vicinity of the project site triggering the requirement for consultation under ESA Section 7 on any federal action that may affect these species or their designated critical habitats. The four federally listed species include: Bull trout, Canada lynx, grizzly bear, and whitebark pine.

In terms of water management, the main consideration for the project is related to changes in the flow regime in and in the vicinity of the project site. The preliminary design contemplates modifications to Ditch Creek, Little Ditch Creek, and Ransack Creek as part of the overall conceptual mine plan. It should be noted that some of the water courses on site (e. g., Ditch Creek), as well as downstream reaches of those water courses are considered critical habitats for Bull trout. As discussed in Section 20.4.3.2, under a U. S. Fish and Wildlife Service Final Rule, Ditch Creek and Hughes Creek were designated as critical habitat as part of the Salmon River Basin Critical Habitat Unit.

Also of note is that based on the project's LOM production schedule, the initial North CPSF, located within Freeman's Gold patented claim boundary, will be used for a two-year period, after which a permit from the US Department of Agriculture – Forest Service (USFS) will be required that would allow placement of mine waste in the South CPSF located within the Salmon-Challis National Forest. It is likely that based on this plan, the mine will be subject to the NEPA review process. An overview of the NEPA review process was described in Section 20.4.2.

As additional baseline data is collected and community and regulatory engagement efforts proceed, changes to project infrastructure design (and estimated costs) may be required at the PFS and future stages including permitting based on the following key studies:

- Fish and fish habitat characteristics for the areas of proposed project disturbance as related to future design, permitting requirements and risks.
- Improved understanding of vegetation/ecosystem and wildlife habitat especially of federally listed (threatened) species on and near the project area.
- Refined understanding of hydrological and hydrogeological conditions related to water balance.
- The quality and quantity of mine contact water is based on geochemical characterization and predictions.
- Traditional land use activities near the project area.

25.13 Capital Cost Estimates

The capital cost estimate conforms to Class 5 guidelines for a pre-feasibility-level estimate with a +50%/-30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2023 dollars based on budgetary quotations for equipment and construction contracts, as well as Ausenco's in-house database of projects and studies as well as experience from similar operations. The calculations are based on the open pit mining operation, the development of a processing plant, infrastructure, tailings storage facility and management facility, and owner's expenses and provisions.

The total initial capital cost for the Lemhi Project is US$190.2 M, and the LOM sustaining cost including financing is US$101.2 M. The capital costs are summarized in Section 21.2.

25.14 Operating Cost Estimates

The operating cost estimate was developed in Q2 2023 dollars from budgetary quotations and Ausenco's in-house database of projects and studies as well as experience from similar operations. Mine operating costs are based on benchmarking to other similar sized operations in western United States, mining 12-16 Mt/a. The accuracy of the operating cost estimate is +50%/-30%. The estimate includes mining, processing, and G&A costs. For more details, refer to Section 21.4.

The overall LOM operating cost is US$670.3 M over 11.2 years, or an average of US$21.53/t of material milled in a typical year.

25.15 Economic Analysis

An economic model was developed to estimate the project's annual pre-tax and post-tax cash flows, sensitivities, and net present value results using a 5% discount rate.

The pre-tax NPV discounted at 5% is US$297 M; the IRR is 26.9%; and payback period is 3.3 years. On a post-tax basis, the NPV discounted at 5% is US$212.4 M; the IRR is 22.8%; and payback period is 3.6 years.

A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using the following variables: gold price, operating costs, initial capital costs, mill recoveries, and mill head grades. The sensitivity analysis revealed that the project is most sensitive to changes in gold price, mill head grade and mill recovery, and to a lesser extent, changes in operating costs and initial capital costs.

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

25.16 Risks

25.16.1 Metallurgical Test Work

There is an inherent risk that the samples tested do not suitably represent the deposit, however the quantities of samples and spatial coverage suggest that this risk is low.

Additional comminution testing should be completed on samples representing mill feed after Year 5 to confirm that the grinding circuit design is adequate for the expansion.

