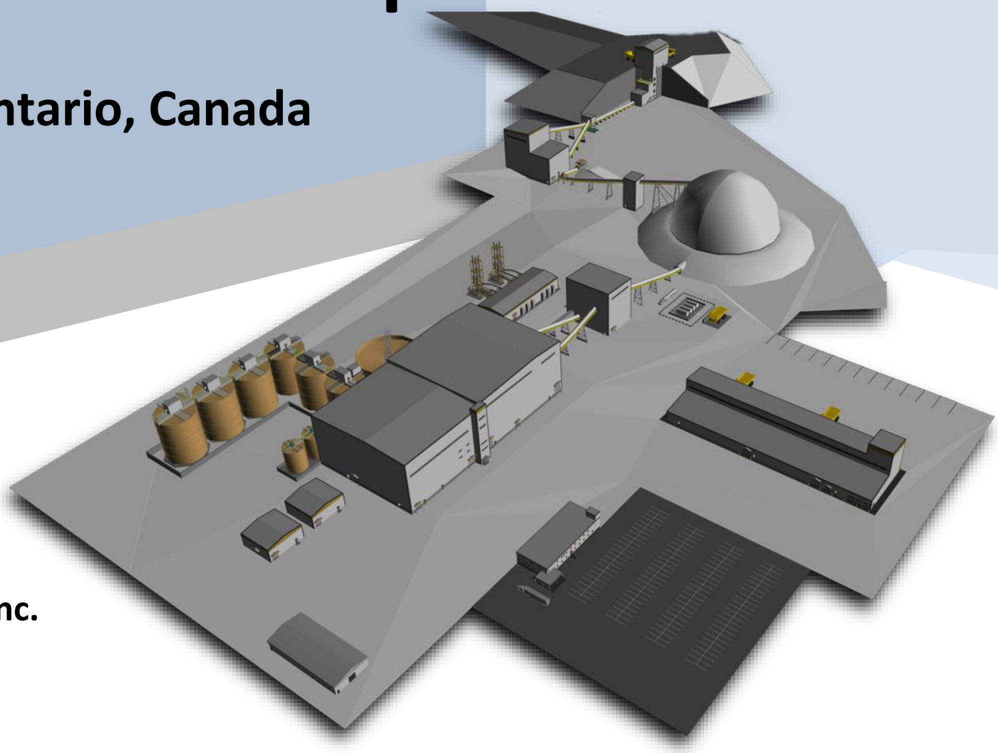
NI 43-101 Technical Report

Hardrock Project, Ontario, Canada



Prepared for:

Greenstone Gold Mines GP Inc.
Suite 500, 365 Bay Street
Toronto, Ontario

Centerra Gold Inc.
Suite 1500, 1 University Avenue
Toronto, Ontario

Premier Gold Mines Limited
Suite 200, 1100 Russell Street
Thunder Bay, Ontario

Prepared by:

G Mining Services Inc.
D200, 7900 Taschereau Blvd.
Brossard, Québec

Louis-Pierre Gignac, P.Eng., G Mining Services Inc.
Glen Schlyter, P.Eng., G Mining Services Inc.
Martin Ménard, P.Eng., G Mining Services Inc.
Rejean Sirois, P.Eng., G Mining Services Inc.
Charley Murahwi, M.Sc., P.Geo., Micon International
Eric Poirier, P.Eng., WSP Canada Inc.
Pierre Roy, M.Sc., P.Eng., Soutex
David G. Ritchie, P.Eng., Amec Foster Wheeler
Marc Rougier, P.Eng., Golder Associates Limited
Craig Johnston, M.Sc., P.Geo., Stantec

Effective Date: October 1, 2016
Issue Date: December 21, 2016

Cautionary Note Regarding Forward-looking Information

Information contained in this report and the documents referred to herein which are not statements of historical facts, may be "forward-looking information" for the purposes of Canadian securities laws. Such forward looking information involves risks, uncertainties and other factors that could cause actual results, performance, prospects and opportunities to differ materially from those expressed or implied by such forward looking information. The words "expect", "target", "estimate", "may", "will", and similar expressions identify forward-looking information. These forward-looking statements relate to, among other things, mineral reserve and resource estimates, grades and recoveries, development plans, mining methods and metrics including strip ratio, recovery process and the expected performance of the HPGR, mining and production expectations including expected cash flows, capital cost estimates and expected life of mine operating costs, the expected payback period, receipt of government approvals and licenses including the timing for submitting a response to the EIS/EA, time frame for construction, financial forecasts including net present value and internal rate of return estimates, tax and royalty rates, expected costs relating to the relocation of certain existing infrastructure, opportunities to improve the LOM average grade from processing material from other Greenstone Gold Property, including Brookbank and the Hardrock underground; and the possibility of any benefit of historical tax positions held by Centerra Gold Inc. ("Centerra") or Premier Gold Mines Limited ("Premier").

Forward-looking information is necessarily based upon a number of estimates and assumptions that, while considered reasonable by the managing partner, Greenstone Gold Mines GP Inc. ("GGM"), Centerra and Premier, are inherently subject to significant political, business, economic and competitive uncertainties and contingencies. There may be factors that cause results, assumptions, performance, achievements, prospects or opportunities in future periods not to be as anticipated, estimated or intended. These factors include the following risks relating to the Hardrock Project, GGM, Centerra and/or Premier: (A) strategic, legal, planning and other risks, including the risks for disagreement between the partners on how to explore, develop, operate and finance the Project, political risk, risks relating to aboriginal claims and consultation issues; resource nationalism including the management of external stakeholder expectations; the impact of changes in, or to the more aggressive enforcement of laws, regulations and government practices; the impact of changes to and the increased enforcement of, environmental laws and regulations; potential defects of title to the property that are not known as of the date hereof; the inability of the Partnership and its partners to enforce their respective legal rights in certain circumstances; risks related to anti-corruption legislation; potential risks related to kidnapping or acts of terrorism; (B) risks relating to financial matters, including the ability of the partners to provide funding to the Partnership in accordance with the terms of the Partnership Agreement; sensitivity of the business to the volatility of gold prices; the imprecision of mineral reserves and resources estimates, and the assumptions they rely on; the accuracy of the production and cost estimates; the ability to obtain financing for the Partnership or by either partner; the impact of global financial conditions, the impact of currency fluctuations, the effect of market conditions on short-term investments, the ability of the partners including Centerra to make payments to the Partnership depends on the cash flow of its subsidiaries; and (C) risks related to operational matters and geotechnical issues; the success of the Partnership's future exploration and development activities, including the financial and political risks inherent in carrying out exploration activities; inherent risks associated with the use of sodium cyanide in the mining operations; the adequacy of insurance to mitigate operational risks; mechanical breakdowns; the occurrence of any labour unrest or disturbance; the ability to accurately predict decommissioning and reclamation costs, including closure costs; the ability to attract and retain qualified personnel; the ability to manage projects effectively and to mitigate the potential lack of availability of contractors; budget and timing overruns and project resources; potential delays in the issuance of permits; potential opposition to the Hardrock Project by local communities or civil groups; potential material increases in project development or operation costs due to increases in key consumables, inflation, imposed demands for infrastructure development or regulatory changes; and the planning, design and costing of the key project infrastructure such as power, water and access.

There can be no assurances that forward-looking information and statements will prove to be accurate, as many factors and future events, both known and unknown could cause actual results, performance or achievements to vary or differ materially, from the results, performance or achievements that are or may be expressed or implied by such forward-looking statements contained herein or incorporated by reference. Accordingly, all such factors should be considered carefully when making decisions with respect to Centerra/Premier, and prospective investors should not place undue reliance on forward-looking information. Forward-looking information in this technical report is as of the issue date, December 21, 2016. Centerra/Premier assumes no obligation to update or revise forward-looking information to reflect changes in assumptions, changes in circumstances or any other events affecting such forward looking information, except as required by applicable law.

Qualified Persons

Prepared by:

(signed and sealed) "Louis-Pierre Gignac"

Louis-Pierre Gignac, P.Eng.,
Co-President - Mining Engineering
G Mining Services Inc.

Date: December 21, 2016

(signed and sealed) "Glen Schlyter"

Glen Schlyter, P.Eng.,
Engineering Manager
G Mining Services Inc.

Date: December 21, 2016

(signed and sealed) "Martin Ménard"

Martin Ménard, M.Sc., P.Eng.,
Senior Electrical Engineer
G Mining Services Inc.

Date: December 21, 2016

(signed and sealed) "Réjean Sirois"

Réjean Sirois, P. Eng.,
Vice President - Geology & Resources
G Mining Services Inc.

Date: December 21, 2016

(signed and sealed) "Charley Murahwi"

Charley Murahwi, M.Sc., P.Geo.,
Senior Geologist,
Micon International

Date: December 21, 2016

Effective Date: October 1, 2016

Qualified Persons

Prepared by:

(signed and sealed) "Eric Poirier"

Eric Poirier, P.Eng.
Director – Electricity and Control
WSP Canada Inc.

Date: December 21, 2016

(signed and sealed) "Pierre Roy"

Pierre Roy, P.Eng., M.Sc.,
Sr. Metallurgist-Mineral Processing Specialist
Soutex

Date: December 21, 2016

(signed and sealed) "David G. Ritchie"

David G. Ritchie, P.Eng.
Principal Geotechnical Engineer,
Amec Foster Wheeler

Date: December 21, 2016

(signed and sealed) "Marc Rougier"

Marc Rougier P.Eng.,
Principal, Mine Stability, Golder Mining
Golder Associates Limited

Date: December 21, 2016

(signed and sealed) "Craig Johnston"

Craig Johnston, M.Sc., P.Geo.
Senior Principal - Mining Sector Leader,
Environmental Services Canada

Date: December 21, 2016

Effective Date: October 1, 2016

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, Louis-Pierre Gignac, do hereby certify that:

1) I am Co-Vice President and Senior Mining Engineer for G Mining Services Inc. (“GMS”) with an office at D- 200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2.

2) I have graduated from McGill University, Canada with a B.Sc. in Mining Engineering in 1999, and from École Polytechnique de Montréal, Canada with a M.Sc.A. in Industrial Engineering in 2002.

3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, licence no. 132995. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum and I am a Chartered Financial Analyst® Charterholder.

4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 15 years, including pit optimization, surface mine design, mineral reserve estimations and mine scheduling. This has included work at Rosebel Gold Mines NV, the Essakane Project, the Merian Project and work on other pre-feasibility and feasibility studies. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for the preparation of Sections 1, 2, 3, 15, 16, 18.7.2, 19, 21.1.5, 21.2.1, 21.2.2, 21.2.4, 22, 25.1, 25.1.2, 25.1.6, 25.2.2, 26.1, 26.1.2 and 27 of the Report.

7) I visited the Project on June 3, 2014.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared sections of Sections 1, 2, 3, 15, 16, 18.7.2, 19, 21.1.5, 21.2.1, 21.2.2, 21.2.4, 22, 25.1, 25.1.2, 25.1.6, 25.2.2, 26.1, 26.1.2 and 27 of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Louis-Pierre Gignac"

Louis-Pierre Gignac, P.Eng.
Co-President
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. ("GGM"), Centerra Gold Inc. ("Centerra") and Premier Gold Mines Limited ("Premier") dated December 21, 2016, with an effective date of October 1, 2016 (the "Report").

I, Glen Schlyter, do hereby certify that:

1) I am a consulting engineer acting as Engineering Manager for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2.

2) I am a graduate of the Royal Military College of Canada with a B.Sc. (Mechanical Engineering) in 1990.

3) I am a Professional Engineer registered with Professional Engineers Ontario, licence no. 90400920, and the Ordre des ingénieurs du Québec, licence no. 5002983. I am a member of the Canadian Institute of Mining, Metallurgy and Petroleum.

4) I have practiced my profession continuously since 1990. I have over 15 years experience in the mining industry in engineering and project management. Prior to joining GMS, I worked as an engineering and project management consultant for various exploration and mining companies (Newmont Mining Corporation Merian Gold Project, Andean Resources Cerro Negro Gold Project, IAMGOLD Corporation Essakane Gold Project and the Canadian Royalties Inc. Nunavik Nickel Project) as well as some other pre-feasibility and feasibility studies. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for Sections 5.4.1, 5.4.4, 5.4.5, 18.1, 18.2.1, 18.5 to 18.7.1, 21.1.1, 21.1.3, 21.1.4, 24, 25.1.4, 25.2.1, 26.1.6 (Other infrastructure) and 26.1.7 of the Report.

7) I visited the Project on June 3, 2014.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 5.4.1, 5.4.4, 5.4.5, 18.1, 18.2.1, 18.5 to 18.7.1, 21.1.1, 21.1.3, 21.1.4, 24, 25.1.4, 25.2.1, 26.1.6 (Other infrastructure) and 26.1.7 of the Report in compliance with NI 43-101 and Form 43 - 101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Glen Schlyter"

Glen Schlyter, P.Eng.
Engineering Manager
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, Martin Menard, do hereby certify that:

1) I am a consulting engineer acting as Electrical Manager for G Mining Services Inc. (“GMS”) with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2.

2) I have a M.Sc (Economics, Finance and Management-Honours) from the Universitat Pompeu Fabra, Spain in 2007 and a B.Eng. (Electrical Engineering) from McGill University, Canada in 2005.

3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, licence no. 139050.

4) I have practiced my profession continuously since 2008. I have over 8 years experience in power engineering and systems control. Prior to joining GMS, I worked as Senior Electrical Engineer and Electrical Project Manager on various mining / construction projects (Merian Gold Project-Newmont Gold Corporation, Cerro Negro Project-Andean Resources Inc, Essakane Gold Project-Iamgold Corporation, Omai Bauxite-Cambior Inc.) as well as work on other pre-feasibility and feasibility studies. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for Section 5.4.2, 5.4.3, 18.3, 18.4, 18.7.3, 21.1.3.1. (Capex – Power plant), 21.2.3 (OPEX – Power cost) and 26.1.6 (Power) of the Report.

7) I visited the Project on June 3, 2014.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 5.4.2, 5.4.3, 18.3, 18.4, 18.7.3, 21.1.3.1. (Capex – Power plant), 21.2.3 (OPEX – Power cost) and 26.1.6 (Power) of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Martin Menard"

Martin Menard, M.Sc., P.Eng.
Electrical Engineering Manager
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. ("GGM"), Centerra Gold Inc. ("Centerra") and Premier Gold Mines Limited ("Premier") dated December 21, 2016, with an effective date of October 1, 2016 (the "Report").

I, Réjean Sirois, do hereby certify that:

1) I am a Geological Engineer acting as Vice-President Geology and Resources for G Mining Services Inc. ("GMS") with an office at D-200, 7900 Taschereau Blvd, Brossard, Québec, J4X 1C2.

2) I am a graduate of l'Université du Québec à Chicoutimi with a B.Sc. (Geological Engineering) in 1983.

3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, licence no. 38754, the Canadian Institute of Mining, Metallurgy and Petroleum ("CIM") and the Prospectors & Developers Association of Canada ("PDAC").

4) I have worked as a geological engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Report is that I have practiced my profession continuously since 1985 and have extensive experience in estimating mineral resources for various types of mineral deposits located in South and North America as well as in Southern and West Africa. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for the preparation of Sections 4.1, 4.5, 5.1.1, 5.3, 5.5, 6.1, 7.1 to 7.4, 8.1, 9.1, 9.2, 10.1, 11.1, 12.1, 14.1, 23, 25.1.1 and 26.1.1 (All generally pertaining to the Hardrock deposit) of the Report.

7) I visited the Project from August 1 to August 4, 2016.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 4.1, 4.5, 5.1.1, 5.3, 5.5, 6.1, 7.1 to 7.4, 8.1, 9.1, 9.2, 10.1, 11.1, 12.1, 14.1, 23, 25.1.1 and 26.1.1 (All generally pertaining to the Hardrock deposit) of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Réjean Sirois"

Réjean Sirois, P.Eng.
Vice-President, Geology and Resources
G Mining Services Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, Charley Murahwi, do hereby certify that:

1) I am employed as a Senior Geologist by and carried out this assignment for Micon International Limited, Suite 900, 390 Bay Street, Toronto, Ontario, M5H 2Y2.

2) I am a graduate of University of Rhodesia, Zimbabwe, with a B.Sc. (Geology) in 1979. Diplome d´Ingénieur Expert en Techniques Minières, Nancy, France, 1987. M.Sc. (Economic Geology), Rhodes University, South Africa, 1996.

3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of Ontario, membership no. 1618 and PEGNL, membership no. 05662, a registered Professional Natural Scientist with the South African Council for Natural Scientific Professions, membership no. 400133/09 and I am also a Fellow of the Australasian Institute of Mining & Metallurgy (FAusIMM), membership no. 300395.

4) I have worked as a mining and exploration geologist in the minerals industry for over 30 years. My work experience includes 14 years on gold, silver, copper, tin and tantalite projects (on and off mine), 12 years on Cr-Ni-Cu-PGE deposits in layered intrusions/komatiitic environments and 6 years in the consultancy business. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for Sections 4.2 to 4.5, 5.1.2, 6.2 to 6.4, 7.5, 8.2, 9.3, 9.4, 10.2, 11.2, 12.2, 14.2, 25.1.1 (All generally pertaining to the Brookbank, Key Lake and Kailey deposits) and 26.2 of the Report.

7) I visited the Project from November 8 to 9, 2011 and on March 19, 2013.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 4.2 to 4.5, 5.1.2, 6.2 to 6.4, 7.5, 8.2, 9.3, 9.4, 10.2, 11.2, 12.2, 14.2, 25.1.1 (All generally pertaining to the Brookbank, Key Lake and Kailey deposits) and 26.2 of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016

(signed and sealed) "Charley Murahwi"

Charley Murahwi, P.Geo., M.Sc.
Senior Geologist
Micon International

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, Marc Rougier, do hereby certify that:

1) I am a Senior Geotechnical Engineer and Principal in the Mine Stability Division of Golder Associates Ltd. (“Golder”), with an office at 6925 Century Avenue, Suite #100, Mississauga, Ontario, Canada L5N 7K2.
2) I am a graduate of Queen’s University, Kingston, Ontario with a B.Sc. in Geological Engineering, in 1991.
3) I am a Professional Engineer registered with Professional Engineers Ontario, licence no. 90423880 and the Ordre des ingénieurs du Québec, licence no. 5055618.
4) I have practiced my profession continuously since 1992. I have over 24 years’ experience in geotechnical engineering for open pits. Prior to joining Golder, I worked for Piteau Associates Engineering Ltd. I have had no prior involvement with the property which is the subject of this Report.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am responsible for Section 16 (Open pit geotechnical / slopes) of the Report.
7) I visited the Project from September 10 to September 11 2013.
8) I am independent of the issuers: GGM, Centerra and Premier.
9) I have read NI 43-101 and Form 43-101F1 and have prepared Section 16 (Open pit geotechnical / slopes) of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Marc Rougier"

Marc Rougier, P.Eng.
Principal, Mine Stability, Golder Mining
Golder Associates Ltd.

SOUTEX

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. ("GGM"), Centerra Gold Inc. ("Centerra") and Premier Gold Mines Limited ("Premier") dated December 21, 2016, with an effective date of October 1, 2016 (the "Report").

I, Pierre Roy, do hereby certify that:

1) I am a Professional Engineer acting as Senior Metallurgist-Mineral Processing Specialist for Soutex Inc. ("Soutex") with an office at 357 Jackson Street, Quebec, Quebec, G1N 4C4.
2) I am a graduate of Laval University with a B.Sc. in Mining Engineering in 1986 and M.Sc. in extractive metallurgy in 1989
3) I am an Engineer registered with the Ordre des ingénieurs du Québec, licence no. 45201 and a Professional Engineer registered with the Professional Engineers Ontario, licence no. 100110987.
4) I have practiced my profession continuously since 1988. I have over 28 years' experience in Metallurgical Engineering. I have been involved in mineral processing since my graduation from University working in Canada. My experience is principally in ore processing and in environment management for the gold, base metal and iron ore industry. Prior to joining Soutex in 2005, I worked for Inmet Mining Corporation at Troilus mine for 9 (nine) years as Metallurgist and Environmental Director, Placer Dome at Kiena Mine for 6 (six) years and at CRM (Centre de recherche minérale du Québec) for 2 (two) years. I have had no prior involvement with the property which is the subject of this Report.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am responsible for Sections 13, 17.1 to 17.5, 18.2.5, 21.2.2, 25.1.3 and 26.1.5 of the Report.
7) I visited the Project from July 21 to 22, 2014.
8) I am independent of the issuers: GGM, Centerra and Premier.
9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 13, 17.1 to 17.5, 18.2.5, 21.2.2, 25.1.3 and 26.1.5 of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Pierre Roy"

Pierre Roy, P.Eng., M.Sc.
Senior Metallurgist-Mineral Processing Specialist
Soutex

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. ("GGM"), Centerra Gold Inc. ("Centerra") and Premier Gold Mines Limited ("Premier") dated December 21, 2016, with an effective date of October 1, 2016 (the "Report").

I, Eric Poirier, do hereby certify that:

1) I am a consulting engineer with WSP Canada Inc. ("WSP"), 1075, 3rd Avenue, Val-d'Or, Quebec, J9P 0J7.
2) I graduated with a B.Sc. in Electrical Engineering and Computer Science Engineering from Université du Québec à Chicoutimi (Chicoutimi, Québec) in 1996 and 1997.
3) I am a Professional Engineer registered with Professional Engineers Ontario, licence no. 100112909 and the Ordre des ingénieurs du Québec, licence no. 120063. I am also a member of the Association of Professional Engineers and Geoscientists of the Province of Manitoba (APEGM), licence no. 33233 and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (NAPEG), licence no. L2229.
4) I have worked as an electrical engineer and project manager for a total of 18 years since graduating from university. My technical expertise includes electrical distribution, cost estimation, automation and instrumentation. I have been involved in many scoping studies and feasibility studies. I have participated in worldwide projects as electrical designer or as multidisciplinary project manager. I have been a consulting engineer for WSP since January 1998. I have had no prior involvement with the property which is the subject of this Report.
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am responsible for Sections 17.6, 17.7, 21.1.2 and 21.1.3.1 (CAPEX – Process Plant) of the Report.
7) I visited the Project on June 3, 2014.
8) I am independent of the issuers: GGM, Centerra and Premier.
9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 17.6, 17.7, 21.1.2 and 21.1.3.1 (CAPEX – Process Plant of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Eric Poirier"

Eric Poirier, P.Eng.
Director – Electricity and Control
WSP Canada Inc.

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, David Ritchie, do hereby certify that:

1) I am an Engineer acting as Principal Geotechnical Engineer for Amec Foster Wheeler with an office at 160 Traders Blvd., Suite 110 Mississauga, Ontario, L4Z 3K7.
2) I am a graduate of Ryerson Polytechnic University with a B.Eng. (Civil Engineering) in 1995 and The University of Western Ontario with a M.Eng. (Geotechnical Engineering) in 2000.
3) I am a Professional Engineer registered with Professional Engineers Ontario licence no 90488198.
4) I have practiced my profession continuously since 1995. I have over 21 years of experience with the planning and design of dams and tailings management facilities. Prior to joining Amec Foster Wheeler, I worked for 14 years at Golder Associates Limited. I have been involved with geotechnical subsurface investigations and dam design at the Hardrock Project since 2014.
5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I am responsible for Sections 18.2.4 and 26.1.4 of the Report.
7) I visited the Project from July 2 to July 3, 2014.
8) I am independent of the issuers: GGM, Centerra and Premier.
9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 18.2.4 and 26.1.4 of the Report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "David Ritchie"

David Ritchie, M.Eng, P. Eng.
Principal Geotechnical Engineer
Amec Foster Wheeler

Stantec

CERTIFICATE OF QUALIFIED PERSON

To accompany the report entitled:

NI 43-101 Technical Report - Hardrock Project, Ontario, Canada prepared for Greenstone Gold Mines GP Inc. (“GGM”), Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”) dated December 21, 2016, with an effective date of October 1, 2016 (the “Report”).

I, Craig Johnston, do hereby certify that:

1) I am a Professional Geoscientist acting as Senior Principal - Mining Sector Leader for Stantec Inc. (“Stantec”) with an office at 100-300 Hagey Boulevard, Waterloo, Ontario, N2L 0A4.

2) I am a graduate of McMaster University with an Honours B.Sc. in Geology in 1990 and of the University of Waterloo with an M.Sc. in Earth Sciences (Hydrogeology) in 1994.

3) I am a Professional Geoscientist registered with the Association of Professional Geoscientists of Ontario (P.Geo), licence no. 538.

4) I have practiced my profession continuously since 1994 and have worked for Stantec over this entire time. I have over 26 years’ experience in geology and hydrogeology and assessing environmental effects related to project developments. I have had no prior involvement with the property which is the subject of this Report.

5) I have read the definition of “qualified person” set out in the National Instrument 43-101 (“NI 43-101”) and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.

6) I am responsible for Sections 5.2, 18.2.2, 18.2.3, 20, 25.1.5 and 26.1.3 of the Report.

7) I visited the Project from July 21 to July 22, 2015.

8) I am independent of the issuers: GGM, Centerra and Premier.

9) I have read NI 43-101 and Form 43-101F1 and have prepared Sections 5.2, 18.2.2, 18.2.3, 20, 25.1.5 and 26.1.3 of the report in compliance with NI 43-101 and Form 43-101F1; and as of October 1, 2016 (the effective date of the Report), to the best of my knowledge, information and belief, the Report contains all scientific and technical information that is required to be disclosed to make the Report not misleading.

Dated this 21st day of December, 2016,

(signed and sealed) "Craig Johnston"

Craig Johnston, M.Sc., P.Geo.
Senior Principal - Mining Sector Leader
Environmental Services Canada
Stantec Inc.

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1. SUMMARY

1.1 Introduction

G Mining Services Inc. (“GMS”) and a team of engineering consultants were retained by Greenstone Gold Mines GP Inc. acting as the managing partner of Greenstone Gold Mines LP (collectively, “GGM”) to prepare a feasibility study (“FS”) and National Instrument 43-101 (“NI 43-101”) technical report (“Report”) for the Hardrock Project (the “Project”) located near Geraldton, Ontario. The objective of this Report and FS is the evaluation of the technical feasibility and economic viability of the development of an open pit mine, processing facilities and related infrastructures. This Report was prepared for GGM and the two partners of GGM, Centerra Gold Inc. (“Centerra”) and Premier Gold Mines Limited (“Premier”).

The Report and FS responsibilities of the engineering consultants are as follows:

- G Mining Services Inc. - overall Report and FS coordination, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, Mineral Resource estimates, Mineral Reserves (pertaining to the Hardrock deposit), mining methods, economic analysis, operating costs pertaining to mining, infrastructure and power capital cost estimate and project execution plan;
- Micon International Limited (“Micon”) - property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification and Mineral Resource estimates (pertaining to the Brookbank, Key Lake and Kailey deposits);
- Stantec Consulting Limited (“Stantec”) - climate and physiology, site water balance (effluent and site runoff), environmental, permitting and social aspects;
- WSP Canada Inc. (“WSP”) and Soutex Inc. (“Soutex”) - metallurgical testing, recovery methods, and mineral processing plant capital and operating cost;
- TBT Engineering Limited (“TBTE”) - re-alignment of Trans-Canada Highway 11 through the Project area of influence, Ministry of Transportation - Ontario (“MTO”) patrol station relocation;
- Amec Foster Wheeler Environment and Infrastructure, a Division of Amec Foster Wheeler Americas Limited (“Amec”) - tailings management, and geotechnical engineering for the Project infrastructure, including the waste rock storage areas, the mineral processing facility and the tailings management facility (“TMF”);
- Golder Associates Ltd. (“Golder”) - rock mechanics and open pit geotechnical studies.

The core aspects of this Report and FS have been subject to independent peer reviews by various industry-recognized third party consultants. Peer reviews have been completed on geology and Mineral Resources, mining and Mineral Reserves, metallurgy and process, geotechnical studies pertaining to surface infrastructure, pit slopes, tailings, waste rock geochemistry and capital expenditures ("CAPEX").

1.2 Property Description and Location

GGM's Greenstone Gold Property, formerly known as the Trans-Canada Property (the "Property") is located approximately 275 km northeast of Thunder Bay, Ontario. The Property consists of four claim groups, Hardrock, Brookbank, Key Lake and Viper, over a distance of more than 100 km, located along, or in close proximity to, the Trans-Canada Highway between the towns of Beardmore and Longlac, Ontario.

The Hardrock claim group includes the Hardrock and Kailey deposit. The Brookbank claim group hosts the Brookbank, Cherbourg and Fox Ear deposits and the Irwin prospect. The Key Lake claim group hosts the past producing Jellicoe mine and the Viper claim group hosts exploration prospects.

The Hardrock Project and deposit, the subject of this Report, is located approximately at Latitude 49°40'N and Longitude 86°56'W in the townships of Lindsley, Errington, Salsberg, McKelvie and Ashmore, and four kilometres south of the town of Geraldton.

1.3 Land Tenure

The Property and Project are held by Greenstone Gold Mines GP Inc., on behalf of the Greenstone Gold Mines LP (the "Partnership"), a 50/50 partnership between Centerra and Premier. As of the date of this Report, the Project consists of a contiguous block of patented claims, mining leases, licences of occupation and staked claims covering a total area of 14,676 hectares ("ha"). All of the claims, leases and licences of occupation are beneficially held by GGM on behalf of the Partnership and are subject to terms under a number of agreements.

1.4 Climate

The Project is located in northern Ontario, which has a continental climate which is typical for temperate regions in the mid-latitudes that are influenced by both polar and tropical air masses. In this climate, seasonal temperature variations are represented by short summers and cold winters. The mean daily temperature is 3.9°C, with annual maximum of 37°C and a minimum of -50.2°C. The mean annual rainfall is 546.4 mm and the mean annual snowfall of 244.5 cm. The annual average wind speed for the area is

11.2 km/h.

1.5 Infrastructure

The Project occurs in a district with active mines and processing facilities located at Hemlo, Thunder Bay, Kapuskasing and Timmins, Ontario and therefore has access to good transportation and regional mining related infrastructure. The Project is located in close proximity to the Trans-Canada Highway 11, TransCanada PipeLines Limited Canadian Mainline (“TCPL Mainline”) natural gas pipeline, a Hydro One electrical substation and a full service regional airport located 12 km north of Geraldton. Geraldton has its own potable water treatment system and water distribution network, which are proposed to be used for the Project.

GGM has established two offices in Geraldton; a field office in Geraldton for core logging, cutting and storage and a second office for public relations.

1.6 History

Gold in the Property area was first discovered south of the Main Narrows of Kenogamisis Lake between 1916 and 1918. This was followed by a number of discoveries in the 1930s, including the Little Long Lac, MacLeod-Cockshutt, Hard Rock and Mosher mines. The Hardrock deposit was mined by the former Hard Rock, MacLeod-Cockshutt and Mosher mines between 1938 and 1970 and produced over 2 Moz of gold at an average grade of approximately 0.14 ounces of gold per ton (4.8 g Au/t).

In the 1980s, Lac Minerals Ltd. (“Lac Minerals”, now Barrick Gold Corporation) undertook studies on the existing underground mineralized zones at the MacLeod-Cockshutt and Hard Rock mines and carried out litho-geochemical sampling, ground geophysical surveys and a diamond drill hole program. Targets, especially those with open pit potential, were investigated. In 1992, Asarco Exploration Company of Canada Limited and then Cyprus Canada Inc. entered into a five-year earn-in agreement with Lac Minerals. Reverse circulation (“RC”) and diamond drilling was carried out, resulting in a historical resource estimate for a number of pit areas and the discovery of a new zone. In the 2000s, Golder carried out stability assessment of the crown pillars at the mines.

Premier acquired the Property in 2008 and carried out drilling in various areas of the Project. This work resulted in a Mineral Resource estimate that was reported in 2010 in a NI 43-101 technical report. Since 2010, Mineral Resources have been updated annually based on further drilling by Premier. In March 2014, a preliminary economic analysis (“PEA”) was prepared for the Project.

On March 9, 2015, Centerra and Premier announced the formation of the Partnership to explore and develop the Greenstone Gold Property, including the Hardrock Project.

1.7 Geology

The Hardrock Project lies within the southern sedimentary unit of the Beardmore-Geraldton Greenstone Belt (“BGB") along the margin between the granite-greenstone Wabigoon Subprovince and the metasedimentary Quetico Subprovince. The Hardrock deposit is characterized by multiple horizons of magnetite-rich chert banded iron formation ("BIF") within a thick sequence of interlayered sandstone-argillite and minor polymictic conglomerate. The sequence is intruded by medium-to coarse-grained diorite sills and feldspar-quartz porphyry dykes, which, together with the sedimentary rocks, are folded by tight to isoclinal, regional F2 folds.

1.8 Deposit Types, Exploration and Drilling

Gold deposits in the Hardrock deposit area are classic examples of epigenetic non-stratiform BIF-hosted gold deposits (historical North Zone and West Zone). Gold mineralization has resulted from the introduction of hydrothermal fluids in zones of high crustal permeability. Most mineralized occurrences in the Hardrock deposit area lie in a zone of deformation to the immediate north of, and genetically linked to, the Tombill-Bankfield Deformation Zone. Numerous Z-folds on various scales were formed in the deformation zone. Auriferous vein systems in the MacLeod-Cockshutt and Hard Rock mines are hosted by one of the Z-folds. This structure plunges shallowly west and is mimicked by minor parasitic folds in the BIF. The mineralization is found in upright sub-vertical axial planes that trend roughly east-west. The fold axes are shallowly west-plunging.

Two main styles of mineralization at the Hardrock deposit are quartz-carbonate stringer mineralization and sulfide replacement mineralization. Quartz-carbonate stringer mineralization generally consists of a series of narrow, tightly asymmetrically folded gold-bearing quartz-carbonate stringers, which are usually attenuated, transposed and dislocated in hook-like segments. The stringers are accompanied by a gold-bearing quartz-sericite-pyrite (±arsenopyrite) alteration halo about the stringers. Sulfide replacement mineralization occurs as variable pyrite, arsenopyrite and pyrrhotite replacement of Fe oxide within the hinge zones of folded BIFs. The auriferous sulfide replacement appears to have migrated outwards along the iron oxide bands from gold-bearing quartz-carbonate stringers occupying brittle axial planar tension fractures; this replacement mineralization yields grades of 7 g Au/t or greater.

Since June 1, 2014, GGM and its predecessor has removed soils and vegetation to expose rocks in the 2016 resource area. The work consisted of three outcrops with detailed geological mapping and channel sampling. The purpose of this work was to verify and establish structural elements and grade continuity at surface. During 2016, GGM conducted induced polarization ("IP") surveys in the 2016 resource area and locally in the Hardrock claim block over past producing mines and known mineralized zones.

Between May 26, 2014 and November 18, 2015, GGM and its predecessor added 157 diamond drill holes on the Hardrock deposit for a total of 54,027 m. Seventy-nine historical diamond drill holes were re-sampled to add new assay results in the 2016 updated Mineral Resource estimate. These holes represent a total of 8,733 m of new footage and 6,411 of new samples in the 2016 database.

The sample preparation, analysis, quality assurance and quality control (“QA/QC”) and security protocols used for the Project follow generally accepted industry standards and the data is valid and of sufficient quality to be used for Mineral Resource estimation.

1.9 Mineral Resource Estimate

1.9.1 Hardrock Project

Definitions for Mineral Resource categories used in this Report are consistent with the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014 (“CIM definitions”) and adopted by NI 43-101. GMS is not aware of any environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate.

This Report is based on an open pit mining scenario of the Project. The in-pit Mineral Resources at the Hardrock deposit are constrained within the design pit using a cut-off grade of 0.30 g Au/t. In addition to in-pit Mineral Resources, an underground Mineral Resource was estimated outside the open pit using a 2.0 g Au/t cut-off grade. The Project open pit and underground Mineral Resources are summarized in Table 1.1.

The Mineral Resource estimate covers a corridor of the Hardrock deposit with a strike length of 5.7 km and a width of approximately 1.7 km, down to a vertical depth of 1.8 km below surface. Mineralized zones were interpreted in 3D using GEMS and Paradigm GOCAD software based on a litho-structural model and the drill hole database. The drill hole database used in the estimate contained 304,940 sampled intervals from 684,116 m of diamond drilling in 1,629 holes and 1,219 assays from 26 channel samples.

Mineral Resources were estimated by applying a minimum true thickness of 3.0 m and using the grade of the adjacent material when assayed or a value of zero when not assayed. High-grade capping on raw assay data was established on a per zone basis and ranged from 15 to 45 g Au/t. Compositing was conducted on drill hole sections falling within the mineralized zones (composite = 1.5 m). Mineral Resources were estimated using 3D block modelling and 3-pass ID^3 interpolation.

Mineral Resources were classified as Indicated only in areas where the maximum distance to drill hole composites was less than 35 m for blocks interpolated in passes 1 and 2 (using a minimum of two drill holes). Mineral Resources were classified as Inferred in areas where blocks were interpolated during passes 1 to 3 and isolated blocks were reclassified as “exploration potential” on a visual basis. No Measured Mineral Resources were estimated for the Project.

Table 1.1: Mineral Resource Estimate (Exclusive of Mineral Reserve) for the Hardrock Deposit

Resource Type		In-Pit	Underground	Total
	Cut-off (g Au/t)	> 0.30 g Au/t	> 2.00 g Au/t	
Indicated	Tonnes (t)	11,444,000	13,692,000	**25,136,000**
	Grade (g Au/t)	0.36	3.91	**2.29**
	Au (oz)	131,200	1,719,900	**1,851,100**
Inferred	Tonnes (t)	170,000	21,507,000	**21,677,000**
	Grade (g Au/t)	0.87	3.57	**3.55**
	Au (oz)	4,800	2,470,400	**2,475,200**

Notes:

1. *2010 CIM definitions were followed for Mineral Resources.*
2. *The effective date of the estimate is August 11, 2016.*
3. *Mineral Resources are exclusive of Mineral Reserves.*
4. *Density data was established on a per zone basis and ranges from 2.72 to 3.26 g/cm^3.*
5. *In-pit Mineral Resources are estimated within the Pit Design shell. Parameters included (all amounts in Canadian dollars): reference mining cost: $1.80/t, incremental bench cost ($/10 m bench): $0.030/t, milling cost: $7.46/t, royalty: 3%, G&A: $1.42/t, rehandling: $0.12/t, sustaining capital: $0.60/t, gold price: $1,625/oz, milling recovery: 90%, pit slope: 55°. Density data was established on a per zone basis and ranges from 2.72 to 3.26 g/cm^3.*
6. *Mineral resources that are not Mineral Reserves do not have demonstrated economic viability.*
7. *Numbers may not add due to rounding.*
8. *Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the inferred resources will ever be converted to a higher category. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*

1.9.2 Other Greenstone Gold Property Deposits

In addition to the Hardrock deposit, Mineral Resources were estimated for the Brookbank, Key Lake and Kailey deposits. Open pit optimization using Whittle software, based on the Lerchs-Grossmann algorithm, was completed to estimate in-pit Mineral Resources for all three deposits. For Brookbank and Key Lake, underground Mineral Resources were also estimated. All these Mineral Resources are effective as of December 31, 2012. There are no Mineral Reserves currently estimated for these deposits. Refer to Table 1.2.

Table 1.2: Summary of Brookbank, Key Lake and Kailey Mineral Resources

Deposit	Mining Method	Category	Tonnes (Mt)	Gold Grade (g/t)	Contained Gold (koz)
Brookbank	Open Pit	Indicated	2.638	2.02	171
		Inferred	0.171	2.38	13
	Underground	Indicated	1.851	7.21	429
		Inferred	0.403	4.02	53
Key Lake	Open Pit	Indicated	2.572	1.17	97
		Inferred	1.345	1.29	56
	Underground	Indicated	0.031	6.48	6
		Inferred	0.058	3.57	7
Kailey	Open Pit	Measured and Indicated	8.630	0.95	265
		Inferred	3.688	0.97	115

Notes:

1. *2010 CIM definitions were followed for Mineral Resources.*
2. *The effective date of the estimates is December 31, 2012.*
3. *Open pit Mineral Resources are reported at a cut-off grade of 0.50 g Au/t and underground Mineral Resources are reported at a cut-off grade of 2.8 g Au/t.*
4. *Mineral Resources are estimated using a long-term gold price of USD 1,455 and an exchange rate of CAD/USD 1.18.*
5. *Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*
6. *Numbers may not add due to rounding.*
7. *Inferred Mineral Resources have a great amount of uncertainty as to their existence and as to whether they can be mined economically. It cannot be assumed that all or part of the inferred resources will ever be converted to a higher category. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.*

GMS is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect these Mineral Resource estimates.

1.10 Mineral Reserves

The Mineral Reserve for the Hardrock Project was estimated based on the open pit mining scenario proposed in this Report and is summarized in Table 1.3. The Mineral Reserve estimate was prepared by GMS.

Table 1.3: Mineral Reserve Estimate (Open-Pit)

Category	Diluted Ore Tonnage (kt)	Diluted Grade (g Au/t)	Contained Metal (koz Au)
Proven	-	-	-
Probable	141,715	1.02	4,647
Total P&P	**141,715**	**1.02**	**4,647**

Notes:

1. *CIM definitions were followed for Mineral Reserves.*
2. *Effective date of the estimate is October 1, 2016.*
3. *Mineral Reserves are estimated at a cut-off grade of 0.33 g Au/t.*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30.*
5. *A minimum mining width of 5 m was used.*
6. *Bulk density of ore is variable but averages 2.83 t/m^3.*
7. *The average strip ratio is 3.87:1.*
8. *Mining dilution factor is 17.3%.*
9. *Numbers may not add due to rounding.*

The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources (“M&I”), and do not include any Inferred Mineral Resources. There are only Indicated Mineral Resources and no Measured Mineral Resources. Therefore, all of the Mineral Reserve estimate is classified as Probable. The Inferred Mineral Resources contained within the mine design are classified as waste.

Open pit optimization was conducted using Whittle software to determine the optimal economic shape of the open pit to guide the pit design process. The Mineral Reserve estimate includes a 17.3% mining dilution at an average grade of 0.15 g Au/t and a 1.4% ore loss factor.

A feasibility level pit slope design study was carried out by Golder. The conclusions of this study have been used as an input to the pit optimization and design process.

1.11 Mining

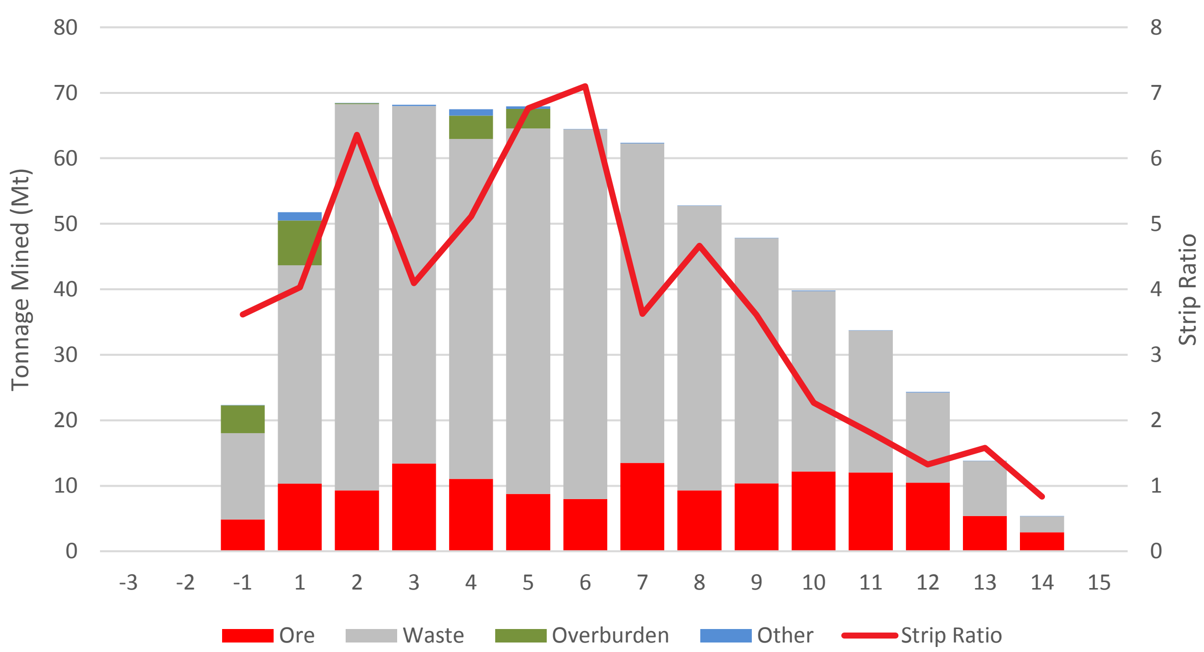
Mining will be carried out using conventional open pit techniques with 10 m benches. An Owner mining open pit operation is planned with hydraulic shovels and mining trucks and includes outsourcing of certain support activities such as explosives manufacturing and blasting.

Production drilling of the 10 m benches will be by blast hole drill rigs with both rotary and down-the-hole ("DTH") drilling capability. Blast holes are loaded with bulk emulsion. The majority of the loading in the pit will be carried out by three hydraulic face shovels, two 26 m^3 and one 19 m^3 and two front-end wheel loaders (21 m^3). The shovels and loaders will be matched with a fleet of 181 t payload mine trucks. The presence of underground stopes was considered when designing the pits mainly for the void in the F-Zone, which is 150 m high and 30 m wide. Most of the other underground openings are backfilled with sand fill or rock fill.

Mining of the Hardrock main pit will occur in four phases (including the borrow pit) with a single phase for the smaller satellite pit to the east. Waste rock will be disposed of in four distinct waste dumps with three located around the pit and one further to the south. The open pit generates 548.9 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) over the life of mine ("LOM") for an average LOM strip ratio of 3.87:1.

The LOM plan (Figure 1.1) details 14.5 years of production, with a four month ramp up and commissioning period, followed by eighteen (18) months at a processing rate of 24,000 t/d ore, increasing to 27,000 t/d ore for the remainder of the mine life.

Figure 1.1: Annual Mine Production



1.12 Mineral Processing and Metallurgical Testing

The process design criteria have been established based on testwork results, GGM and vendor recommendations or requirements and on standard industry practices.