The impact on leach extraction at the coarser grind size selected for the expansion should be investigated further by testwork on material representing the later years of operation.

The completed test programs identified a small number of variability samples that contained elevated copper levels which appeared to compromise gold leach extraction. A review of copper levels in the resource should be conducted to determine the prevalence of this potential risk. Additional testing should be conducted on material with elevated copper levels to better understand the effect on metallurgical performance, and if warranted, determine processing methods to mitigate any negative effects.

Pressure filtration testing has not been completed on a representative tailings sample. This testing should be completed to adequately design the tailings filtration process proposed as an expansion item in Year 2.

25.16.2 Mineral Resource Estimate

The Lemhi Property carries risks inherent in utilizing significant amounts of historical drilling. Specific risks center on the poor reproducibility of assay results from the 2012 LGT core twinning program as compared with historical RC hole results. Confirmation drilling completed in 2012 by LGT included twin holes of historic drill holes with both core and RC drilling methods. The results from the LGT twin holes indicate that 2012 core drilling returned a number of erratic and a few lower grade intersections for a number of holes versus historical RC drilling within the same mineralized zones. Historically these variances were also observed in comparisons between historical core holes and historical RC holes whereby the core holes returned lower overall assays for a particular interval.

Additionally, assaying both halves of split core has indicated that gold values can also vary significantly within a particular core interval, this is further confirmed by the duplicate analyses received to date in the 2020 Freeman Phase 1 drilling program. LGT's duplicate sampling using the 2012 pulps and rejects showed significant variances between fire assay and metallic screen assay results of as much as 300%. LGT duplicate sampling has also indicated variances of between 200% and 400%. Brewer (2019) concludes that while these variances are not the norm, they do indicate that the Lemhi Gold Deposit exhibits some significant nugget effects. The 2020 drill program has identified a significant number of occurrences of visible gold in several core holes, likely further indicative of potential nugget effects.

The issue of poor assay value reproducibility is poorly understood and requires further investigation. The discrepancy can, at least in large part, be explained by the indications of potential nugget effect in this deposit, along with the uncertainty of accurately "twinning" unsurveyed historical drill holes and, the inherent grade variance within a deposit that does have some mineralization related to quartz veining.

The inclusion of 1980s FMC drill holes increases the risk of a slightly biased estimate in areas that rely on the 1980s FMC data. To this end, it is recommended that further infill drilling be completed in areas that significantly rely on the 1980s FMC data to increase the confidence level in those areas.

The authors are not aware of any other significant material risks to the MRE other than the risks that are inherent to mineral exploration and development in general. The authors of this report are not aware of any specific environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that might materially affect the results of this mineral resource estimate and there appears to be no obvious impediments to developing the MRE at the Lemhi Gold Project.

25.16.3 Mining Methods

The project is at a scoping level of engineering. There has been limited geotechnical, hydrogeological, and geochemical information and data collected across the project. Further field work, lab work, and modelling are required to advance engineering to the next stages of pre-feasibility or feasibility. It can be anticipated that further field drilling and advancement of the project engineering will materially alter the existing mine plan, reducing the plan's risk and identifying and exploiting the potential opportunities that arise.

Risks to the preliminary economic assessment (PEA) defined mill feed quantities, gold grades, associated waste rock quantities, and the estimated costs to exploit include changes to the following factors and assumptions:

- Metal Prices
  - Decreases in metal prices may increase the economic cut-off grade, or reduce the size of the open pit, with either outcome reducing the size of the resource base to include into the mine plan.
- Interpretations of mineralization geometry and continuity in mineralization zones
  - Decreases in the resource base could significantly alter the mine plan.
- Geotechnical and hydrogeological assumptions
  - Geotechnical sampling, testwork, and analysis may show a required shallowing of pit slope angles, which likely would in turn increase the overall LOM stripping ratio to access the resource.
  - Hydrogeological sampling, testwork, and analysis may identify needs for a more onerous (costly) pit water management and pit slope depressurization solution.
- Geochemical assumptions for mined resource and waste materials
  - Geochemical sampling, testwork, and analysis, specifically in the open pit waste rock, may identify a more onerous (costly) PAG management solution.
- Ability of the mining operation to meet the annual production rate and anticipated grade control standards and recoveries
  - Reduced selectivity with the mining fleet, reduced mining or milling recoveries, or increased mining dilution would result in an increased cost of achieving the planned PEA metal production.
- Operating cost assumptions and cost creep
- Ability to meet and maintain permitting and environmental license conditions, and the ability to maintain the social license to operate
- Ability to access capital for project financing.

25.16.4 Recovery Methods

This study was performed with limited metallurgical testing, potentially resulting in the following:

- Grinding equipment was selected based on available comminution test data and may be undersized if actual hardness values are higher than the design values.
- The gold recovery flowsheet was selected based on the metallurgical testing data available and may not be optimal.
- Process conditions, residence times, and reagent usage may change with further testing.

- A provision for tailings filtration has been added in Year 2, however there is limited data to support the design requirements of this process unit.

Testing to date is based on limited composite samples. Testing on variability samples from the deposit may have different metallurgical characteristics than assumed for this study.

25.16.5 Infrastructure

25.16.5.1 Water Supply

Hydrogeological information is limited for the water wells to evaluate the sustainability of groundwater as a water supply source to the project.

25.16.5.2 CPSF

The North CPSF assumed geotechnical parameters are lower, which could result in a smaller facility that does not have a two-year storage capacity. Due to the steep terrain in the project area, there is limited expansion capabilities for the CPSFs.

25.16.6 Site Geotechnical

Risks related to the site infrastructure include the following:

The project infrastructure presented has no historical geotechnical information, therefore, construction material properties, subsurface conditions, waste material properties have been assumed. The risks to proposed infrastructure, excluding the open pit, are:

- Infrastructure ground conditions, geological containment, and stability of the proposed CPSFs areas are unknown as a geotechnical program has not been completed.
- There is no geochemical analysis to define the non-acid generating (NAG) / potentially acid generating (PAG) of waste materials.
- There is a possibility for cost could increase if the geotechnical, hydrogeological, and geochemical characteristics foundation, construction materials, and waste materials are different from the criteria considered in this study that could impact the capital, sustaining capital and operating costs for the project.

25.16.7 Commodity Prices

The ability of mining companies to fund the advancement of their projects through exploration and development is influenced by commodity prices. Variations in the commodity prices may lead to reduced or elevated revenues compared to those projected in this study.

25.16.8 Environmental Permitting

Both the United States and the State of Idaho employ rigorous but well-defined processes to complete environmental assessments and overall permitting for activities in their respective jurisdictions and it would be anticipated that the Lemhi Project would fall under these processes and timelines.

The main risks associated with the permitting schedule for the project include:

- Potential lack of support from community and local Indigenous tribes
- Potential impacts to fish and fish habitat, particularly Bull trout critical habitat, that cannot be readily avoided, resulting in difficulties in receiving authorization under the NEPA process
- Potential delays or barriers to obtaining approvals for the purpose of long-term LOM waste disposal (waste rock and tailings) within the Salmon-Challis National Forest
- Potential impacts to listed/threatened species and the requirement under the legislation to ensure that the species are preserved, and the site infrastructure/activities are not likely to jeopardize their existence or adversely modify their designated habitat (e.g., Bull trout)
- Potential location of mine infrastructure in the headwaters of a salmon fishery
- Potential mine effluent characteristics that may require water treatment throughout the mine life.

The timely implementation of the recommendations presented in Section 26.7 will help to quantify, qualify, and potentially mitigate these risks to the PFS stage of the project as well as future permitting and schedule.

25.17 Opportunities

25.17.1 Metallurgical Test Work

Reducing leach time from 36 to 24 hours could reduce capital needed for leaching; additional test work is needed to confirm the effects on gold recovery.

25.17.2 Mineral Resource Estimate

There are areas of the inferred block model that, although the drilling was of sufficient density for an improved classification to indicated or better, the areas were dominated by 1980s drillholes; the authors review demonstrated presence of systematic bias toward high values, which is related to increased risk. Some pointed infill drilling in these areas dominated by 1980s drillholes might improve confidence in the data and lower the risk, allowing for a higher classification.

In addition, the rock and soil sampling along with some limited exploration drilling outside of the main conceptual pit area shows areas with a number of good results but with little follow-up exploration drilling. With further work, including drilling, there are opportunities to increase the mineral resources on the project.

25.17.3 Recovery Methods

There may be opportunities to improve the process flowsheet for the project once suitable metallurgical testing is completed. The studies should include engineering trade-off studies to confirm the following:

- target grind size and comminution circuit selection
- leaching retention time
- review of plant layout to incorporate any recommendations generated by the work described above.

Further opportunities exist to confirm that the gold recovery circuit selected in this process design is optimal for the life of mine with respect to both capital and operating costs.

25.17.4 Environmental Permitting

Opportunities, as listed below, should be considered as the project continues along the development path.

1. The timely initiation of community, Indigenous, and regulatory engagement regarding proposed project, anticipated impacts (both positive and adverse) and proposed impact mitigation, including discussions with communities on potential benefits of the project.
2. The timely initiation of targeted environmental and socio-economic baseline studies that will inform impact mitigation and risk reduction measures associated with infrastructure footprint, and adoption of appropriate low impact and sustainable technologies.
3. Entering discussions with federal and state agencies regarding the utility and use of historic baseline data.
4. Regarding hydrological, hydrogeological, and geochemical studies, there are opportunities to work closely and collaborate with the geotechnical, water resources, and mineralized material processing engineering teams and hence, reduce effort and costs.

26 RECOMMENDATIONS

26.1 Overall Recommendations

The Lemhi Project demonstrates positive economics, as shown by the results presented in this technical report. Continuing to develop the project through to pre-feasibility study is recommended. Table 26-1 summarizes the proposed budget to advance the project through the pre-feasibility stage.

Table 26-1: Cost Summary for the Recommended Future Work

Item	Budget (US$M)
Exploration and drilling	4.00
Metallurgical testwork	0.15
Mining methods	2.20
Process and infrastructure engineering	0.80
Site-wide assessment & CPSF geotechnical studies	0.96
Environmental, permitting, social and community recommendations	0.99
Total	**9.10**

Note: Totals may not sum due to rounding

26.2 Exploration and Drilling

Historical drilling and the recent Freeman drilling have defined a significant zone of gold mineralization at the Lemhi Gold Project. Prior 3D modelling has shown the deposit to be of significant size and open in several directions, which was confirmed with the 2020-2022 drilling. Prior to 2020, little surface exploration has been conducted at the Lemhi project since the late 1980s. Certainly no modern exploration techniques have been employed to either extend the known mineralization or identify new mineralization along strike.

A significant mineralized zone has been intersected by numerous drill holes between 1984 and 2022 and a modern MRE has been established. The work to date indicates that there is potential to expand the current MRE and there is potential for new discoveries with further exploration drilling. The MRE can be improved by additional drilling to increase confidence in the MRE, upgrade the classification and reduce the reliance on FMC 1980s drill hole data.

A follow-up exploration program would include both infill and exploration drilling to expand the resource base at Lemhi, further metallurgical drilling and studies, a property-wide soil and rock sampling program, geological mapping, trenching and remote sensing surveys such as Worldview 3 alteration mapping and a structural interpretation of LiDAR surveys completed by the Idaho LiDAR Consortium (processing of LiDAR survey is ongoing by Boise State University).