Prior to the start of the FS, between 2011 and 2013, mineralogy, grindability and gold recovery testwork was performed by SGS Lakefield Research Limited (“SGS Lakefield”) and McClelland Laboratories, Inc. (“McClelland”). The SGS Lakefield testwork showed that the ore is composed mainly of quartz and plagioclase with minor amounts of pyrite and arsenopyrite, the gold occurs mainly as native gold, the ore is in the category of medium hardness to moderately hard, a portion of the gold can be recovered by gravity concentration and gold can be recovered to a bulk concentrate. The subsequent McClelland testwork showed that gold recovery increased with finer grind size but was not affected by cyanide concentration.

In the course of the PEA and FS, additional testwork was carried out by SGS Lakefield, JKTech Pty Ltd and FLSmidth. Primarily, high pressure grinding roll ("HPGR") tests were required to confirm the ore amenability for high pressure grinding, to select the equipment and estimate the operating costs. Grindability, head grade determination, mineralogy, magnetic separation, gravity recovery, flotation, cyanidation, cyanide destruction, solid-liquid separation and other tests were completed. Additional thickening and rheology testwork was carried out to determine the sizing and operating parameters of a pre-leach thickener.

The HPGR testing program included laboratory scale tests (batch and locked-cycle tests) to determine the amenability of the ore to HPGR milling and yield data to perform a preliminary sizing; abrasion tests to provide the data necessary to predict the service life of the rolls and a large scale pilot plant test to adequately size the equipment. Bond grindability testing was performed to evaluate the BWI reduction of the HPGR product compared to the feed. A detailed comminution trade-off study recommended two stage crushing followed by HPGR and ball milling over other typical comminution flowsheets such as crushing followed by semi-autogenous (“SAG”) milling and ball milling, to reduce the risk in not meeting the design throughput and increase energy efficiency.

A multivariate linear regression analysis was used to determine the correlation between the residual gold grade and the ore body mineralogical composition. The results of the cyanidation tests conducted on composites were used as the basis for the analysis. The residual gold grade from the cyanidation testwork was found to be highly correlated to the gold, arsenic and sulphur head sample grades, and somewhat less predominantly to grind size.

The gold recovery process for the Project consists of a crushing circuit (gyratory and cone), a grinding circuit (HPGR and ball mill), pre-leach thickening, a leach and carbon in pulp (“CIP”) circuit, cyanide destruction and tailings disposal, carbon elution and electrowinning, carbon regeneration and a gold refinery. The plant is designed to operate at a throughput of 27,000 t/d. The process operation schedule is 24 hours per day, 365 days per year, with an overall availability of 92%.

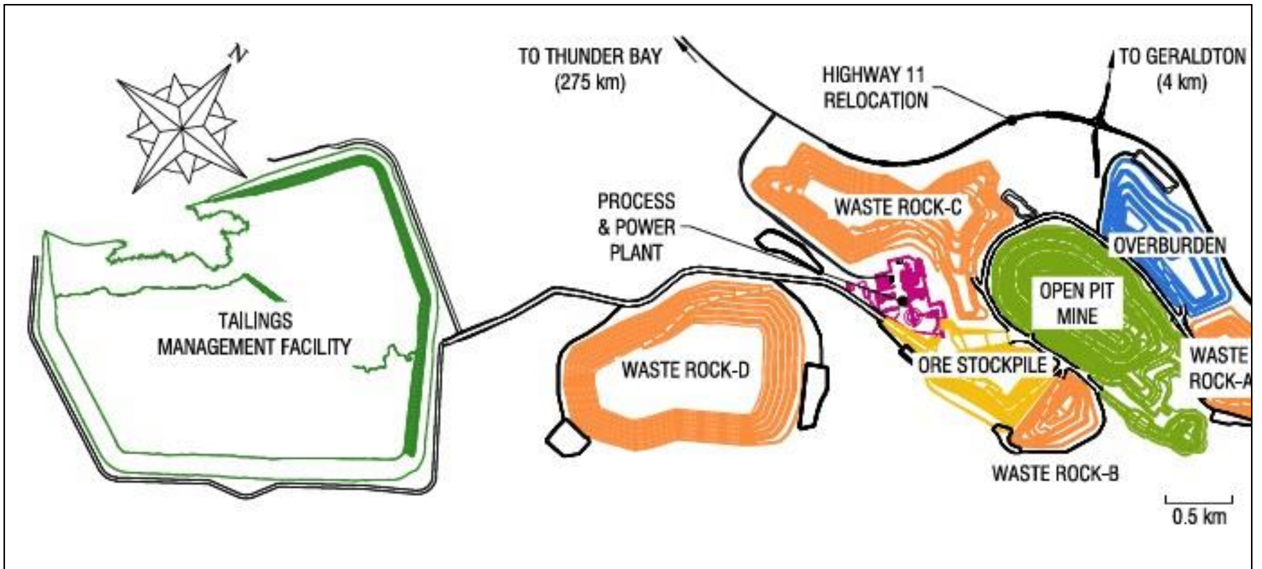
Gold production averages 356 koz for the first four full years of production (Year 2 to 5) with an average head grade of 1.27 g Au/t and an average metallurgical recovery of 90.6%.

1.13 Mine Infrastructure and Services

The Project will require infrastructure to support mining and processing. General infrastructure for the Project will include:

- Site access and haul roads;
- Workshop and maintenance facility;
- Warehousing for spare parts and reagents;
- Administration building including a dry facility, gatehouse and parking area;
- Explosive reagent storage;
- Fuel storage and distribution;
- Recycling and sorting facility;
- Potable water and sewage systems;
- Fire water systems;
- Site security and fencing.

Figure 1.2: Hardrock Site General Arrangement



Title: Hardrock Site General Arrangement	Author: GMS Date: 16/10/01

Existing infrastructure within the footprint of the property limits that will need to be relocated includes:

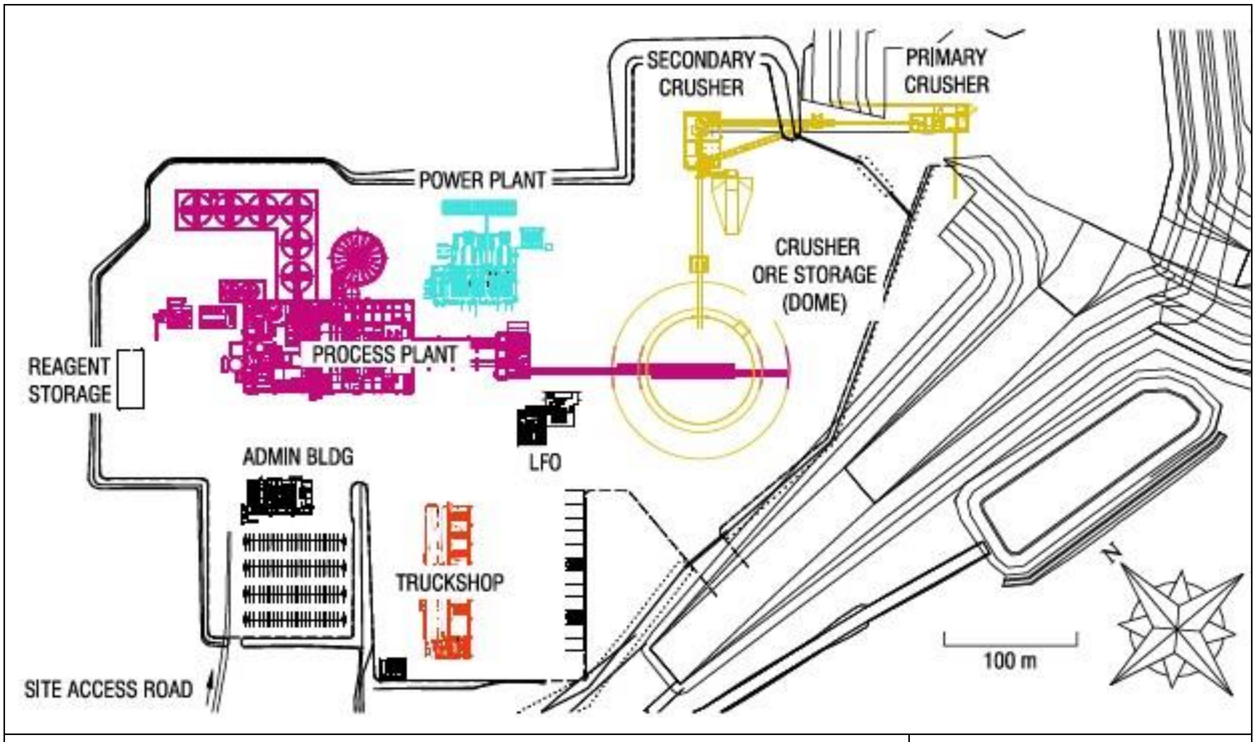
- Trans-Canada Highway 11;
- Existing Hydro One 115 kV transmission station;
- OPP station;
- MacLeod high tailings (portion covering the open pit mine);
- MTO patrol station.

Existing infrastructure within the footprint of the property limits that will need to be purchased and / or dismantled includes:

- Portions of a golf course;
- Gas station;
- MacLeod-Cockshutt (MacLeod-Mosher) mine headframe;
- MacLeod townsite and Hardrock townsite housing.

The existing Hydro One grid is insufficient for powering the processing facilities and associated infrastructure. A 65 MW natural gas-fired power plant will be constructed which will include a natural gas pipeline originating from the existing TCPL Canadian Mainline pipeline directly to the site power plant.

Figure 1.3: Process Plant and Mine Infrastructures



Title: Process Plant and Mine Infrastructures	Author: GMS Date: 16/10/01

1.14 Tailings Management Facility

The TMF dams have been designed to meet the requirements of the Lakes and River Improvement Act Ministry of Natural Resources ("MNR, 2011") and the Canadian Dam Association guidelines ("CDA, 2014") with a relatively low permeability core along with filters and transition zones upstream of the main embankment constructed of geochemically benign mine rock.

The TMF site is located approximately five kilometres southwest of the process plant site and was selected to minimize the disturbance to fish bearing water bodies, maximize the use of natural containment and optimize Project economics. Prior to construction of the TMF, Goldfield Creek will be diverted around the north side of the TMF into a permanent channel designed to provide fisheries compensation.

The site has a positive water balance, and as such, the TMF will be developed initially with only one of two cells capturing runoff to minimize the surplus water requiring treatment. It is planned to complete tailings deposition early in one cell to allow for progressive rehabilitation and shedding of runoff from the system.

Closure of the TMF involves lowering of the spillways and vegetation of the exposed beaches. Runoff will be directed through emergency spillways constructed in natural ground when deemed suitable for discharge to the environment.

1.15 Environmental Studies

Environmental baseline studies were initiated for the Project in 2013 and were used to identify environmental constraints during the development of preliminary layouts and designs for the Project. This includes consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective. This environmental baseline was the basis for determining incremental changes and predicting environmental effects associated with the Project.

A draft environmental impact statement / environmental assessment (“EIS/EA”), which also includes a conceptual closure plan, has been completed and submitted to regulatory agencies, Aboriginal groups and the public for review and comment. Project interactions were analyzed for 13 valued components (“VCs”) to determine potential environmental effects associated with the Project for construction, operation and closure phases. In addition to the VCs, the effects assessment also considered effects of the environment on the Project, accidents and malfunction scenarios and cumulative effects. The draft EIS/EA contained preliminary recommendations for follow-up monitoring and environmental management plans and included measures related to both compliance and EIS/EA monitoring for all phases of the Project.

A conceptual closure plan was developed as part of the EIS/EA to provide an early opportunity to discuss the closure approach and initial costing. The conceptual closure plan includes preliminary details on closure that may be refined following EIS/EA approval through further discussion with regulatory agencies. At the end of mining operations, the main features requiring closure will include the main open pit, water management and drainage systems, waste rock storage areas, TMF, site access roads and buildings and associated infrastructure. After the closure works have been completed, a post-closure monitoring program will be carried out to verify that the closure objectives and criteria have been met and confirm that the Project can proceed to final close out status.

Since completing the draft EIS/EA, GGM has carried out slight modifications of Project components in response to agency comments, which generally form the basis for the final mine plan used for this Report and FS. Additional refinements may be made for the final EIS/EA and during detailed engineering.

Active consultation with stakeholders (community members, agencies and interested parties) and Aboriginal communities has been undertaken throughout Project planning including the preparation of the draft EIS/EA, and will continue as the Project progresses.

GGM continues to work with Aboriginal communities to understand potential effects of the Project on traditional land uses and activities and is committed to working towards Long Term Relationship Agreements ("LTRAs").

1.16 Execution Plan

The Project will be executed using an “Owner-managed” project delivery model. All aspects of engineering, procurement and construction for the Project will be managed directly by the Owner. Detailed engineering and a portion of the procurement will be outsourced. The Project construction period is 23 months and the total pre-production period is estimated at 42 months which includes detailed engineering, procurement, construction and commissioning activities up to commercial production being declared. The peak construction workforce on site is estimated at 650 people.

The operating organization consists of three departments: mine, including mine operations, geology, engineering and maintenance; process and power plant; and general and administrative including human resources, environment, health and safety, site services and accounting. The planned peak total operating workforce is 544 employees (reached in Year 4).

1.17 Capital Cost Estimate

The initial capital cost ("CAPEX") for Project construction, equipment purchases, pre-production activities and other payments is estimated to be CAD 1,247M, as shown in Table 1.4. The CAPEX includes a contingency of CAD 131M, which is 11.8% of the total before contingency.

Table 1.4: Capital Expenditures Summary

Category	Total Costs (M CAD)
Infrastructure	62.6
Power & Electrical	72.4
Water & Tailings Management	79.9
Mobile Equipment	178.1
Infrastructure Repositioning	45.6
Process Plant General	343.1
Construction Indirect Costs	175.4
General Services - Owner's Costs	59.8
Preproduction, Start up, Commissioning	94.1
Contingency	131.3
Other Costs	4.5
Total Capital Cost	**1,246.9**

Sustaining capital is required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation. The sustaining capital is estimated at CAD 257M.

The total salvage value is estimated at CAD 38M, and includes mining equipment purchased during operations that will not have been utilized to its useful life, a residual value for some of the process plant major equipment and a residual value for the power plant as the units will have a remaining useful life of 10 to 15 years at the end of operations.

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance and post closure monitoring. The reclamation cost is funded with cash outflows provisioned in the economic model from Year 3 to Year 14 and spent over three years at the end of operations. The total reclamation and closure cost is estimated to be CAD 54M.

1.18 Operating Cost Estimate

Operating costs ("OPEX") are summarized in Table 1.5. The OPEX includes mining, processing, general and administration ("G&A"), transportation and refining, other costs and royalties. The average OPEX is CAD 705/oz Au or CAD 20.95/t milled over the LOM. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs averages CAD 780/oz Au over the LOM.

Table 1.5: Operating Costs Summary

Category	Total Costs (M CAD)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz Au)
Mining	1,412	10.03	338
Processing	1,061	7.54	254
G&A	205	1.45	49
Transport & Refining	13	0.09	3
Other Costs	56	0.40	13
Royalties	203	1.45	49
Total Operating Cost	**2,950**	**20.95**	**705**
Closure & Reclamation	54	0.38	13
Sustaining Capital	257	1.82	61
All-in Sustaining Cost	**3,261**	**23.16**	**780**

1.19 Economic Analysis

The base case economic model has been developed using a long-term gold price assumption of USD 1,250/oz and an exchange rate of CAD/USD 1.30. The gold price and exchange rates are supported by third party forecasts.

Gold production over the LOM is 4,193 koz based on an average processing recovery of 90.2%. Gold production begins during the pre-production period and is treated as pre-production revenue which partially offsets pre-production costs.

The economic model excludes any Project or equipment financing assumptions. The Project funding is assumed to be through equity for the purposes of the Report. The economic results are calculated as of the start of the pre-production CAPEX phase, which includes detailed engineering and procurement. All prior costs are treated as sunk costs.

The Partnership is not subject to income taxes and the partners, Centerra and Premier, will bear the responsibility for paying tax on profits generated by the Partnership. The post-tax results in this Report are based on the assumption that GGM is a taxable Canadian entity and tax is calculated based on the tax rules in Ontario. The calculations do not reflect the benefit of any historical tax positions held by either Centerra or Premier (if any).

The before-tax Project cash flow over the Project life is estimated at CAD 2,325M. The Project before-tax net present value ("NPV") at a discount rate of 5% is estimated to be CAD 1,095M with a before-tax internal rate of return ("IRR") of 17.9%.

The total after-tax cash flow over the Project life is estimated to be CAD 1,636M. The Project after-tax NPV at a discount rate of 5% is estimated to be CAD 709M. The after-tax Project cash flow results in a 4.5-year payback period from the commencement of commercial operations with an after-tax IRR of 14.4%. Table 1.6 is a summary of the Project economics.

Figure 1.4: Annual Gold Production

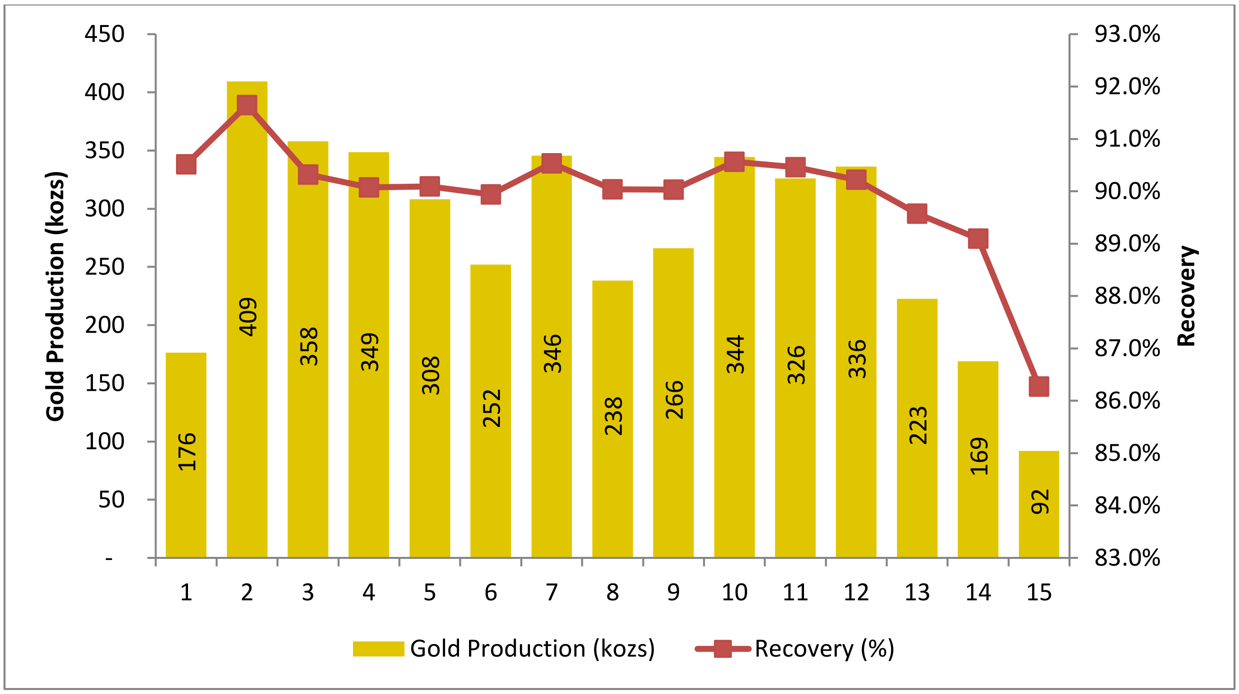


Table 1.6: Project Economics Result Summary

Project Economics		Results
Production Summary		
Tonnage Mined	Mt	691
Ore Processed	Mt	142
Average Head Grade	g Au/t	1.02
Gold Processed / Contained Gold	koz	4,647
Recovery	%	90.2%
Gold Production	koz	4,193
Cash Flow Summary		
Gross Revenue	M CAD	6,795
Mining Costs (including rehandle)	M CAD	(1,412)
Processing Costs	M CAD	(1,061)
G&A Costs	M CAD	(205)
Royalty, Refining and Other Costs	M CAD	(272)
Total Operating Costs	M CAD	(2,950)
Operating Cash Flow Before Taxes	M CAD	3,845
Initial CAPEX	M CAD	(1,247)
Sustaining Capital	M CAD	(257)
Total Capital	M CAD	(1,504)
Salvage Value	M CAD	38
Closure Costs	M CAD	(54)
Taxes (mining, provincial and federal)	M CAD	(690)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M CAD	2,325
NPV 5% Before-Tax	M CAD	1,095
Project Before-Tax Payback Period	Years	3.9
Project Before-Tax IRR	%	17.9%
After-Tax Results		
After-Tax Undiscounted Cash Flow	M CAD	1,636
NPV 5% After-Tax	M CAD	709
Project After-Tax Payback Period	Years	4.5
Project After-Tax IRR	%	14.4%

Table 1.7 is a summary of the Project NPVs at various discount rates

Table 1.7: Project Net Present Values at Various Discount Rates

Discount Rate	Before-Tax Project NPV (M CAD)	After-Tax Project NPV (M CAD)
5%	1,095	709
6%	933	587
7%	791	481
8%	667	387

A sensitivity analysis was performed for ±10% and ±15% variations for gold price, exchange rate, operating costs and initial capital expenditure.

The Project is most sensitive to gold price followed by exchange rate, initial capital costs and finally operating costs. The CAD/USD exchange rate is slightly less sensitive than the gold price in USD/oz as some of the initial capital expenditures are in US dollars. The sensitivity on gold grade is identical to that of the gold price and is therefore not presented in the following figures.

The results of the sensitivity analysis on after-tax undiscounted NPV and IRR are presented in Table 1.8.

Table 1.8: Project After-Tax Sensitivities

	NPV 5%			IRR		
Feasibility Study (FS) Variable	**-15 % (CAD M)**	**FS (CAD M)**	**+15 % (CAD M)**	**-15 % (% IRR)**	**FS (% IRR)**	**+15 % (% IRR)**
Operating Costs	873	709	543	16.3	14.4	12.4
Capital Costs	824	709	590	17.4	14.4	12.1
Exchange Rate (CAD/USD)	314	709	1,093	9.6	14.4	18.5
Gold Price	293	709	1,113	9.2	14.4	19.0

1.20 Interpretation and Conclusions

The completion of this Report and the FS has confirmed the technical feasibility and economic viability of the Project, based on an open pit mining operation with average gold production at 288,000 ounces per year over a 14.5 year LOM.

The principal conclusions by area are detailed below.

- Geology and Mineral Resources
  - Understanding of the Project geology and mineralization, together with the deposit type, is sufficiently well established to support Mineral Resource and Mineral Reserve estimation.
  - Cut-off grades of 0.30 g Au/t for the in-pit resource and 2.00 g Au/t for the underground resource are appropriate for reporting Mineral Resources for the Project.
  - At a cut-off grade of 0.30 g Au/t, the in-pit Indicated Mineral Resources are estimated to be 131.9 Mt grading 1.10 g Au/t for 4.7 Moz of gold. In-pit Inferred Mineral Resources are estimated to be 170 kt grading 0.87 g Au/t for 4.8 koz of gold.
  - At a cut-off grade of 2.00 g Au/t, the underground Indicated Mineral Resources are estimated to be 13.7 Mt grading 3.91 g Au/t for 1.7 Moz of gold. Underground Inferred Mineral Resources are estimated to be 21.5 Mt grading 3.57 g Au/t for 2.5 Moz of gold.
  - Definitions for Mineral Resource categories used in this Report are consistent with the CIM definitions and adopted by NI 43-101.
- Mining and Mineral Reserves
  - The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies.
  - At a cut-off grade of 0.33 g Au/t, Probable Mineral Reserves are estimated to be 141.7 Mt with an average grade of 1.02 g Au/t for 4.65 Moz of gold.
  - The LOM plan details 14.5 years of production, with a four month ramp up and commissioning period followed by eighteen (18) months at a throughput rate of 24,000 t/d, increasing to 27,000 t/d for the remainder of the mine life.
  - The open pit generates 548.9 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) for a strip ratio of 3.87:1.
  - The Mineral Reserve estimate stated herein is consistent with CIM definitions. The Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources.
- Metallurgical Testing and Mineral Processing
  - The process design criteria have been established based on testwork results, Owner and vendor recommendations or requirements and on standard industry practices.

    - The processing options for the Project were selected based on the results of this testwork and are well known technologies that are currently used in the mining industry.
    - The gold recovery process for the Project consists of a crushing circuit, a grinding circuit (HPGR and ball mill), pre-leach thickening, a leach and CIP circuit, cyanide destruction and tailings disposal, carbon elution and electrowinning, carbon regeneration, and a gold refinery. The process plant is designed to operate at a throughput of 27,000 t/d.
    - Overall metallurgical recovery is 90.2%.
- Infrastructure
    - Existing infrastructure within the footprint of the property limits will need to be relocated or purchased and dismantled. The most significant relocation is that of the TransCanada Highway 11.
    - Power availability from the existing grid is deemed insufficient. Construction of a natural gas-fired power plant is planned.
- Environmental Considerations
    - A draft EIS/EA, which also included a conceptual closure plan, has been completed and submitted to regulatory agencies, Aboriginal groups and the public for review and comment.
    - The results of the draft EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects.
    - There are no issues identified to date that would materially affect the ability of GGM to extract minerals from the Project; however, Agency comments on the draft EIS/EA received to date and potential future conditions of approval could require refinements to Project components or additional mitigation measures to be implemented.
    - GGM continues to work with Aboriginal communities to understand potential effects of the Project on traditional land uses and activities and is committed to working towards LTRAs.
- Capital and Operating Costs
    - The estimate was developed according to AACE International Standards for a Level 3 estimate with a target accuracy of ± 15%.
    - The initial CAPEX for Project construction, including processing, mine equipment purchases and pre-production activities, infrastructures and other direct and indirect costs is estimated to be CAD 1,242M. The total initial capital includes a contingency of CAD 131M, which is 11.8%

of the total CAPEX. Other costs during the construction period of CAD 5M bring the total initial capital to CAD 1,247M.

- Sustaining capital required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation is estimated at CAD 257M.
- A salvage value of CAD 38M is estimated for some mining equipment, processing equipment and power plant that will not have been utilized to their useful life.
- The total reclamation and closure cost is estimated to be CAD 54M.
- The average operating cost is CAD 705/oz Au or CAD 20.95 per tonne milled over the life of the mine. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs average CAD 780/oz Au over the mine life.

1.21 Risks and Opportunities

GGM's risk identification and assessment process is iterative and has been applied throughout the FS phase. Risks are identified in relation to Project objectives and the internal and external context at the time of each assessment, and are summarized into the Hardrock Project risk register. All aspects of the Project (technical, environmental, community, financial, health and safety, etc.) are assessed in order to provide a business or enterprise level perspective.

Risk treatment plans are developed for each risk in order to reduce the risk's probability and/ or impact to an acceptable or practical level. Certain risk mitigation activities were completed as planned during the FS Phase, while other actions are planned for detailed engineering, construction, operations or closure as appropriate. These mitigation plans are incorporated in the project execution plans and where required in the CAPEX and OPEX budgets. The key risks areas that are being managed through current controls and future phase mitigation plans include stability of historic tailings, tailings management facility, relocation of Highway 11, water management, project execution (people and systems), environmental assessment (EA) and permits, process plant ramp-up and mining of voids.

There are several opportunities to improve overall Project economics and sustainability.

- Revenue related potential opportunities:
  - The use of the Hardrock process plant and TMF for the future processing of gold from other GGM Property deposits such as Brookbank, or a potential future Hardrock underground resource to improve the LOM average grade.
  - Extend the LOM by the addition of potential newly defined resources / reserves from the Property and any marginal-grade Hardrock material stockpiled during the LOM.
  - The use of the Hardrock process plant and TMF to process some portion of the existing surface historic tailings in order to recover gold, generate revenue, and also potentially mitigate environmental liabilities related to sulphides, arsenic and other contaminants.
  - Connecting the natural gas power plant to the grid, and selling spare power generation to the grid during times of shutdowns or excess capacity.
- OPEX related potential opportunities:
  - A potential blend of liquid natural gas ("LNG") and diesel as a fuel source is possible for the mine haul trucks. Currently, the mine fleet uses 100% diesel.
  - The use of new, commercially available technologies such as automated mine haulage equipment to increase operational efficiencies and reduce OPEX.
  - Use of RC drilling and other studies early in the Project to provide a better understanding of deposit continuity resulting in better control of dilution, reducing the amount of waste processed and therefore improving OPEX.
- CAPEX related potential opportunities:
  - Obtain unused or high quality / refurbished used equipment for the process plant.
  - Consideration of site construction labour efficiencies through the use of pre-fabricated or modular structures, equipment packages and concrete foundations.
  - Consider the use of alternative lower cost sources for materials and equipment for the mine, processing and infrastructures development.
  - Consider the possibility of major equipment vendor / manufacturer financing or leasing arrangements that serve to improve Project economics.

1.22 Recommendations

Given the technical feasibility and positive economic results of the FS, GMS recommends that GGM continue the work necessary to support a decision to fund and develop the Project.

GGM plans the following principal tasks in the next phase of development:

- Completing the financing plan to fund the construction period;
- Continuing stakeholder engagement activities to establish LTRAs;
- Submission and approval of a EIS/EA;
- Securing all required environmental and construction permits; and
- Managing and mitigating key risks and pursuing opportunities to improve project economics.

The cost for this phase of the work are estimated at approximately CAD 12M.

The list of specific recommendations that follow applies to this and successive phases of work. The cost of addressing each of these recommendations have not been individually estimated however are generally considered to be within the scope of Project CAPEX, sustaining capital, closure and OPEX outlined in this Report.

1.22.1 Exploration and Geology:

- Use a second laboratory as an independent review on 5 to 10% of its pulps in future sampling programs;
- Refine the contaminant models of arsenic and sulfur which are used to modulate the expected metallurgical gold recovery.

1.22.2 Open-Pit Mining

- Conduct additional pit slope geotechnical work such as detailed review of variation in structural fabric orientation to identify possible localized sub-domains with stronger controls on achievable bench face angles; and conduct sensitivity analyses on slope saturation and lower effective shear strength. Additional laboratory testing such triaxial testing and intact shear strength of foliation is recommended;
- A runout assessment study using specialized software is recommended to further validate the waste dump set-back criteria.

1.22.3 Water and Environment

- Complete the evaluation of flood protection berms where Project infrastructure is located in proximity to floodlines as a risk mitigation measure;
- Complete additional investigations around the eastern extension of the open pit to evaluate soil and rock permeability and need for mitigation measures to reduce inflows and potential for flooding due to high water levels within Kenogamisis Lake.;
- Consider the options to manage historical tailings that need to be relocated to allow possible future processing as a source of low grade mill feed;
- Manage the potential geotechnical and environmental issues associated with the construction of the Highway 11 deviation over top of historical tailings. Clearly define the divisions of responsibility for highway related engineering, construction, geotechnical engineering, and environmental engineering;
- Continue the implementation of the environmental follow up / monitoring programs described in Section 20 related to air, noise, water, fish, fauna, wildlife and social and the implementation of environmental management plans;
- Advance the design of the drainage and seepage collection systems and ponds to maximize seepage collection, conveyance, and storage potential;
- Refine the water balance to optimize storage requirements within the underground workings, open pit and TMF to equalize flows and discharges to the mine effluent treatment plant;
- Advance geochemical testing and characterization studies and incorporate production and operational management into the Conceptual Waste Rock Management Plan;
- Complete additional geochemical testing of historical tailings to allow better prediction of potential effects to water quality as a result of the relocation and storage of the tailings within the TMF.

1.22.4 Tailings Management Facility

- Conduct supplemental geotechnical investigations and laboratory testing for better definition of strength and consolidation properties of the interbedded silt layers encountered in the subsurface soils near the southwest and southeast dams;
- Conduct deformation modelling of critical dam sections to confirm sufficiently robust protection against core cracking;

- Perform settling and consolidation testing to better understand tailings behavior and density progression to optimize the TMF design as the currently assumed properties are believed to be conservative;
- Conduct further studies of the geochemistry of the ore and tailings to allow optimization of the TMF design, operation, and closure planning.;
- Conduct detailed tailings deposition planning to optimize the dam raising schedule and inner dam construction requirements;
- Conduct detailed water balance modelling to confirm design assumptions and set operating guidelines for the TMF pond. Adequate mill make-up water supply storage will be required before winter;
- Conduct site-specific seismic hazard analysis to determine appropriate earthquake design parameters for the dam design;
- Finalize geotechnical investigations to support construction documents.

1.22.5 Metallurgy and Processing

- Conduct additional metallurgical tests including:
  - Cyanide destruction optimization testwork to confirm reagents and operating conditions. Investigate the possibility of realizing the cyanide destruction and the precipitation of arsenic in two stages;
  - Tests to validate oxygen consumption in the leaching tanks;
  - Abrasion tests to confirm liner and steel ball consumptions in the grinding mills;
  - Consider additional pilot plant tests with a potential HPGR vendor;
  - Additional tests for equipment sizing, as required;
  - Testwork to investigate the possibility of thickening the tailings prior to cyanide destruction to increase cyanide recovery.

1.22.6 Power and Other Infrastructure

- Continue to integrate the planning and execution of the infrastructure relocation program and other external infrastructure interfaces, to ensure alignment with the project development schedule and budget, including the Trans-Canada Highway 11 realignment, the relocation of the Hydro One Geraldton Transmission Station and the natural gas distribution pipeline.

- Continue to monitor evolving climate change regulations and evaluate the impact of climate change regulations on processing OPEX related to the consumption of natural gas for power generation, and re-evaluate the heat recovery tradeoffs to consider the cost impact of carbon taxes and/or credit trading and whether any further potential increases in overall project thermal efficiency through the use of heat recovery in the power plant could mitigate the impact of the additional potential costs of carbon emissions regulation.

1.22.7 Project Execution

- Put in place the Project delivery organization as proposed and described in Section 24 of this Report, and implement the associated project controls and management systems for effective project delivery;
- Develop and implement the operations organization required to execute the Project general and administrative and mining preproduction functions;
- Refine and detail hiring plan for project execution team to assure all positions are staffed in a timely manner.

1.23 Brookbank, Key Lake and Viper Recommendations

Consider additional exploration on the surrounding deposits, such as Brookbank underground, as an eventual source of high grade mill feed material when the average grade dips in Year 6 and Years 8 and 9. These potential mines would need to be mined concurrently with the Hardrock Project open pit given the high milling rates.

- Brookbank – 9,000 m drill program, surface stripping and detail mapping, followed by a resource update to include all new information. The cost for this program would be approximately CAD 1.5M.

2. INTRODUCTION

On March 9, 2015 Centerra Gold Inc. ("Centerra") and Premier Gold Mines Limited ("Premier") formed a 50/50 partnership to facilitate the joint ownership, exploration and future development of the Trans-Canada Property (subsequently renamed the Greenstone Gold Property) (the "Property"), which includes the Hardrock Project (the "Project"). The partnership was originally called TCP Limited Partnership and was subsequently changed to Greenstone Gold Mines LP (the "Partnership"). The Partnership is managed by Greenstone Gold Mines GP Inc. which acts on behalf of the Partnership ("GGM"). On behalf of the Partnership and its partners, Centerra and Premier, GGM commissioned a team of engineering consultants to prepare and issue a feasibility study ("FS") as well as a technical report ("Report") to be prepared in accordance with the National Instrument 43-101 ("NI 43-101") Standards of Disclosure for Mineral Projects. The objective of this Report and FS is the evaluation of the technical feasibility and economic viability of the development of an open pit mine at the Hardrock deposit, including processing facilities and related infrastructures.

The scope of this Report and FS includes the geology and Mineral Resources of GGM's Greenstone Gold Property claim groups, including Hardrock, Brookbank, Key Lake and Viper. The Mineral Reserves, mining, infrastructure, processing and financial analysis sections of this Report consider the Hardrock deposit only.

The Report and FS responsibilities of the engineering consultants follow:

- G Mining Services Inc. ("GMS") - overall Report and FS coordination, property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, Mineral Resource estimates, Mineral Reserves (pertaining to the Hardrock deposit), mining methods, economic analysis, operating costs pertaining to mining, infrastructure and power capital cost estimate and project execution plan;
- Micon International Limited ("Micon") - property description and location, accessibility, history, geological setting and mineralization, deposit types, exploration, drilling, sample preparation and security, data verification, and Mineral Resource estimates (pertaining to the Brookbank, Key Lake and Kailey deposits);
- Stantec Consulting Limited ("Stantec") - climate and physiology, site water balance (effluent and site runoff), environmental, permitting, and social aspects;
- WSP Canada Inc. ("WSP") and Soutex Inc. ("Soutex") - metallurgical testing, recovery methods, mineral processing plant capital and operating cost;

- TBT Engineering Limited ("TBT") - re-alignment of Trans-Canada Highway 11 through the Project area of influence, Ontario Ministry of Transportation ("MTO") patrol station;
- Amec Foster Wheeler Environment and Infrastructure, a Division of Amec Foster Wheeler Americas Limited ("Amec") - tailings management, and geotechnical engineering for the Project infrastructures, including the waste rock storage areas, the mineral processing facility and the tailings management facility ("TMF");
- Golder Associates Ltd. ("Golder") - rock mechanics and open pit geotechnical studies.

A summary of the qualified persons ("QP") responsible for each section of the Report is detailed in Table 2.1.

Table 2.1: Summary of Qualified Persons

	QP	Company	Report Sections
1	Louis-Pierre Gignac, P.Eng.	GMS	1, 2, 3, 15, 16, 18.7.2, 19, 21.1.5, 21.2.1, 21.2.2, 21.2.4, 22, 25.1, 25.1.2, 25.1.6, 25.2.2, 26.1, 26.1.2, 27
2	Glen Schlyter, P.Eng.	GMS	5.4.1, 5.4.4, 5.4.5, 18.1, 18.2.1, 18.5 - 18.7.1, 21.1.1, 21.1.3, 21.1.4, 24, 25.1.4, 25.2.1, 26.1.6 (Other infrastructure), 26.1.7
3	Martin Menard, P.Eng.	GMS	5.4.2, 5.4.3, 18.3, 18.4, 18.7.3, 21.1.3.1. (Capex – Power plant) and 21.2.3 (OPEX – Power cost), 26.1.6 (Power)
4	Réjean Sirois, P.Eng.	GMS	4.1, 4.5, 5.1.1, 5.3, 5.5, 6.1, 7.1 to 7.4, 8.1, 9.1, 9.2, 10.1, 11.1, 12.1, 14.1, 23, 25.1.1, 26.1.1 (All generally pertaining to the Hardrock deposit)
5	Charley Murahwi, P.Geo.	Micon	4.2 to 4.5, 5.1.2, 6.2 - 6.4, 7.5, 8.2, 9.3, 9.4, 10.2, 11.2, 12.2, 14.2, 25.1.1 (All generally pertaining to the Brookbank, Key Lake and Kailey deposits), 26.2
6	Eric Poirier, P.Eng.	WSP	17.6, 17.7, 21.1.2, 21.1.3.1 (CAPEX – Process Plant)
7	Pierre Roy, P.Eng.	Soutex	13, 17.1 – 17.5, 18.2.5, 21.2.2, 25.1.3, 26.1.5
8	David Ritchie, P.Eng.	Amec	18.2.4, 26.1.4
9	Marc Rougier, P.Eng.	Golder	16 (Open pit geotechnical / slopes)
10	Craig Johnston, P.Geo.	Stantec	5.2, 18.2.2 and 18.2.3, 20, 25.1.5, 26.1.3

2.1 Sources of Information and Data

Unless otherwise stated, all the information and data contained in the Report or used in its preparation has been provided by GGM, and all currencies are expressed in Canadian dollars (CAD). Technical data provided by GGM for this Report is the result of work performed or verified by GGM staff or their consultants as follows:

- Infrastructure relocation estimates;
- Project execution strategy, schedule, organization, budgets and estimates pertaining to construction indirect costs;

The QPs who prepared the Report relied on information provided by the following sources who are not QPs for this Report:

- Roscoe Postle Associates Inc. ("RPA") validated the key input parameters for the Report economics;
- TBT Engineering Limited ("TBT") provided the technical information and estimates related to the relocation of Highway 11 and the MTO patrol station;
- Delta Energy LLC provided energy cost studies and forecasts used by GMS to develop the cost of self-generated electricity and perform energy cost tradeoff studies, including the liquefied natural gas fuel study for the mine fleet;
- Eagle Mapping Ltd provided 0.5 m accuracy topography in digital format of the Project area which was used to determine the ground surface shapes used in open pit and infrastructure / processing/ tailings management facility earthworks quantity estimates;
- SGS Minerals Services, ThyssenKrupp and SimSAGe provided metallurgical reporting and studies as referenced in Section 13 - Mineral Processing and Metallurgical Testing, managed principally by GGM;
- SGS Mineral Services provided laboratory geochemical and mineralogical testing, managed principally by GGM;
- Golder relied on the oriented core data collected by MD Engineering ("MDE") for the evaluation of the open pit geotechnical parameters and pit slope studies. Golder validated the MDE methodology and validated <5% of the total oriented core. Golder has no reason to believe that the remainder were not also collected in a professional manner;
- Union Gas provided engineering studies, market information and a construction cost estimate for a natural gas branch pipeline from the TCPL Canadian Mainline pipeline to the Project site. Union Gas was managed principally by GGM.

2.2 Site Visit

The following QPs visited the Project site as detailed below:

- Louis-Pierre Gignac, P.Eng., GMS, visited the site on June 3, 2014;
- Glen Schlyter, P.Eng., GMS, visited the site on June 3, 2014;
- Martin Menard, P.Eng., GMS, visited the site on June 3, 2014;
- Réjean Sirois, P.Eng., GMS, visited the site from August 1 to 4, 2016;
- Charley Murahwi, P.Geo., Micon, visited the site from November 8 to 9, 2011 and on March 19, 2013;
- Eric Poirer, P.Eng., WSP, visited the site on June 3, 2014;
- Pierre Roy, P.Eng., Soutex, visited the site on June 3, 2014;
- David Ritchie, P.Eng., Amec, visited the site from July 2 to 3, 2014;
- Marc Rougier, P.Eng., Golder, visited the site from September 10 to 11, 2013;
- Craig Johnston, P.Geo., Stantec, visited the site from July 21 to 22, 2014.

2.3 Units of Measure, Abbreviations and Nomenclature

The units of measure presented in this Report, unless noted otherwise, are in the metric system.