The proposed Phase 1 program for 2023 would be comprised of 8,000 m of core drilling (HQ and PQ) in at least 40 holes along with geological mapping, soil, and rock sampling, trenching in areas where mineralization has been identified at surface, along with various remote sensing studies to guide a modern structural interpretation. The estimated cost of the proposed Phase 1 exploration program is US$4.0 M (C$5.0M).

26.3 Metallurgical Testwork

Additional metallurgical testing should be conducted to confirm the following:

- Comminution properties across the deposit, particularly in later years.
- The effect of a coarser primary grind size on metallurgical performance for material mined after the proposed expansion.
- The effect of elevated copper grades in the feeds on metallurgical performance, should elevated copper grades be identified in a significant portion of the resource.
- The potential to reduce the leach residence time to 24 hours, confirmed through triplicate testing on samples.
- Mercury concentrations in samples.

The above scope of metallurgical testing is estimated to be US$0.15 M.

26.4 Mining Methods

The following recommendations are made with regards to advancing the mine engineering of the Lemhi Project to a pre-feasibility study, with estimated budget for each recommended program included:

- Targeted open pit geotechnical drilling using triple-tube HQ holes and televiewer with oriented cores (US$1.5 M):
    - 6-8 drillholes, 500 m length.
    - Installation of vibrating wire piezometers in select holes.
    - Laboratory testing for intact rock strength (unconfined compressive strength tests, point load tests, and indirect tensile strength tests) and for discontinuity strength (direct shear tests).
    - Build-up of 3D fault and rock mass fabric models.

Packer testing should be conducted to determine pit hydrogeology, hydraulic conductivity and refine pit water inflow estimates.

- Further hydrogeological and hydrological characterization are required in the pit areas. (no cost, covered elsewhere).
- Condemnation drilling of the footprints identified for the waste rock storage facilities, as well as site infrastructure; condemnation drilling is done to ensure no valuable mineralization exists below these planned facilities, so that it is not locked in the ground from future potential exploitation. (US$0.5M).

- Drill penetration and blast fragmentation studies, testing properties in all lithologies, as well as within mineralized areas and within waste rock. It is possible to utilize exploration and geotechnical drill core for rock samples, and no additional drilling has been planned for these studies in the estimated budgets. (US$0.02M).
- Updates to designs of open pits, waste storage facilities, stockpiles, and mine haul roads incorporating results from all other recommended work programs. (US$0.15M).
- Mine operational and cost trade-off studies examining contractor vs. owner equipment fleet, lease vs. purchase equipment fleet, cost comparisons of various equipment class sizes, and utilization of electrically driven mine equipment (including trolley systems for haulers) over diesel driven units. (US$0.03M).

A budget of $2.2 M is estimated for the above work programs and studies.

26.5 Process and Infrastructure Engineering

The estimated cost for process and infrastructure engineering for the PFS is US$0.80M Engineering deliverables would include:

- PFS Trade Off Studies targeting NPV and IRR improvement scenarios
- Process Plant Engineering, through criterion, lists, drawings, MTOs and cost estimates
- PFS Cost Estimating
- PFS project execution planning
- Technical Report Support.

26.6 Site-wide Assessment and CPSFs Geotechnical Field and Laboratory Program

Due to the conceptual nature of this study and the paucity of information available at the time of writing, assumptions have been made regarding the layout, MTOs, and construction of the proposed co-placement storage facility (CPSF). Construction material geotechnical properties are required to perform slope stability analyses and other geotechnical assessments to confirm that the CPSF can be built as designed. In addition, a detailed tailings and waste rock deposition plan will be required which may lead to the conceptual staging requiring adjustment to contain the given waste material capacities.

- Geological and geotechnical site investigations and laboratory program should be carried out for infrastructure, process plant, and CPSFs that shall include drilling, test pitting, and in-situ and laboratory testing, to understand foundation, tailings, and waste rock characteristics, construction material properties, and groundwater levels.
- Seepage and stability analyses for the CPSF needs to be investigated with information gathered from the field and laboratory programs.
- Hydrological information should be gathered from site-specific climate studies to detail site surface water management and site water balance.