A list of the main abbreviations and terms used throughout this Report is presented in Table 2.2**.**

Table 2.2: List of Abbreviations

Abbreviations	Full Description
3SD	Three standard deviations
A	Ampere
AA	Atomic Absorption
ABA	Acid-Base Accounting
AECO	Alberta Energy Company
AERT	Aboriginal Environmental Review Team
Ag	Silver
AISC	All-In Sustaining Cost
Amec	Amec Foster Wheeler Americas Limited
APV	Aquatic Protection Value
ARD	Acid Rock Drainage
As	Arsenic
Au	Gold
AZA	Anumbiigoo Zaagi'igan Anishinaabek
BGB	Beardmore-Geraldton Greenstone Belt
BIF	Banded-iron Formation
BNA	Bongwi Nevaashi Anishinaabek
BWI	Ball Mill Work Index
BZA	Biinjitiwaabik Zaaging Anishinaabek
°C	Degree Celsius
C	Carbon
Ca	Calcium
CAD	Canadian Dollar
CAPEX	Capital Expenditures
CCTV	Closed circuit television
CEA	Canadian Environmental Assessment
CEAA 2012	Canadian Environmental Assessment Act 2012
CHP	Combined Heat and Power
CHVI	Cultural Heritage Value or Intertest
CIA	Cultural Impact Assessment

Abbreviations	Full Description
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definitions	CIM Definition Standards for Mineral Resources and Mineral Reserves dated May 10, 2014
CIP	Carbon in Pulp
CN	Cyanide
COG	Cut-off Grade
CoV	Coefficient of Variation
CRM	Certified reference material
CSD	Critical solid density
Cu	Copper
DCF	Discount Cash Flow
DD	Diamond Drilling
DDH	Diamond Drill Hole
DGPS	Differential Global Positioning System
DTH	Down-the-hole
DWT	Drop Weight Test
EA	Environmental Assessment
EAA	Environmental Assessment Act
EDF	Environmental Deign Flood
E-GRG	Extended Gravity Recoverable Gold
EIS	Environmental Impact Statement
EM	Electromagnetic
EMP	Environmental Management Plans
ETP	Effluent Treatment Plant
˚F	Degree Fahrenheit
FA	Fire Assay
Fe	Iron
FEL	Front-End-Wheel Loaders
FS	Feasibility Study
Ft	Foot or feet
G	Giga - (000,000,000’s)
g	Gram
g/t	Grams per tonne
g Au/t	Grams of gold per tonne
g/L	Grams per litre
G&A	General & Administration

Abbreviations	Full Description
GGM	Greenstone Gold Mines GP Inc. (the managing partner) and Greenstone Gold Mines LP (the partnership), collectively referred to as Greenstone Gold Mines
GHG	Greenhouse gas
GMS	G Mining Services Inc.
gpm	Gallons per minute (US)
GPS	Global Positioning System
GRG	Gravity Recoverable Gold
h/d	Hours per day
h/wk	Hours per week
h/y	Hours per year
ha	Hectares
h	Hour
HCl	Hydrochloric Acid Solution
HDPE	High-Density Polyethylene
HG	High Grade
HONI	Hydro One Networks Inc.
hp	Horsepower
HPC	Hazard Potential Classification
HPGR	High Pressure Grinding Rolls
HSE	Health, Safety and Environmental
HVAC	Heating, Ventilation and Air Conditioning
Hz	Hertz
ICMI	International Cyanide Management Institute
ICPAES	Inductively Coupled Plasma Atomic Emission Spectroscopy
ICPMS	Inductively Coupled Plasma Mass Spectroscopy
ID^3	Inverse Distance Cube Interpolation
IDF①	Inflow Design Flood
IDF②	Intensity Duration Frequency
IEC	International Electrotechnical Commission
IESO	Independent Electricity System Operator
in	Inch (imperial unit)
IP	Induced Polarization
IR	Information Requests
IRR	Internal Rate of Return
ISO	International Organization for Standardization
IT	Information Technology

Abbreviations	Full Description
ITRB	Independent Tailings Review Board
JV	Joint Venture
k	Kilo - (000's)
kg	Kilograms
kg/t	Kilograms per tonne
koz	Thousands of troy ounces
kV	Kilovolts
km	Kilometre
km/h	Kilometre per hour
kPa	Kilopascal
KPIs	Key Performance Indicators
kV	kilovolt
kW	Kilowatt
kWh	Kilowatt hour
kWh/t	Kilowatt hour per tonne
L	Litre
LAA	Local Assessment Areas
LEL	Lowest Effect Level
LG	Low Grade
LIMS	Low Intensity Magnet Separation
LNG	Liquid Natural Gas
LOM	Life of Mine
LTRAs	Long Term Relationship Agreements
M	Mega or Millions (000,000's)
MARC	Maintenance and Repair Contract
masl	Metres above sea level
m	Metre
m/min	Metre per minute
m/s	Metre per second
m^2	Square metre
m^3	Cubic metre
m^3/h	Cubic metre per hour
MCC	Motor Control Centers
mg	Milligram
MG	Medium Grade
mg/L	Milligram per litre
MHT	MacLeod High Tailings
min	Minute

Abbreviations	Full Description
ml	Millilitre
mm	Millimetre
MMAH	Ministry of Municipal Affairs and Housing Act
MMER	Metal Mining Effluent Regulations
MNDN	Ministry of Northern Development and Mines
MNO	Métis Nation of Ontario
MNRF	Ministry of Natural Resources and Forestry
mo	Month
MOECC	Ministry of the Environment and Climate Change
Moz	Millions of troy ounces
MOWL	Maximum Operating Water Level
MPa	Megapascal
MRE	Mineral Resource Estimate
Mt	Million tonnes
MTO	Ministry of Transportation - Ontario
MVA	Megavolt-ampere
MW	Megawatt
N	Newton
NaCN	Sodium Cyanide
NI 43-101	National Instrument 43-101 - Canadian Standards of Disclosure for Mineral Projects
Non-PAG	Non-Potentially Acid Generating
NPI	Net Profit Interest
NPV	Net Present Value
NQ	Drill Core Diameter (47.6 mm)
NSR	Net Smelter Return
NTS	National Topographic Systems
NVR	Network Video Recorder
Ø	Diameter
OG	Original
OK	Ordinary Kriging Methodology
OPEX	Operating Expenditures
OPP	Ontario Provincial Police
O.Reg	Ontario Regulation
oz	Troy Ounce (31.10348 grams)
P80	Dimension, in size distribution, for which 80 percent of the material is smaller
PAG	Potentially Acid Generating

Abbreviations	Full Description
PDA	Project Development Area
PEA	Preliminary Economic Assessment
PFS	Pre-feasibility Study
Premier	Premier Gold Mines Limited
Pb	Lead
PMF	Probable Maximum Flood
PLC	Programmable Logic Controller
POE	Power Over Ethernet
POX	Pressure Oxidation
ppb	Parts per Billion
ppm	Parts per Million
psi	Pounds per square inch
PV	Present Value
PWQOs	Provincial Quality Objectives
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
R&D	Research and Development
RA	Repeat Assays
RAA	Regional Assessment Area
RC	Reverse Circulation
RoM	Run-of-mine
RQD	Rock Quality Designation
RPA	Roscoe Postle Associates Inc.
rpm	Revolutions per minute
RSMIN	Red Sky Métis Independent Nation
RWI	Rod Mill Work Index
S	Sulfur
SAG	Semi-autogenous Grinding
SAR	Species at Risk
SCC	Standard Council of Canada
Sec	Second (time)
SEL	Severe Effect Level
SMC	SAG Mill Comminution
SMU	Selective Mining Unit
SOCC	Species of Conservation Concern
SPT	Standard Penetration Tests
STP	Sewage Treatment Plant
t	Tonnes (1,000 kg) (metric ton)

Abbreviations	Full Description
t/y	Tonnes per year
t/d	Tonnes per day
t/h	Tonnes per hour
t/m^3	Tonnes per cubic metre
TBTE	TBT Engineering Limited
TCPL	TransCanada-PipeLines Limited
TK	Traditional Knowledge
TMF	Tailings Management Facility
TK	Traditional Knowledge
ToR	Terms of Reference
TRLU	Traditional Land and Resource Use
TS	Transmission Station
µm	Micron (10^{-6} metre)
UCoG	Underground Cut-off Grade
USD	United States Dollar
V	Volt
VC	Valued Components
VFD	Variable Frequency Drive
VLF-EM	Very Low Frequency Electromagnetic
VSA	Vacuum Swing Adsorption
WHIMS	Wet High Intensity Magnetic Separation
wk	Week
WRSAs	Waste Rock Storage Area
WSP	WSP Canada Inc.
XRF	X-ray Fluorescence
y	Year
% w/w	Percent weight by weight

3. RELIANCE ON OTHER EXPERTS

This Report has been prepared by GMS for GGM, Centerra and Premier. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available to GMS at the time of preparation of this Report,
- Assumptions, conditions, and qualifications as set forth in this Report, and
- Data, reports, and other information supplied by GGM and other third party sources.

For the purpose of this report, GMS has relied on ownership information provided by GGM. GMS has not researched property titles or mineral rights for the Project and expresses no opinion as to the ownership status of the property.

GMS has relied on GGM for guidance on applicable taxes, royalties, and other government levies or interests, applicable to revenue or income from the Project.

Except for the purposes legislated under provincial securities laws, any use of this Report by any third party is at that party's sole risk.

4. PROPERTY DESCRIPTIONS AND LOCATIONS

GGM's Greenstone Gold Property (formerly the Trans-Canada Property) includes four claim groups: Hardrock, Viper, Brookbank and Key Lake, of which Hardrock is the core project.

4.1 Hardrock Project

4.1.1 Location

The Hardrock Project is located in northern Ontario, Canada, approximately 275 km northeast of Thunder Bay on NTS sheets 42 E/10 and 42 E/11, and approximately at Latitude 49º 40'N and Longitude 86º 56'W (Figure 4.1). The Hardrock Project is located in the townships of Lindsley, Errington, Salsburg, McKelvie and Ashmore.

The Hardrock deposit area covered by the Mineral Resource estimate in this Report is located in the townships of Errington and Ashmore on NTS sheet 42E/10, approximately four kilometres south of the town of Geraldton. The approximate geographic centre coordinates of the Hardrock deposit resource area are 49°40'47"N and 86°56'32"N (UTM coordinates: 504175.9E and 5503024N, NAD 83, Zone 16).

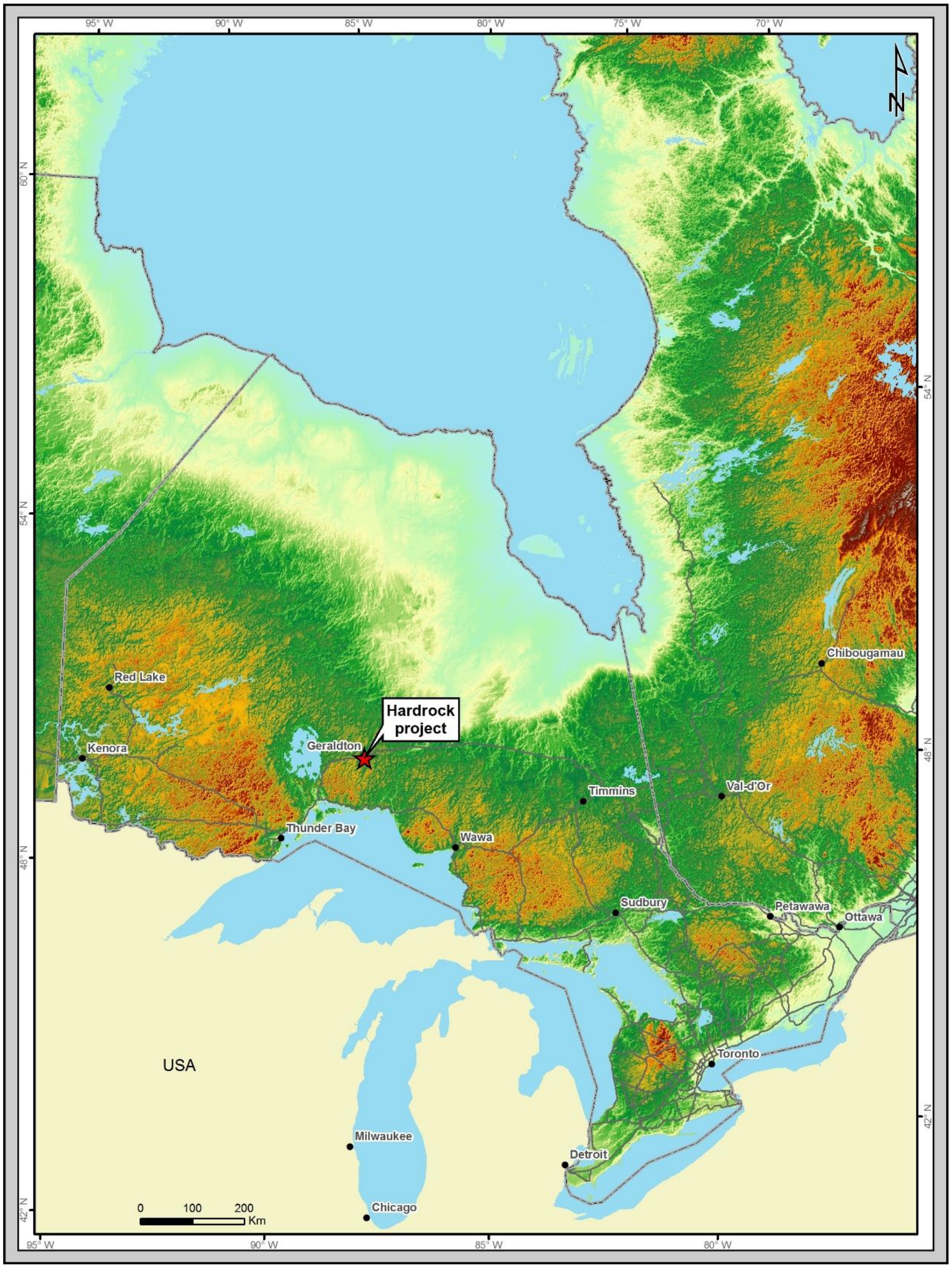
4.1.2 Property Holdings

On October 12, 2016, the Hardrock Project consisted of a contiguous block of patented claims, mining leases, licences of occupation and staked claims covering an aggregate area of 14,676.416 ha (Figure 4.2 to Figure 4.4). All of the claims, leases and licences of occupation are held by GGM on behalf of the Partnership. The patented claims, leases and licences of occupation for the Hardrock Project are subject to terms under a number of agreements.

A leasehold patent of mining rights or of surface rights, or of both mining rights and surface rights is a conveyance or grant of possession of land for a set length of time. There is usually a requirement to pay rent.

Freehold Patent means a grant from the Crown in fee simple. The patent cannot be terminated by the Ministry of Northern Development and Mines ("MNDM"), except for voluntary surrender or non-payment of mining lands taxes.

Figure 4.1: Location Map for the Hardrock Project



Source: Innovexplo, 2015

Table 4.1 is a summary of patented claims, mining leases, licences of occupation and staked claims at the Hardrock Project.

Table 4.1: Summary of Patent Claims, Mining Leases, Licenses of Occupation and Stake Claims - Hardrock Project

Type	Numbers	Hectares
Unpatented Claims	55	9,877.257
Mining Lease	27	791.156
Part-Patented; Part Licence of Occupation	46	992.980
License of Occupation	32	515.223
Patented Claims	141	2,499.800
Total		**14,676.416**

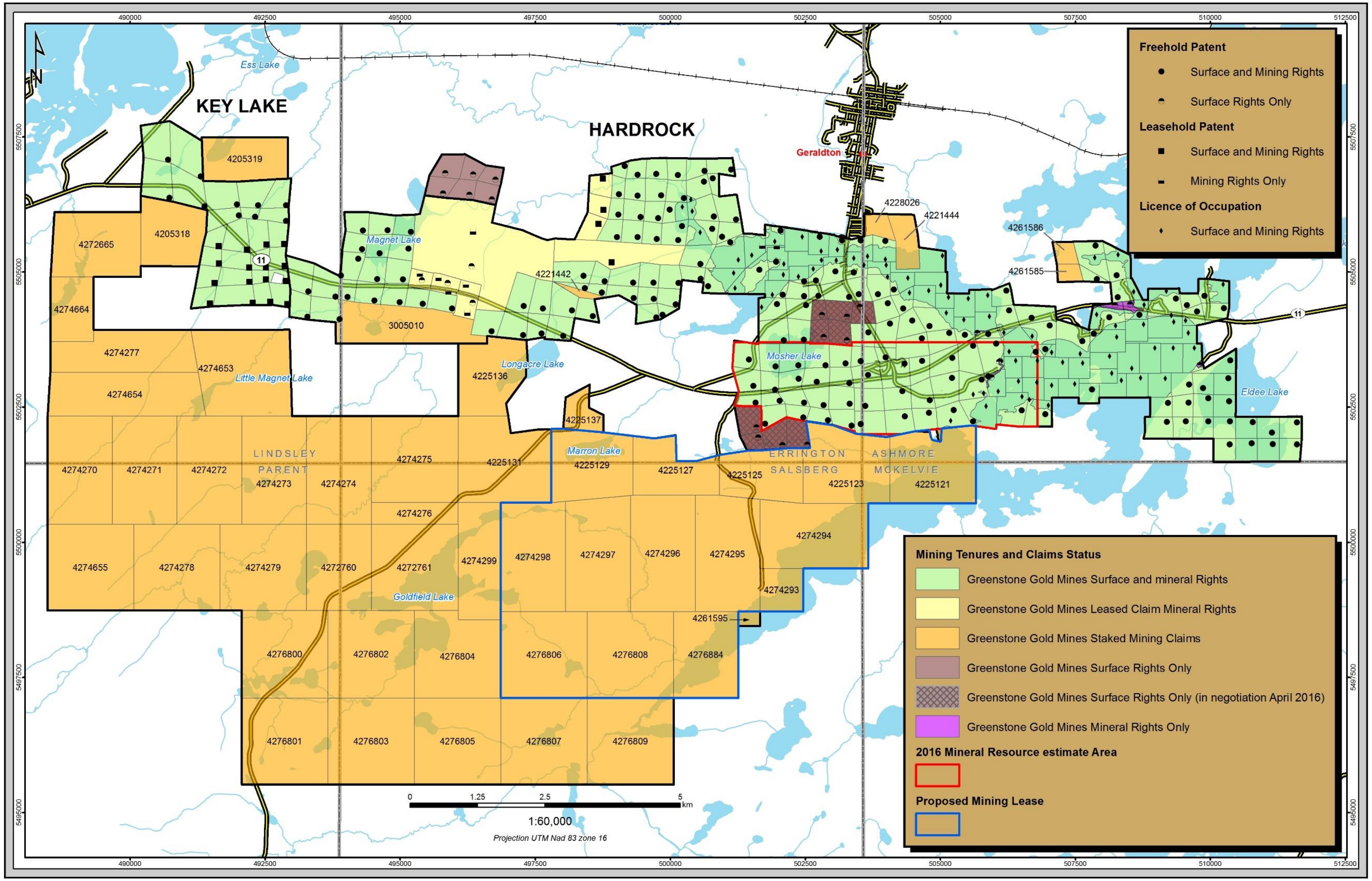
4.1.3 Agreements and Encumbrance

4.1.3.1 Agreement with Roxmark Mines Ltd - 2007

This agreement is summarized from the Premier 2007 Annual Information Form available on the SEDAR website.

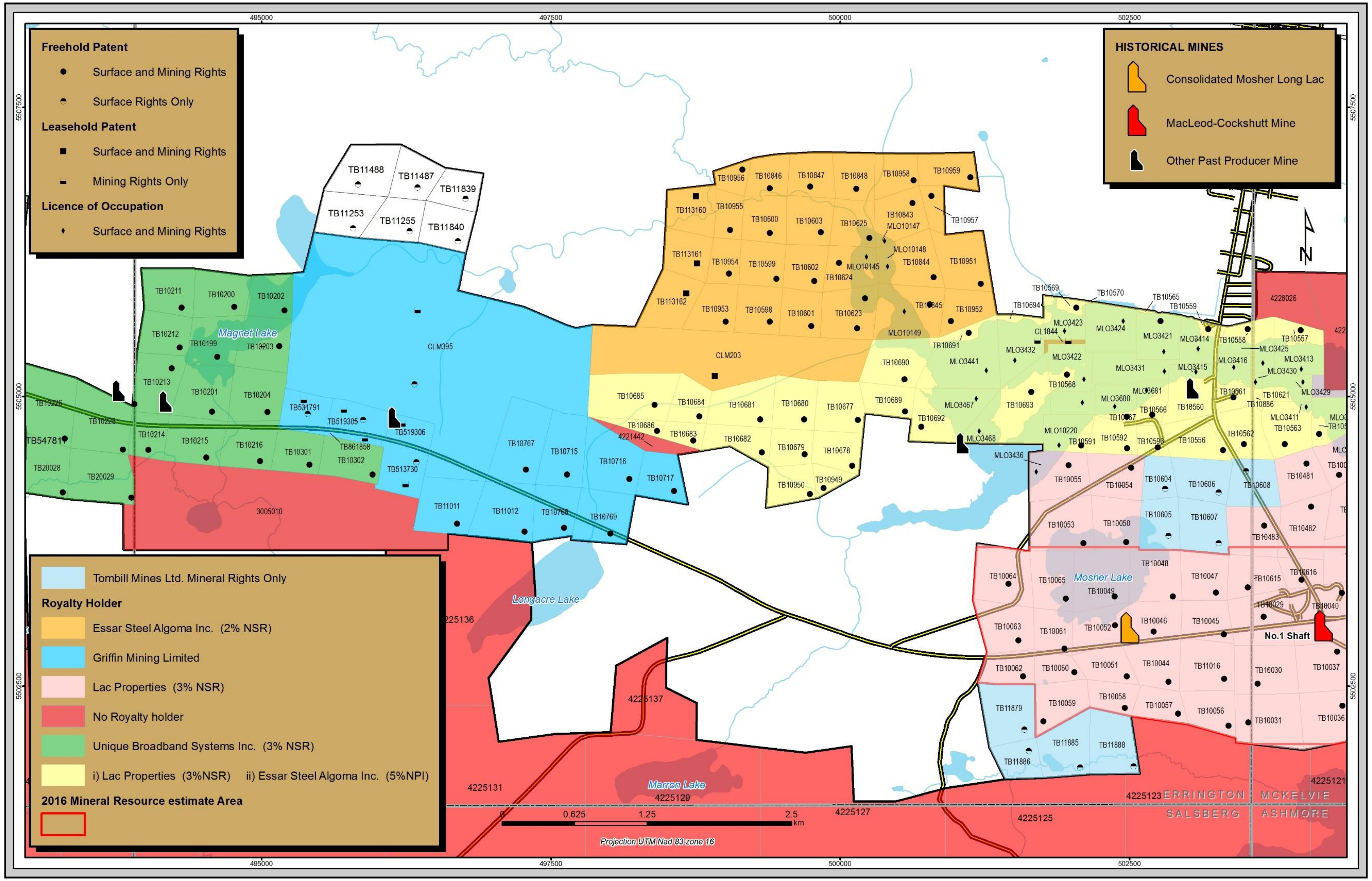
Pursuant to a Letter of Intent dated September 24, 2007, Premier entered into an agreement with Roxmark Mines Limited ("Roxmark"; now amalgamated with Goldstone Resources Inc.), whereby Premier acquired the right to earn up to a 70% interest in certain mineral claims which are 100% owned by Roxmark (the "Geraldton Project"). The Geraldton Project consists of 113 mineral claims located in Ashmore and Errington Townships in the Geraldton Greenstone belt of northwestern Ontario. Under the terms of the September 24, 2007 Letter of Intent, Premier could earn an initial 51% interest in the Geraldton Project by paying to Roxmark CAD 500,000, issuing 250,000 common shares of Premier, and incurring CAD 7,000,000 in exploration expenses on the Geraldton Project over a four-year term (the "Geraldton Earn-In").

Figure 4.2: Location Map - Mining Titles and Mineral and Surface Rights - Hardrock and Key Lake Projects



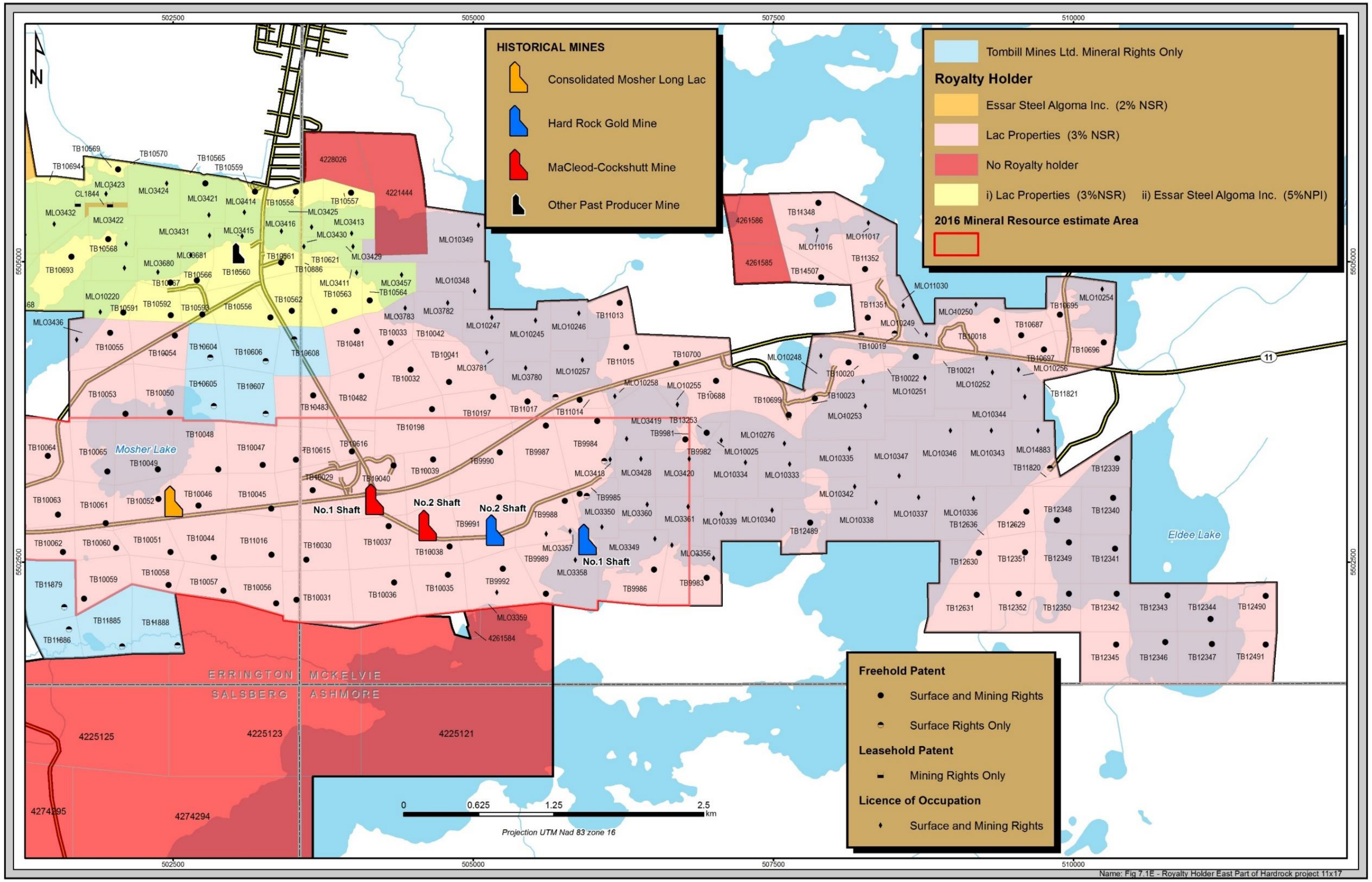
Source: Innovexplo, 2015 with modifications by GGM, 2016

Figure 4.3: Location Map of the Hardrock Project - Mining Titles Subject to an NSR (West part)



Source: Innovexplo, 2015 with modifications by GGM, 2016

Figure 4.4: Location Map of the Hardrock Project - Mining Titles Subject to an NSR (East part)



Source: Innovexplo, 2015 with modifications by GGM, 2016

4.1.3.2 Claims Staked by Premier and Roxmark - 2007

Between October 1 and November 2007, Premier staked four claims located in Errington and Ashmore townships, covering an aggregate area of 288 ha**.** Premier personnel established the claim boundaries by GPS. These claims are covered by the Roxmark JV Agreement and, therefore, are owned 70% by Premier and 30% by Roxmark.

4.1.3.3 Agreement with Lac Properties Inc. - 2008

This agreement is summarized from the Premier 2008 Annual Information Form available on the SEDAR website.

Effective December 18, 2008, Premier entered into an agreement (the "G-L Agreement") with Lac Properties Inc. ("Lac Properties") pursuant to which Premier purchased from Lac Properties, a 100% interest in the mining claims commonly known as the Geraldton, Ozone Creek and Eva Summer properties (the "G-L Properties") together with certain equipment and other assets related thereto (the "G-L Assets"). Premier satisfied the purchase price for the G-L Assets by:

- Issuing to Lac Properties 500,000 common shares of Premier;
- Paying to Lac Properties the amount of CAD 1,000,000;
- Depositing CAD 1,000,000 in an environmental reclamation trust fund in connection with the G-L Assets;
- Entering into a royalty agreement with Lac Properties which provides for, among other things, the payment to Lac Properties of a 3% net smelter return royalty ("NSR") in respect of the G-L Assets (this NSR was later acquired by Franco-Nevada Corporation, see Section 4.1.3.7 below);
- Assuming certain liability and obligations of Lac Properties in respect of the G-L Assets.

No royalty shall be payable for or with respect to reasonable quantities of product which are not sold but which are used for assaying, treatment, amenability, metallurgical, test work, piloting or other analytical processes or procedures.

4.1.3.4 Agreement with Roxmark Mines Limited - 2008

This agreement is summarized from the Premier 2009 Annual Information Form available on the SEDAR website.

Pursuant to a Letter of Intent dated September 18, 2007 and amended on July 18 2008, (the "Roxmark Letter of Intent") between Premier and Roxmark (now amalgamated with Goldstone Resources Inc.), the companies agreed to form a joint venture in the Thunder Bay Mining Division, Ontario with respect to:

- Certain mining claims that were 100% owned by Roxmark (the "Roxmark Claims");
- Certain mining claims to be staked by Premier on behalf of the joint venture (the "Staked Claims");
- Certain mining claims commonly known as the Geraldton property that were acquired by Premier from Lac Properties (the "Lac Claims", and, together with the Roxmark Claims and the Staked Claims, the "Hardrock Project").

Pursuant to the Roxmark Letter of Intent, Premier acquired the right to earn a 51% interest in the Roxmark Claims by paying CAD 500,000 to Roxmark, issuing 250,000 common shares of Premier to Roxmark, and incurring CAD 7,000,000 in exploration expenses on the Hardrock Project over a four-year term (the "Roxmark Earn-In"). In addition, Premier was granted the option, immediately upon fulfilling the Hardrock Earn-In, to increase its interest in the Roxmark Claims to 70% by paying an additional CAD 250,000 to Roxmark, issuing an additional 150,000 common shares of Premier to Roxmark and electing to bring the Hardrock Project to a production decision within five years of completing the Hardrock Earn-In period (the "Roxmark Option Obligations"). If Roxmark proposes to dispose of any of the common shares of Premier issued to it under the Roxmark Letter of Intent, Premier has the right to place such common shares within a 10-day period following the notice of such disposition.

Under the terms of the Roxmark Letter of Intent, upon Premier's acquisition of the Lac Claims in December 2008, Premier was deemed to have given its notice of intention to exercise its option to increase its interest in the Roxmark Claims to 70%, subject to satisfaction of the Roxmark Earn-In and the Roxmark Option Obligations, with Roxmark acquiring a 30% interest in the Lac Claims upon such satisfaction.

Premier satisfied the Hardrock Earn-In and the Hardrock Option Obligations, and, as a result, Premier and Roxmark held a 70% interest and a 30% interest, respectively, in the Hardrock Project. Roxmark was subsequently acquired by, and amalgamated with, Goldstone Resources Inc.

4.1.3.5 <u>Acquisition of Goldstone Resources Inc. by Premier- 2011</u>

This acquisition is summarized from the Premier 2011 Annual Information Form available on the SEDAR website.

On August 16, 2011, Premier acquired all of the outstanding common shares of Goldstone Resources Inc. (“Goldstone”) pursuant to an arrangement (the "Goldstone Arrangement") under Section 182 of the Ontario Business Corporate Act. Goldstone's principal asset was its 30% interest in the Hardrock Project. As a result of the Goldstone Arrangement, Premier then held a 100% interest in the Hardrock Project.

Pursuant to the terms of the Goldstone Arrangement, all of the common shares of Goldstone were transferred to Premier, and former holders of common shares of Goldstone received 0.16 of a Premier common share plus CAD 0.0001 in cash in exchange for each common share of Goldstone so transferred. All of the outstanding common shares of Goldstone were then transferred from Premier to a wholly-owned subsidiary of Premier ("Premier Subco") incorporated for the purposes of the Goldstone Arrangement, and Goldstone was amalgamated into Premier Subco to form a new amalgamated corporation with the name Goldstone Resources Inc. ("Goldstone Amalco"). As a result of the Goldstone Arrangement, Goldstone Amalco became a wholly-owned subsidiary of Premier. Under the terms of the Goldstone Arrangement, holders of options exercisable for common shares of Goldstone received a fully-vested option (each a "Premier Replacement Option") granted by Premier to acquire 0.16 of a Premier common share plus the fractional amount of a Premier common share that, immediately prior to the effective time of the Goldstone Arrangement, had a fair market value equal to CAD 0.0001 in cash in exchange for each outstanding option to acquire Goldstone common shares. The outstanding warrants (the "Goldstone Warrants") to acquire Goldstone common shares remained outstanding in accordance with their terms, which provided that each warrant will be exercisable to acquire 0.16 of a Premier common share plus CAD 0.0001 in cash for each such Goldstone Warrant exercised. Pursuant to the Goldstone Arrangement, the Corporation issued an aggregate of 16,814,553 Premier common shares and paid an aggregate of CAD 10,512.61 to former holders of common shares of Goldstone. In addition, 197,026 Premier common shares are issuable upon exercise of Premier Replacement Options (which have exercise prices ranging from CAD 2.3436 to CAD 5.995 per share). The common shares of Goldstone were delisted from the Toronto Stock Exchange at the close of business on August 19, 2011, and Goldstone Amalco has ceased to be a reporting issuer.

4.1.3.6 Consolidation of the Hardrock Project - 2011

The completion of the Goldstone Arrangement had the effect of consolidating a 100% interest in the Hardrock Project for Premier. Premier also acquired, through its acquisition of Goldstone, a portfolio of exploration properties in the Geraldton-Beardmore Greenstone Belt, including interests in the Brookbank, Leitch-Sand River, Northern Empire, Nortoba-Tyson and Key Lake projects. The Key Lake Project adjoins the western end of the Hardrock Project and hosts the historical Jellico Mine of Jelex Mines Limited. Following completion of the Goldstone Arrangement, Premier has referred to its Geraldton-Beardmore property portfolio as the "Trans-Canada Property" of which Hardrock is the core project. The Trans-Canada

Property would later be renamed by Premier and its joint venture partner, Centerra, as the Greenstone Gold Property.

Figure 4.5: Location of Premier Gold Projects within the Trans-Canada- Property in 2011

Source: Innovexplo, 2015

4.1.3.7 Royalties Underlying Roxmark's Agreement Before 2007

As indicated, there are a number of underlying agreements and royalties that apply to some of the mining titles constituting the Hardrock Project. A number of mining titles owned by Roxmark in the past were subject to a 3% NSR in favour of Lac Properties and a 5% net profit interest in favour of Algoma Steel Inc. ("Algoma Steel"). Lac Properties is a wholly-owned subsidiary of Barrick Gold Corporation ("Barrick"). The Lac Properties royalty previously owned by Barrick is now owned by Franco-Nevada Corporation. Algoma Steel was acquired by Essar Global in 2007. Following its purchase by Essar Global, Algoma Steel announced that its name had been changed to Essar Steel Algoma Inc. A number of mining titles owned by Roxmark in the past are subject to a 2% NSR in favour of Essar Steel Algoma Inc.

In one deal, Roxmark, through its subsidiary Beaurox Mines Limited ("Beaurox"), purchased the interest held by Ateba Mines Inc. ("Ateba") in the former producing Magnet mine in Errington Township (Northern Miner, November 30, 1992). Ateba acquired an interest in the Magnet mine through a joint venture with Roxmark in the late 1980s. Under the agreement, Beaurox discharged Ateba's debt to Roxmark and the private European Mining Finance Company for a 1% NSR on the first 350,000 t of ore processed from the mine. The name of the European Mining Finance was changed to Griffin Mining Limited in January 1998 following a change in management and the business of the company to that of mining.

In another agreement, Roxmark purchased the mining rights on the Bankfield mine property from Golden Trio Minerals Ltd for 1,000,000 Roxmark shares and a 3% NSR royalty (Northern Miner, November 30, 1992). During the first year of commercial production, Roxmark can reduce the NSR to 1.5% with a CAD 500,000 cash payment. Following a 1:2 reverse-split in 1994, Golden Trio Minerals Ltd changed its name to PCS Wireless Inc. and changed its name again in 1999 to Unique Broadband Systems Inc.

4.1.3.8 Agreement with Tombill Mines Ltd - 2014

On December 10, 2014, Premier reached an agreement with Tombill Mines Ltd for the purchase of the surface rights of adjacent lands for CAD 500,000. These lands represent nine patented mining claims (TB 10604 to TB 10608 and TB 11879, TB 11885, TB 11886, and TB 11888) located in Errington and Ashmore townships. At the time of the Report, these surface rights were not yet acquired by GGM as it was still under legal procedures regarding the application of consent to consolidate part of these claims.

4.1.3.9 Partnership with Centerra for the Greenstone Gold Property

Effective March 9, 2015, Centerra and Premier, formed the 50/50 Partnership for the exploration, development and operation of the Trans-Canada Property (later renamed the Greenstone Gold Property). Under the Partnership, the Partnership's assets comprised the following claim groups: Hardrock, Brookbank, Key Lake and Viper (Figure 4.6 taken from Brousseau et al., 2015). The Partnership's assets do not include the Beardmore/Northern Empire property/mill or the Leitch/Sands River and Nortoba-Tyson properties (all of which remain solely owned by Premier).

Pursuant to a contribution agreement with the Partnership, Premier Hardrock arranged for all claims, leases and licences of occupations comprising the Greenstone Gold Property (other than assets related to Beardmore/Northern Empire, Leitch/Sands River and Nortoba-Tyson) to be transferred to the Partnership in exchange for a 50% interest in the Partnership (through the holding of general partnership units). Centerra entered into contribution agreement with the Partnership whereby it agreed to contribute CAD 85,000,000 to the Partnership for its 50% interest in the Partnership (through the ownership of limited partnership units). The CAD 85,000,000 was distributed to Premier Hardrock pursuant to the terms of the amended and restated limited partnership agreement entered into between Centerra, Premier and GGM (as managing partner) dated March 9, 2015 (the "Partnership Agreement").

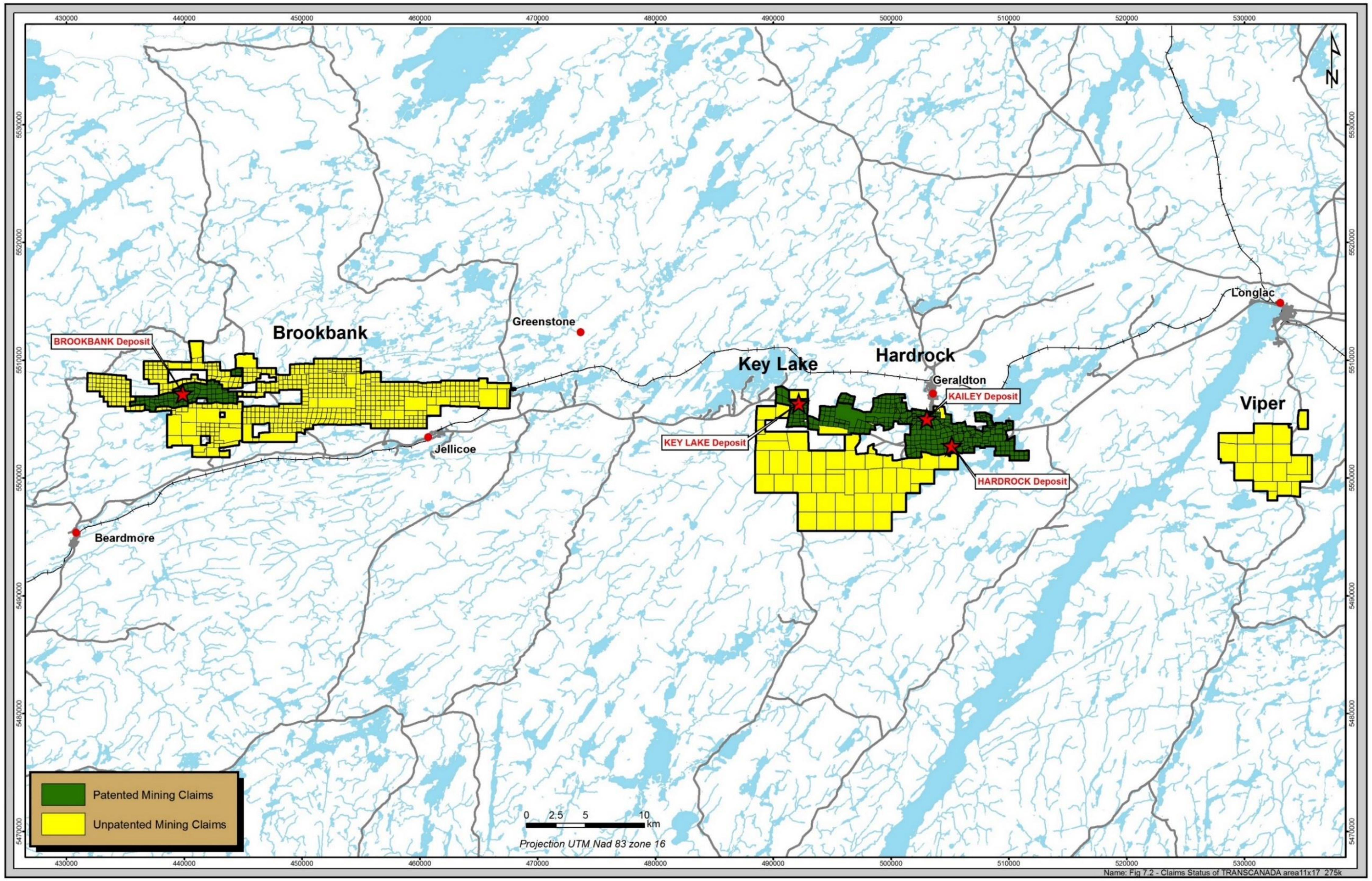
As contemplated in the Partnership Agreement, Centerra paid Premier an additional CAD 11 million in September 2015 in respect of a contingent payment which was based on the updated mineral resource calculation for the Hardrock Project. These funds were distributed by the Partnership to Premier.

In accordance with the Partnership Agreement, Centerra committed to solely fund up to CAD 185M in capital to develop the Greenstone Gold Property (following which all funding for the Partnership will be made on a pro-rata basis). A portion of these funds were used to complete a comprehensive technical and economic feasibility study including an updated Mineral Resource estimation for the Hardrock deposit. Subject to the satisfaction of certain feasibility and project advancement criteria, which are still outstanding, the remainder of the funds will be used towards the construction and development of the Hardrock deposit in accordance with annual programs and budgets approved by GGM, as managing partner for the Partnership, from time to time. Centerra and Premier have formed a joint board of directors for GGM to oversee future exploration, development and operations of the Partnership.

4.2 Viper

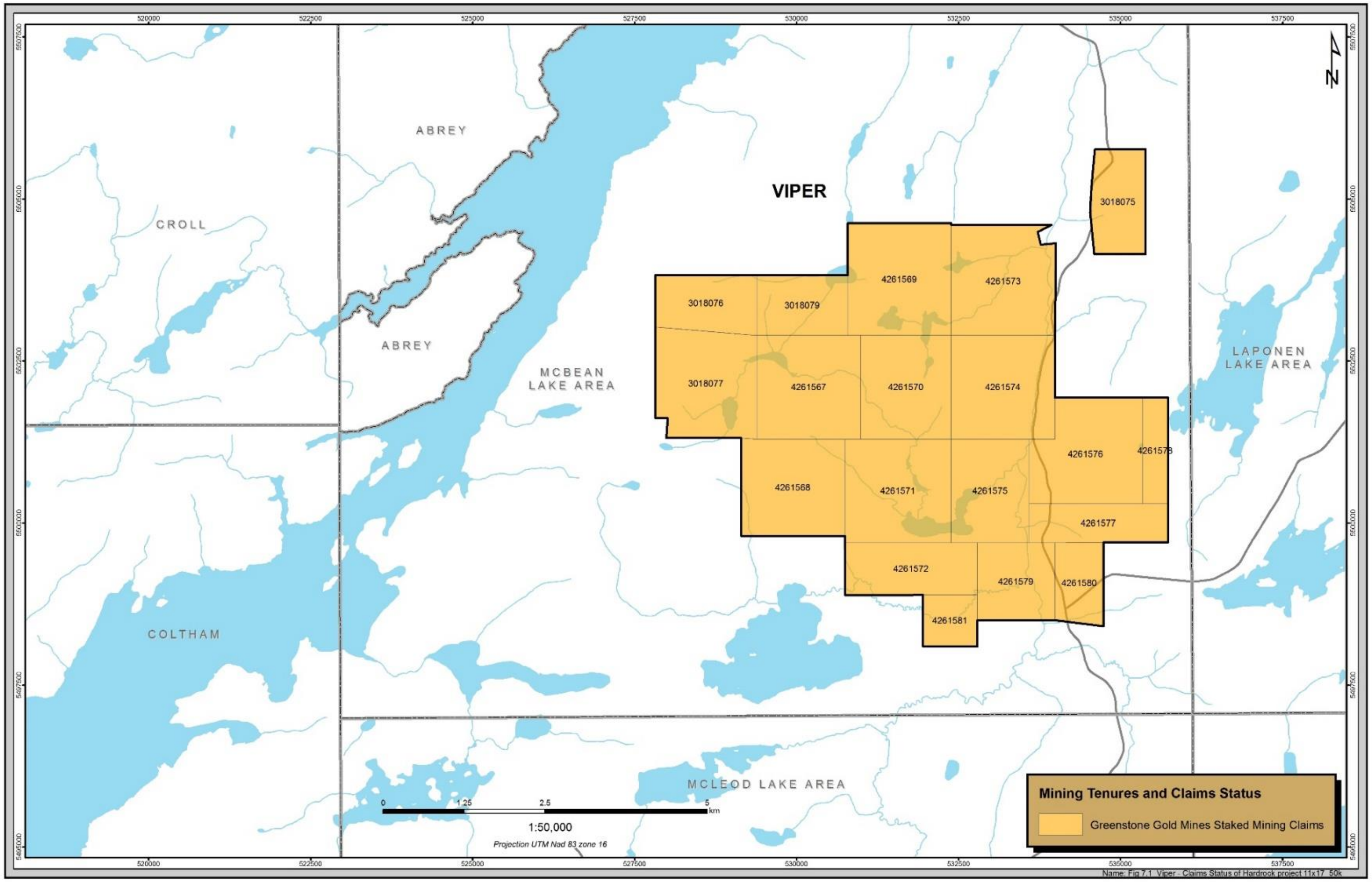
The Viper claims were staked by Premier in October 2013, three additional claims were staked in May 2014 and an additional isolated claim was added in October 2015 (Figure 4.7). The Viper claim group is 100% owned by GGM. The Viper Project is made up of eighteen contiguous and one isolated unpatented claims, totalling 3,551.84 ha.

Figure 4.6: Projects of the Trans-Canada Property as of March 2015



Source: Innovexplo, 2015

Figure 4.7: Viper Project Claims Map



Source: Innovexplo, 2015

4.3 Brookbank Project

The Brookbank project area hosts the Brookbank, Cherbourg and Fox Ear deposits and the Irwin prospect.

4.3.1 Location

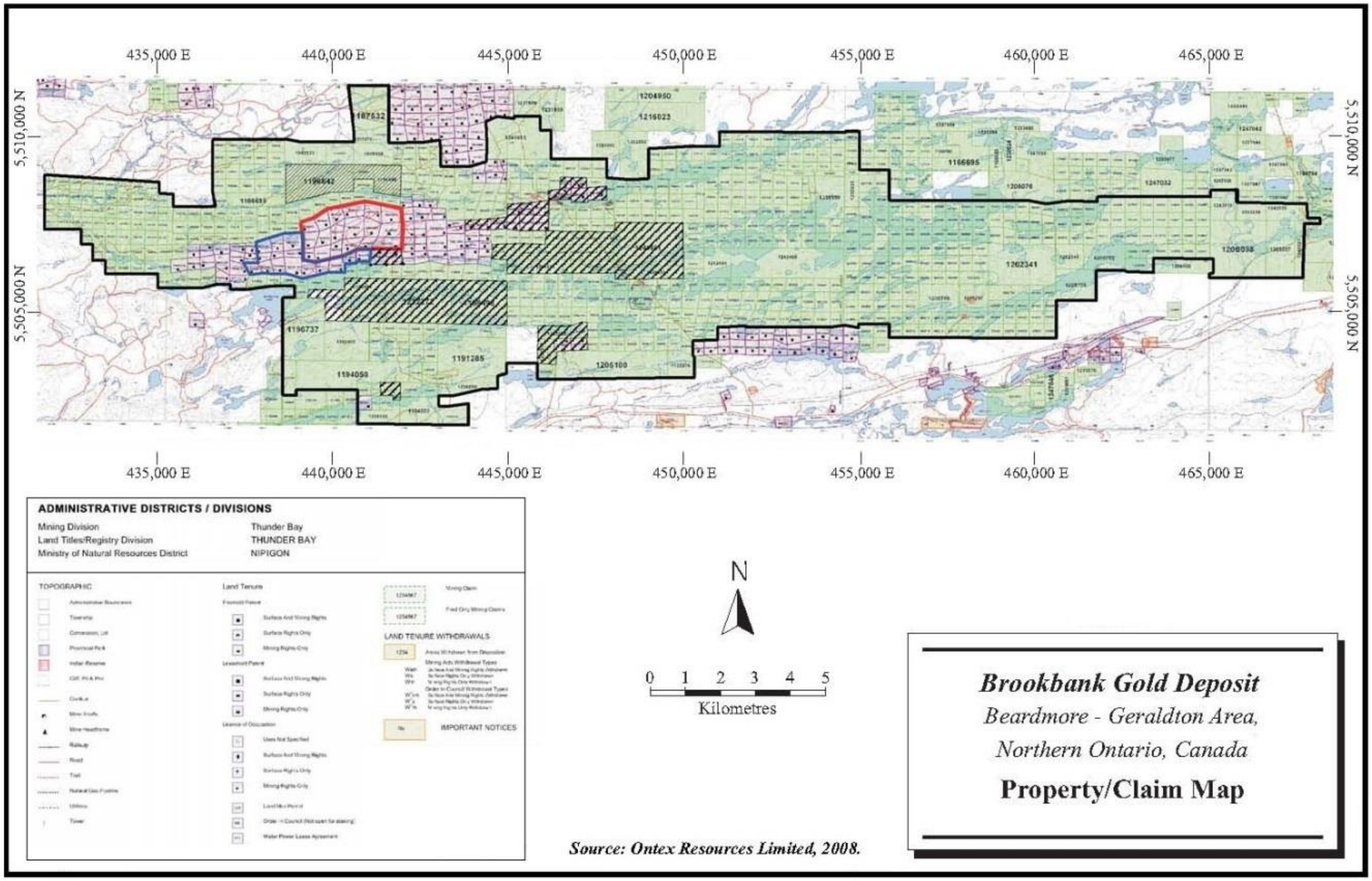
The approximate centre of the Brookbank project is located at 440 100 m E, 5 507 000 m N, using NAD 83, Zone 16 coordinates, or 49º43'N and 87º05'W using geographic coordinates. The project area is located within 1; 50,000 scale NTS map sheet 42E/12 and lies 10 km (straight distance) to the northeast of the town of Beardmore. By road, it is approximately 14 km east of Beardmore along the Trans-Canada Highway and 12 km north of the highway by gravel road. Beardmore is about 205 km by the Trans-Canada Highway northeast of the airport in Thunder Bay, Ontario.

4.3.2 Agreements and Encumbrance

The Brookbank project consists of 686 mining leases and staked claims totalling 15,080.217 ha. Figure 4.8 shows all the mining leases and staked claims for the Brookbank project.

GGM owns 100% of the eighteen leases that cover the Brookbank deposit with the remaining portion of the project leases being subject to two Joint Venture ("JV") agreements with Metalore Resources Limited ("Metalore"). The first JV is a GGM 74% / Metalore 26% split with the second a GGM 79% / Metalore 21% split.

Figure 4.8: Brookbank Project Claims Map



4.4 Key Lake

The Key Lake project hosts the past-producing Jellicoe mine.

4.4.1 Location

The Key Lake project is located 12 km west of the town of Geraldton, a few hundred metres north of the Trans-Canada Highway 11. Its geographical coordinates are 49º 41' N and 87º 31' W. Geraldton is about 275 km by highway northeast of Thunder Bay, Ontario.

As shown in Figure 4.2, the Key Lake project area is adjacent to the Hardrock Project area.

4.4.2 Agreements and Encumbrance

The Key Lake project is 100% owned by GGM. It consists of twenty-eight unpatented and patented claims and leases totalling 807.453 ha. The area covered by the patented claims, leases and unpatented claims for the Key Lake project is shown in Figure 4.2 under section 4.1.3.1.

4.5 Permits

Permits are required to undertake surface stripping and trenching and drilling when the drill site encroaches on Provincial Highway No. 11. Table 4.2 to Table 4.4 list all the permits in place for Hardrock, Key Lake, Brookbank and Viper as of October 12, 2016.

Table 4.2: Permits for Hardrock and Key Lake

Permit	Permit No.	Issued by	Effective Date	Expiry Date
Closure Plan	-	MNDM	5-Apr-12	-
ECA (Air/Noise)	9088-94LRDR	MOE	29-Apr-13	-
ECA (Dewatering)	6096-8XZPUV	MOE	23-Oct-12	-
Land Use Permit	1176-1003064	MNR	1-Jan-15	31-Dec-19
Exploration Permit (Renewal)	PR-13-10133-A2R	MNDM	16-May-16	16-May-19
Encroachment Permit	EC-2014-61T-61	MTO	16-Mar-14	16-Mar-25
Exploration Permit	PR-13-10134	MNDM	17-Apr-14	16-Apr-17

Table 4.3: Permits for Brookbank

Permit	Permit No.	Issued by	Effective Date	Expiry Date
Exploration Plan	PL-16-10610	MNDM	3-Jul-16	3-Jul-18
Exploration Permit	PR-16-10892	MNDM	15-Jul-16	15-Jul-19

Table 4.4: Permits for Viper

Permit	Permit No.	Issued by	Effective Date	Expiry Date
Exploration Plan	PL-16-10607	MNDM	30-June-16	30-June-18
Exploration Permit	PR-16-10886	MNDM	20-Jul-16	20-Jul-19

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

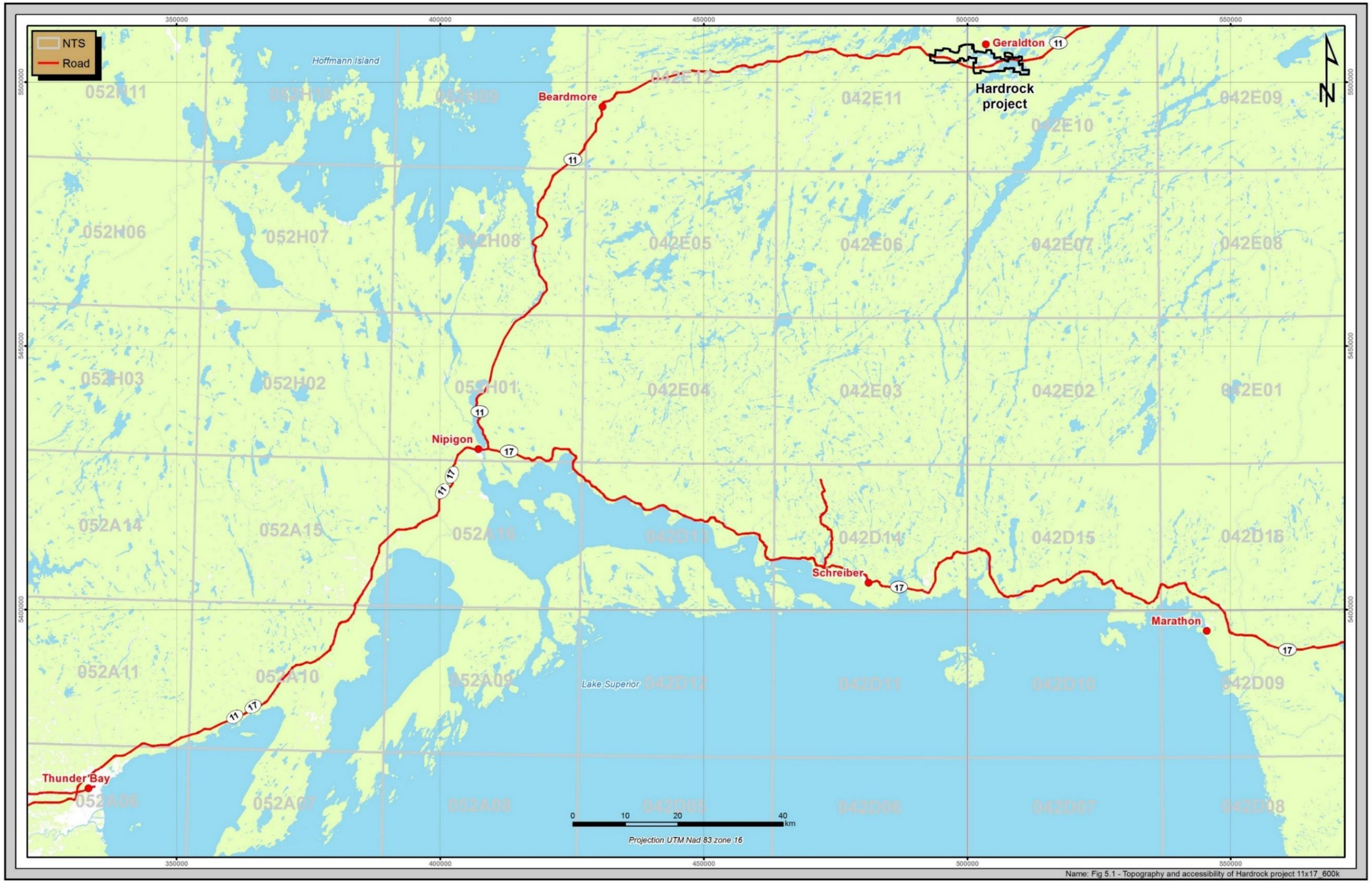
5.1 Accessibility

5.1.1 Hardrock

The Hardrock Project is located in the Municipality of Greenstone in the Province of Ontario, near the town of Geraldton. The area is accessible year-round via paved roads from Geraldton or Highway 11, which crosses the property from east to west (Figure 5.1). The closest major city is Thunder Bay, Ontario, located 275 km to the southwest, and it can be reached by Trans-Canada Highway 11. Public roads are maintained by various levels of government.

The south portion of the Hardrock Project is accessed via Highway 11. The remainder of the Hardrock Project can be easily accessed by four-wheel drive vehicles via numerous logging/bush roads that branch off of the paved highways. Drill roads provide excellent access to the areas being explored GGM. Those areas of the Hardrock Project not serviced by roads can be accessed by ATV, on foot or by boat during the summer and by snowmobile in the winter.

Figure 5.1: Hardrock Project Access Routes



Source: Innovexplo, 2015

5.1.2 Brookbank/ Key Lake/Viper

The Brookbank, Key Lake and Viper projects are also located in the Municipality of Greenstone in the Province of Ontario, between the towns of Beardmore and Geraldton, and are accessible year-round via paved roads from Beardmore/Geraldton or Highway 11 (Trans-Canada Highway).

5.2 Climate

The Project is located in northern Ontario, which has a continental climate which is typical for temperate regions in the mid-latitudes that are influenced by both polar and tropical air masses. In this climate, seasonal temperature variations are represented by short summers and cold winters.

The nearest permanent weather monitoring station is located approximately 14 km north of the Project at the Greenstone Regional Airport, which services Geraldton and surrounding area. Weather statistics for the period 1971 to 2000 indicate a mean daily temperature of 3.9°C. Temperature ranges between a maximum of 37°C and a minimum of -50.2°C with a mean annual rainfall of 546.4 mm and the mean annual snowfall of 244.5 cm. On average, precipitation was recorded for 167 days during the course of a year. The annual average relative humidity in the morning is about 83.6%. The annual average wind speed for the area is about 11.2 km/h and the most frequent wind direction, on an annual basis, is from the west. In the summer, winds blow most frequently from the west and south, while in the fall to winter the most frequent direction is from the west.

5.3 Local Resources

The Hardrock Project benefits from local human resources and services in the town of Geraldton. Geraldton has a population of approximately 1,900 people and is part of the Municipality of Greenstone, which also includes Longlac, Nakina, Beardmore and an extensive area of unincorporated territory. Greenstone itself has an approximate population of 6,000 people. Although there has been no mining activity in the immediate area since 1970, the area has a skilled workforce for the future mining activities. Geraldton has all of the services typical for a town of that size including hospital, emergency services, school, sports centre, food, lodging, wireless, and wireline telecommunications.

5.4 Infrastructure

GGM has established a field office in the town of Geraldton near the Hardrock Project itself for core logging/cutting and core storage. An independent sample preparation facility is located in Geraldton.

GGM has also established a second office in the commercial district of Geraldton for public relations.

Other significant infrastructure includes the Trans-Canada Highway, a TransCanada Pipelines Limited (TransCanada) gas pipeline, a Hydro One electrical substation, and a full service regional airport located 12 km north of Geraldton.

The Hardrock Project occurs in a mining friendly district with active mines and milling facilities located at Hemlo, Thunder Bay, Kapuskasing and Timmins with good transportation and regional mining related infrastructure.

There are adequate surface rights for the planned mining related infrastructure – waste rock storage areas, tailings management facility, processing and administration facilities - as depicted in Figure 5.2. The arrangement of mining related infrastructure will be constrained by the surrounding lakes and watercourses.

5.4.1 Roads

The Hardrock Project is accessible year-round via paved roads from Geraldton or Highway 11, which crosses the property in an east-westerly direction.

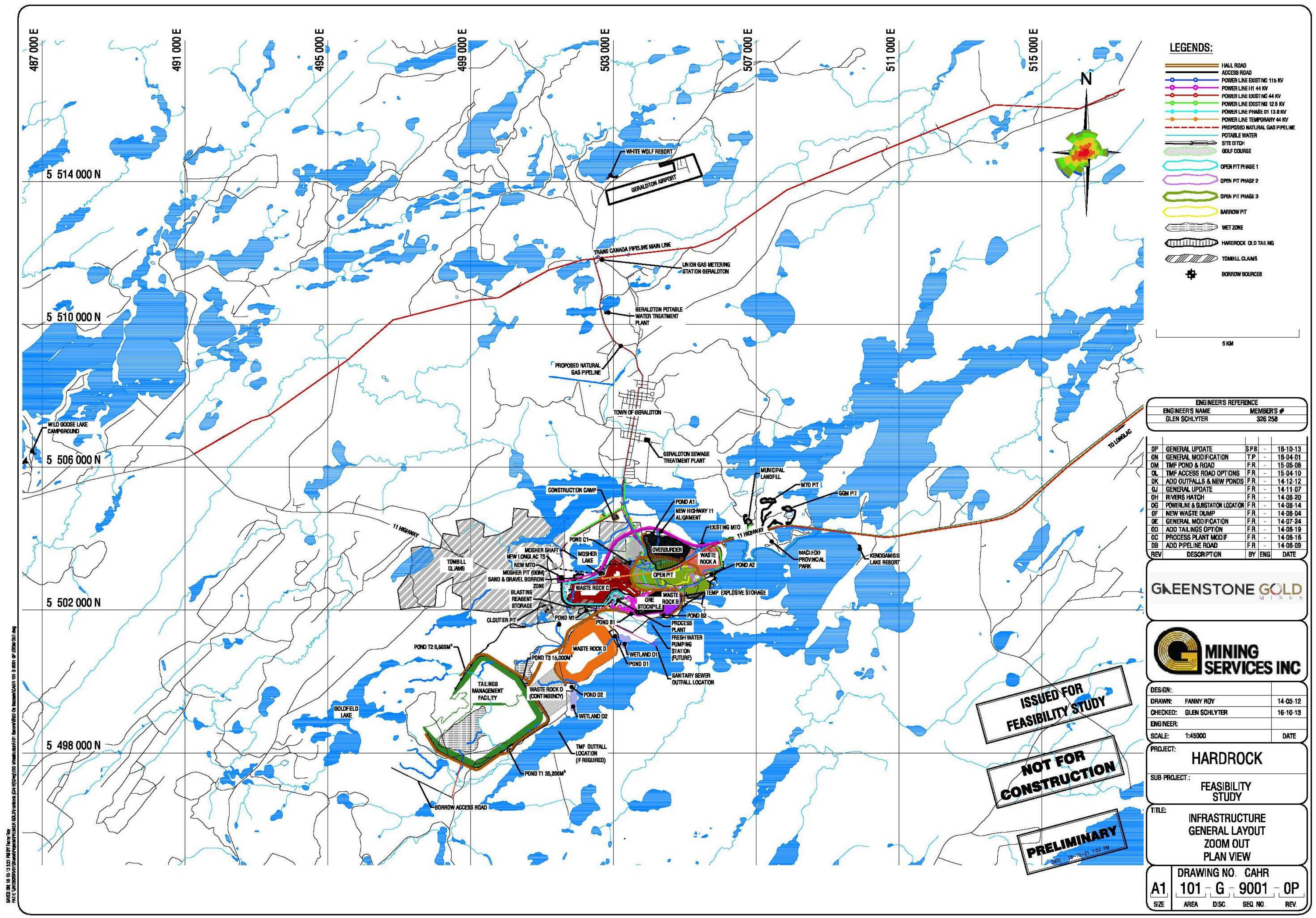
5.4.2 Natural Gas

The Hardrock Project is located approximately 9 km south of TransCanada's Canadian Mainline, an important natural gas transmission artery linking the natural gas hubs in Alberta to Canada's eastern coast (Figure 5.2).

5.4.3 Power Supply and Distribution

The future Hardrock Project is currently transected by a 115 kV transmission line, identified as A4L, which is the property of Hydro One. The A4L transmission line stretches from the Alexander SS to the Longlac TS over a distance of approximately 152 km.

Figure 5.2: Infrastructure General Layout



5.4.4 Water

The town of Geraldton has its own potable water treatment system and water distribution network. The plan is to use the Municipality's potable water for the Hardrock Project.

5.4.5 Sewage

The town of Geraldton has its own sewage treatment facility. The collecting network for sewage, however, does not come south of the Kenogamisis Lake.

5.5 Physiography

The Project lies within the Boreal Shield, a Canadian Ecozone where the Canadian Shield and the boreal forest overlap. Precambrian bedrock at or near the surface plays an important role in shaping the biophysical landscape. Lakes, ponds and wetlands are abundant in this landscape and drainage patterns are typically dendritic, with sporadic angular drainage as influenced by bedrock outcrops.

The topography in the Project area is relatively flat to gently rolling with local relief up to 20 m, largely attributed to glacial deposits that blanket the bedrock. There are no distinct topographic features that stand out in relief. Lower lying areas are characterized by swamps and ponds with overall very poor drainage throughout the area. The surrounding land has an altitude of about 335 masl. The largest lake adjacent to the Project is Kenogamisis Lake and it bounds the Project to the south, east and north. This lake elevation is about 330 masl.

6. HISTORY

6.1 Hardrock

This section provides a summary of the historical work carried out on the Hard Rock, MacLeod-Cockshutt and Mosher mines. Table 6.1 presents the statistics on gold production, diamond drilling and underground development for all three mines. A detailed chronological summary of the historical work carried out on these mines since 1980 is provided on Table 6.7.

In 1931, following the discovery of gold by W.W. Smith at Discovery Point on Kenogamisis Lake, F. MacLeod and A. Cockshutt staked the ground adjoining the Hard Rock Gold Mines Limited property to the west. Surface exploration led to the discovery of gold-bearing quartz veins in 1931. The discovery of larger mineralized zones in 1933 led to the organization of a new company, MacLeod-Cockshutt Gold Mines Limited. In 1934, shaft sinking began with the No. 1 shaft; followed by the No. 2 shaft, 600 m to the southeast, in 1936. The MacLeod-Cockshutt mine became the fifth producing gold mine in the Little Long Lac area on April 19, 1938 when a mill with a rated capacity of 600 t/d was brought into operation. In 1967, MacLeod-Cockshutt Gold Mines Limited, Consolidated Mosher Mines Limited and Hard Rock Gold Mines Limited were amalgamated to form MacLeod Mosher Gold Mines Limited. Underground operations continued until July, 1970. The mine had produced 1,546,980 ounces of gold at an average grade of approximately 0.14 oz Au/ton (4.8 g Au/t). This total accounts for about half of all the gold produced by the 10 mines in the Geraldton gold camp between 1934 and 1970.

In the 1980s, Lac Minerals Ltd ("Lac Minerals", now Barrick) undertook studies on the existing underground mineralized zones at the MacLeod-Cockshutt and neighbouring Hard Rock mines and carried out litho-geochemical sampling (Gray, 1994). Starting in 1987, Lac Minerals conducted ground geophysical surveys, followed by 77 diamond drill ("DD") holes, totalling approximately 50,000 ft (15,240 m). Targets, especially those with open pit potential, were investigated (e.g., Hard Rock D and F; North and South Porphyry; and Porphyry Hill zones). In 1992, Asarco Exploration Company of Canada Limited ("Asarco") entered into a five-year earn-in agreement with Lac Minerals and in 1993 carried out a program of reverse circulation ("RC") overburden drilling and diamond drilling, the latter largely focusing on the near-surface portion of the F-Zone and targets along the plunging nose of the albite porphyry.

Figure 6.1: Map of the Hardrock Project - 2016 Resource Estimate Area (red outline) Representing Limits of Historical Work

Source: Innovexplo, 2015 with modifications by GGM, 2016

Table 6.1: Gold Production, Diamond Drilling and Underground Development Statistics-Hardrock, MacLeod-Cockshutt, Mosher Long Lac and MacLeod Mines

	Hard Rock Mine	MacLeod-Cockshutt Mine	Mosher Long Lac Mine	MacLeod-Mosher Mine	TOTAL
Years of production	1938-1951	1938-1967	1962-1966	1967-1970	
Ore milled (short tons)	1,458,375	9,403,145	2,710,657	1,656,413	15,228,590
Ore milled (metric tonnes)	1,323,038	8,530,533	2,459,108	1,502,698	13,815,377
Au grade (oz/ton)	0.185	0.145	0.122	0.109	0.141
Au grade (g/t)	6.33	4.98	4.18	3.74	4.83
Gold ounces	269,081	1,366,404	330,265	180,576	2,146,326
Silver ounces	9,009	90,864	34,604	17,321	151,798
Total length of surface DDH (m)	14,021.4	16,933.5	1,083.0	0.0	32,037.9
Total length of underground DDH (m)	67,423.6	224,168.5	59,591.1	1,043.0	352,226.2
Total length of drifting (m)	10,572.0	32,698.9	7,292.3	7,259.2	57,822.4
Total length of crosscutting (m)	3,608.5	8,976.1	3,267.2	3,369.3	19,221.1
Total length of raising (m)	1,878.5	10,589.7	2,467.4	4,300.1	19,235.7

Source: Ferguson et al., 1971; Mason and White, 1986

As a result of this work, a geological resource was estimated for the Porphyry Hill, West and East pits as follows (Gray, 1994):

- Pit Resource: 1,920,000 short tons grading 0.079 oz Au/ton (with strip ratio, including overburden, of 4.76:1)
- Ramp Resource: 1,160,000 short tons grading 0.127 oz Au/ton

These "resources" are historical in nature and should not be relied upon. It is unlikely that they conform to current NI 43-101 criteria or to CIM definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

Asarco continued their exploration program into 1994, completing reverse circulation holes in overburden, sonic holes in tailings, and an additional 40,000 ft of diamond drilling, mostly on the aforementioned targets (Gray, 1994). Cyprus Canada Inc. ("Cyprus") assumed Asarco's role in the Lac Minerals agreement in 1996 and drilled 24 holes, leading to the discovery of the B-Zone (Mason and White, 1997).

The agreement ended in 1997. Lac Minerals began a rehabilitation program.

In 2000, Golder was retained by Lac Properties Inc. (“Lac Properties”) to conduct a stability assessment of the F-Zone crown pillar of the MacLeod-Cockshutt mine (Telesnicki and Steed, 2007). From November 27 to December 12, 2000, Golder conducted a field investigation to determine whether caving had occurred above the stoping. One investigation borehole (369.5 m) was drilled in order to perform this investigation. The study also included a literature review of the properties of the mined material at the Hard Rock mine, rock mass classification of the rock core from the investigation borehole, an empirical analysis using the Scaled Span crown pillar stability assessment, an analysis using the CPillar crown pillar stability assessment, numerical modelling to determine the stability of the crown pillar using PHASE software, and a correlation of numerical modelling results with the field investigation and conclusions.

Investigation drilling at the MacLeod-Cockshutt mine allowed Golder to confirm that the crown pillar overlying the workings was intact at the time of the study. No unravelling or caving of the crown pillar above the working was observed. Classification of the rock mass overlying the workings indicated the quality to be "good" to "very good". Empirical, analytical and numerical modelling of the stability of the crown pillar overlying the mined zone indicated that the crown pillar is stable. Due to the depth of the mine workings and quality of the rock mass, it was not considered probable that significant caving can occur or will have an influence on the overlying ground surface.

In 2002, Golder was retained by Lac Properties to conduct a stability assessment of the crown pillar of the Hard Rock Mine (Soni and Steed, 2002). A total of 16 investigation boreholes (2,116.8 m) were drilled to determine whether caving in the crown of the stope had occurred. The study comprised a literature review of the properties of the mined material at the Hard Rock Mine, rock mass classification of the rock core from the investigation boreholes, an empirical analysis using the Scaled Span crown pillar stability assessment, an analytical analysis using the CPillar crown pillar stability assessment, numerical modelling to determine the stability of the crown pillar using PHASE software, and a correlation of numerical modelling results with the field investigation and conclusions.

Investigation drilling at the Hard Rock Mine indicated that the crown pillar overlying the workings was intact at the time of the study. No unraveling or caving of the crown pillar above the working was observed by Golder and no unexpected geometries were encountered. Classification of the rock mass overlying the workings indicated the quality to be "good". Empirical, analytical and numerical modelling of the stability of the crown pillar overlying the mined zone indicated the crown pillar to be stable, even when conservative values were used for stope geometries, for strength, and for rock mass classification, thus ensuring an additional built-in factor of safety.

In 2007, six geotechnical diamond drill holes totalling 1,208.1 m were drilled by Lac Properties in the crown pillars (Murahwi et al., 2011; 2013).

Almost all drilling in 2009 was done in the vicinity of the former Hard Rock, MacLeod-Cockshutt and Mosher mines, following Premier's acquisition of Lac Properties' claims in late 2008. Premier drilled a total of 91,802 m in 346 holes, with work focused on the North Iron Formation Area, the Hard Rock-Porphyry Hill Area and the Hard Rock-East Pit Area.

There were two areas where overburden stripping and related work were carried out. The GP-Zone, located north of the Trans-Canada Highway approximately one kilometre west of the Geraldton turn-off, was stripped, washed and sampled. No mapping was done. The second area, the TAZ Zone, located approximately 1.5 km west-southwest of the Little Long Lac mine, was stripped, washed and sampled. No mapping was done.

In March 2010, Reddick et al. (2010) published a new Mineral Resource estimate for the Hardrock deposit and a supporting NI 43-101 technical report. The technical report defined the Mineral Resources as several closely spaced zones considered best suited to open pit mining. The minimum cut-off grade, block size and depth below surface used to constrain the resources were applied with the assumption of a resource with bulk mineable characteristics. Contained metal and Mineral Resource estimates are summarized in Table 6.2.

Table 6.2: Mineral Resources - Hardrock Area (Reddick et al,. 2010)

Mineral Resources Class	Tonnage (Millions of tonnes)	Cut Au Grade (g Au/t)	Tonnage (Millions of short tons)	Cut Au Grade (oz/ton)	Contained Gold, Cut (oz)
Indicated	11.6	1.82	12.7	0.053	675,000
Inferred	7.3	1.81	8.1	0.053	425,000

In 2010, three different areas on the Hardrock Project were stripped:

- The East MacLeod Zone, which is located 500 m due east of the MacLeod-Cockshutt No.1 Headframe along the Trans-Canada Highway (stripping, washing, mapping and sampling).
- The Headframe Zone, which is located at the base of the MacLeod-Cockshutt No.1 Headframe at the intersection of the Trans-Canada Highway 11 and Highway 584 (stripping and power washing).

- The Portal Zone, which is located 500 m southwest of the MacLeod-Cockshutt No. 1 Headframe (stripping, power washing, sampling). Gold grades ranged from trace values to 13 g Au/t. A structural study was carried out based on observations from the stripped outcrops and drill core.

A regional prospecting program was completed during the summer of 2010. Prospective targets were selected from regional magnetic anomalies. Prospecting covered the majority of the active claim group. Various regions of the property yielded gold values in trace amounts to 3 g Au/t.

Diamond drilling continued in 2010 on and around the old Hard Rock, MacLeod and Mosher mine sites. Drilling was accelerated in 2010 with 11 drills operating on the Hardrock Project in Q4. A total of 114,611 m was drilled in 279 holes. Some limited definition drilling was completed based on the 2009 data and, thereafter, regional exploration became a more important focus, with exploration on magnetic targets and other target surroundings historical mine sites on the property. The main zones that were drilled in 2010 were the North, F and SP-Zones. New discoveries were made, namely, the F2-Zone. The F2-Zone was originally discovered when the bottom level was drifted on the 13th level. No follow-up occurred below that level.

In 2011, Premier drilled 204 DD holes with a total length of 107,413 m. The drill program expanded the SP-Zone and F-Zone, and identified new discoveries including the high-grade Tenacity South Zone.

The zones mentioned above are described in detail in section 7.5.3

Murahwi et al. (2011) prepared an updated Mineral Resource estimate for the Hardrock deposit and a supporting NI 43-101 technical report. Contained Mineral Resource estimates in such report are summarized in Table 6.3.

Table 6.3: Mineral Resources - Hardrock Deposit (Murahwi et al., 2011)

Material	Resource Classification	Cut-off Grade (g Au/t)	Estimated Gold Grade (g Au/t)	Tonnes	Contained Gold (oz)
Open Pit	Measured	0.83	2.446	6,865,000	540,000
Open Pit	Indicated	0.83	2.280	5,833,000	427,500
Open Pit	Measured+Indicated	0.83	2.370	12,698,000	967,500
Open Pit	Inferred	0.83	2.483	615,000	49,200
Underground	Measured	2.80	5.993	2,312,000	445,800
Underground	Indicated	2.80	5.827	5,757,000	1,078,500
Underground	Measured+Indicated	2.80	5.875	8,069,000	1,524,300
Underground	Inferred	2.80	5.397	6,187,000	1,073,500
OP + UG	Measured	-	3.340	9,177,000	985,800
OP + UG	Indicated	-	4.042	11,590,000	1,506,000
OP + UG	Measured+Indicated	-	3.732	20,767,000	2,491,800
OP + UG	Inferred	-	5.133	6,802,000	1,122,700

Premier drilled 125 DD holes between January and October, 2012 for a total length of 68,549 m. Diamond drilling focused primarily on testing specific target areas of the Fortune Zone and its possible extensions, the HGN and P-Zones. The Fortune and HGN zones comprise multiple, en-echelon, narrow-vein veined zones located in close proximity to the historical Hard Rock Mine workings. The primary vein zones were identified over a plunge length of approximately two kilometres and appear to coalesce at depth but remain open further to the west.

A NI 43-101 technical report by Murahwi et al. (2013) presented an updated Mineral Resource estimate for the Hardrock deposit. Contained metal and Mineral Resource estimates are summarized in Table 6.4.

Table 6.4 Mineral Resources - Hardrock Deposit (Murahwi et al., 2013)

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Hardrock Project	Open Pit (0/P)	Measured (M)	12.737	1.41	576
		Indicated (I)	33.920	1.55	1,685
		Subtotal M & I	**46.657**	**1.51**	**2,261**
		Inferred	6.615	1.74	370
	Underground (U/G)	Measured (M)	0.315	5.84	60
		Indicated (I)	4.730	5.42	829
		Subtotal M & I	**5.045**	**5.48**	**889**
		Inferred	16.009	5.91	3,040

Between October 31, 2012 and August 9, 2013, a total of 153 DD holes (72,776.4 m) were drilled on the Hardrock deposit. These holes were included in an updated Mineral Resource estimate prepared by InnovExplo Inc. (InnovExplo) in 2013 and presented in a NI 43-101 technical report (Brousseau et al., 2013). Premier released the updated Mineral Resource estimate on October 29, 2013.

Table 6.5: Mineral Resources - Hardrock Deposit (Brousseau et al, 2013)

Resource Type	Parameters	Area		Total
		In-Pit	Underground	
	Cut-off (g Au/t)	> 0.50 g Au/t	> 3.00 g Au/t	
Indicated	Tonnes (t)	50,228,100	5,522,200	55,750,300
	Grade (g Au/t)	1.46	5.01	1.81
	Au (oz)	2,351,947	889,022	3,240,968
Inferred	Tonnes (t)	17,792,500	16,918,700	34,711,200
	Grade (g Au/t)	1.50	5.38	3.39
	Au (oz)	858,982	2,925,065	3,784,047

Between August 10, 2013 and December 31, 2013, Premier added 144 DD holes on the Hardrock deposit for a total of 66,606.7 m. None of these holes were included in the 2013 Mineral Resource estimate by Brousseau et al. (2013).

On March 2014, a PEA for the Hardrock Project was published. The study results indicated that 89,332,152 tonnes grading 1.18 g Au/t (3,392,559 oz Au) will be mined to surface over a nominal 15-year mine life (St-Laurent et al., 2014). The results of the financial analysis for the Hardrock Project indicated that the resource could be extracted at an estimated average operating cost of CAD 23.72/t and a total estimated (initial and sustaining) capital cost of CAD 767.89M. Using the consistent gold price of USD 1,250/oz and a currency exchange rate of CAD 1.00 = USD 0.95, the PEA stated the Project would generate a positive cash flow with an NPV of CAD 518.70M (discounted at 5%) and an IRR of 23% before taxes and CAD 358.97M (discounted at 5%) and an IRR of 19% after taxes.

Between January 1, 2014 and May 26, 2014, Premier added 38 DD holes on the Hardrock deposit for a total of 12,653.6 m (Brousseau et al., 2014). Thirteen DD holes from 2013 were also deepened in 2014 representing a total of 2,867.3 m of new footage. Seven historical DD holes were re-sampled to add new assay results in the 2014 Mineral Resource estimates. These holes were not previously sampled and had therefore been rejected from the 2013 database (Brousseau et al., 2013). These holes represent a total of 5,709 m of new footage in the 2014 database. InnovExplo included the new data in its updated Mineral Resource estimate presented in a NI 43-101 technical report by Brousseau et al. (2014). Premier released the updated Mineral Resource on August 25, 2014.

Contained metal and Mineral Resource estimates are summarized in Table 6.6.

Table 6.6: Mineral Resources - Hardrock Deposit (Brousseau et al., 2014)

Resource Type	Parameters Cut-off (g Au/t)	Area		Total
		In-Pit	Underground	
		> 0.50 g Au/t	> 3.00 g Au/t	
Indicated	Tonnes (t)	83,867,800	5,169,300	89,037,100
	Grade (g Au/t)	1.47	5.40	1.70
	Au (oz)	3,972,542	897,814	4,870,356
Inferred	Tonnes (t)	10,225,080	12,921,700	23,146,700
	Grade (g Au/t)	1.53	5.40	3.69
	Au (oz)	501,349	2,242,288	2,743,638

Premier carried out two small drilling programs in the area of the past producing Bankfield Mine (Brousseau et al., 2014). The Bankfield Mine is located on the Hardrock Project in the west-central part of Errington Township, extending into Lindsley Township and enclosing the southwest part of Magnet Lake. This historical mine is situated about 10 km west-southwest of the Town of Geraldton. Between December 15, 2013 and January 24, 2014, two DD drill holes were drilled for a total of 1,043 m. Between April 22, 2014 and May 17, 2014, six DD holes were added in this area totalling 2,513 m. None of these holes were included in the Mineral Resource estimate prepared by Brosseau et al in 2014.

Since June 1, 2014, Premier has been stripping in the 2014 resource area, east of MacLeod Shaft No. 1 (Brousseau et al., 2014). The work consists of three stripping areas with detailed geological mapping and channel sampling. The channels are five metres apart in the east-west direction and sampled to the extent of the outcrop every one metre. The purpose of this work was to verify and establish structural elements and grade continuity at surface.

During 2014, a total of 128 mechanical test pits were completed on the Hardrock Project to evaluate the overburden thickness. Results of these test pits were not used in the 2014 Mineral Resource estimate update (Brousseau et al., 2015).

All Mineral Resource estimates reported for the Hardrock Project in this section are superseded by the current Mineral Resource estimate in Section 14 of this Report.

Table 6.7: Historical Work Executed on the Hardrock Deposit Area since 1980

Year	Company	Activity	Comments *	Reference
1980	Long Lac Minerals Ltd.	Studies of existing underground reserves; Lithological reconnaissance		Gray, 1994
1982	Lac Minerals Ltd Mining Corporation of Canada	"Ore reserves" and "ore potential" in the Hard Rock and MacLeod-Mosher mines	- Historical "reserves" of 1,300,000 tons at 0.140 oz Au/ton (Proven and Probable ore) * 80% of total ore located below Level 13 of the Mosher winze (No. 3 shaft) Mineralization of the down-plunge of the F-Zone and South Zone	Jarvi, 1982
1987	Lac Minerals Ltd	Line cutting; Ground magnetometer, VLF EM, and IP surveys; Diamond drilling (37 DDH = 6,218.9 m)	- DDH program targeted the open pit potential of the Hard Rock D and F-Zones, North and South Porphyry, and Homestake-Hill Several IP anomalies were partially tested	Gray, 1994 2012 Premier Gold's Prospectus
1988	Lac Minerals Ltd	Diamond drilling (40 DDH = 9,052.6 m)	- DDH program targeted the open pit potential of the Hard Rock D and F-Zones, North and South Porphyry, and Homestake-Porphyry Hill	Gray, 1994 2012 Premier Gold's Prospectus
1992	Asarco Exploration of Canada Ltd Lac Minerals Ltd	Agreement between Asarco and Lac Minerals	- Asarco acquired 95 patented claims and 52 licences, including the former MacLeod-Cockshutt, Mosher-Longlac and Hard Rock mines	Mason and White, 1993
1993	Asarco Exploration of Canada Ltd Lac Minerals Ltd	106 reverse circulation overburden (RCO) drill holes (1,483.2 m); Diamond drilling (28 DDH = 5,125.2 m); Geological resource estimate	- RCO drilling program was a reconnaissance test for anomalous gold values in glacial till Diamond drilling program tested IP targets associated with iron formations and the near-surface portion of the F-Zone Pit resource: 1,920,000 tons at 0.079 oz Au/ton with strip ratio of 4.76:1 * Ramp resource: 1,600.000 tons at 0.127 oz Au/ton *	Gray, 1994 Mason and White, 1993
1994	Asarco Exploration of Canada Ltd Lac Minerals Ltd	17 reverse circulation overburden (RCO) drill holes (395.6m); 21 sonic drill holes (304.8m); Diamond drilling (78 DDH = 11,961.9 m)	- RCO drilling program was a reconnaissance test for anomalous gold values in glacial till Sonic drilling program tested the MacLeod-Mosher tailings Diamond drilling program consisted of infill drilling within a potential open pit zone (F-Zone, North Porphyry Zone, South Porphyry Zone, and No. 2 Vein) and testing of the near-surface portions of the C-Zone and North Zone.	Gray,1994

Year	Company	Activity	Comments *	Reference
1995	Asarco Exploration of Canada Ltd Lac Minerals Ltd	Pre-feasibility study; Mineral resource estimate	- Pit resource: 2,900,000 tons at 0.086 oz Au/ton * Underground resource: 1,400,000 tons at 0.131 oz Au/ton *	Reddick et al., 2010 Mason and White, 1995b
1995	Lac Minerals Ltd	Diamond drilling (7 DDH = 1,024.4 m)	- Diamond drilling program to test some of the crown pillars of old stopes in the past producing mines	Murahwi et al., 2011 and 2012
1996	Lac Properties Inc. Cyprus Canada Inc	Project joint-venture; Diamond drilling (24 DDH = 1,024.4 m); Metallurgical work on the previous sonic holes; Samples from tailings; Environmental assessment work	- Diamond drilling program defined the previous open pit area identified by Lac Minerals and Asarco	Reddick et al., 2010
1997	Lac Properties Inc. Cyprus Canada Inc.	Diamond drilling (1 DDH = 185.0 m) Geological resource estimate	- Pit resource: 9,800,000 tons at 0.047 oz Au/ton * Tailings resource: 11,200,000 tons at 0.023 oz Au/ton *	Reddick et al., 2010
2000	Lac Properties Inc.	Diamond drilling (1 DDH = 369.5 m)	- Diamond drilling program tested the F-Zone crown pillars at the past producing MacLeod-Cockshutt Mine	Telesnicki and Steed, 2007
2002	Lac Properties Inc.	Diamond drilling (16 DDH = 2,116.8 m)	- Diamond drilling program tested some crown pillars at the past producing Hard Rock Mine	Soni and Steed, 2002
2008	Premier Gold Mines Limited	Acquisition of the Lac Claims		Premier Gold
2009	Premier Gold Mines Limited	Diamond drilling (346 DDH = 91,802 m); Overburden stripping with power washing, mapping and sampling	- Diamond drilling program focused on the North Iron Formation Area, Porphyry Hill Area and East Pit Area Two areas were stripped (GP-Zone and TAZ Zone)	Premier Gold
2010	Premier Gold Mines Limited	Diamond drilling (279 DDH = 114,611 m); Overburden stripping with power washing, mapping, and sampling; Regional prospecting program	- Three areas were stripped (East MacLeod Zone, Headframe Zone and Portal Zone) Diamond drilling focused on the same area as in 2009 The main zones drilled were North, F, SP, NN, and K Discovery of the F2 and Z zones New Mineral Resource estimate and a supporting NI 43-101 technical report	Premier Gold Reddick et al., 2010
2011	Premier Gold Mines Limited	Diamond drilling (204 DDH = 107,413 m)	- Diamond drilling program resulting in the expansion of the SP, F, P and K zones Discovery of the Tenacity South Zone Updated Mineral Resource estimate and a supporting NI 43-101 technical report	Premier Gold Murahwi et al., 2011

Year	Company	Activity	Comments *	Reference
2012	Premier Gold Mines Limited	Diamond drilling (125 DDH = 68,549 m)	- Diamond drilling program focused on the Fortune, HGN and P-Zones Updated Mineral Resource estimate and supporting NI 43-101 technical report	Premier Gold Murahwi et al., 2013
2012-2013	Premier Gold Mines Limited	Diamond drilling (153 DDH = 72,776.4 m) (from Oct. 31, 2012 to Aug. 9, 2013) (144 DDH = 66,606.7 m) (from Aug. 10, 2013 to Dec. 31, 2013)	- Updated Mineral Resource estimate and supporting NI 43-101 technical report	Premier Gold Brousseau et al., 2013
2014	Premier Gold Mines Limited	Preliminary Economic Assessment	- Using the consistent gold price of USD1,250 per ounce and a currency exchange rate of CAD1.00 = US0.95, the Project generates a positive cash flow with an NPV of CAD518.70M (discounted at 5%) and an IRR of 23.02% before taxes and CAD358.97M (discounted at 5%) and an IRR of 19.02% after taxes.	Premier Gold St-Laurent et al., 2014
2014	Premier Gold Mines Limited	(38 DDH = 12,653,6 m) (from Jan. 01, 2014 to May. 26, 2014)	- Updated Mineral Resource estimate and supporting NI 43-101 technical report	Premier Gold Brousseau et al., 2014
2015	Premier Gold Mines Limited and Centerra Gold Inc.	Formation of a 50/50 Partnership	- New NI 43-101 Technical Report	Premier Gold Centerra Gold Brousseau et al., 2015

*Note: *Unless specifically indicated as reported in a NI 43-101 technical report, all "resources" listed in the table are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM definitions, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.*

6.2 Brookbank

6.2.1 Exploration History

The following summary of exploration activities on the Brookbank Project is excerpted and updated from the Scott Wilson RPA (2009, now RPA) NI 43-101 technical report, who adapted it from Thompson (2006), and is restricted to those leases and claims covering the Brookbank, Cherbourg and Fox Ear zones.

The earliest known work on the Brookbank Project is a program of surface trenching and limited diamond drilling carried out in 1934 by Connell Mining and Exploration Co. Ltd. ("Connell Mining"). A total of 17 trenches, plus numerous test pits, exposed a rusty shear zone in mafic flows over a strike length of 396 m. Gold values from samples in this zone were low and erratic, and the results for the diamond drilling are not known. Work was suspended in late 1935.

In 1944, Noranda Exploration Company Limited ("Noranda") completed detailed mapping, trenching and 1,860 m of diamond drilling in 40 holes to test the Brookbank Zone. Brookbank-Sturgeon Mines Limited ("Brookbank-Sturgeon"), a predecessor company to Ontex Resources Limited ("Ontex"), acquired the claims covering the current property in 1950; however, there is no record of the work performed (if any) by Brookbank-Sturgeon.

Between 1974 and 1975, Lynx Canada Explorations Limited ("Lynx") completed geological mapping, ground magnetic surveys and diamond drilling over a portion of the property. In 1974, Lynx carried out surface mapping and a magnetometer survey on the eastward extension of the Noranda showing. In the following year, Lynx completed six DD holes totalling 376 m to test a thin siliceous band along the metavolcanic-metasedimentary contact.

In 1981, Metalore optioned the property from Brookbank-Sturgeon and completed line-cutting followed by an electromagnetic ("EM") survey over the entire grid and a very low frequency electromagnetic ("VLF-EM") survey over selected portions of the property. Metalore subsequently drilled 30 DD holes totalling 3,567 m.

Between late 1982 and early 1983, Metalore drilled three widely spaced DD holes totalling 330 m to test the metavolcanic-metasedimentary contact on the Brookbank West property and one 453 m DD hole on the Fox Ear property.

From September 1983 to March 1984, Metalore completed an additional 62 DD holes totalling 6,946 m, including four wedges. In July 1984, Metalore commissioned a combined helicopter-borne magnetometer, gamma ray spectrometer, and VLF survey over its holdings in Sandra, Irwin and Walters townships, including the Brookbank project.

From 1984 to 1985, Metalore drilled 23 DD holes, including 14 wedges, on the Brookbank Zone totalling 4,421 m, six DD holes on the Cherbourg Zone totalling 6,684 m, and 26 DD holes on the Fox Ear Zone totalling 2,202 m.