- Hydrogeological information from desktop studies and site investigations should be gathered to better understand subsurface flow regimes and potential pit dewatering.
- Development of factual and interpretive geotechnical reports.

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

26.6.1 Co-placement Storage Facilities (CPSF)

To bring the design and analysis of the CPSFs, along with supporting infrastructure (access roads, surface water management) to support a pre-feasibility-study the following activities are recommended:

- Acquire satellite imagery for site.
- Update geochemical characterization of tailings, waste rock and construction materials.
- Develop seepage predictions and seepage control measures for the CPSFs.
- Optimize the tailings and waste rock handling and deposition strategy, including trafficability of material handling equipment for the CPSFs.
- The stability model should be reviewed and updated, as required, with consideration of the final stacking plan using updated data about the material properties of the waste using laboratory results along with foundations for the CPSFs.
- Perform a liquefaction assessment with consideration of updated information on material properties for the tailings along with foundation for CPSFs.
- Solicit additional budgetary quotes for earthworks and geosynthetics (i.e., geomembrane, geotextile, and piping) to get more accurate pricing for the next cost estimates.
- Develop PFS level design of CPSF.
- Develop cost estimates (i.e., capital, sustaining capital, and operating costs) for site vegetation suppression, earthworks, and material placement costs for CPSFs

The estimated cost for the recommended work is US$120,000.

26.7 Environmental, Permitting, Social and Community Recommendations

The following recommendations are made with regard to the design and implementation of environmental and socio-economic baseline studies. Qualified professionals should be retained to design and oversee the implementation of each of these studies. A review of historical baseline data (collected in 1995 and 2012) should be undertaken as part of the design and scoping of these studies, prior to field implementation. These studies and activities will be necessary to support the project to the PFS stage and provide a strong basis for future EIS preparation and permitting.

26.7.1 Water Resources:

- Compile a multi-year seasonal hydrological and meteorological monitoring plan for key areas within the study area to further characterize the hydrological conditions and to develop a future water balance model. The water balance model will be used as a predictive tool regarding the quality and quality of water available to support mineral processing as well as prediction of effluent quality and quantity. Consideration should be given to establishing a site-specific meteorological station, based on the adequacy of continuing to use data from regional stations.
- Development and implementation of a surface water and groundwater monitoring, sampling, and testing plan focusing on areas that will be potentially affected by mine infrastructure based on current infrastructure plans (refer to Section 18). As part of this program, and as an effort to reduce costs, the condition of the existing monitoring well network (nine monitoring wells) should be reviewed, and the wells rehabilitated if warranted. Consideration should be given to establishing additional wells based on an adequacy assessment.
- The existing conceptual hydrogeological model should be reviewed and updated, based on monitoring and testing results, and should provide the basis for the future development of a three-dimensional numerical groundwater model that will support advanced feasibility design phases and EIS. The model should provide emphasis on seasonal recharge of the freshwater aquifers within and near the project area and the potential drawdown from future pit development and dewatering activities as well as pit water inflows.

Estimated costs for the above recommendations are $500,000, assuming that existing groundwater monitoring wells can be rehabilitated and utilized. Cost savings can be realized for hydrogeological characterization work by coordinating closely with geotechnical and exploration drilling teams and their drilling programs.

26.7.2 Geochemistry

- A geochemical assessment of the ARD/ML risk for the project should be implemented utilizing the existing geological model for the site and sampling of fresh drill core sampled intervals, if available. Generally, the program should consist of the collection of the following samples:
  - Collection of around 200 to 300 waste rock samples based on the site geological and structural model.
  - Three to six tailings samples, collected during future mineralogical test work.
  - Three to six mineralogical rock samples.
  - Several overburden samples.
- Range of analytical tests to include elemental analysis, acid-base accounting, shake flask extraction (short term leach), NAG pH, minerology, and humidity cell testing (minimum 40 weeks).
- Development of preliminary source terms for the weathering of waste rock, mineralized material, tailings, and pit walls for use in water balance modelling.
- Preliminary interpretation of results and assessment of requirement for site-specific mine rock management practices and water treatment.