In 1986, Metalore concentrated on the Cherbourg Zone and completed 43 drill DD holes for a total of 4,368 m. On October 1, 1986, Metalore entered into an exploration and development agreement with Hudson Bay Mining and Smelting Co., Ltd. (“Hudson Bay”).

In 1987, Hudson Bay drilled 44 DD holes for a total of 11,203 m on the Brookbank Zone and 10 DD holes for a total of 2,777 m on the Fox Ear Zone. Mineralogical studies and preliminary metallurgical testing were completed on one mineralized sample and approximately 70 drill collars were located and surveyed.

Metalore’s agreement with Hudson Bay was terminated in 1988 because of an ownership dispute between Metalore and Ontex. In October 1998, Ontex acquired a release of Metalore’s right to earn an interest in the Brookbank leases, subject to a 1% NSR due to Metalore upon production.

In July 1989, Placer Dome Inc. (“Placer”) and Metalore signed an option agreement to which Ontex was not a party. From early August to late November of that year, Placer completed a program consisting of power stripping/trenching, detailed geological mapping, channel sampling and diamond drilling. Placer exposed an area of about 650 m x 15 m and took 215 channel samples totalling 244 linear metres. Detailed mapping was completed at an Imperial scale of one inch to ten feet. During 1989, drilling at the Brookbank zone consisted of 18 DD holes totalling 7,010 m to test the lateral and down-dip extensions to a vertical depth of 670 m. A Sperry Sun gyro-log system was used to confirm downhole deviations for 13 of the DD holes drilled in 1989 and 15 of the pre-existing holes. Additional Placer drilling at Cherbourg consisted of five DD holes totalling 1,437 m with a further two DD holes totalling 984 m drilled at Fox Ear. Placer dropped its option due to ongoing litigation between Ontex and Metalore.

From 1990 through to 1996, the Brookbank project was the subject of Superior Court of Ontario litigation between Ontex and Metalore [Ontex Resources Ltd. v. Metalore Resources Ltd. (1990), 75 O.R. (2d) 513 (Gen. Div.), with an appeal allowed in part (1993) 13 O.R. (3d) 229, 103 D.L.R. (4th) 158, 12 B.L.R. (2d) 226 (C.A.)]. Costs were subsequently awarded to Ontex [(1996), 45 C.P.C. (3d) 237 (Ont. Assmt. Officer)].

Between 1993 and 1994, Metalore completed four DD holes totalling 533 m on the Brookbank Zone, fifteen DD holes totalling 2,107 m at Cherbourg and seven DD holes (including one wedge) totalling 3,323 m at Fox Ear. In 1994, reviews of the data by both Micon and J.R. Trussler & Associates, on behalf of Metalore, were positive and additional work was recommended by both companies. However, the ongoing litigation between Ontex and Metalore precluded work being done.

In October 1998, Ontex and Metalore announced a settlement whereby Ontex acquired a release of Metalore's right to earn an interest in the Brookbank leases and Ontex took over as the operator of the Brookbank deposit and all of the Metalore property in the area.

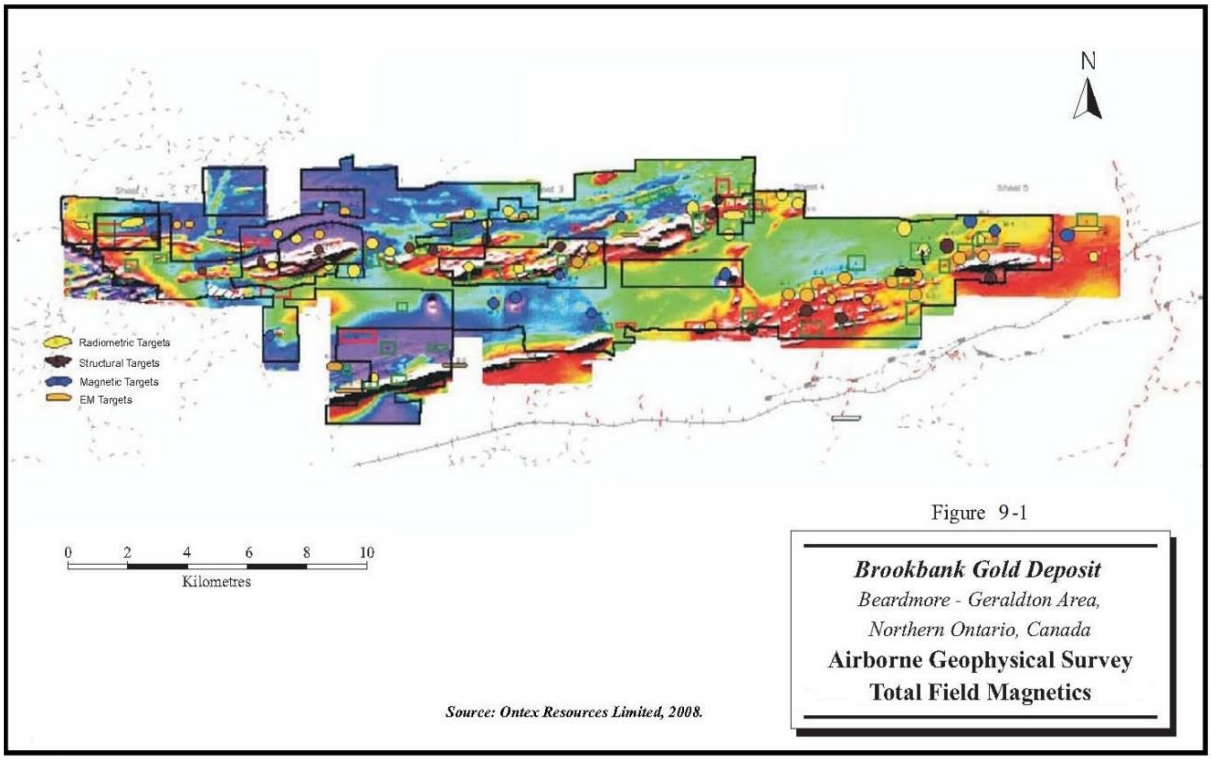
From 1999 till 2009, all exploration on the property was conducted by Ontex. The most significant of all of Ontex's exploration programs was achieved in September 1999 when Geoterrex-Dighem Ltd. completed a combined helicopter borne magnetic, VLF-EM and radiometric survey along 1,807 line kilometres over the entire property in a north-south direction. The airborne program included the collection and delivery of total field and calculated vertical gradient magnetics, VLF-EM, resistivity and radiometrics K/Th/U ratio. The results are summarized in Figure 6.2.

The airborne survey results are reflective of geology and favourable structure and alteration but are not a direct guide to mineralization. The Brookbank deposit geophysical signature is very subtle and is too subdued to be a reliable guide to the direct location of further mineralization along the favourable structural break between known gold zones. The geophysical signature, however, can be used to locate alteration on structural breaks that might contain mineralization.

The geophysical targets shown in Figure 6.2 have been used to guide the test-drilling and evaluation programs that have been completed on the Brookbank deposit to date. Almost all of the completed drilling is in the central part of the claim area. Other targets to the east and west of the Brookbank-Cherbourg-Fox Ear zone remain to be investigated in greater detail.

Since it's acquisition of the Brookbank deposit in March 2015, approximately 95% of GGMs exploration expenditures on the Brookbank deposit have been on diamond drilling, acquisition and claims protection. The details of the drilling are described in Section 10 - Drilling.

Figure 6.2: Major Helicopter Borne Geophysical Targets on the Brookbank Property



On December 18, 2009, Ontex and Roxmark announced that their respective shareholders had voted in favour of the merger transaction between the two companies. In connection with the merger, Ontex announced that the shareholders approved a one-for-three share consolidation, the election of additional directors and a name change from Ontex to Goldstone.

In June 2011, Premier and Goldstone announced that they had entered into a definitive agreement whereby Premier would acquire all of the outstanding common shares of Goldstone. Under the terms of the deal, each Goldstone shareholder would receive 0.16 of a Premier common share plus CAD 0.0001 in cash for each Goldstone share held.

On August 16, 2011, Premier completed the previously announced acquisition of Goldstone for approximately CAD 104M. The acquisition of Goldstone allowed Premier to add the Key Lake, Brookbank, Northern Empire and Leitch-Sand River projects to its portfolio of projects within the Trans-Canada Property (now called the Greenstone Gold Property) as well as add the remaining portion of the Hardrock Project it did not hold.

On March 9, 2015, Centerra and Premier announced the formation of the Partnership to explore and develop the Greenstone Gold Property.

6.2.2 Production History

There has not been any historical production from the Brookbank project area.

6.2.3 Previous Resource Estimates

Scott Wilson RPA completed a previous Mineral Resource estimate on the Brookbank project in 2009 for Ontex. This estimate is summarized in Table 6.8 and is contained in a NI 43-101 technical report dated May 4, 2009, and entitled "Technical Report on the Brookbank Gold Deposit, Beardmore-Geraldton Area, northern Ontario, Canada". The Scott Wilson RPA Brookbank Mineral Resource estimate is summarized in Table 6.8

Table 6.8: Scott Wilson RPA 2009 Mineral Resource Estimate for the Brookbank Project

Zone	Indicated Mineral Resources			Inferred Mineral Resources		
	Tonnes	Cut Au (g Au/t)	Cut Au (oz)	Tonnes	Cut Au (g Au/t)	Cut Au (oz)
Brookbank	1,217,400	8.8	345,600	813,100	7.4	192,800
Cherbourg	79,900	10.1	25,900	141,200	8.1	37,000
Fox Ear	34,500	4.3	4,700	54,200	3.7	6,500
Total	**1,331,800**	**8.8**	**376,200**	**1,008,500**	**7.3**	**236,300**

Notes:

1. *A minimum mining width of 1.5 m;*
2. *A minimum grade of 1.0 g Au/t for the Fox Ear deposit wireframe;*
3. *A minimum grade of 2.0 g Au/t for the Brookbank and Cherbourg deposits wireframes;*
4. *Grade capping was at 40 g Au/t for Brookbank, 13 g Au/t for Cherbourg and no capping for the Fox Ear deposit; assays were capped prior to compositing;*
5. *A long-term gold price of USD 850/oz and a USD/CAD exchange rate of 1.10 were used.*

The 2009 Mineral Resource estimate is compliant with the December 11, 2005 CIM Definition Standards for Mineral Resources and Mineral Reserves as required by NI 43-101 at that time. The 2009 Mineral Resource estimate is superseded by the current Mineral Resource estimate in Section 14 of this Report.

6.3 Key Lake

6.3.1 Exploration History

Drilling by Placer at Key Lake in the 1980s identified extensive zones of gold mineralization but these were initially considered too low grade to be economic (McCormack, L.V. 1984). Placer conducted additional drilling in 1990 before abandoning the project Subsequently, Cyprus confirmed two shallow mineralized shoots with average grades greater than 1 g Au/t (Gasparetto and Stevenson, 1996). Roxmark carried out some drilling in 2010/2011 and identified wide mineralized intervals, such as 1.6 g Au/t (0.047 oz Au/ton) over a drilled length of 30 m in KL-11-109 (including 11.9 g Au/t over 0.3 m). Higher grade intervals, such as 5.6 g Au/t (0.16 oz Au/ton) over 16.1 m in KL-11-112 (including 31.6 g Au/t over 1.85 m) were also encountered. There has been no drilling below a vertical depth of about 250 m.

6.3.2 Production History

The Key Lake deposit area includes the past-producing Jellicoe Mine. The Jellicoe Mine produced 5,620 oz of gold from 1939 to 1941 and an additional 55 oz in 1949 (Mason and White, 1986). The ore bodies comprised a series of veins, each with a maximum strike length of about 100 m and average width of 0.6 m. The mine workings extend discontinuously for about 1,000 m along strike at depths less than 150 m.

6.3.3 Previous Resource Estimates

There are no previously published resources for the Key Lake deposit.

6.4 Kailey

6.4.1 Exploration History

Kailey is located at the former Little Long Lac Mine. In 1917, gold was discovered in the glacial drift along the shore near the Main Narrows on Little Long Lake. In 1932, claims were staked by various individuals. Sudbury Diamond Drilling Co. drilled the area of the gold discovery and outlined a commercial ore shoot. In 1933, Little Longlac Gold Mines Ltd. was formed to develop the mine. A three compartment shaft was sunk to 137.16 m. In 1934, an electric powerline reached the mine and a 150 t/d mill was built. Between 1935 and 1940; underground development continued in the form of shaft sinking, drifting, winze sinking, cross-cutting, etc. Diamond drilling was extensive. In 1941, the discovery of scheelite in the ore resulted in handpicking of the tungsten rich material. In 1942, the underground development continued. A small mill was built to treat the tungsten. Between 1943 and 1952, the underground development continued and diamond drilling was extensive. In 1953, the mining operations continued until the end of the year. Salvage of equipment and mill clean-up followed. Between 1954 and 1956, limited production resulted from clean-up. In 1967, a new entity, also called Little Longlac Gold Mines Ltd., drilled 1,524 m to test the iron formation.

6.4.2 Production History

The Kailey project area includes the past-producing Little Long Lac Mine. The Little Long Lac Mine produced 1,615,247 tonnes at a grade of 11.7 g Au/t for a total of 605,499 oz of gold from 1934 to 1956.

6.4.3 Previous Resource Estimates

There are no previously published resources for the Kailey Project

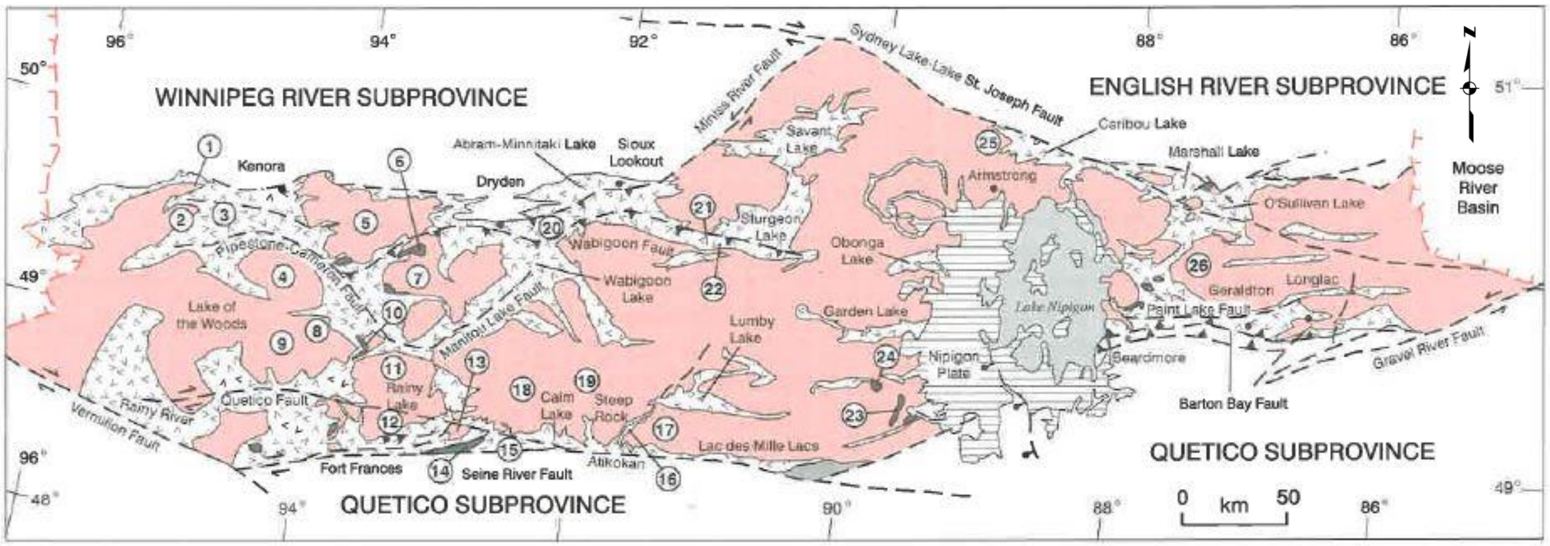
7. GEOLOGICAL SETTING AND MINERALIZATION

7.1 Hardrock Regional Geological Setting

The Hardrock Project lies within the granite-greenstone Wabigoon Subprovince of the Archean Superior craton, in eastern Canada (Figure 7.1). The Wabigoon Subprovince, averaging 100 km wide, is exposed for some 900 km eastward from Manitoba and Minnesota, beneath the Mesoproterozoic cover of the Nipigon Embayment, to the Phanerozoic cover of the James Bay Lowlands (Card and Poulsen, 1998). The Wabigoon Subprovince is bounded on the south by the metasedimentary Quetico Subprovince, on the northwest by the plutonic Winnipeg River Subprovince, and on the northeast by the metasedimentary English River Subprovince. The Wabigoon-Quetico Subprovince boundary is a structurally complex, largely faulted interface.

The Wabigoon Subprovince can be subdivided into western greenstone-rich domains in the Lake of the Woods-Savant Lake and Rainy Lake Areas, a central dominantly plutonic domain, and an eastern greenstone-rich domain in the Beardmore-Geraldton Area (Blackburn et al., 1991). Deformation and syn- to post-tectonic plutonism occurred in the interval 2711 to 2685 Ma. Based on limited geochronological data, the diverse arc-type volcanic sequences in the eastern Wabigoon Subprovince are thought to be mainly Neoarchean, some as old as 2769 Ma (Anglin et al., 1988).

Figure 7.1: Plan Map of Major Geological Elements -Wabigoon Subprovince (From Card and Poulsen, 1998)



PROTEROZOIC

Mafic intrusions (Logan sills) and clastic sedimentary rocks (Sibley Group)

ARCHEAN

Undivided plutonic rocks

Mafic, ultramafic, and anorthositic intrusions

Greenstone belt

Limit of Phanerozoic cover................

Thrust fault (teeth on upper plate).....

Strike-slip fault.................................

Normal fault (symbols on downthrown side).........

Fault..

1. High Lake stock
2. Snowshoe Bay batholith
3. Canoe Lake stock
4. Aulneau batholith
5. Dryberry batholith
6. Mulcahy gabbro
7. Atikwa batholith
8. Heronry Lake pluton
9. Sabaskong batholith
10. Katimiagamak gabbro
11. Lateral Lake stock
12. Bear Pass pluton
13. Ottertail stock
14. Bad Vermilion anorthosite
15. Seine Group
16. Steep Rock Group
17. Marmion Lake batholith
18. Irene-Eltruit batholith
19. White Otter pluton
20. Sandybeach Lake pluton
21. Pike Lake gabbro
22. Beidelman Bay sill
23. Lac des Iles intrusion
24. Tib gabbro
25. Caribou Lake gneiss
26. Onaman Lake batholith

GSC

7.2 Beardmore Geraldton Belt

The following description of the geological setting of the Beardmore-Geraldton Greenstone Belt ("BGB") is mostly modified and summarized from Tóth et al. (2013, 2014b), except where noted.

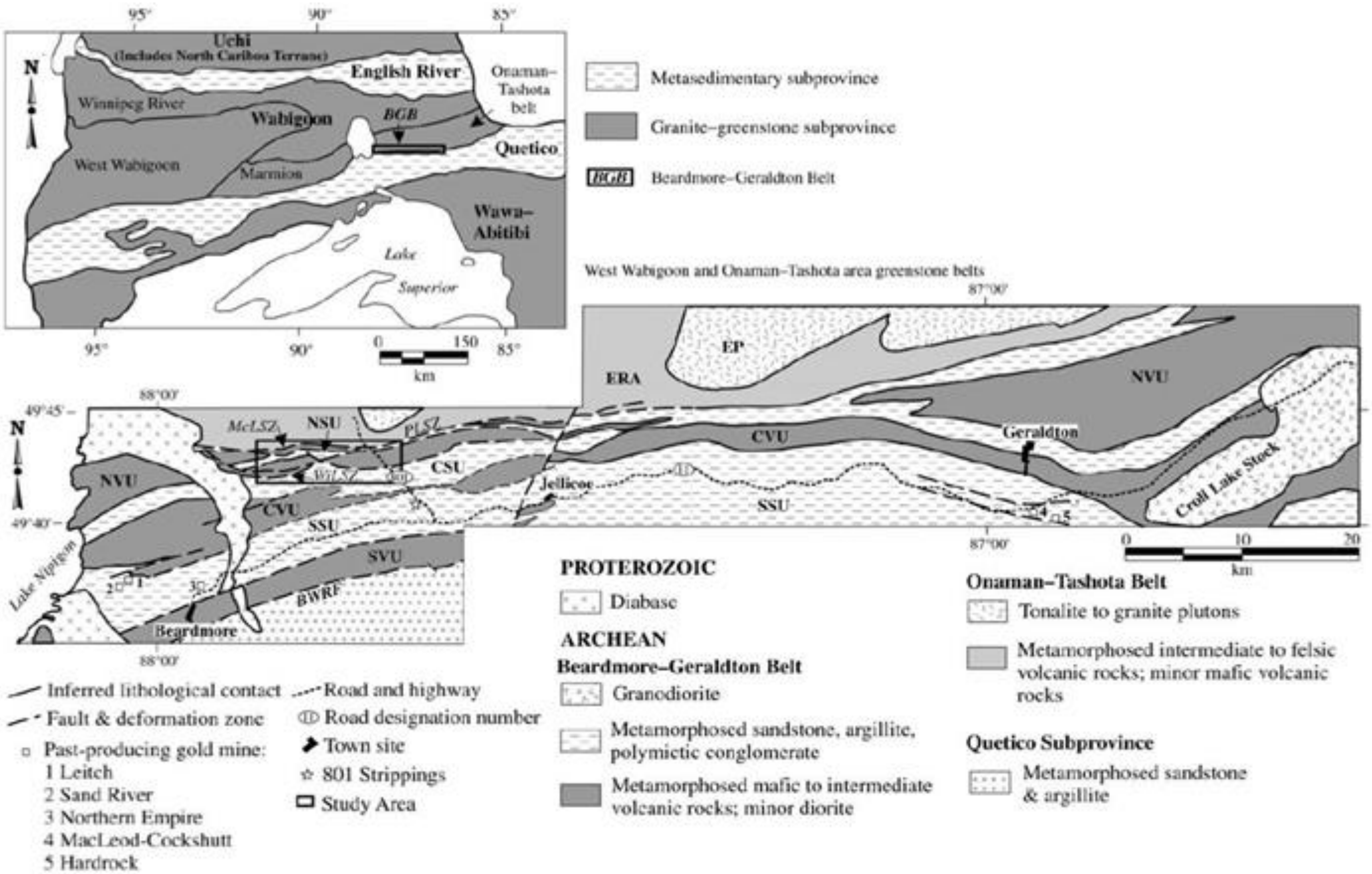
The BGB is located east of Lake Nipigon along the margins of the granite-greenstone Wabigoon Subprovince and the metasedimentary Quetico Subprovince (Figure 7.2). This 90 km long greenstone belt is composed of three metavolcanic and three metasedimentary units that are bounded by shear zones.

The BGB underwent four deformation events that are summarized in Table 7.1 (Tóth et al,. 2013, 2014a). The deformation of the belt started with D1 thrusting and the formation of isoclinal, recumbent F1 folds and strong, axial-planar S1 foliation. During D2 north-to-south compression, F1 folds were refolded by tight, upright, west-plunging, regional F2 folds, which have an east-trending, steeply dipping, axial planar S2 foliation (Lafrance et al., 2004). The last ductile deformation event recorded by these rocks was D4 dextral transcurrent faulting. Previous studies suggest that gold was emplaced during D4 dextral shear (Pye, 1952; Horwood and Pye, 1955; Anglin, 1987; Macdonald, 1988; Lafrance et al., 2004; DeWolfe et al., 2007; Lavigne, 2009). This was disputed by Tóth et al. (2013) who suggested that gold was emplaced either prior to or early during D2.

Table 7.1: Summary of Deformation and Gold Mineralization Events - Beardmore–Geraldton Greenstone Belt (Lafrance et al., 2004; Tóth et al. 2013, 2014a, 2014b).

Regional Deformation Style	Description of Structures	
	Fold	Foliation
Gold mineralization D_1 thrusting	Isoclinal, recumbent F_1 folds; up to 1 m in amplitude	Strong S_1; appears in some mafic dikes and quartz-feldspar porphyry; typically bedding-parallel in sedimentary rocks
D_2 north-south compression	Tight upright regional F_2 folds; plunge: 20°W to 70°W; amplitude up to several kilometres	East-trending, steeply dipping S_2; axial planar to F_2 folds; parallel or slightly clockwise or anticlockwise of bedding
Gold mineralization D_3 sinistral transcurrent shear	Tight to open S-shaped F_3 folds; amplitude up to tens of centimetres	East-trending, steeply dipping S_3 crenulation cleavage; axial planar to F_3 folds
Gold mineralization? D_4 dextral transpression	Z-shaped F_4 folds; plunge: 20°W to 60°W; amplitude up to several kilometres	East-northeast-trending, steeply dipping regional S_4; axial planar to F_4; regionally oriented anticlockwise to bedding
	Dextral east-trending shear zones localized along S_2 and lithological contacts	
	Z-shaped F_4' drag folds overprinting S_4 foliation in shear zones	Sinistral slip S_4' crenulation cleavage; axial planar to F_4'

Figure 7.2: Regional Geology Map - Beardmore-Geraldton Belt (From DeWolfe et al., 2007)



Note: Compiled from maps by Pye (1951), Horwood and Pye (1951), Mackasey (1975, 1976), Mackasey et al. (1976), Carter (1985), Beakhouse (1989), Kresz and Zayachivsky (1991), Kresz and Aiken (1991), Kresz (1991), Shanks (1993), and DeWolfe (2002). Inset shows the southwestern Superior Province. ERA, Elmhirst-Rickaby assemblage; EP, Elmhirst pluton; NVU, northern volcanic unit; NSU, northern sedimentary unit; CVU, central volcanic unit; CSU, central sedimentary unit; SSU, southern sedimentary unit; PLSZ, Paint Lake shear zone; McLSZ, McCambly Lake shear zone; WiLSZ, Windigokan Lake shear zone; BWRF, Black Water River Fault

7.3 Local Geological Setting (Geraldton Area)

The following discussion on the geological setting of the Geraldton Area was mostly taken from Lafrance et al. (2004) except where noted.

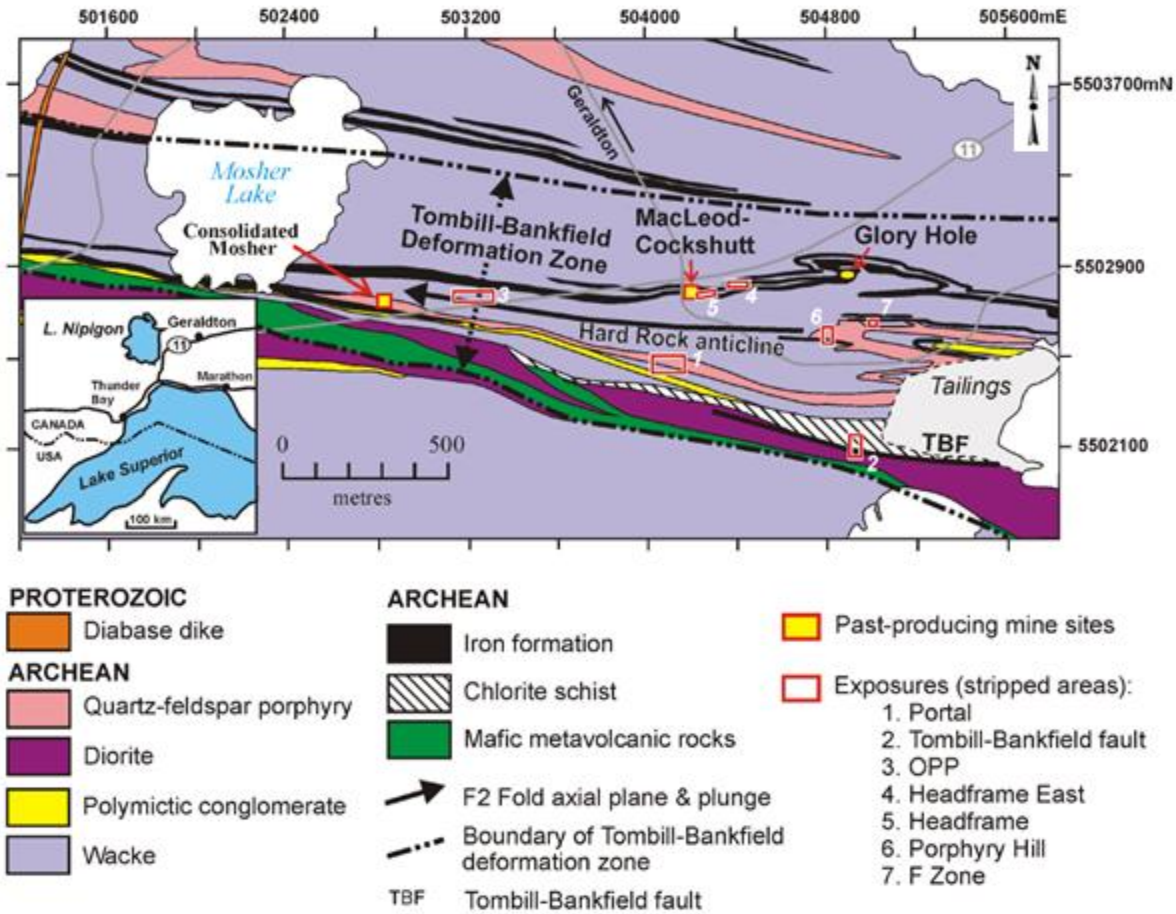
The Hardrock deposit lies within the southern sedimentary unit. The southern sedimentary unit in the Geraldton Area is characterized by multiple horizons of magnetite-rich chert banded iron formation ("BIF") within a thick sequence of interlayered sandstone-argillite and minor polymictic conglomerates. The sequence is intruded by medium- to coarse-grained diorite sills and feldspar-quartz porphyry dykes, which, together with the sedimentary rocks, are folded by tight to isoclinal, regional F2 folds, e.g., the Ellis Syncline and Hard Rock Anticline (Figure 7.3) (Pye, 1951; Horwood and Pye, 1951). The folded feldspar-quartz porphyry dykes are near parallel to lithological contacts along the limbs of the folds, but they cut across bedding and are transposed parallel to cleavage in the hinge of the folds (Figure 7.4 a).

At the MacLeod-Cockshutt Mine, several small isoclinal F1 folds in iron formation (Figure 7.4 b) are folded by parasitic F2 S-folds on the north limb of the Hard Rock Anticline. They have a strong rod-like coaxial lineation, which plunges 20°W to 40°W, subparallel to the strike of the axial plane cleavage (S2) of the regional folds. S2 strikes 100° and dips 85°SW. In the polymictic conglomerate, volcanic clasts are stretched parallel to S2 on both vertical and horizontal sections, whereas granitic clasts remained equi-dimensional to slightly elongate parallel to S2.

The Tombill-Bankfield Deformation Zone (Pye, 1952) is a one kilometre wide high-strain zone that extends from the Hard Rock and MacLeod-Cockshutt mines to the Bankfield Mine near Highway 11 (Figure 7.3). The deformation zone is parallel to S2, and it overprints the Ellis Syncline and Hard Rock Anticline. F3 folds have a steeply dipping cleavage (S3) that crenulates S2 and strikes 20° to 30° anticlockwise of S2 and transposed bedding. S2 has a strike of 100° in the conglomerate sandstone matrix away from the clasts, but as it wraps around the clasts, it changes orientation to that of the elongate granitic clasts, resulting in geometry similar to dextral asymmetrical strain shadows around rigid objects (Figure 7.4 d).

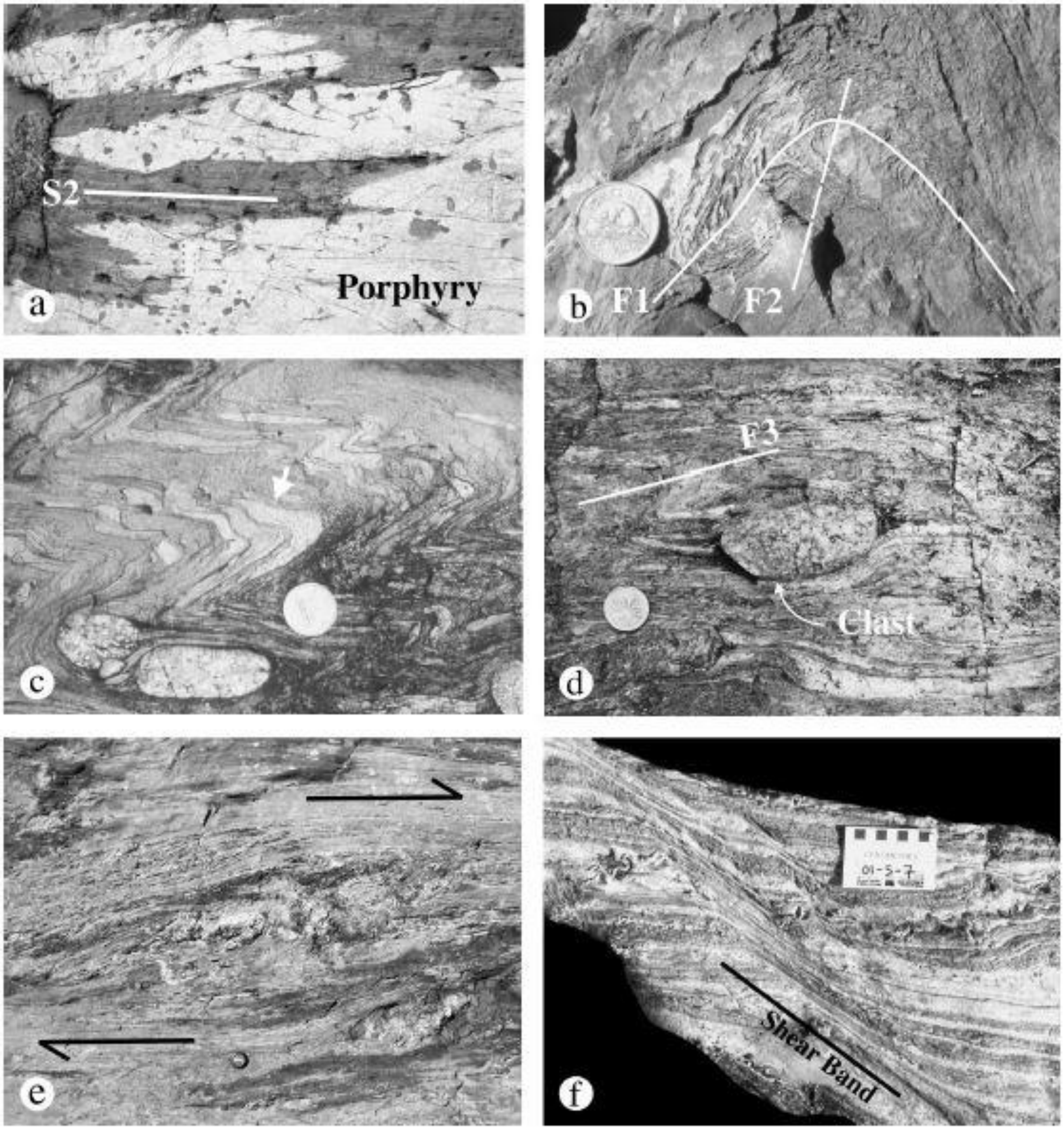
Shear is distributed heterogeneously across the Tombill–Bankfield Deformation Zone. A shallowly plunging lineation defined by millimetre-wide ridges and grooves in deformed feldspar–quartz porphyries, and by chloritic and sericitic streaks in all other rock types, occurs along S2 and is folded by F3 folds. Cascades of F3 Z-folds are bounded by localized narrow shears parallel to S2 (Figure 7.4 e). South of the Hard Rock and Mosher Mines, the shear deformation is concentrated in anastomosing, <20 m wide, chloritic shear zones cutting through dioritic intrusions and mafic volcanic rocks. Steeply dipping dextral shear bands oriented clockwise of S2 are also indicative of dextral shear (Figure 7.4f).

Figure 7.3: Geological Map of Geraldton Area



Note: (after Horwood and Pye, 1951 and Pye, 1951, with modifications by Lafrance et al., 2004, Lafrance et al., 2012, and Tòth et al. (2013, 2014b)). UTM Zone 16, NAD27 coordinate system.

Figure 7.4: Photos of Geraldton Area (From Lafrance et al., 2004)



(a) Transposed quartz–feldspar porphyry dyke in the hinge of the Hard Rock Anticline. (b) F1 overprinted by F2 on north limb of Hard Rock Anticline. Coin for scale (21 mm diameter). (c) Z F3 fold in polymictic conglomerate. S2 in volcanic clast (arrow) and conglomerate matrix are folded by the fold. Coin for scale (21 mm diameter). (d) Rotated granitic clast in polymictic conglomerate. Volcanic clasts are strongly deformed parallel to S2, which wraps around the stronger granitic clast. The long axis of the clast is parallel to the axial plane of a F3 fold. Coin for scale (18mm diameter). (e) Cascade of Z F3 folds bounded by shears parallel to S2. Camera lens has a diameter of 3 cm. (f) Dextral shear band cutting across reactivated S2 cleavage in Tombill–Bankfield Deformation Zone. Rectangular card is 9 cm in length.

7.4 Mineralization

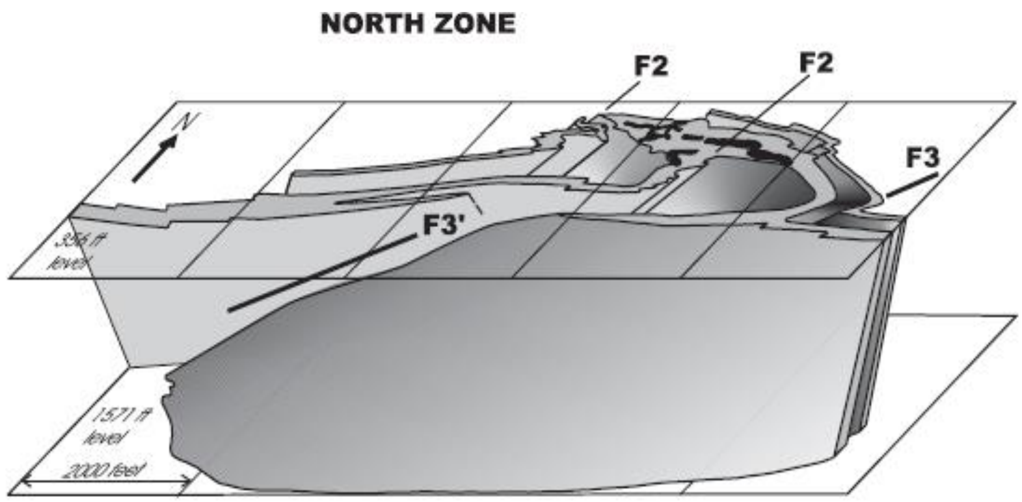
The following discussion on mineralization was taken from Smyk et al. (2005).

Gold mineralization in the BGB has resulted from the introduction of hydrothermal fluids in zones of high crustal permeability (Smyk et al., 2005). Permeability was generated by prolonged, multiple periods of deformation, which focused not only fluids, but magmatic activity and intrusions. In the Hardrock deposit area, a major zone of deformation in which the gold mines are located has been alternatively termed the Bankfield-Tombill Fault Zone (Pye, 1951; Horwood and Pye, 1951) or the Tombill-Bankfield Deformation Zone (Lafrance et al., 2004, and herein).

Most mineralized occurrences in the Hardrock deposit area lie in a zone of deformation to the immediate north of, and genetically linked to, the Tombill-Bankfield Deformation Zone. This zone of deformation varies from 600 to 100 m in total width (Figure 7.3), while the crush zone of the Tombill-Bankfield Fault proper ranges from metres to hundreds of metres in width.

Gold mineralization is associated with D3 brittle shear zones and folds overprinting regional F2 folds (Lafrance et al., 2004). The plunge of the mineralized zones is parallel to F3 fold axes and to the intersection of D3 shear zones with F2 and F3 folds. On a subprovince scale, regional folds cut by D3 dextral shear zones are promising targets for discovering the next generation of large gold deposits.

Figure 7.5: Block Diagram of North Zone at the MacLeod-Cockshutt and Hard Rock Mines showing Ore Pods in Black (From Lafrance et al., 2004)



The diagram in Figure 7.5 was drawn using level mine plans published in Horwood and Pye (1955), and shows the overprinting of an F2 S-fold by an F3 Z-fold on the north limb of the Hard Rock Anticline. Ore pods are shown in black.

7.4.1 Alteration Associated with Mineralization

The Geraldton Gold Camp is underlain by a lithologically heterogeneous package of rocks with anomalous volumes of mafic and felsic intrusions and BIF. Conglomerate occurs along the TBDZ, where most of the gold mines are located. All these rocks are highly strained and have attained lower greenschist facies metamorphism. Despite lithological constraints, it can be demonstrated that chemical alteration near the gold mines often consists of enrichment in Au, Si, K, Ba and CO_2, and depletion in Mg and Ca (Lavigne, 2009).

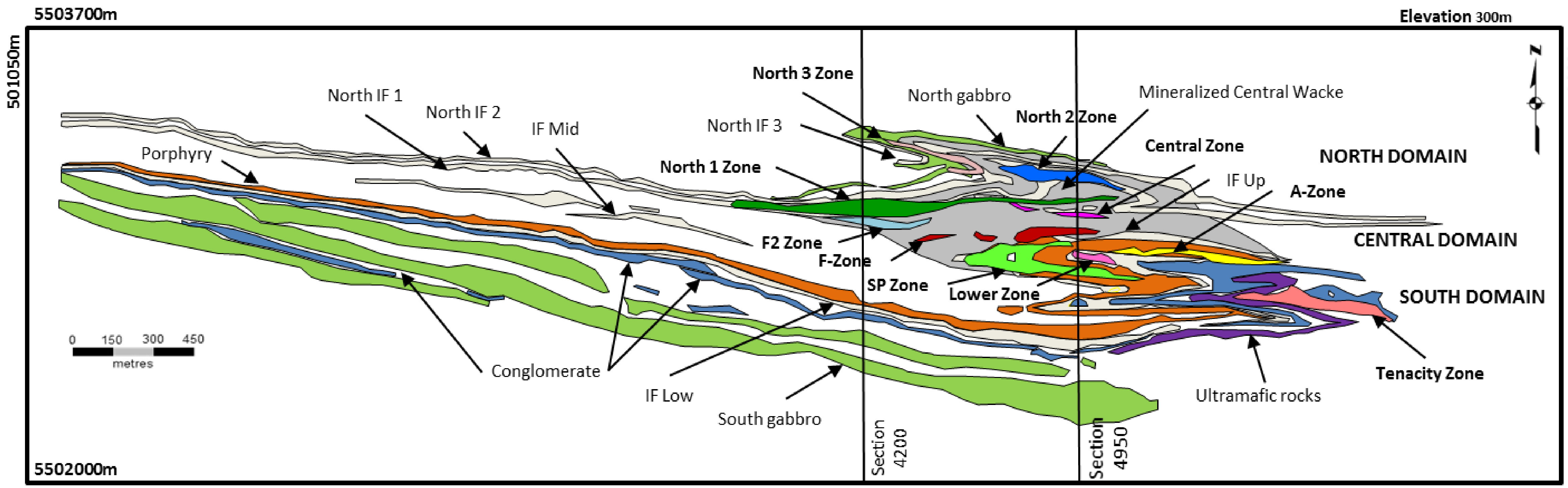
7.4.2 Identification of Gold Mineralization

The interpretation of the mineralized zones was based mainly on an update of the litho-structural model developed by InnovExplo prior to the current Mineral Resource estimate mandate. In the updated model, lithological domains and mineralized zones are located inside three distinct structural domains (Figure 7.6 to Figure 7.8).

- A North Domain consisting of a refolded (F3 overprinting F2) sequence of BIF and greywacke, with minor porphyry and gabbros. Three BIF units are present, denoted by “IF” in the unit names, interlayered with the Mineralized Central Wacke and the undifferentiated greywackes. The North Gabbro is located between the two northernmost BIF units, and has been subjected to (at least) the F3 folding episode. From top to bottom, the units are as follows:
    - North IF 3;
    - North Gabbro;
    - North IF 2;
    - North IF 1.
- In the North Domain, mineralization appears to be preferentially spatially associated with the complex refolded area affecting the BIFs and the North Gabbro. Gold mineralization occurs within all rock types but shows a preferential association with the BIFs and gabbro. The three mineralized zones are as follows:
    - North 1 Zone;
    - North 2 Zone;
    - North 3 Zone.

- A Central Domain consisting mainly of an undifferentiated greywacke sequence and a mineralized portion of this greywacke, defined as the Mineralized Central Wacke, which are both likely sheared and folded. Three mineralized zones have been defined within the Central Domain to constrain zones of higher grade gold mineralization inside the Mineralized Central Wacke. From south to north, the three mineralized zones are as follows:
    - F-Zone;
    - F2-Zone;
    - Central Zone.
- A South Domain characterized by a tightly folded (F2) stratigraphic sequence, consisting of the following units from top to bottom:
    - Upper Greywacke;
    - Mid BIF;
    - Upper BIF;
    - Porphyry;
    - Lower BIF;
    - Mid Conglomerate;
    - Mid Ultramafic;
    - Mid Greywacke;
    - Lower Conglomerate;
    - Lower Greywacke.
- Five mineralized zones have been defined within the South Domain, in which gold mineralization appears primarily associated with the “main” anticline (Hardrock Anticline) and preferentially within both BIFs. These mineralized zones are as follows (from south to north):
    - Tenacity Zone;
    - SP2-Zone;
    - SP-Zone;
    - Lower Zone;
    - A-Zone.
- The South Gabbro unit marks the southern limit of the deposit and is interpreted to be spatially associated with the Tombill-Bankfield Deformation Zone, but shows no evidence of mineralization.