The estimated costs for the above are US$200,000.

26.7.3 Fish and Fish Habitat and Aquatic Studies

- Develop and implement multi-year and seasonal baseline fish and fish habitat for key waterbodies within and downstream of the project area.
- Develop and implement multi-year baseline aquatics study that includes physical and chemical parameters, aquatic sediments, tissue residues, and aquatic life (invertebrates, algae, macrophytes) for key areas within the project area and for reference areas.
- Based on the results of the above, develop plans that effectively mitigate potential impacts to fish and fish habitat based on an assessment of alternatives, including mine waste facilities and realignment of Ditch Creek and modification to other water bodies.

The estimated costs for the above are US$100,000.

26.7.4 Terrestrial and Wildlife Monitoring

- Develop and implement a seasonal baseline vegetation/ecosystem and wildlife/wildlife habitat survey plan for key areas within the project area with special emphasis on listed and threatened species under the Endangered Species Act.
- Indigenous tribal and other land users should be offered the opportunity to become closely involved in the development and execution of wildlife baseline studies, especially in relation to traditional and current use of the land for harvesting.

The estimated costs for the above are US$100,000.

26.7.5 Air Quality and Noise

- Baseline conditions for air quality and noise should be established for near field and further afield operations.

The estimated cost for the above is US$20,000.

26.7.6 Near Surface Soil Characteristics

- Near surface soil textures and chemistry should be established for the project area as part of the baseline program.

The estimated cost of the above is US$10,000.

26.7.7 Socio-Economic, Cultural Baseline Studies and Community Engagement

- Develop and implement Class I and Class II cultural baseline studies.
- Develop and implement socio-economic baseline study.
- Initiate community engagement and tribal consultation to understand current land and resource use at or near the project area and potential impacts (positive and negative) to same due to project development.

The estimated costs for the above are US$50,000.

26.7.8 Environmental Constraints Mapping

- To assist in the development of the project at the PFS stage, environmental constraints mapping should be developed and continuously updated, based on the results of historical and future baseline environmental and land use studies. This mapping should be utilized to limit risks at the design stages of the project.

The estimated cost for the above task is US$10,000.

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Appendix 1 – List of Claims

FREEMAN GOLD CORP. (o/a Lower48 Resources Idaho LLC)

FMAN: CSE

LIST OF CLAIMS

Unpatented Claims

Name	Number of Unpatented Claims	Serial Numbers
L48-1 to L48-231	231*	IMC230832 to IMC231050; IMC231481 to IMC231484
BHLK#91 to BHLK#128 BHLK#130 to BHLK#141 BHLK#126A,BHLK#127A, BHLK#128A	53*	IMC229036 to IMC229088
BHLK12 to 15, 17 to 21,23 to 27, 46 to 51 BHLK#2-#6, #16,#28, #41 to #45 BHLK#52 to #58, #63 to #67, #73, #200	46**	IMC205844 to IMC205893, IMC205895 to IMC205899 IMC205918 to IMC205923; IMC220145 to IMC220170
MOON#100, MOON#101	2*	IMC202837 to IMC202838
TOTAL	**332**	

Patented Claims (11)*

Name	Mineral Survey Number	Acres
Ditch Creek Placer Hamilton Placer Marysville Placer Conola Lode Copperstain Lode	MS 1120 MS 2512 MS 784 A and B	477.75 19.79 20.66
Beauty Consolidated Lode Atlanta Lode Fraction Lode Chamaleon Lode Chamaleon Millsite Ironstone Lode	MS 784 A and B	97.75
TOTAL	**11**	

*Patented and unpatented claims owned 100% by Lower 48 and parent Freeman Gold
**Unpatented claims owned by BHLK and under option to purchase 100% by Lower 48 and parent Freeman Gold