Figure 7.6: Plan View of Litho-structural Model showing Mineralized Zones at Elevations 300 m and -200 m (Projection: UTM NAD 83, Zone 16



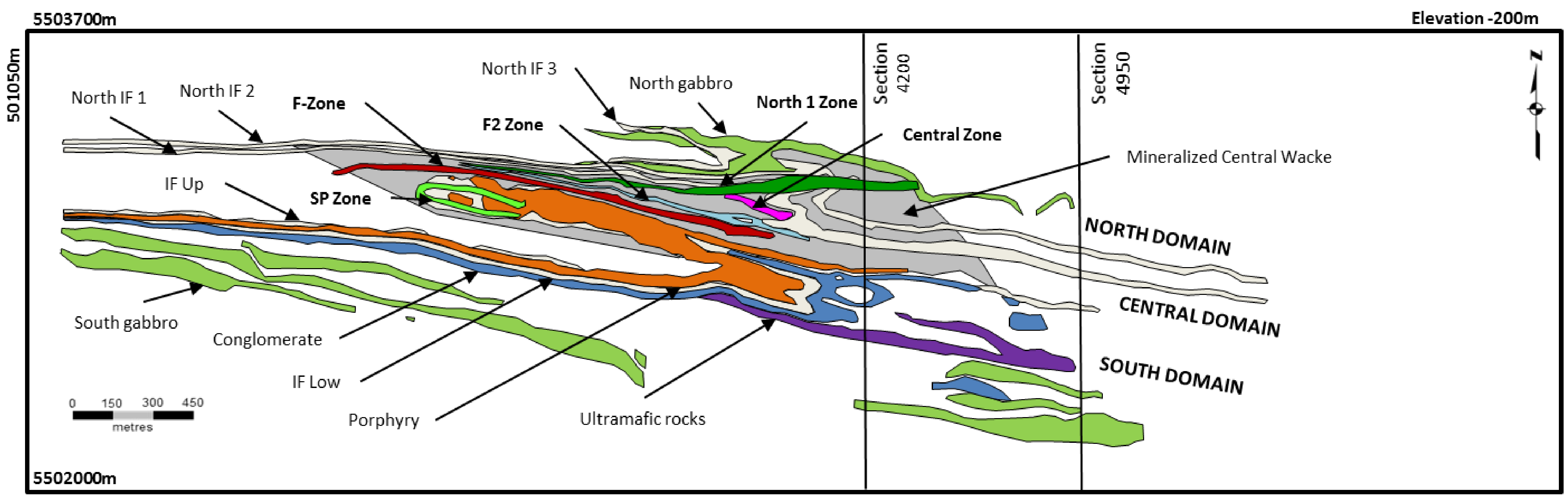


Figure 7.7 : Litho-structural Model showing Various Mineralized Zones (Cross section 4200, looking west)

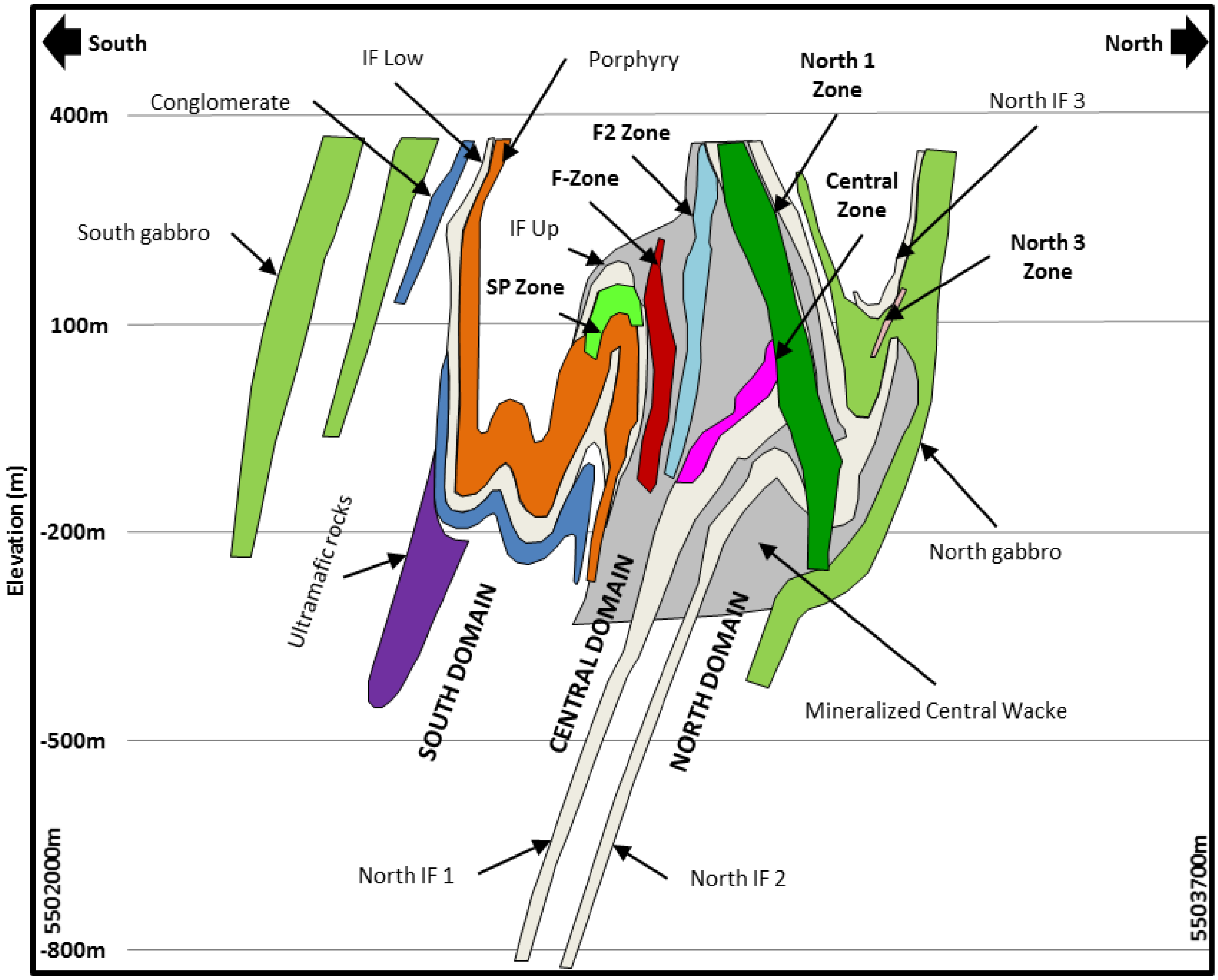
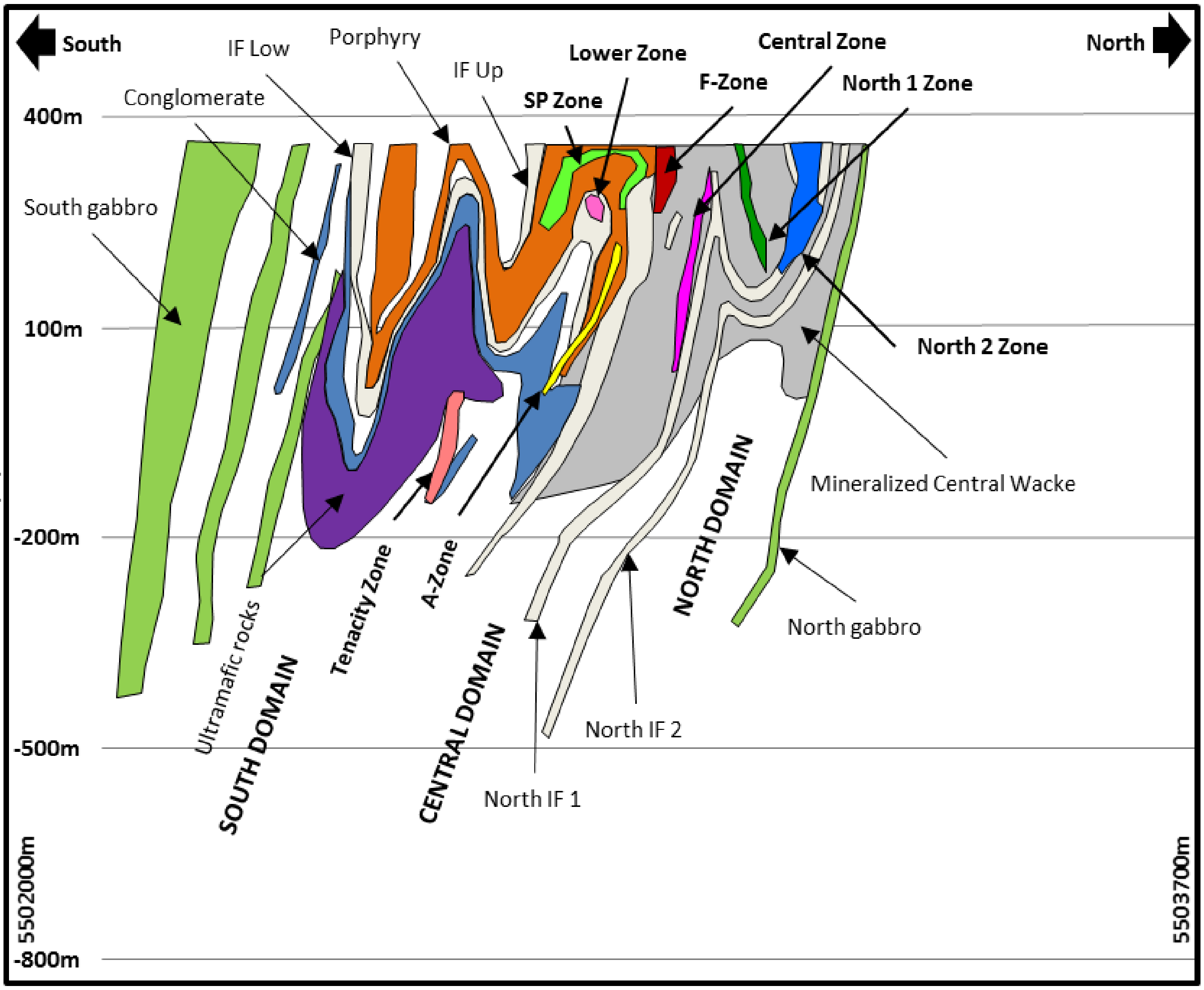


Figure 7.8: Litho-structural Model showing Various Mineralized Zones (Cross section 4950, looking west)



7.4.3 Style of Gold Mineralization

The following discussion on the style of gold mineralization was mostly taken from Davie (1995).

Quartz-carbonate Stringer Mineralization

Zones which are categorized as quartz-carbonate stringer mineralization include F-Zone, F2-Zone, A-Zone, SP-Zone, Central Zone and Tenacity Zone. Mineralization within these zones generally consists of a series of narrow, tightly asymmetrically folded gold-bearing quartz-carbonate stringers, which are usually attenuated, transposed and dislocated in hook-like segments. The stringers are accompanied by a gold-bearing quartz-sericite-pyrite (±arsenopyrite) alteration halo about the stringers. It is the accumulation of a number of stringers and associated alteration halos that constitutes the zones. Individual stringers and their associated alteration haloes within the mineralized zones are often high grade with minute flecks and clusters of visible gold. Assay results of up to, and often greater than, 30 g Au/t are attainable from some stringers. Overall, zones having average grades of 4 g Au/t as individual stringers are too narrow and discontinuous to consider mining as separate higher grade zones.

The quartz-carbonate stringers and veins display parallel to crosscutting relationships in varying lithologies; however, not unlike the sulfide replacement-type mineralization, they appear to show an affinity towards rocks with higher Fe contents. When in the sediments, the mineralized zones often occur within or proximal to lean iron formations, and variable amounts of pyrite, arsenopyrite and pyrrhotite appear to replace the Fe oxides in the quartz-sericite alteration halos about the stringers. When the mineralization occurs in porphyry, the porphyry displays a similar alteration assemblage with the sulfides having replaced the 0.5 to 1% disseminated hematite content noted in the less altered, hematite-stained porphyry.

All evidence indicates that the mineralized zones have undergone identical deformation to that displayed by the lithologies and individual veins. As a result, the mineralized zones appear to be the preserved portions of isoclinally and asymmetrically folded mineralized zones occurring at or near the hinge lines of major and minor fold axes. An understanding of this deformation is critical in determining which drill hole extrapolations have the best probability of intersecting mineralization.

Sulfide Replacement Mineralization

Zones which are categorized as sulfide replacement mineralization include the North 1, North 2 and North 3 zones, and the SP-Zone. The nature of the mineralization within these zones is best understood from the historical work completed on the North 1 Zone. Mineralization within these zones occurs as variable pyrite, arsenopyrite and pyrrhotite replacement of Fe oxide within the hinge zones of folded BIFs. The auriferous sulfide replacement appears to have migrated outwards along the iron oxide bands from gold-bearing quartz-carbonate stringers occupying brittle axial planar tension fractures. This replacement mineralization yields grades of 7 g Au/t or greater.

7.4.4 Mineralization by Zone

The following descriptions of mineralization were provided by GGM's Manager, Geology, Ben Cleland, P.Geo.

Following the initial discovery of gold at the Hard Rock Mine in 1934, and during subsequent exploration and mining over the next 80 years, many different naming systems have been used for the mineralized zones. Table 7.1 summarizes the evolution of the nomenclature.

Table 7.1: Nomenclature of Mineralized Zones at Hardrock

<table>
<tr><th></th><th>2016 Name</th><th>Former Names</th><th>Historical Name</th><th>Description</th></tr>
<tr><td rowspan="4">NORTH DOMAIN</td><td rowspan="2">North 1 Zone</td><td>New North Zone</td><td rowspan="2">North Zone</td><td rowspan="2">Iron formation sulfide replacement</td></tr>
<tr><td rowspan="2">North Zone</td></tr>
<tr><td>North 2 Zone</td><td>n/a</td><td>Iron formation sulfide replacement</td></tr>
<tr><td>North 3 Zone</td><td>North Wall Zone</td><td>n/a</td><td>Iron formation sulfide replacement</td></tr>
<tr><td rowspan="3">CENTRAL DOMAIN</td><td>F-Zone</td><td>F-Zone</td><td>F-Zone</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td>F2-Zone</td><td>Fortune (F2) Zone</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td>Central Zone</td><td>n/a</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke</td></tr>
<tr><td rowspan="5">SOUTH DOMAIN</td><td>Tenacity Zone</td><td>Tenacity Zone</td><td>B-Zone</td><td>Quartz-carbonate stringers in greywacke and conglomerate</td></tr>
<tr><td>SP2-Zone</td><td rowspan="2">SP-Zone</td><td>n/a</td><td>Quartz-carbonate stringers in greywacke and minor Iron formation sulfide replacement</td></tr>
<tr><td>SP-Zone</td><td>South Zone / Trench Zone</td><td>Quartz-carbonate stringers in porphyry and greywacke and minor Iron formation sulfide replacement</td></tr>
<tr><td>Lower Zone</td><td>P-Zone</td><td>P-Zone</td><td>Quartz-carbonate stringers</td></tr>
<tr><td>A-Zone</td><td>A-Zone</td><td>A-Zone</td><td>Quartz-carbonate stringers in greywacke and lesser porphyry</td></tr>
</table>

7.4.4.1 North Domain

North 1 and 2 Zones

The North 1 and North 2 zones both represent two main types of mineralization, fracture filling and replacement. They are characterized by the presence of massive sulfides, but the fracture filling type contains greater amounts of quartz and carbonate.

The North 1 Zone is an amalgamation of mineralized areas of the historical North Zone located at the Z-fold hinge of the main iron formation, and the New North Zone located further west.

The North 2 Zone is located along the northern synclinal limb of the historical North Zone and encompasses the majority of its mined resource.

North 3 Zone

Mineralization is primarily quartz-carbonate stringers concentrated at the synclinal hinge contact between the upper iron formation and the northern gabbro and enveloping greywacke. Gold mineralization is focused in areas with intercalated bands (1 to 50 cm wide) composed of all three lithologies, indicating tight isoclinal folding. Mineralization is accompanied by moderate chlorite and sericite alteration in the gabbro and greywacke, and weak to moderate fuchsite alteration in the gabbro. Mineralization is associated with arsenopyrite and pyrite sulfides in all three lithologies.

7.4.4.2 Central Domain

F-Zone

The F-Zone mineralization lies proximal to the northern contact between the quartz-feldspar porphyry and greywacke. Gold mineralization is associated with trace to 5% pyrite and lesser arsenopyrite and pyrrhotite and moderate to minor sericite, chlorite and carbonate alteration.

F2-Zone

The F2-Zone horizon is composed of multiple, en-echelon, narrow vein zones located between the F-Zone to the south and the North 1 Zone to the north. Gold mineralization is associated with trace to 5% pyrite, with lesser arsenopyrite and pyrrhotite, and moderate to minor sericite, chlorite and carbonate alteration.

Central Zone

The Central Zone is a lens within the greywacke envelope adjacent to the North 1 Zone and subparallel to the south limb of the North IF-1 unit. Similar to the F2-Zone, the Central Zone is characterized by quartz-carbonate stringers with trace to 2% pyrite and lesser arsenopyrite, hosted in greywacke with moderate to minor sericite, chlorite and carbonate alteration.

7.4.4.3 South Domain

Tenacity Zone

The Tenacity Zone is marked by moderately to intensely silicified and veined greywacke host rocks, adjacent to folded altered ultramafic and conglomerate units. Gold mineralization is associated with trace to 5% pyrite and lesser pyrrhotite and arsenopyrite, and accompanied by sericite and chlorite alteration in sediments or talc and serpentine alteration in ultramafics.

SP and SP2-Zones

The mineralization is partly quartz-carbonate stringer and partly sulfide replacement, and occurs at the contact between the porphyry and the lean iron formation/greywacke unit of the southern limb of the main porphyry anticline. The mineralization is located along the southern limb, proximal to the hinge of a parasitic asymmetrical Z-fold of the contact. Quartz-carbonate stringer mineralization is predominantly found in the porphyry and greywacke and is associated with trace to 5% pyrite and lesser arsenopyrite. Sulfide replacement mineralization is localized at the contact margins between porphyry and iron formation, and consists of 2 to 10% blebby pyrite.

Lower Zone

Mineralization is primarily quartz-carbonate stringers located in the hinge of the Lower BIF with intercalated greywacke. Gold mineralization is associated with trace to 5% pyrite as stringers and blebs, contained in veinlets with 10 to 30% quartz and carbonate. Alteration is strong to moderate chloritization. The mineralized zone is often crosscut by moderately chlorite- and fuchsite-altered gabbro.

A-Zone

The mineralization consists mainly of gold-bearing, irregularly folded, quartz-carbonate stringers that are generally less than 10 cm wide. Most of this gold occurs freely in the quartz-carbonate stringers, although some is associated with pyrite. The mineralization occurs within a folded and fractured greywacke and conglomerate, and stops in the northern limb of the porphyry. Gold mineralization is associated with trace to 10% pyrite and lesser arsenopyrite, accompanied by carbonate and sericite alteration.

7.5 Other Greenstone Gold Property Deposits (Brookbank, Key Lake and Kailey)

7.5.1 Regional Geological Setting

The regional geological setting described in Subsection 7.1 for the Hardrock Project and summarized in Figure 7.1 is applicable to the Greenstone Gold Property (formerly the Trans-Canada Property) as a whole, which includes the Brookbank, Key Lake and Kailey Projects.

7.5.2 Brookbank Project Local Geology

The Brookbank project geology is summarized in Figure 7.9.

The following is an excerpt from the 2013 Technical Report by Micon.

The Brookbank project is underlain predominantly by east-west trending and steeply south to vertically dipping metavolcanic and metasedimentary rocks. Metavolcanic rocks consist of massive and pillowed, locally amygdaloidal, flows of basaltic composition along with related tuffaceous rocks. Pillowed flows exhibit tops to the north. They are locally intercalated with coarser-grained rocks of similar composition that have been interpreted as either intrusions or coarse-grained volcanic phases at the centre of thicker basaltic flows. The metavolcanic rocks are locally intruded by quartz-feldspar porphyritic dykes.

Mafic metavolcanic rocks are fault-bounded against domains of metasedimentary rocks. The northern domain consists of a polymictic conglomerate with pebble- to boulder-sized, rounded to sub-rounded clasts in a feldspar-quartz-sericite matrix. Clasts consist of volcanic and intrusive rock types of various compositions, quartz pebbles and jasper, the latter suggesting affinity with Timiskaming Formation conglomerates in the Timmins (Porcupine) Mining District.

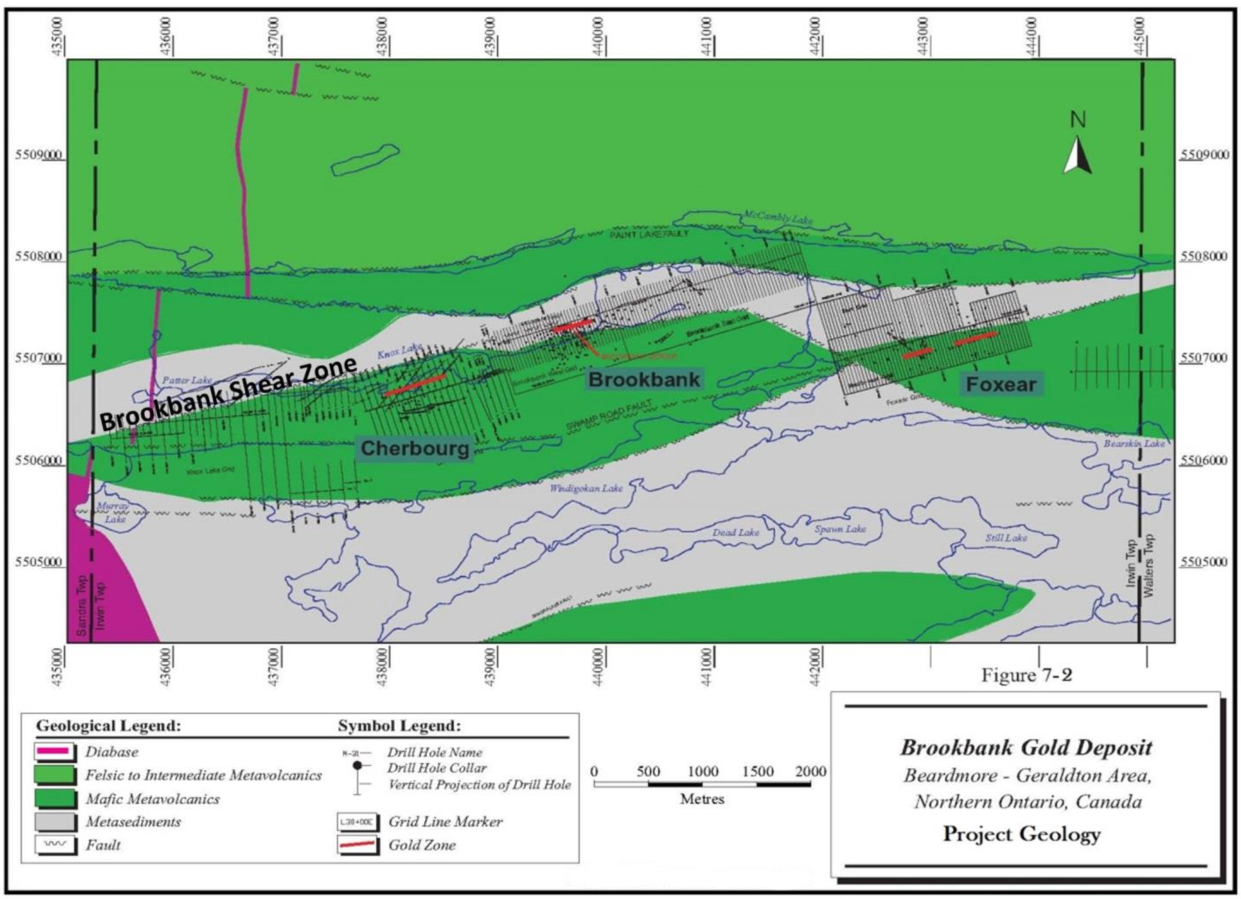
Metasedimentary domains south of Windigokan Lake also contain polymictic conglomerate as well as feldspathic and quartzose sandstone and wacke, siltstone, minor argillite and hematitic iron formation.

Felsic to intermediate pyroclastic rocks and flows occur in the north part of the property and are fault-bounded with mafic metavolcanic rocks across the Paint Lake Fault. They consist of tuff breccia, pyroclastic breccia and tuff, and massive to porphyritic rhyolite flows.

Intermediate to mafic intrusions cut the metavolcanic and metasedimentary rocks in the central part of the Brookbank property. They consist of quartz diorite, diorite and gabbro. North-trending, flat-lying, locally porphyritic diabase dykes of Keweenawan age cut the metavolcanic and metasedimentary rocks along the western boundary of the property in Sandra Township and along the western boundary of Irwin Township.

The Brookbank project is transected by an east-west trending zone of extensive heterogeneous brittle and ductile deformation and hydrothermal alteration, which is referred to as the "Brookbank shear zone" (Figure 7.1). Deformation is locally in excess of one kilometre wide and consists of anastomosing bands of intense fissile shearing, quartz veining and fracturing with associated ductile deformation around domains of less deformed metavolcanic and metasedimentary rocks. The deformation can be traced for a minimum of 10 km along strike through Irwin Township and remains open in either direction.

Figure 7.9: Brookbank Project Geology Map



Source: Modified after Ontex Resources Ltd, 2008.

7.5.3 Brookbank Project Mineralization

The 6.5 km long Brookbank shear zone hosts the Brookbank, Cherbourg, and Fox Ear deposits (Figure 7.9). The deposits occur along lithological contacts between mafic volcanics and metasediments.

Other areas of gold mineralization are present in one or more of the localized deformation bands within the hanging wall mafic volcanics, which are generally parallel to the Brookbank main zone within the Brookbank shear zone structure.

The zones of mineralization at Brookbank, Cherbourg and Fox Ear occur within one of several bands of intense deformation and hydrothermal alteration at or near the contact between domains of mafic flows and polymictic conglomerates. Hydrothermal alteration accompanying the mineralization consists of silicification, carbonatization, sericitization, chloritization, hematization and sulfidization (Figure 7.10). This alteration is commonly marginal to the mineralized quartz-carbonate veins, fractures and stockworks and may exceed 50 m in width locally.

Figure 7.10: Exposure of the Brookbank Mineralized Corridor showing Intense Hydrothermal Alteration

Micon, 2013

Mineralogical studies indicate that the precious metal mineralization consists of gold-silver particles with an approximate gold to silver ratio of 80:20. The gold occurs primarily as late fracture-controlled mineralization. The mineralization forms elongate lenticular particles associated with grain boundaries and possibly crystallographic planes. The gold generally consists of fine grained free gold particles, although there is very little visible gold even in areas of plus 30 g Au/t assays. Gold values are highest in the quartz–carbonate veinlets/stringers.

Sulfide mineralization (pyrite and minor chalcopyrite) is also present within the sheared host rock and quartz veinlets.

7.5.4 Key Lake Project Local Geology

The Key Lake project is located within the Beardmore-Geraldton Greenstone Belt of the Wabigoon Subprovince of the Superior Province. The project area is within the southern metasedimentary sub-belt on the southern limb of a west-plunging syncline. The mineralized zone at Key Lake is 550 to 800 m northeast of the Tombill-Bankfield Fault and diverges from it toward the west. It is about 2.5 km south of the contact with the central metavolcanic sub-belt.

Metagreywacke is the predominant rock type in the area and occurs in a series of turbidites. A thick section of fine to coarse-grained altered wacke hosts most of the gold mineralization. A bed with granule- to pebble-size clasts may be a matrix-supported metaconglomerate or a vitric lapilli tuff. Magnetite-rich argillite occurs to the north and south of the mineralized zone. BIF's occur further north.

The metasedimentary rocks have been intruded by one or more thin (0.5 to 3 m) porphyritic aphanitic felsic dykes which are spatially related to gold mineralization. Gabbro and diorite dykes occur in some areas and Proterozoic diabase dykes crosscut all other rock units.

7.5.5 Key Lake Project Mineralization

Gold occurs in altered metagreywacke (arkose), felsic dykes and in thin veins cutting these rocks. Gold-bearing altered rocks typically have more than trace amounts of pyrite and/or arsenopyrite. Mason and White (1986) reported sphalerite and silver. Accessory chalcopyrite has been identified in some holes. A variety of veins are present including quartz with angular bits of white carbonate typically along vein margins, white and grey massive quartz, and dark grey veinlets usually less than 3 mm thick composed of quartz and/or very fine grained arsenopyrite. Visible gold occurs in veins in both metagreywacke and felsic dykes but is not common and rarely occurs in wall rock.

Alteration occurs within and extends beyond the zone of gold mineralization. Widespread dolomite/ankerite alteration was detected by staining (Gasparetto and Stevenson, 1996).

7.5.6 Kailey Project Local Geology

The local geological setting described in Subsection 7.3 for the Geraldton area and summarized in Figure 7.1 is applicable to the Kailey project.

The Kailey deposit is located at the former Little Long Lac Gold Mine, about 1.7 km north of the Hardrock Mineralized Corridor. It lies within a broad synclinal belt of greywacke, slates, conglomerates and iron formation that extend westwards to Lake Nipigon. The sediments overlie a thick series of lavas, and both are intruded by igneous rocks of various ages and types. At Little Long Lac Gold Mine, the sediments follow a westerly pitching drag fold on the northern limb of the syncline. Subsequent to the folding, east-west zones of shearing developed and formed channel ways for gold-bearing solutions.

8. DEPOSIT TYPES

8.1 Hardrock

8.1.1 Epigenetic Banded Iron-Formation-Hosted Gold Deposits

This discussion on deposit types was mostly taken from Kerswill (1993). Gold deposits in the Hardrock deposit area are classic examples of epigenetic non-stratiform BIF-hosted gold deposits (historical North Zone and West Zone).

Important common features of BIF-hosted gold deposits include a strong association between native gold and iron sulfide minerals, the presence of gold-bearing quartz veins and/or shear zones, the occurrence of deposits in structurally complex terranes, and the lack of lead and zinc enrichment in the ores.

8.1.2 Non-Stratiform Type

In non-stratiform deposits, gold is restricted to late structures (quartz veins and/or shear zones) and/or sheared sulfide BIF immediately adjacent to such structures. Mineralization is confined to discrete, commonly small, shoots separated by barren (gold- and sulfide-poor), typically oxide BIF. Mineralized rocks are generally less deformed than associated rocks. Iron-sulfide minerals are in many cases relatively undeformed and unmetamorphosed. Pyrite and/or sheared pyrrhotite have clearly replaced other pre-existing iron-rich minerals, notably magnetite. Arsenic-bearing minerals are common, but not always present. If they are present, a strong positive correlation generally exists between gold and arsenic. Alteration is usually typical of that associated with "mesothermal vein" gold deposits. Mineralization is relatively silver-poor, and gold grains generally have gold/silver ratios of >8.0. Non-stratiform deposits are relatively common, typically small and, compared with stratiform deposits, difficult to evaluate and mine. Examples of non-stratiform deposits are the North ore zone at the MacLeod-Cockshutt Mine, the Central Patricia mine and portions of the Pickle Crow mine (all in Canada), numerous deposits in Western Australia, including Hill 50, Nevoria and Water Tank Hill mines, and several deposits in Zimbabwe, including the Lennox Mine.

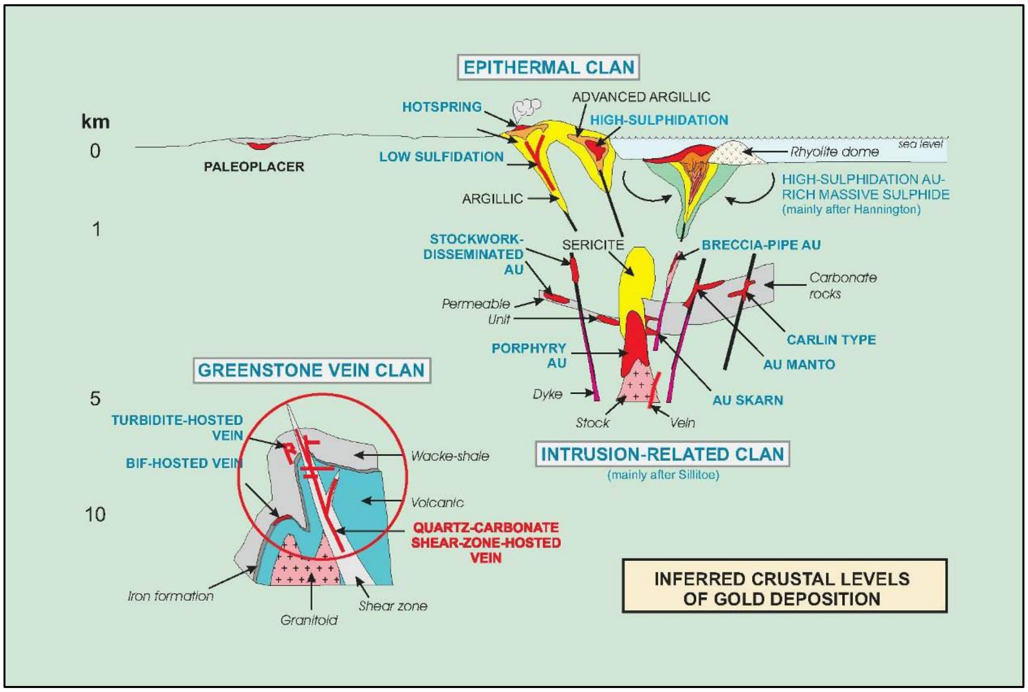
Non-stratiform deposits contain sulfide-rich alteration zones immediately adjacent to late structures and are similar to mesothermal vein-type gold deposits. Late quartz veins and/or shear zones are present in most known BIF-hosted gold deposits. The distributions of gold-bearing veins and sulfide-rich zones are commonly controlled by fold structures. Major faults ("breaks") of regional scale have been recognized near many non-stratiform deposits.

Irregular, massive lenses of sulfides and quartz occur in a folded series of greywacke and iron formation in the Hard Rock and MacLeod-Cockshutt mines (Horwood and Pye, 1951). These massive replacement lenses (up to 65%, sulfides) cut the Z-folded iron formation and are related to quartz-carbonate veins up to 0.6 m wide. Veins are usually barren of gold mineralization except where they contain sulfides, consisting primarily of pyrite, arsenopyrite and pyrrhotite. Mineralization is preferentially concentrated in the wall rocks outward from the quartz veins and ore is locally banded due to the selective replacement of the less competent wacke laminae in the iron formation by sulfides. The main ore zone (the North or No. 30 Zone, and the West Zone), mined in the Hard Rock and MacLeod-Cockshutt mines, was of this type (Horwood and Pye, 1951). The grade from these zones was generally higher than the grades in the larger F-Zone (associated with greywacke).

8.1.3 Greenstone-Hosted Quartz-Carbonate Vein Deposits

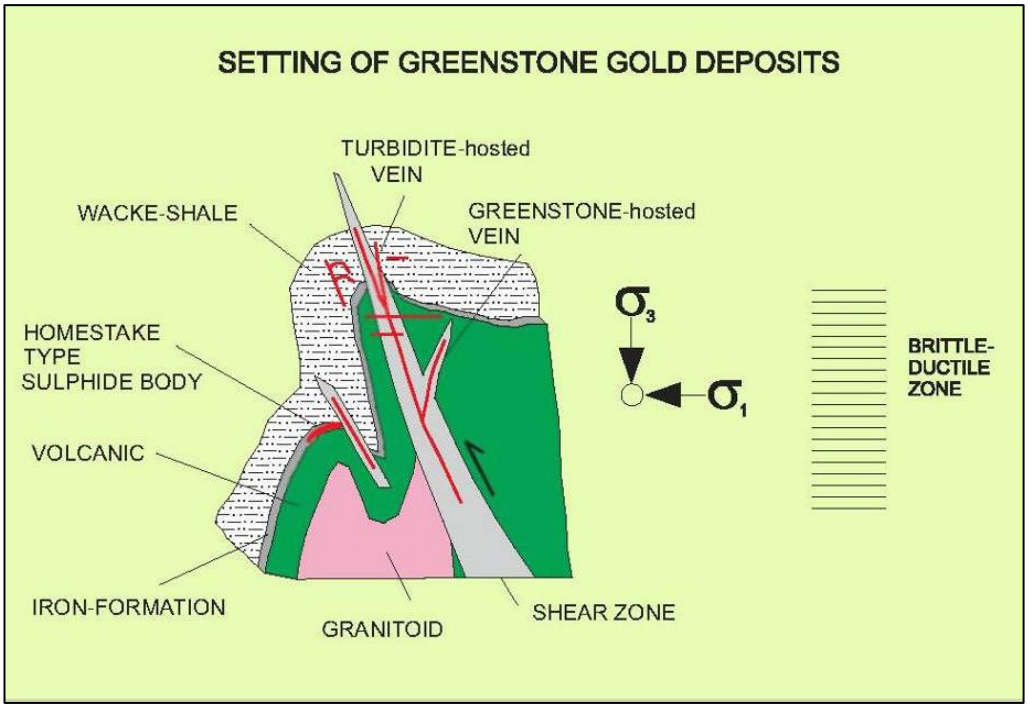
Greenstone-hosted quartz-carbonate vein deposits occur as quartz and quartz-carbonate veins, with valuable amounts of gold and silver, in faults and shear zones located within deformed terrains of ancient to recent greenstone belts commonly metamorphosed at greenschist facies (Dubé and Gosselin, 2007). Greenstone-hosted quartz-carbonate vein deposits are a subtype of lode gold deposits (Poulsen et al., 2000) (Figure 8.1). They are also known as mesothermal, orogenic. They consist of simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins in moderately to steeply dipping, compressional brittle-ductile shear zones and faults, with locally associated extensional veins and hydrothermal breccias. They can coexist regionally with iron formation-hosted vein and disseminated deposits, as well as with turbidite-hosted quartz-carbonate vein deposits (Figure 8.2). They are typically distributed along reverse-oblique crustal-scale major fault zones, commonly marking the convergent margins between major lithological boundaries such as volcano-plutonic and sedimentary domains. These major structures are characterized by different increments of strain, and consequently several generations of steeply dipping foliations and folds resulting in a fairly complex geological collisional setting.

Figure 8.1: Inferred Crustal Levels of Gold Deposition - Different Types of Lode Gold Deposits and Inferred Deposit Clan



From Dubé et al., 2001; Poulsen et al., 2000

Figure 8.2: Schematic Diagram Illustrating Setting of Greenstone-hosted Quartz-carbonate Vein Deposits



From Poulsen et al., 2000

The crustal scale faults are thought to represent the main hydrothermal pathways towards higher crustal level. However, the deposits are spatially and genetically associated with higher order compressional reverse-oblique to oblique brittle-ductile high-angle shear zones commonly located less than 5 km away and best developed in the hanging wall of the major fault (Robert, 1990). Brittle faults may also be the main host to mineralization as illustrated by the Kirkland Lake Main Break; a brittle structure hosting the 25 Moz Au Kirkland Lake deposit.

Stockworks and hydrothermal breccias may represent the main host to the mineralization when developed in competent units such as granophyric facies of gabbroic sills. Due to the complexity of the geological and structural setting and the influence of strength anisotropy and competency contrasts, the geometry of the vein network varies from simple such as the Silidor deposit, Canada, to more commonly fairly complex with multiple orientations of anastomosing and/or conjugate sets of veins, breccias, stockworks and associated structures (Dubé et al., 1989; Hodgson, 1989, Robert et al., 1994, Robert and Poulsen, 2001).

Economic grade mineralization also occurs as disseminated sulfides in altered (carbonatized) rocks along vein selvages. Deposit shoots are commonly controlled by: 1) the intersections between different veins or host structures, or between an auriferous structures and an especially reactive and/or competent rock type such as iron-rich gabbro (geometric ore shoot); or 2) the slip vector of the controlling structure(s) (kinematic ore shoot). For laminated fault-fill veins, the kinematic ore shoot will be oriented at a high angle to the slip vector (Robert et al., 1994; Robert and Poulsen, 2001).

At the district scale, the greenstone-hosted quartz-carbonate-vein deposits are associated with large-scale carbonate alteration commonly distributed along major fault zones and associated subsidiary structures (Dubé and Gosselin, 2007). At the deposit scale, the nature, distribution and intensity of the wall-rock alteration is largely controlled by the composition and competence of the host rocks and their metamorphic grade. Typically, the alteration haloes are zoned and characterized, at greenschist facies, by iron-carbonatization and sericitization with sulfidation of the immediate vein selvages (mainly pyrite, less commonly arsenopyrite).

The main gangue minerals are quartz and carbonate with variable amounts of white micas, chlorite, scheelite and tourmaline. The sulfide minerals typically constitute less than 10% of the mineralization The main ore minerals are native gold with pyrite, pyrrhotite and chalcopyrite without significant vertical zoning (Dubé and Gosselin, 2007).

The structurally controlled, high grade veins spatially related to the Hard Rock Porphyry in the Hard Rock and MacLeod-Cockshutt mines are similar to quartz-carbonate-sericite veins that host gold in many gold camps in Ontario (Porcupine, Kirkland Lake and Red Lake). The veins related to the Hard Rock Porphyry do not host significant tonnages of ore from past production, despite their locally high grades. Numerous thin, gold-bearing quartz stringers occur along shear fractures in zones of faulting, folding and shearing at the contact with wacke and Hard Rock Porphyry. When stringers merge, elongate replacement or blow-out lenses up to 1 m long are formed. Normally, they occur as thin highly contorted veinlets which follow both shear and tension fractures and locally have a gash-like character. Carbonate (ankeritic-dolomite), sulfides (pyrite, pyrrhotite, arsenopyrite and chalcopyrite) and tourmaline are found to be associated with the quartz. Zones A through H were of this type (Horwood and Pye, 1951).

The greywacke (turbidite) associated mineralization is typically characterised by narrow, often sheeted, millimetre- to centimetre-scale veins with attendant but highly variable degrees of carbonate-sericite-pyrite alteration. This style of mineralization forms wide, low grade zones in the former Hard Rock, MacLeod-Cockshutt and Mosher mines. The F-Zone was the most spectacular zone, accounting for an orebody of some 10,000,000 t at 0.15 oz/ton Au (Macdonald, 1983b). The F-Zone produced the bulk of the tonnage that came from these mines from the 1950s to 1970.

8.2 Other Greenstone Gold Deposits

8.2.1 Brookbank

Economic concentrations of gold in the Beardmore-Geraldton area are typical of Archean epigenetic hydrothermal gold deposits normally considered to be mesothermal lode gold deposits. The gold mineralization is primarily located in areas of high strain and deformation with brittle structures providing a pathway and also hosting mineralization as veins or replacement zones with associated alteration. There are also low grade zones that locally have less obvious structural control, less veining, and less intense hydrothermal alteration on a hand specimen scale, but these clearly have strong deposit scale structural controls.

Gold mineralization on the Brookbank deposit is hosted within bands of intense deformation at the contact zone between domains of mafic flows and polymictic conglomerate. This contact zone straddles the 6.5 km east-west trending Brookbank shear zone. The mineralization occurs within quartz-carbonate veinlets/stringers, fractures and/or stockworks associated with hydrothermal alteration (Figure 8.3).

Figure 8.3: Exposure of the Brookbank Deposit-Quartz Carbonate Veins/Stringers, Fractures/Stockworks

Micon, 2013.

Taking into account the deposit model discussed above, previous and current exploration activities have been focused on the contact zone between the sedimentary formation and the volcanic assemblage within the confines of the Brookbank shear zone.

8.2.2 Key Lake

The Key Lake deposit consists of several lenticular bodies in an echelon arrangement following a north-westerly direction. The mineralization is of a volcanoclastic-exhalative nature. Post mineralization processes have concentrated the mineralization into isolated high grade patches/pockets.

8.2.3 Kailey

Kailey is located at the location of former Little Long Lac Mines. The deposit at the Little Long Lac Mine occurred in the large Z-shaped minor fold on the north limb of the Barton syncline. The fold plunges 45 to 55° to the west. Numerous smaller flexures are superimposed, some of which are believed to have been formed during a later period of deformation. The deposit consist of more or less parallel quartz veins and stringers within fracture zones in massive arkose. For the most part, the sulfides are confined to narrow selvages and books of altered wall rock along and within the individual veins, although small amounts are commonly enclosed by the vein quartz itself. The quartz veins have, along their walls, narrow selvages, generally less than half an inch thick, of highly sheared and sericitized arkose impregnated with small amounts of finely divided sulfides, chiefly pyrite and arsenopyrite.

9. EXPLORATION

9.1 Hardrock Geological Mapping and Channel Sampling

Exploration work performed by Premier before June 2014 was summarized in Section 6.0 – History.

Since June 1, 2014, Premier and after March 2015 GGM has been removing soils and vegetation to expose rocks in the 2016 resource area. The work consisted of three outcrops with detailed geological mapping and channel sampling. The location of these strippings is shown on Figure 9.1 and the locations of the channels are shown on Figure 9.2 to Figure 9.4. The purpose of this work was to verify and establish structural elements and grade continuity at surface. The detailed geological mapping of these outcrops is provided in Section 7.4-Stripping.

- On the Porphyry Hill Stripping, a total of 539 m was channelled including 468 samples.
- On the F-Zone Stripping, a total of 186.9 m was channelled including 128 samples.
- On the Headframe East Stripping, a total of 597 m was channelled including 623 samples.

9.1.1 Procedures and Parameters

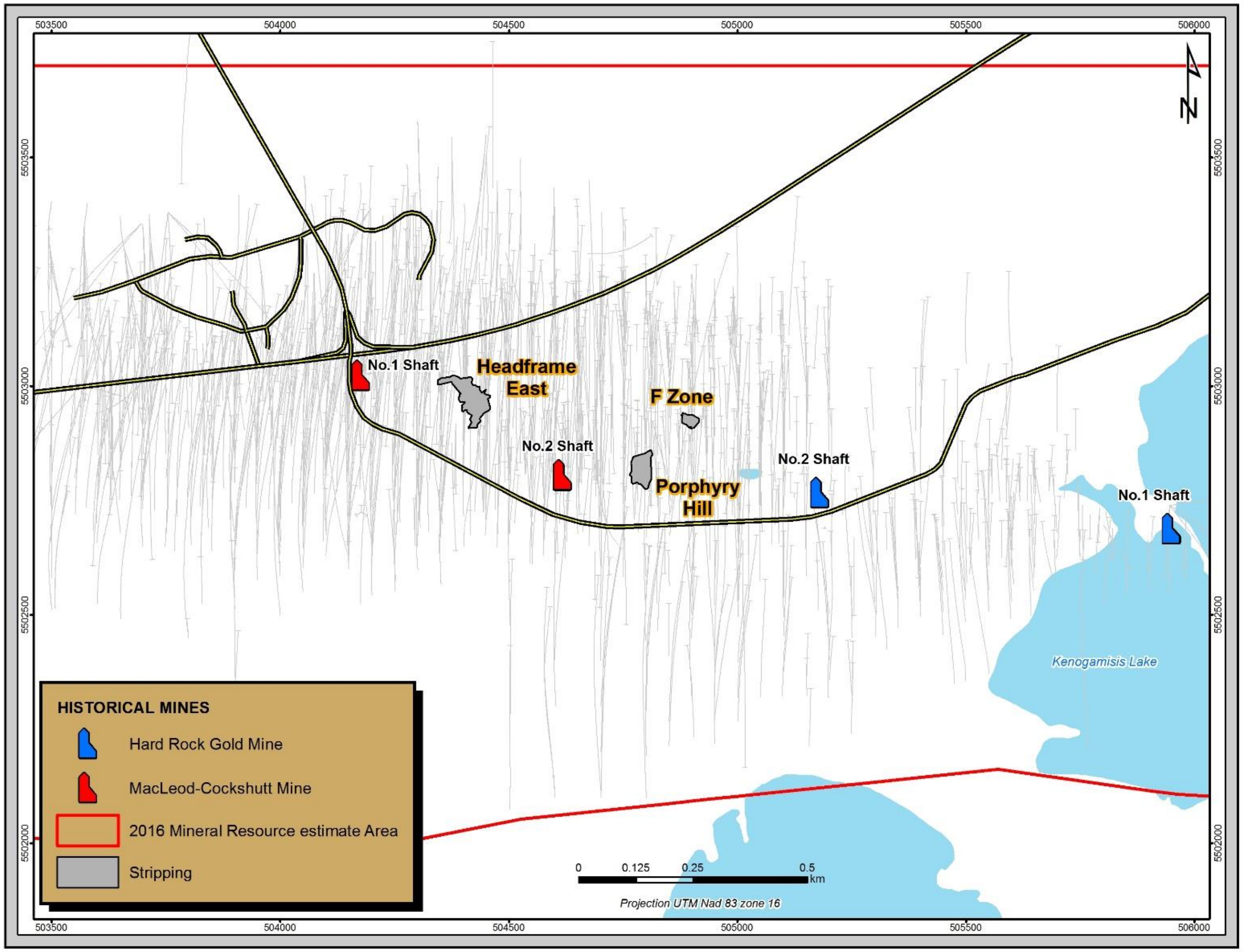
The Headframe East and Porphyry Hill stripping areas were chosen to be sampled because they were naturally stripped of overburden. Minor amounts of overburden were removed around their edges to increase the amount of visible bedrock. The F-Zone stripping was chosen because of its geological significance since it is the surface showing of the F-Zone which was not mined out in previous operations. All channel samples were cut parallel to each other with 5 m spacing. Each channel was cut south to north along the same easting. There were minor deflections along the eastings if there were any significantly deep bedrock sections into which the saws could not cut. These deflections are generally less than 1 m off the original easting. The channels were surveyed at each start and end using a Trimble RTK and integrated to the database as horizontal drill holes.

9.1.2 Sampling Methods

The channels were cut using a two-bladed rock saw that cut to an average depth of 8 cm. Samples were taken on average every 0.3 to 1.5 m evenly along each northing. The sample lengths varied to allow for samples to end on lithological boundaries. The samples were subsequently removed with a small crowbar and placed directly into sample bags. Cuts were made perpendicular to the channel to demonstrate where samples started and ended. Sample tags were placed in the perpendicular cuts.

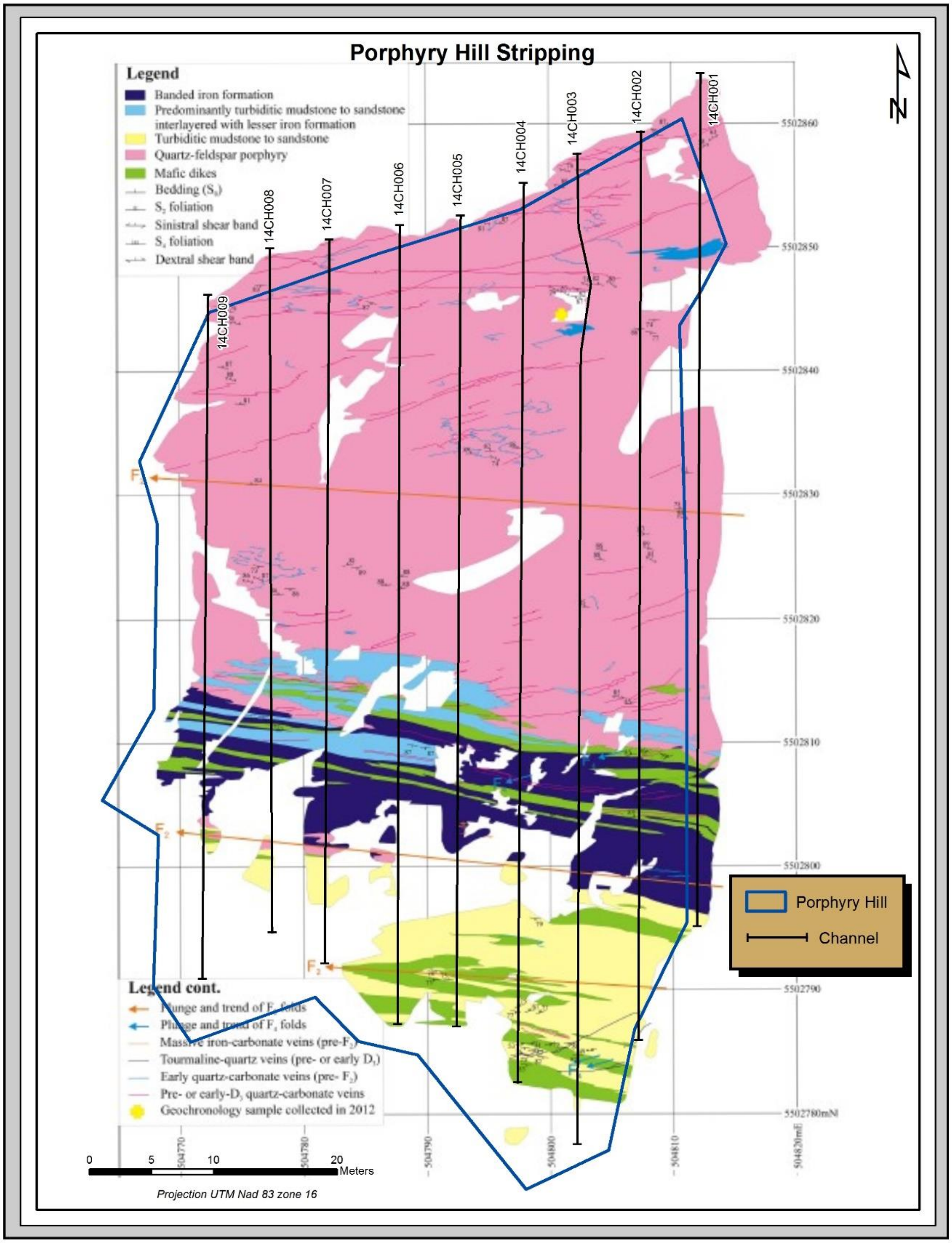
GMS is of the opinion that the channel samples from the stripping program are valid and of sufficient quality to be used in the Mineral Resource estimation herein.

Figure 9.1: Location of Strippings



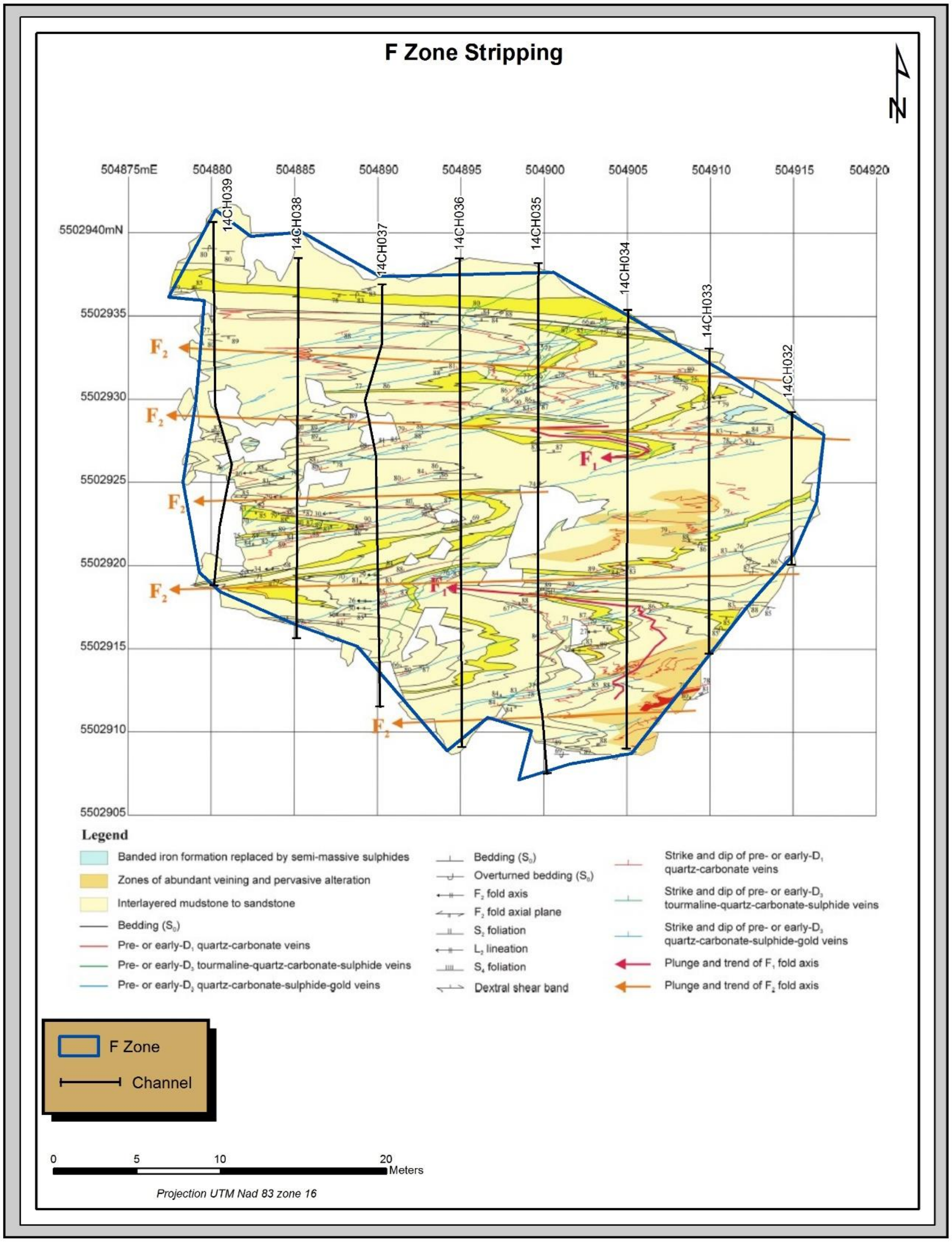
Source: GGM, 2016

Figure 9.2: Channeling on the Porphyry Hill Stripping



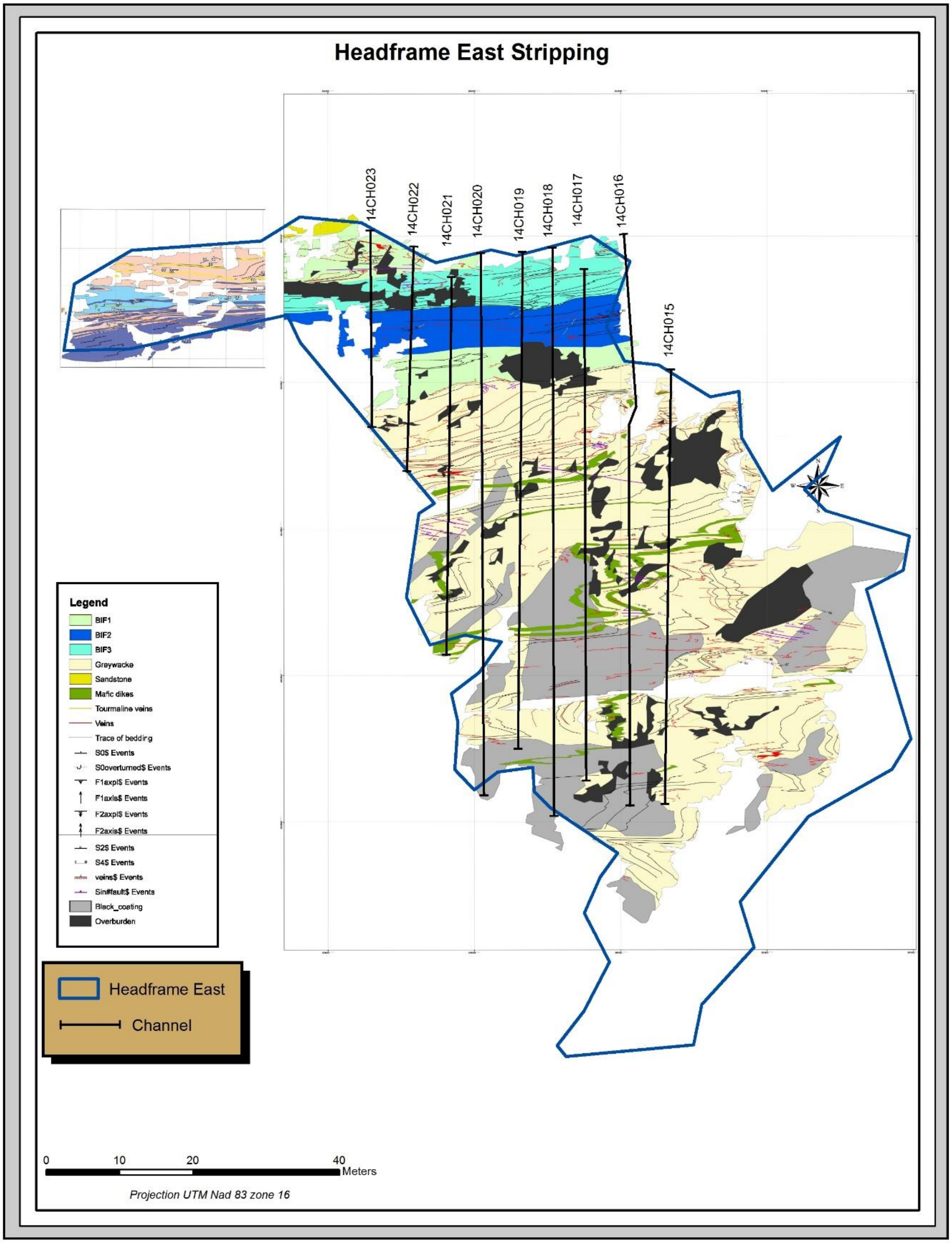
Source: GGM, 2016

Figure 9.3: Channeling on the F-Zone Stripping



Source: GGM, 2016

Figure 9.4: Channeling on the Headframe East Stripping

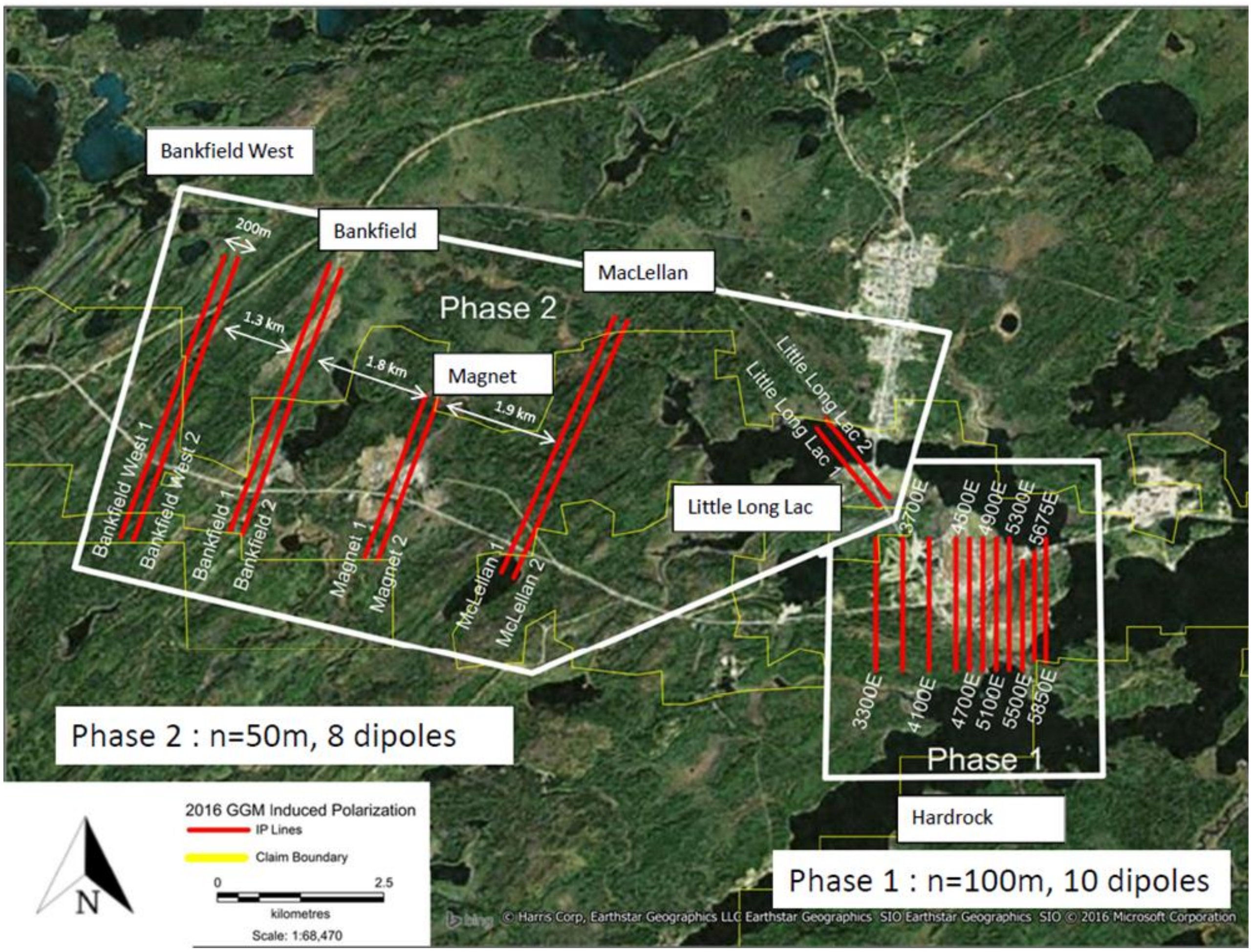


Source: GGM, 2016

9.2 Hardrock Geophysical Survey

During 2016, GGM conducted induced polarization (IP) surveys in the 2016 resource area and locally in the Hardrock claim block over past-producing mines and known mineralized zones. The work consisted of two phases of IP. Phase 2 was conducted in March and April 2016 and totalled 34 km of IP surveys divided into five series of two lines 200 m apart over Little Long Lac, MacLellan, Magnet, Bankfield and Bankfield West in order to build a geophysical signature for resistivity and chargeability over the known deposits. Phase 1 was conducted in June 2016 and totalled 23 km over the Hardrock deposit.

Figure 9.5: Induced Polarization Survey Location - Hardrock Claim Block



9.3 Viper 2016 Regional Exploration Work

In July 2016, GGM conducted 18.5 km of IP surveys on the Viper claim block over known mineralisation. Soil sampling was executed during the summer of 2016, totalling 38 humus samples, 27 B Horizon samples and 23 C Horizon samples. A reconnaissance mapping program was carried out in the southern part of the claim block along with the collection of 56 grab samples. Relogging of 11 drill holes added 688 new samples to the Viper database.

9.4 Brookbank 2016 Regional Exploration Work

In the summer of 2016, an orientation till/soil survey was done on a 200 x 100 m spacing, totalling 183 B Horizon samples and 80 C Horizon samples over the Brookbank deposit. A second till/soil survey was done over the Patter Lake area totalling 38 B Horizon samples and 13 C Horizon samples. A third till/soil survey was conducted over an area near the Brookbank East outcrop on a 200 x 100 m spacing. On the historical stripped outcrop of Brookbank East, the following work was done: 1.8 km of ground magnetics, mapping and channel sampling. Finally, 14 holes in the area surrounding Brookbank East were relogged. A total of 926 new core samples were taken from these historical holes. Prospecting was also conducted over some portion the Brookbank claim, mostly near the known resource area.

10. DRILLING

10.1 Hardrock

Over the years, different drill core diameters have been used on the Hardrock deposit. Recent drill holes at the Hardrock Project are mostly drilled with NQ diameter core. Table 10.1 summarizes the core diameter used in different years.

Table 10.1: Number of Drill Holes and Core Size per Year

Year Drilled	DDH Count	Core Size
1987	34	BQ
1988	33	BQ
1993	27	BQ
1994	76	BQ
1995	7	BQ
1996	24	Unknown
2009	340	NQ
2010	243	NQ
2011	166	NQ
2012	126	NQ
2013	278	NQ
2014	128	NQ
2014	1	PQ
2015	117	NQ
Unknown	29	Unknown
Total	**1629**	

10.1.1 Drilling and Re-sampling Included in the 2016 Mineral Resource Estimate Update

The close-out date of the database is **November 18, 2015** and corresponds to the completed and validated diamond drill holes as of this date.

Between May 26, 2014 and November 18, 2015, GGM added 157 diamond drill holes on the Hardrock deposit for a total of 54,027 m. One diamond drill hole (MM043) included in the 2014 Mineral Resource Estimate was also deepened, from 456 m to 655 m, representing a total of 199 m of new footage.

Seventy-nine historical diamond drill holes were re-sampled to add new assay results in the 2016 updated Mineral Resource Estimate. These holes represent a total of 8,733 m of new footage and 6,411 of new samples in the 2016 database.

Figure 10.1 shows the locations of the drill holes included in the 2016 updated Mineral Resource Estimate presented in Section 14.1. The new drill holes (red), the re-sampled diamond drill holes (blue) and extended drill hole (yellow) that are included in the 2016 updated Mineral Resource Estimate are presented in Figure 10.1.

Figure 10.2 shows the locations of the condemnation drill holes drilled in the area of the Hardrock deposit. A total of 55 condemnation diamond drill holes totalling 8,512 m were drilled by GGM.

10.1.2 Collar Locations, Orientations and Down Hole Surveys

Collar locations for the drill holes on the Hardrock Project were determined using a cut grid or a hand-held GPS. Subsequent to completion, the collars were located, depending on the years drilled, using either a GPS, a Trimble and more recently since 2014 the more precise Trimble RTK survey instrument. A total of 55% of the holes drilled prior to 2013 have been surveyed using a hand-held GPS. Table 10.2 summarizes the numbers of drill holes in the Hardrock resource database relative to the collar survey method used. The drill holes are also divided by years of drilling.

Table 10.2: Drill Hole Database of the Hardrock Resources Relative to the Collar Survey Method used

Coordinates	X – Y	Z	DDH COUNT	1987	1988	1993	1994	1995	1996	2009	2010	2011	2012	2013	2014	2015	UK
SURVEY METHOD	GPS	Use LIDAR 2014	886	26	25	15	37	7	16	132	176	147	100	148	29		28
	GPS	GPS	11						1	8		1		1			
	Leica GPS (JDBARNES)	Leica GPS (JDBARNES)	32			7	25										
	APS	APS	1												1		
	TRIMBLE RTK	TRIMBLE RTK	698	8	7	5	14		7	200	67	18	26	129	99	117	1
	Unknown	Unknown	1		1												
		Total	**1629**	**34**	**33**	**27**	**76**	**7**	**24**	**340**	**243**	**166**	**126**	**278**	**129**	**117**	**29**

Whenever it was possible, casings were left in the ground for the holes on the Hardrock Project. A collar re-survey campaign, using the Trimble RTK survey instrument, took place in the summer of 2014 for a total of 536 drill holes for which casing was found. Of these 536 resurveyed collars, 489 were previously surveyed by a handheld GPS. Following the ranking system described below, the Trimble survey replaced the original survey improving the precision of the collar location for 30% of the drill holes in the database.

Once the holes were drilled, the drill hole azimuth and precise UTM coordinates were determined by placing an APS unit on the drill casing. The downhole dip and drill hole orientations were surveyed using a gyroscope unit (REFLEX Gyro™). The UTM Coordinate System, NAD 83, Zone 16, is used to record the locations (x, y, and z) of the drill collars.

10.1.3 Core Logging Procedures

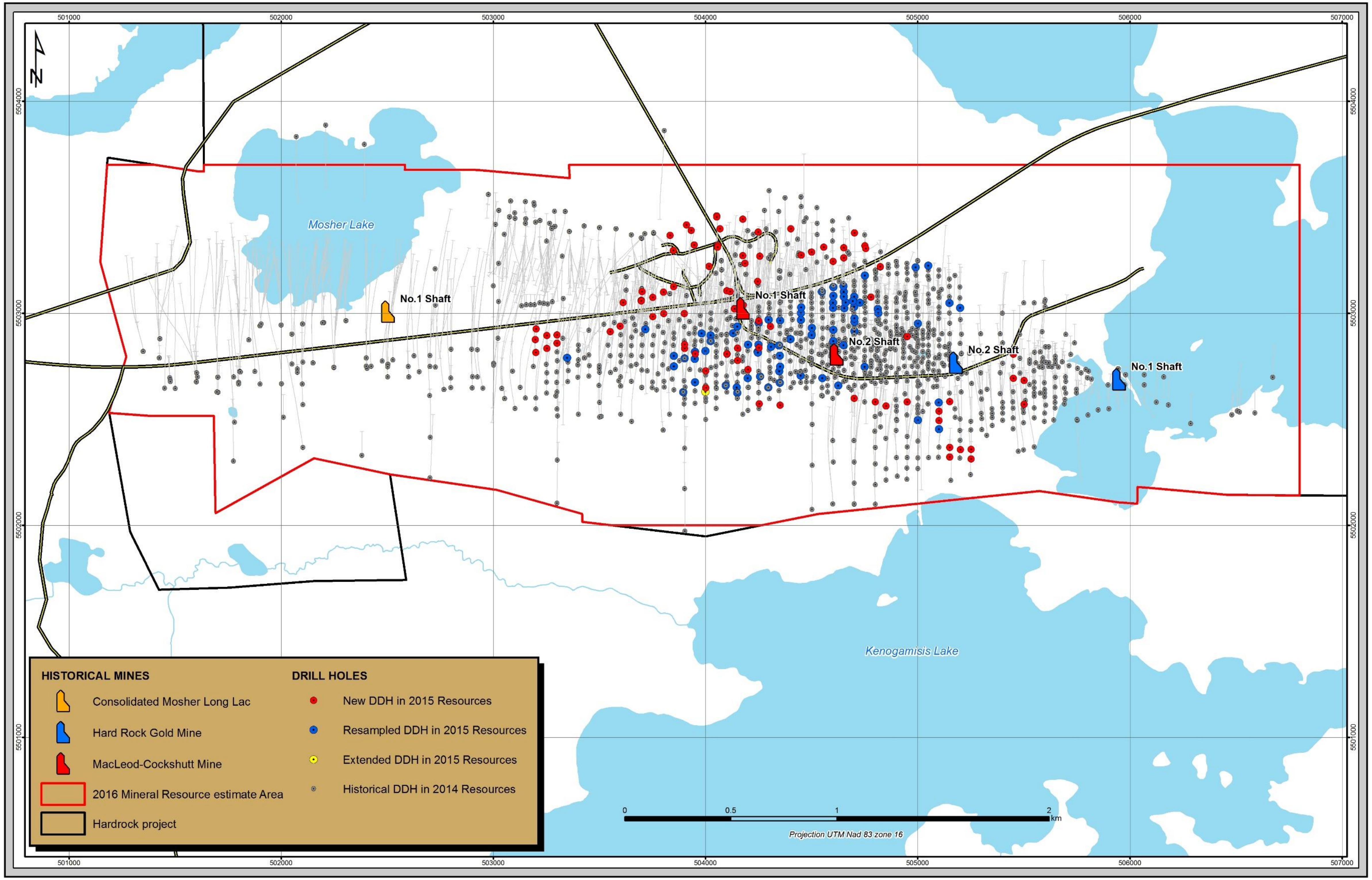
The first time the core is handled is at the drill by the driller helper who takes the core from the core tube and places it in core boxes, marking off every 3 m. Once a core box is full, the helper wraps the box with tape or wire depending on the preference of the drilling company. At the end of each shift, the core is delivered to the core shack. GGM personnel remove the wire or tape and bring the boxes to the logging trailers. The technicians rotate the core so that all pieces slant one way, at about a 45° angle. They check that distances are correctly indicated on the wooden blocks placed every 3 m. If there is a mistake on any of the blocks, the Project Manager is informed and the Drill Foreman brought in. If it is an easy fix, the blocks are moved, but if not, the drill rods are removed and counted to assess what happened. The core is measured in each box and the box labelled. Red lines are drawn along the centre of the core to provide a reference for the core cutters. Geological technicians and geologists are then responsible for taking photographs of the core.

Rock quality designation (“RQD”) is done by either geologists or the geological technicians. Any breakage under 10 cm is recorded. Core from the Hardrock deposit is of very good quality and recovery is high.

Samples were generally taken along the entire length of the holes (continuous sampling). Sample length typically ranged from 0.5 to 1.5 m. The sampled core must be considered representative and of good quality. Once logged and/or ticketed, the core is stored outside in racks until it is brought into the cutshack for sawing. The core of each selected interval is cut in half using a typical table-feed circular rock saw, with one half placed in a numbered plastic bag for shipment to the laboratory, and the other half returned to the core box as a witness (reference) sample. A tag bearing the sample number is left in the box at the beginning of the sampled interval. The core box is then brought to the core compound for storage. GGM drill core is stored at Geraldton.

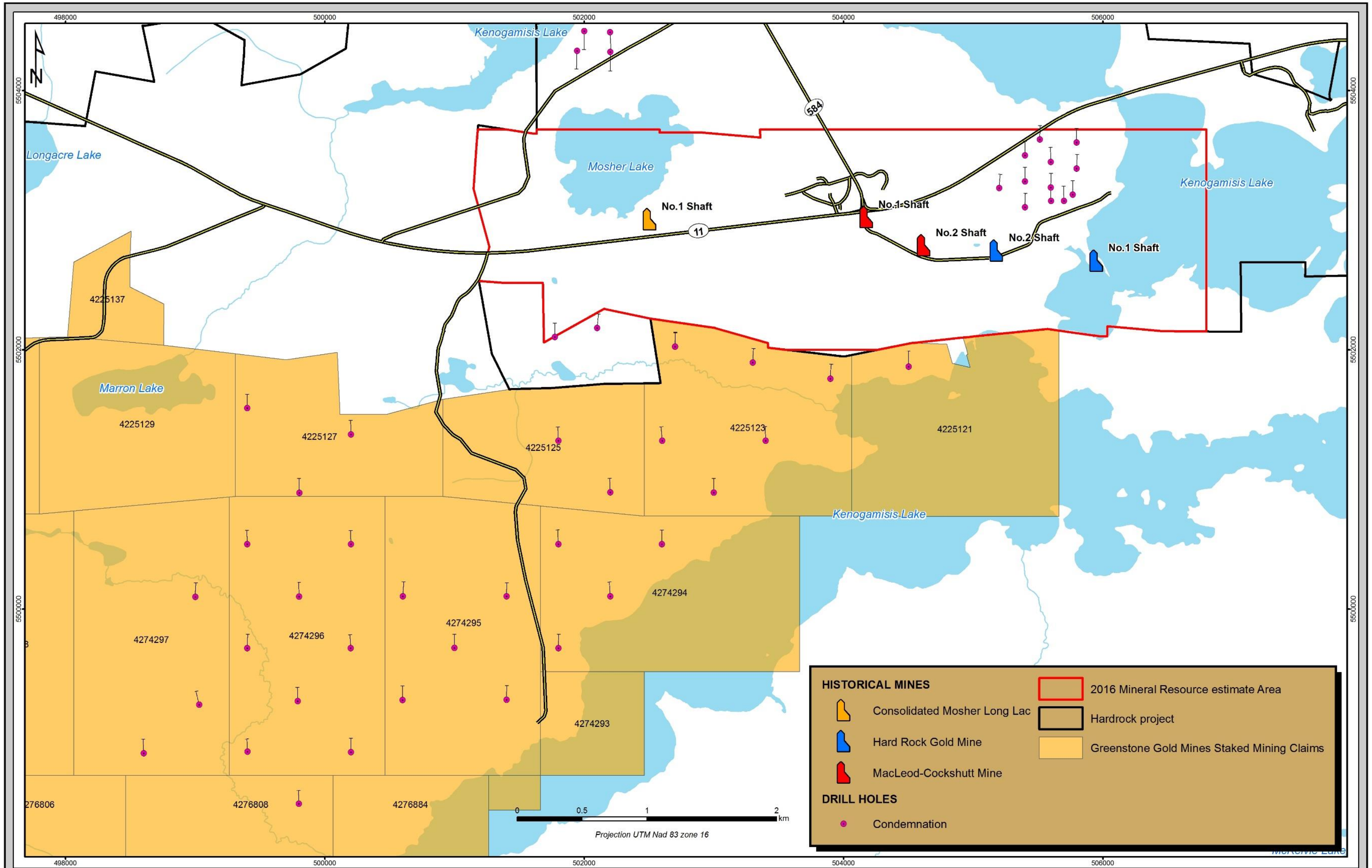
GMS is of the opinion that the core samples from the Hardrock Project drilling programs are valid and of sufficient quality to be used in the Mineral Resource estimation herein.

Figure 10.1: Location of Greenstone Gold Mines Drill Holes in the Mineral Resource for the Hardrock Deposit



Source: Innovexplo, 2015 with modifications by GGM, 2016

Figure 10.2: Location of Greenstone Gold Mines Condemnation Drill Holes in the Hardrock Deposit Area



Source: GGM, 2016

10.2 Other Greenstone Gold Property Deposits Brookbank and Key Lake

Previous operators, notably Ontex between 1999 and 2009, did most of the drilling on other deposits on the Greenstone Gold Property. Between 2010 and 2012, Premier drilled two holes on the Brookbank property (1,359 m) and eight holes on the Key Lake mineral deposit (3,190 m). Premier also re-surveyed a number of existing holes. As of the effective date of this Report, GGM has not completed any additional drilling.

10.2.1 Procedures

Drill hole collar positions were established on grid cut-lines aligned perpendicular to the strike of the mineralized zones. The details of drilling done by the previous operators, including Lac, Asarco and Cyprus, are scarce. The core was BQ size. One of the authors of the 2010 Technical Report, J. Reddick, P. Geo, worked on the property for Asarco in 1993 and 1994, and also reviewed the property data when it was held by Cyprus in 1997. All exploration work, drilling procedures and assaying procedures are believed to have been conducted in accordance with standard industry practice at the time.

All of the diamond drilling between 2007 and 2009 was contracted to Chibougamau Diamond Drilling (Chibougamau Drilling) based in Chibougamau, Quebec. The drill rigs were mounted on skids and dragged into position using a skidder or bulldozer. A Reflex Instruments down-hole survey tool, provided by the drill contractor, was used with surveys typically taken every 50 m. Surveys that are more detailed were taken at the request of the site geologist. This provided both dip and azimuth readings, but in areas with iron-rich rocks, the azimuth readings are unreliable. As of November 2007, a Reflex Instruments Maxibore tool was used for down-hole surveys. In May and June 2010, Premier changed to an Icefields Gyro survey tool to achieve more efficient and more accurate survey data.

In 2009 and 2010, many of the Lac, Asarco and Cyprus drill holes were located by Premier and surveyed using either a hand-held or Trimble GPS survey instrument for collars, and a gyroscope for dip and azimuth on drill holes that had casings.

10.2.2 Drill Core Sampling

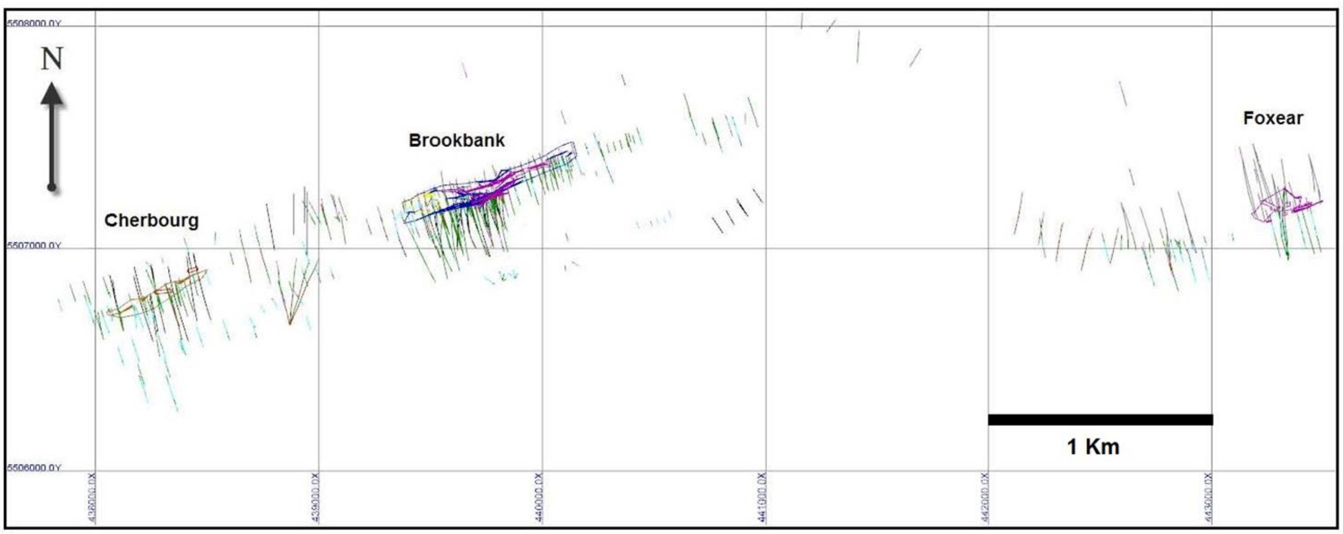
The geologist prepared a detailed geological log including lithology, alteration, mineralization and structures. The geologist then identified and marked the beginning and the end of the sampling intervals. Upon completion of the logging and demarcating the sample intervals, technicians sawed the core longitudinally in half with a diamond saw, except for material which was highly fractured and contained clay minerals, which was divided manually with hammer and chisel. One half of the core was bagged, tagged with a sample number and then sealed; the other half was put back in the core boxes and kept as a reference and check sample in the event that duplicate assays are required. Generally, samples of 1 m length were taken in longer sections of similarly mineralized rocks; however, sample size was reduced to as low as 0.4 m in areas of particular interest, or where lithology and mineralization were distinct.

Premier re-sampled and analyzed the holes drilled by their predecessor as part of their validation of previous work.

10.2.3 Brookbank Drill Results

The drill holes completed on the Brookbank deposit, to date, are shown in Figure 10.3.

Figure 10.3: Brookbank Drill Hole Layout

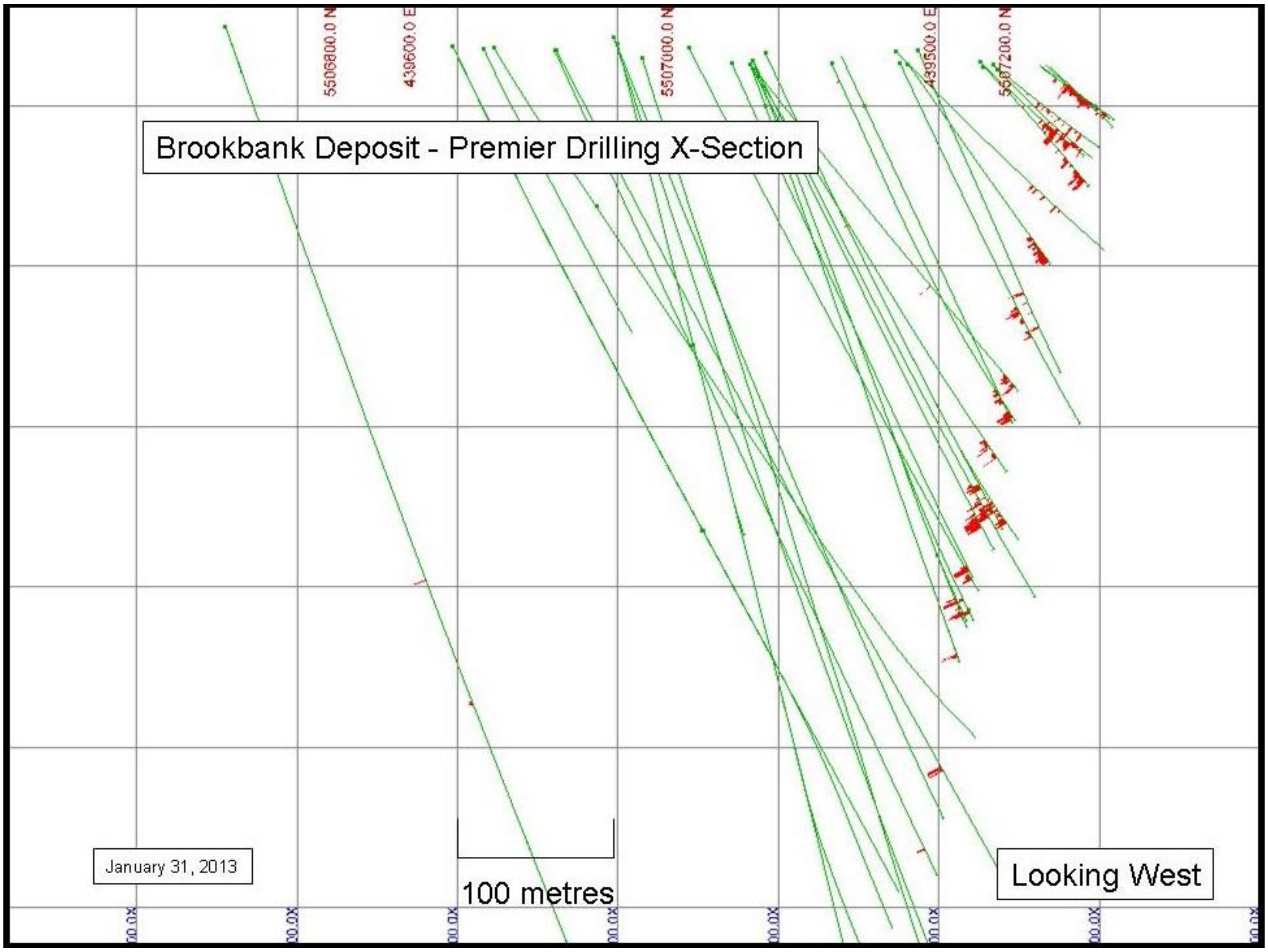


The most significant results are summarized as follows:

- Mineralization on the Brookbank main zone deposit is continuous along strike for over a kilometre with the exact limits yet to be established.
- Using a mineralization envelope of 0.1 g Au/t, the mineralized horizon is cone-shaped in section and is sub-vertical with a slight inclination to the south as demonstrated in Figure 10.4.
- The true thickness of the mineralized envelope varies from 20 to 50 m at/or close to surface and to 1 to 2 m at a 750 m (approximate) vertical depth from surface.

- Within the 0.1 g Au/t envelope, the drill intercepts with mineralization of potential economic interest vary in grade from 0.5 to 15 g/t. The variation is completely random. The true width varies between 50% and 75% of the core length, depending on the angle of intersection.

Figure 10.4: Representative Section through the Brookbank Deposit



10.2.4 Key Lake Drill Results

The drill holes completed on the Key Lake deposit to date are shown in Figure 10.5. Drill results show that the mineralization is in lenticular bodies in an echelon pattern over a strike distance of about 2 to 3 km. Gold grades are highly variable from as low as 0.5 g/t to as high as 50 g/t over true thicknesses varying between 0.5 m and 10 m. A typical section through the deposit is shown in Figure 10.6. The lenticular bodies have strike lengths varying between 100 and 300 m.

Figure 10.5: Key Lake Project Drill Hole Layout

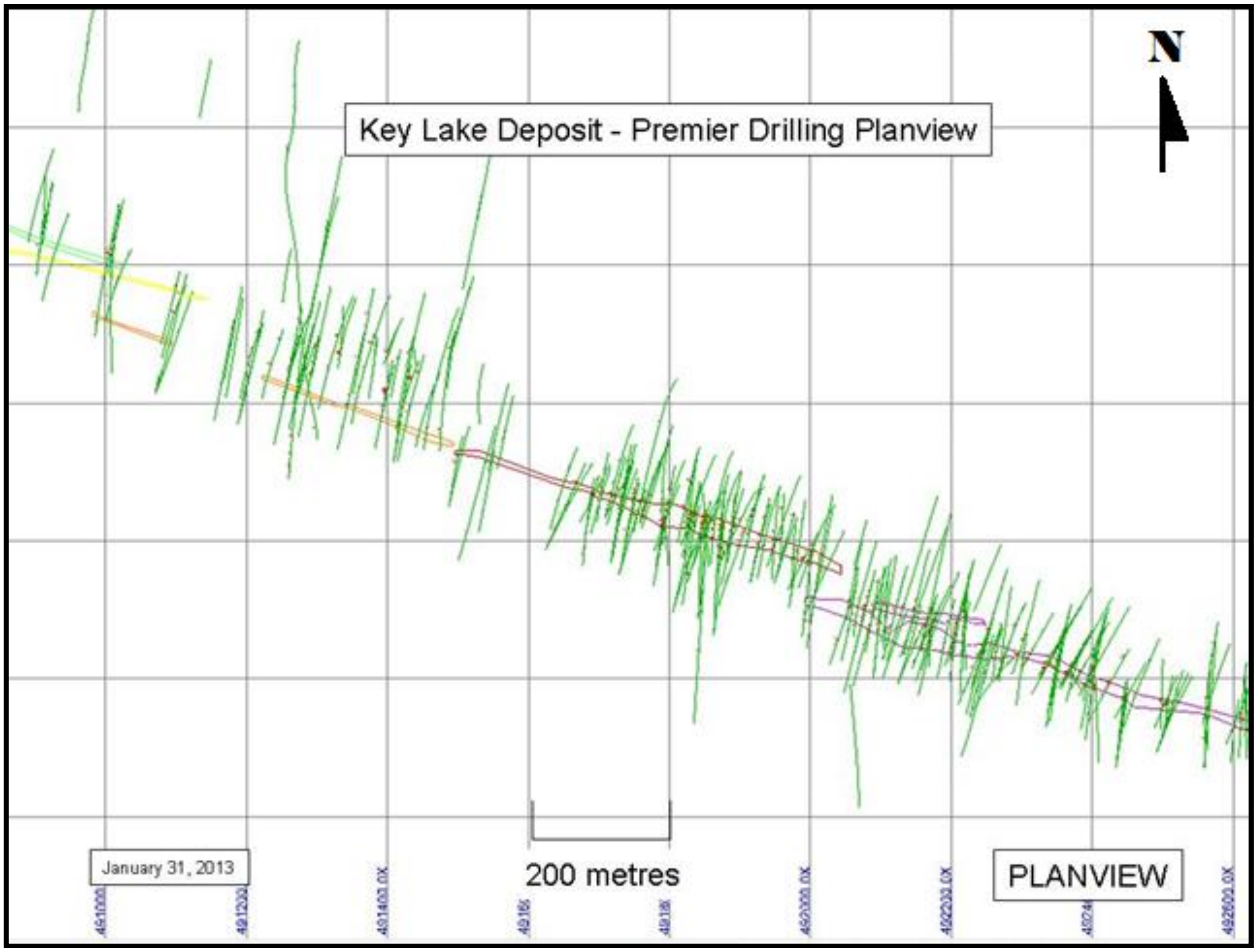
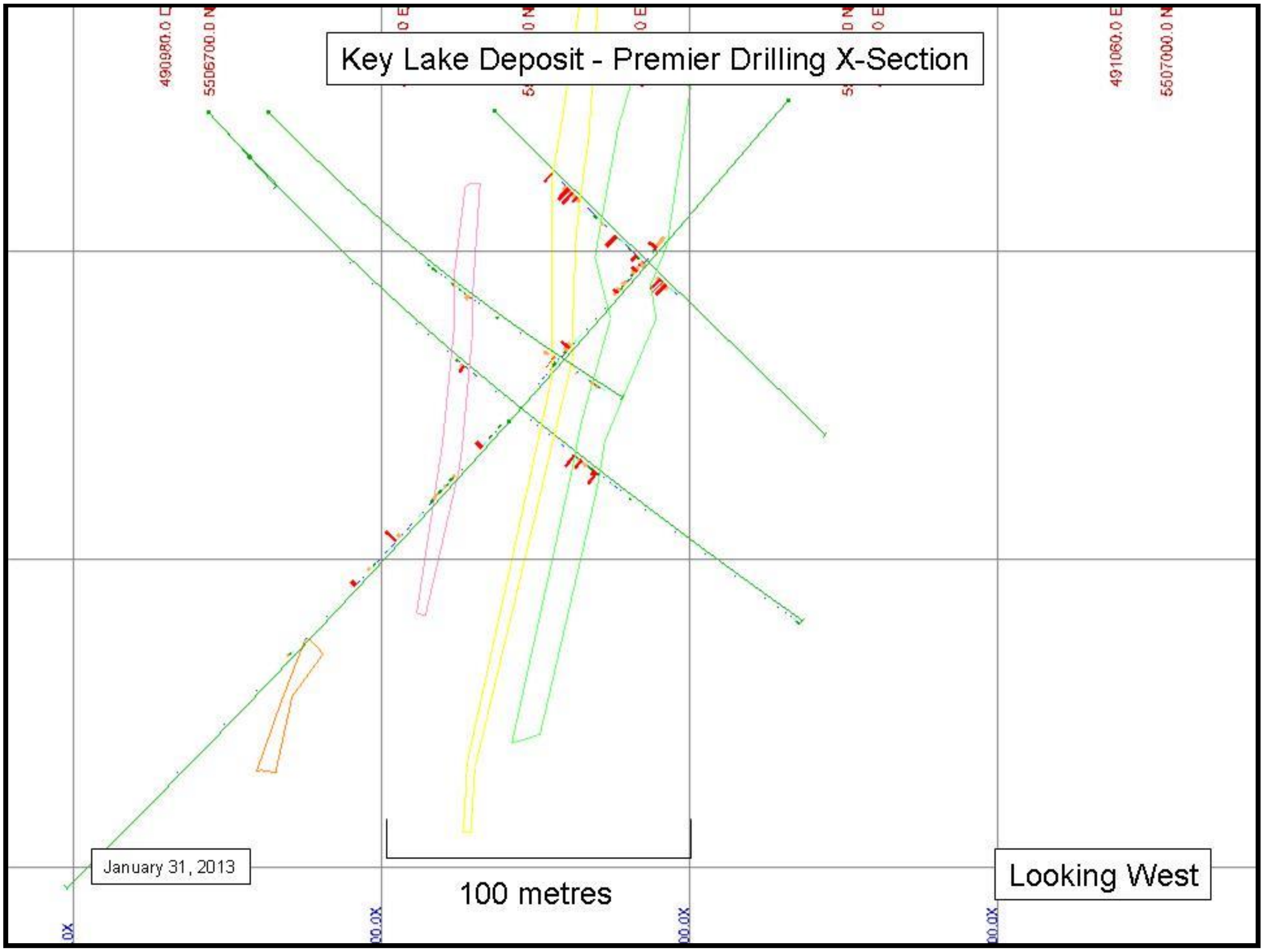


Figure 10.6: Representative Section through the Key Lake Deposit



10.2.5 <u>Micon Comments</u>

During the Brookbank and Key Lake site visits, Micon reviewed drilling procedures, sampling and drill hole logs. Core recovery is excellent throughout the deposit. There are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of the assay results.

Overall, Micon considers the data obtained from the exploration and drilling programs to be reliable.

11. SAMPLE PREPARATION, ANALYSES AND SECURITY

11.1 Hardrock

11.1.1 Laboratory Accreditation and Certification

The International Organization for Standardization ("ISO") and the International Electrotechnical Commission ("IEC") form the specialized system for worldwide standardization. ISO/IEC 17025 General Requirements for the Competence of Testing and Calibration Laboratories sets out the criteria for laboratories wishing to demonstrate that they are technically competent, operating an effective quality system, and able to generate technically valid calibration and test results. The standard will form the basis for the accreditation of competence of laboratories by accreditation bodies. ISO 9001 is for management support, procedures, internal audits and corrective actions. It provides a framework for existing quality functions and procedures.

The main difference between ISO/IEC 17025 and ISO 9001 is the accreditation and certification. ISO/IEC 17025 stands for accreditation, which means the recognition of competence of specific technical competence. ISO 9001 stands for certification, which means accordance with a standard assessed by management systems, certified by any independent body that is internationally agreed.

The Geraldton facility belonging to Activation Laboratories Ltd ("Actlabs Geraldton") was used for the entire drilling and channelling programs. Actlabs Geraldton has received ISO 9001:2008 certification through Kiwa International Cert GmbH. Actlabs Geraldton is an independent commercial laboratory.

All re-assayed batches (pulps) were sent to ALS-Chemex in Thunder Bay. ALS-Chemex laboratory is part of the ALS Global Group and has ISO 9001 certification and ISO/IEC 17025 accreditation through the Standards Council of Canada. ALS is an independent commercial laboratory.

11.1.2 Sample Preparation by GGM Personnel

All quality control samples are prepared and bagged in advance by GGM personnel. The GGM employee in the core cutting facilities places one half of the ticket into a bag with the sample and staples the other half in the box. One half of each quality control sample ticket is placed in the appropriate type of control sample bag, which was prepared beforehand. A list of quality control samples and their numbers/locations is posted on the wall in the coreshack and regularly updated by GGM personnel. Five to seven samples are placed in a rice bag and the contents identified on the outside of the bag. Each bag and its contents are recorded on a notepad and placed in a plastic holder once complete. These slips are picked up each morning by a GGM employee and recorded in an Excel spreadsheet. Once the batches are complete, GGM personnel deliver the bags to Actlabs Geraldton and no third party is involved in transportation.

Economic samples (drilling and channelling programs) are sent in batches of 34 samples. Each purchase order covers one batch of 34 samples consisting of:

- 30 regular samples;
- 1 field duplicate sample;
- 1 field blank;
- 1 certified reference material (standard) with a low gold value;
- 1 certified reference material (standard) with a high gold value.

As a quality control check, Actlabs Geraldton adds a 35th sample to every field batch received in the form of a coarse duplicate of the last regular sample (the 30th sample), constituting a second pulp prepared from the reject. The quality of the reject is monitored to ensure that proper preparation procedures are used during crushing. For the fusion process, Actlabs Geraldton adds seven additional quality control samples (two analytical blanks, two certified reference materials and three pulp duplicates), bringing the fusible batch to a total of 42. The pulp duplicates are necessary to ensure that proper preparation procedures are used during pulverization.

At Actlabs Geraldton, the maximum furnace charge of 42 samples ensures that GGM samples are not mixed with others.

11.1.3 Fire Assay Sample Preparation (Actlabs Geraldton)

Samples are received at Actlabs Geraldton, sorted and bar-coded. They are then placed in the sample drying room and dried at 60°C. Any samples that are damaged upon receipt (i.e., punctured sample bag, loose core) are documented and the client is informed with pictures.

Samples are crushed to 90% passing 10 mesh and split with a Jones riffle, and a 250 g split is pulverized to 95% passing 150 mesh. Sieve tests are performed on the crusher at the beginning of each day. Sieve tests are performed on the pulps on the first and fiftieth sample of each work order. If there is a failure, the samples are re-milled to ensure that they pass. There is a pulp duplicate made every 30th sample in sample prep and a coarse reject duplicate every 50th. These are all reported in Section 11.1.7 of this Report. Samples are then sent for fire assay.

11.1.4 Metallic Sieve Sample Preparation (Actlabs Geraldton)

All samples containing visible gold are prepared with metallic sieve sample preparation procedures.

A representative 2,000 g split (Code 1A4-2000) is sieved at 100 mesh (149 microns) with fire assays performed on the entire +100 mesh and two splits on the -100 mesh fraction. The total amount of sample and the +100 mesh and -100 mesh fractions are weighed for assay reconciliation. Measured amounts of cleaner sand are used between samples and saved to test for possible plating out of gold on the mill. Alternative sieving mesh sizes are available, however, the finer the grind the more likelihood of gold loss by plating out on the mill.

11.1.5 Fire Assay Procedures (Actlabs Geraldton)

The following description for the fire assay procedures was supplied by Actlabs Geraldton. Samples (50 g each) are sent to the fire assay area numbered and in order (usually 1 to 34+1). A rack of 42 crucibles is then labelled with an assigned letter code and numbered one to 42. The mixture is placed in a fire clay crucible. The mixture is then preheated to 850°C, intermediate at 950°C and finished at 1,060°C, with the entire fusion process lasting sixty minutes. The crucibles are then removed from the assay furnace and the molten slag (lighter material) is carefully poured from the crucible into a mould, leaving a lead button at the base of the mould. The lead button is then placed in a preheated cupel which absorbs the lead when cupelled at 950°C to recover the Ag (doré bead) + Au.

All samples were assayed using fire assay with atomic absorption (“AA”) finish (1A2-50 code).

The entire Ag doré bead is dissolved in aqua regia and the gold content is determined by AA.

AA is an instrumental method of determining element concentration by introducing an element in its atomic form to a light beam of appropriate wavelength causing the atom to absorb light. The reduction in the intensity of the light beam directly correlates with the concentration of the elemental atomic species. On each tray of 42 samples there are two blanks, three sample duplicates and two certified reference materials, one high and one low (QC = 7 out of 42 samples).

All samples assaying grades over 5.0 g Au/t with AA were re-run with gravimetric finish to ensure accurate values. After the fire assay procedures, Au is separated from the Ag in the doré bead by parting with nitric acid. The resulting gold flake is annealed using a torch. The gold flake remaining is weighed gravimetrically on a microbalance.

11.1.6 Fire Assay Procedures with Gravimetric or Atomic Absorption Finish (ALS-Chemex-Thunder Bay)

The fire assay technique uses high temperature and flux to "melt" the rock and allows the gold to be collected. Lead formed from the reduction of litharge is traditionally used as the collecting medium for silver and gold. The test sample is intimately mixed with a suitable flux that will fuse at high temperature with the gangue minerals present in the sample to produce a slag that is liquid at the fusion temperature. The liberated precious metals are scavenged by the molten lead and gravitate to the bottom of the fusion crucible.

Upon cooling, the lead button is separated from the slag and processed in a separate furnace for a high temperature oxidation (cupellation) where the lead is removed, leaving the precious metals behind as a metallic bead called a prill. Traditionally, this prill was then partially dissolved in nitric acid (parted) to remove silver and the remaining gold determined by weighing (gravimetry). Alternatively, the prill can be dissolved in a mixture of hydrochloric and nitric acid (aqua regia) and the concentration determined by spectroscopic methods (AAS, ICPAES or ICPMS).such as atomic absorption spectroscopy ("AAS"), inductively coupled plasma atomic emission spectroscopy ("ICPAES") or inductively coupled plasma mass spectroscopy ("ICPMS"). The concentration is normally expressed as parts per million ("ppm"), which is equivalent to grams per tonne ("g/t").

For the AA finish method, a pulp sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead is digested in 0.5 ml dilute nitric acid in the microwave oven. The 0.5 ml concentrated hydrochloric acid is then added and the bead is further digested in the microwave at a lower power setting. The digested solution is cooled, diluted to a total volume of 4 ml with de-mineralized water, and analyzed by AAS against matrix-matched standards.

For the gravimetric finish method, a pulp sample is fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents in order to produce a lead button. The lead button containing the precious metals is cupelled to remove the lead. The remaining gold and silver bead is parted in dilute nitric acid, annealed and weighed as gold. Silver, if requested, is then determined by the difference in weights.

At the ALS-Chemex laboratory, the batch size for all fire assay method is 84 including six internal QC. Therefore 78 client samples can be done per batch.

The maximum furnace charge of 78 client samples ensures that GGM samples are not mixed with others.

11.1.7 Results of Quality Control

Analysis of the previous year's monitoring reports revealed that database accuracy was tested by adequate control samples incorporating certified reference material ("CRM"), blanks and duplicates. In a few instances where standards failed, appropriate investigations were conducted and re-assaying was conducted whenever it was deemed necessary. Table 11.1, Table 11.2 and Table 11.3 were extracted from previous Technical Reports on the Project and summarize the CRM results from 2013 to 2015. GMS did not identify any flaws in the QA/QC results.

Table 11.1: Results for Standards used by Premier during the 2012-2013 Drilling Program on the Hardrock Deposit

Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Laboratory	Analytical method	Amount of results	Lower process limit (±10%)	Upper process limit (±10%)	Outliers	(%) passing quality control
CDN-GS-5F	CDN Resource Laboratories LTD	5.300	Actlabs Geraldton	FA/AA	228	4.770	5.830	11	95.2%
CDN-GS-5K	CDN Resource Laboratories LTD	3.840	Actlabs Geraldton	FA/AA	376	3.456	4.224	27	92.8%
CDN-GS-7A	CDN Resource Laboratories LTD	7.200	Actlabs Geraldton	FA/AA	2	6.480	7.920	2	0.0%
CDN-GS-7B	CDN Resource Laboratories LTD	6.420	Actlabs Geraldton	FA/AA	583	5.778	7.062	40	93.1%
CDN-GS-8A	CDN Resource Laboratories LTD	8.250	Actlabs Geraldton	FA/AA	201	7.425	9.075	16	92.0%
SF67	Rocklabs Ltd	0.835	Actlabs Geraldton	FA/AA	227	0.752	0.919	18	92.1%
SG40	Rocklabs Ltd	0.976	Actlabs Geraldton	FA/AA	227	0.878	1.074	5	97.8%
SJ53	Rocklabs Ltd	2.637	Actlabs Geraldton	FA/AA	131	2.373	2.901	5	96.2%
SN60	Rocklabs Ltd	8.595	Actlabs Geraldton	FA/AA	204	7.736	9.455	15	92.6%
TOTAL					**2179**			**139**	**93.6%**

Source: Innovexplo, 2013

Table 11.2: Results for Standards used by Premier during the Drilling Program on the Hardrock Deposit from August 12, 2013 to December 31, 2013

Hardrock Project (From August 12, 2013 to December 31, 2013)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-2SD)	Upper process limit (+2SD)	Outliers	(%) passing quality control
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/GRAV	1191	3.33	4.37	11	99.08%
CDN_GS_6C	CDN Resources Laboratories LTD	5.95	FA/GRAV	477	4.99	6.91	12	97.48%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/GRAV	555	5.43	7.31	22	96.04%
CDN_GS_8A	CDN Resources Laboratories LTD	8.25	FA/GRAV	3	7.05	9.45	0	100.00%
SF67	Rocklabs Ltd	0.835	FA/GRAV	256	0.793	0.877	85	66.80%
SN60	Rocklabs Ltd	8.318	FA/GRAV	249	7.694	8.942	16	93.57%
TOTAL				**2731**			**146**	**94.65%**

Source: Innovexplo, 2015

Table 11.3: Results for Standards used by Premier during the Drilling Program on the Hardrock Deposit from January 2, 2014 to May 26, 2014

Hardrock Project (From January 2, 2014 to May 26, 2014)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-2SD)	Upper process limit (+2SD)	Outliers	(%) passing quality control
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/GRAV	207	3.33	4.37	3	98.55%
CDN_GS_5K	CDN Resources Laboratories LTD	3.85	FA/AA	160	3.33	4.37	2	98.75%
CDN_GS_6C	CDN Resources Laboratories LTD	5.95	FA/GRAV	114	4.99	6.91	4	96.49%
CDN_GS_6C	CDN Resources Laboratories LTD	3.85	FA/AA	26	4.99	6.91	0	100.00%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/GRAV	111	5.43	7.31	6	94.59%
CDN_GS_7B	CDN Resources Laboratories LTD	6.37	FA/AA	53	5.43	7.31	0	100.00%
SF67	Rocklabs Ltd	0.835	FA/GRAV	20	0.793	0.877	1	95.00%
SN60	Rocklabs Ltd	8.318	FA/AA	66	7.694	8.942	9	86.36%
TOTAL				**757**			**25**	**96.70%**

Source: Innovexplo, 2015

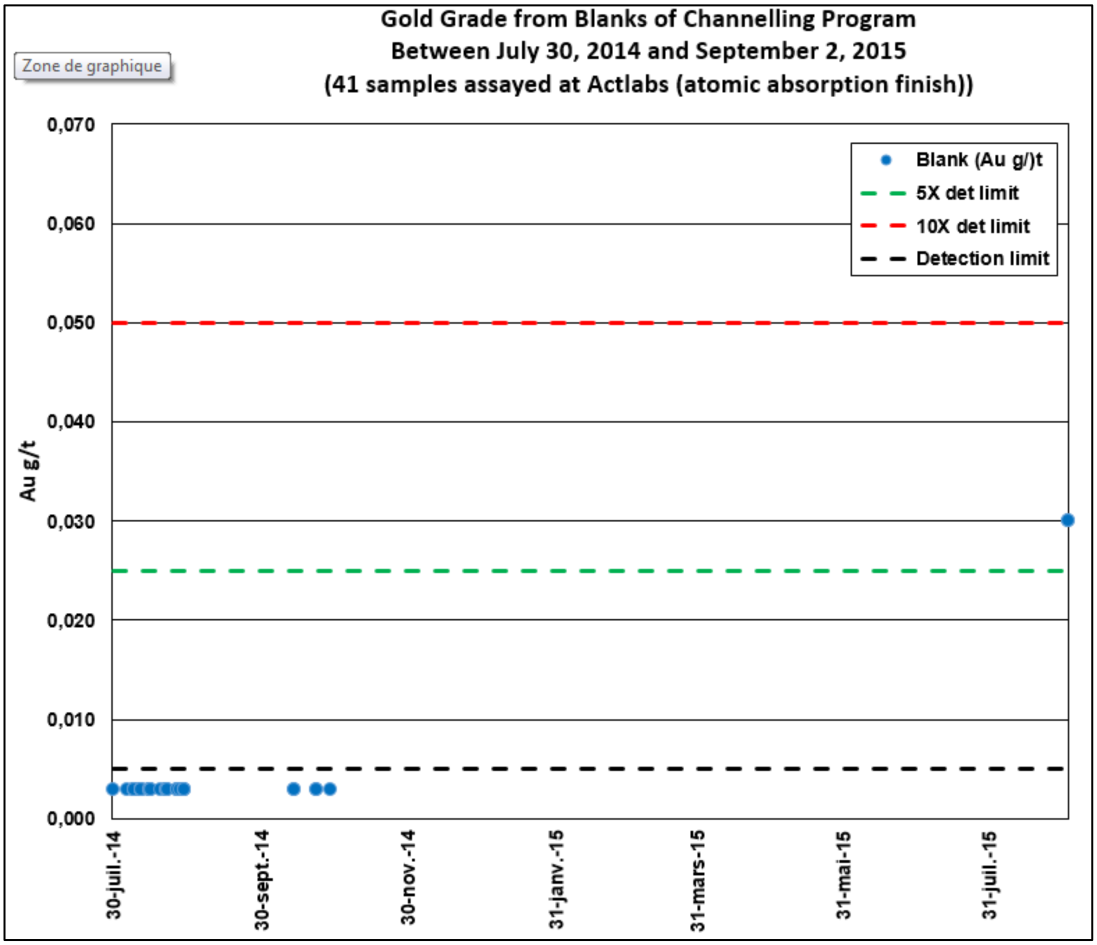
11.1.7.1 Blanks

The field blank used in the drilling program is from a barren sample of crushed white marble. One field blank is inserted for every 34 samples.

According to GGM's QA/QC protocol, if any blank yields a gold value above 0.05 g Au/t (10x detection limit for AA finish), the batch containing the blank should be re-assayed.

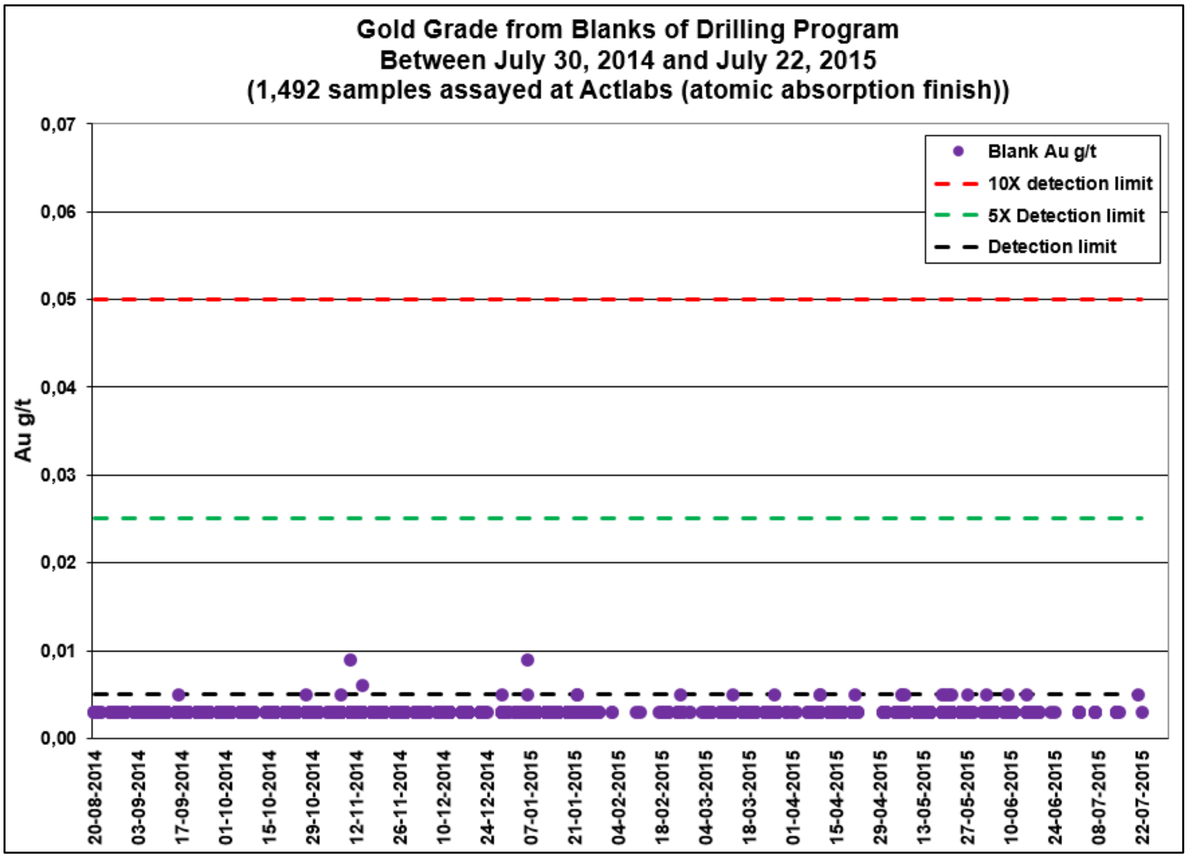
For the channelling program that ran from July 30, 2014 to September 2, 2015 on the Hardrock deposit, none of the 41 blank results (10x detection limit for AA finish) yielded a gold value above 0.05 g Au/t (Figure 11.1).

Figure 11.1: Results of Blank Samples used for Quality Control during Channelling Program Hardrock Deposit between July 30, 2014 and September 2, 2015. Detection Limit = 0.005 g Au/t for AA Finish



For the drilling program that ran from July 30, 2014 to July 22, 2015 on the Hardrock deposit, none of the 1,492 blank results (10x detection limit for AA finish) yielded a gold value above 0.05 g Au/t (Figure 11.2).

Figure 11.2: Results of Blank Samples used for Quality Control during Drilling Program on the Hardrock Deposit between July 30, 2014 and July 22, 2015. Detection Limit = 0.005 g Au/t for AA Finish



11.1.8 Certified Reference Material (Standards)

Two CRMs were inserted for every 34 samples during the channelling and drilling programs. Nine standards were used, with gold grades ranging from 0.417 to 8.595 g Au/t as follows:

- CDN-GS-P4B with a theoretical value of 0.417 ± 0.023 g Au/t;
- CDN-GS-P7J with a theoretical value of 0.722 ± 0.036 g Au/t;
- CDN-GS-1L with a theoretical value of 1.160 ± 0.050 g Au/t;
- CDN-GS-2P with a theoretical value of 1.990 ± 0.075 g Au/t;
- CDN-GS-5K with a theoretical value of 3.840 ± 0.140 g Au/t;
- CDN-GS-6C with a theoretical value of 6.030 ± 0.280 g Au/t;
- CDN-GS-7B with a theoretical value of 6.420 ± 0.230 g Au/t;

- SF67 with a theoretical value of 0.835 ± 0.021 g Au/t;
- SN60 with a theoretical value of 8.595 ± 0.223 g Au/t.

GGM quality control protocol stipulates that if any analyzed standard yields a gold value above or below three standard deviations ("3SD") of the certified grade for that standard, then the Project Manager is informed and must decide whether the batch containing that standard should be re-analyzed. All re-analyzed batches (pulps) were sent to ALS-Chemex in Thunder Bay.

The results of all standards used in the Hardrock channelling program carried out from July 30, 2014 to September 2, 2015 are summarized in Table 11.4, and those used in the drilling program from July 30, 2014 to July 22, 2015 are summarized in Table 11.5.

Overall, more than 97.50% of the available results for standards passed the quality control criteria for the channelling program, while more than 97.55% passed for the drilling program.

GMS is of the opinion that all results of the standards are reliable and valid.

Table 11.4: Results for Standards used by GGM during Channelling Program on Hardrock Deposit July 20, 2014 - September 2, 2015

Hardrock Project (From July 30, 2014 to September 2, 2015)								
Standard (CRM)	**Standard Supplier**	**Certified gold value (g/t)**	**Analytical method**	**Amount of results**	**Lower process limit (-3SD)**	**Upper process limit (+3SD)**	**Outliers**	**(%) passing quality control**
CDN_GS_2P	CDN Resources Laboratories LTD	1.99	FA/AA	2	1.765	2.22	0	100.00%
CDN_GS_5K	CDN Resources Laboratories LTD	3.84	FA/AA	39	3.46	4.24	2	94.87%
CDN_GS_6C	CDN Resources Laboratories LTD	6.03	FA/AA	40	5.31	6.75	0	100.00%
CDN_GS_7B	CDN Resources Laboratories LTD	6.42	FA/AA	1	5.73	7.11	0	100.00%
TOTAL				**80**			**2**	**97.50**

Table 11.5: Results for Standards used by Premier during the Drilling Program on Hardrock Deposit from July 30, 2014 to July 22, 2015

Hardrock Project (From July 30, 2014 to July 22, 2015)								
Standard (CRM)	Standard Supplier	Certified gold value (g/t)	Analytical method	Amount of results	Lower process limit (-3SD)	Upper process limit (+3SD)	Outliers	(%) passing quality control
CDN_GS_P4B	CDN Resources Laboratories LTD	0.417	FA/AA	474	0.348	0.486	21	95.57%
CDN_GS_P7J	CDN Resources Laboratories LTD	0.722	FA/AA	70	1.01	1.31	2	97.14%
CDN_GS_1L	CDN Resources Laboratories LTD	1.16	FA/AA	71	1.01	1.31	1	98.59%
CDN_GS_2P	CDN Resources Laboratories LTD	1.99	FA/AA	114	1.77	2.22	3	97.37%
CDN_GS_5K	CDN Resources Laboratories LTD	3.84	FA/AA	804	3.46	4.24	18	97.76%
CDN_GS_6C	CDN Resources Laboratories LTD	6.03	FA/AA	589	5.47	6.59	12	97.96%
CDN_GS_7B	CDN Resources Laboratories LTD	6.42	FA/AA	531	5.72	7.12	8	98.48%
SF67	Rocklabs Ltd	0.835	FA/AA	177	0.772	0.898	1	99.44%
SN60	Rocklabs Ltd	8.595	FA/AA	145	7.926	9.264	7	95.17%
TOTAL				**2975**			**73**	**97.55%**

11.1.9 Duplicates

The quality control protocol requires that a coarse duplicate be prepared for the 30th sample in each batch. The duplicate is prepared by taking half of the crushed material derived from the original sample. By measuring the precision of the coarse duplicates, the incremental loss of precision can be determined for the coarse-crush stage of the process, thus indicating whether two sub-samples taken after primary crushing is adequate for the given crushed particle size to ensure a representative sub-split.

Duplicates are used to check the representativeness of results obtained for a given population. To determine reproducibility, precision (as a percentage) is calculated according to the following formula:

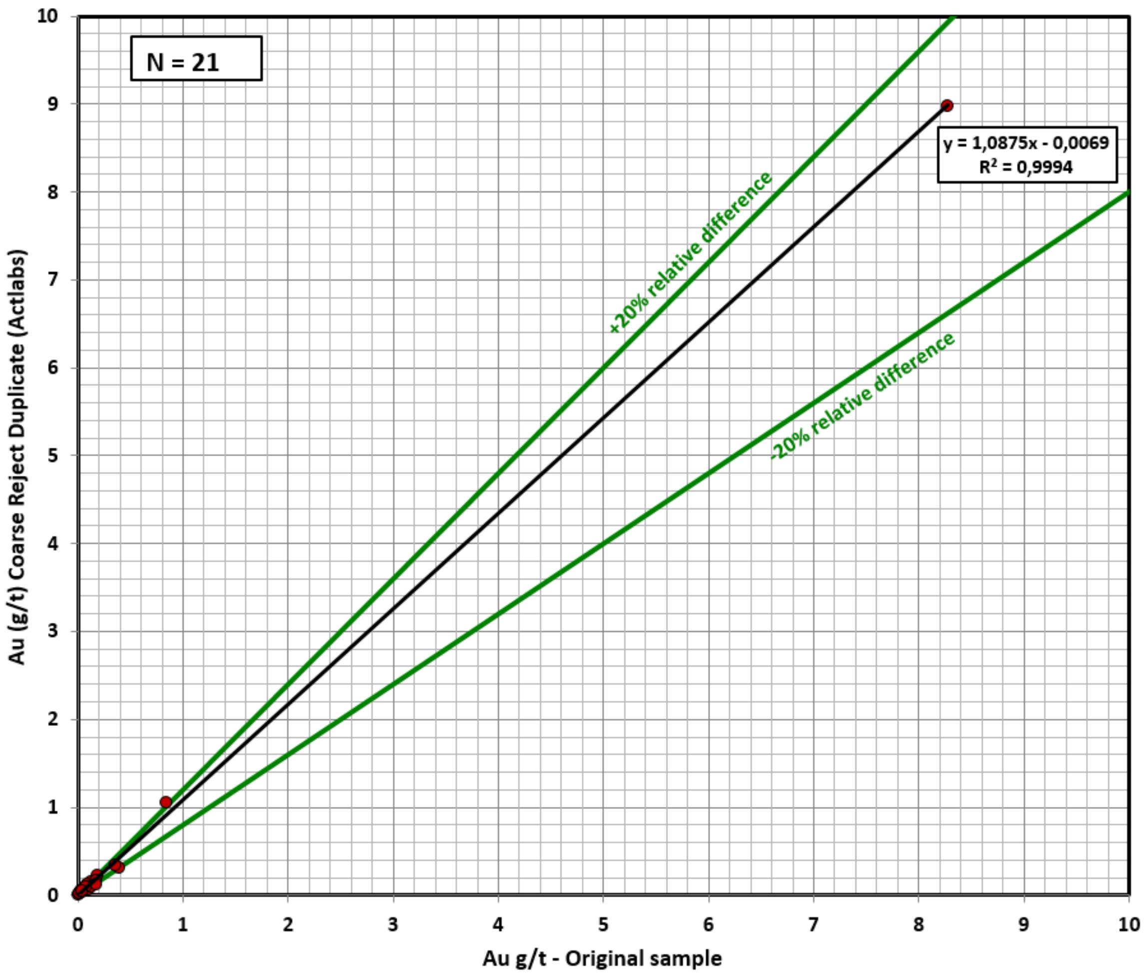
$$\text{Precision (\%)} = \frac{\text{(Duplicate Sample Gold Grade – Original Sample Gold Grade)}}{\text{Average Between Duplicate Sample Gold Grade and Original Sample Gold Grade}} \times 100$$

Precision ranges from 0 to 200% with the best being 0%, meaning that both the original and the duplicate sample returned the same grade.

A total of 21 original-coarse crush duplicate pairs (channelling) were identified in the database corresponding to the period between July 30, 2014 and September 2, 2015. Figure 11.3 shows a linear regression slope of 1.0875 and a correlation coefficient of 99.9%.

The correlation coefficient (%) is given by the square root of R^2 and represents the degree scatter of data around the linear regression slope. The results obtained indicate an excellent reproducibility of gold values with a gravimetric finish at Actlabs Geraldton. For gold values greater than 1 g Au/t, no outlier is observed on the graph because no duplicate pair is outside the lines marking a ±20% relative difference.

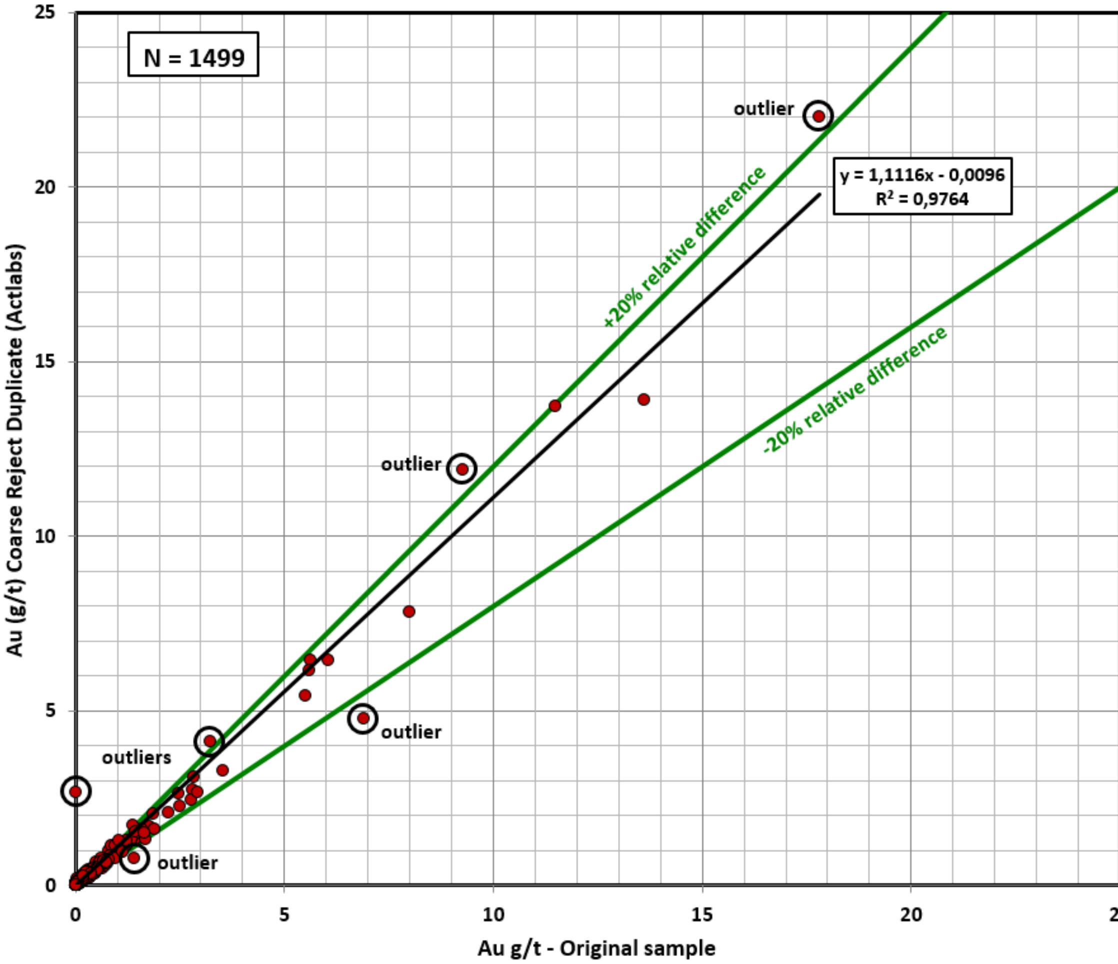
Figure 11.3: Linear Graph Comparing Original Samples and Crush Coarse Duplicate Samples (duplicate pairs) between July 30, 2014 and September 2, 2015 (channelling)



A total of 1,499 duplicate pairs (drilling) were identified in the database corresponding to the period between July 30, 2014 and July 22, 2015. Figure 11.4 shows a linear regression slope of 1.1116 and a correlation coefficient of 98.8%. The results obtained indicate an excellent reproducibility of gold values with AA finish at Actlabs Geraldton. For gold values greater than 1 g Au/t, only six outliers are observed on the graph because these duplicate pairs are outside the lines marking a ±20% relative difference.

GMS is of the opinion that the results obtained for the Hardrock coarse duplicates are reliable and valid.

Figure 11.4: Linear Graph Comparing Original Samples and Crush Coarse Duplicate Samples (duplicate pairs) for the Period between July 30, 2014 and July 22, 2015 (drilling)



11.1.10 Conclusions

A statistical analysis of the QA/QC data provided by GGM did not reveal any significant analytical issues. GMS is of the opinion that the sample preparation, analysis, QA/QC and security protocols used for the Hardrock Project follow generally accepted industry standards and that the data is valid and of sufficient quality to be used for Mineral Resource estimation.

11.2 Brookbank and Key Lake

Historical sample preparation, analysis and security protocols prior to 2010 are described in an earlier Technical Report by Scott Wilson RPA dated May 4, 2009, and entitled "*Technical Report on the Brookbank Gold Deposit, Beardmore-Geraldton Area, northern Ontario, Canada*". The following descriptions are applicable as from 2010 onwards.

11.2.1 Protocols Before Sample Dispatch and Quality Control

Drill core sampling protocols are described in Subsection 10.2.2. Quality control was achieved by inserting two standards (one high and one low), a blank and a duplicate for every batch of 34 samples sent to the assay laboratory. Other than the sampling and insertion of control samples, there was no other action taken at site.

Sample batches were placed into rice bags, sealed and transported to Actlabs sample preparation facilities in Geraldton in trucks by Premier staff. Sample pulps were shipped to the Actlabs in Thunder Bay for analytical work. Actlabs is independent of Premier and provides analytical services to the mining and mineral exploration industry worldwide. It is ISO 17025 accredited.

11.2.2 Security

The Premier Project Manager, a P. Geo, supervised all aspects related to sampling, recording, packaging and transportation of samples to the laboratory.

11.2.3 Sample Preparation and Analysis

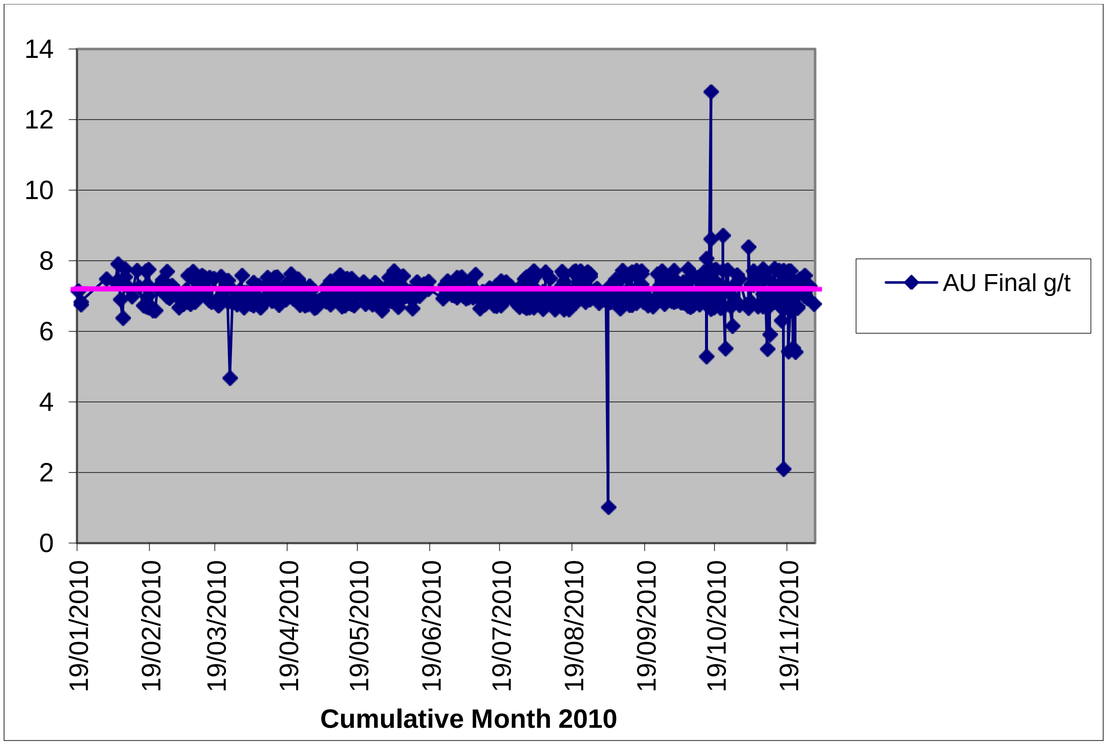
The Actlabs sample preparation and analysis procedures used for the Brookbank and Key Lake projects are similar to those already described above for the Hardrock Project.

11.2.4 Results of Quality Control

The performance of control samples (i.e. standards, blanks and duplicates) was assessed upon receipt of the results for each batch. Any results falling outside the failure limit of +/-3SD were rejected pending investigation into the source of error. Repeat analyses were conducted whenever significant failures were observed.

Micon has reviewed the control and monitoring charts compiled by Premier from 2009 to April 2013 and is satisfied that adequate measures were in place to ensure the accuracy of assays used in the resource databases. An example of the control charts compiled by Premier is shown in Figure 11.5.

Figure 11.5: Control Chart for one of Premier's Standard with Certified Value of 7.2 g Au/t



From the above figure, it is evident that failures were rarely encountered and hence, repeat analyses were not frequent.

11.2.5 Micon Comments

Micon considers that the sample preparation, security, and analytical procedures used at Brookbank and Key Lake are adequate to ensure credibility of the assays. The QA/QC procedures and protocols employed by GGM are sufficiently rigorous to ensure that the sample data are appropriate for use in Mineral Resource estimation. However, Micon recommends that GGM use a second laboratory as an umpire on 5 to 10% of its pulps in future sampling programs.

12. DATA VERIFICATION

12.1 Hardrock

The diamond drill hole database used for the 2016 Mineral Resource Estimate (the “2016 MRE”) presented herein was provided by Greenstone Gold Mines (“GGM”) and is referred to as the “GGM database” in this section. A drilling program in the Hardrock deposit resource area ended on July 20, 2015, and the database close-out date for the resource estimate update was established as November 18, 2015. The last hole included in the database is MM754B. A significant re-sampling program was also completed in 2015 by GGM, including 6,411 new samples from 79 historical diamond drill holes. These were added to the GGM database for the resource estimate update herein. The 2014-2015 stripping program was also included in the update.

GMS’s data verification included visits to the Hardrock field sites (outcrops and drill collars), as well as to the logging facilities. It also included an independent re-sampling of selected core intervals and a review of drill hole collar locations, assays, the QA/QC program, downhole surveys, the information on mined-out areas and the descriptions of lithologies, alterations and structures. The site visit was completed by Réjean Sirois between August 1 and 4, 2016.

12.1.1 Historical Work

The historical information used in this Report has been taken mainly from reports produced before the implementation of NI 43-101 Canadian Standards of Disclosure for Mineral Projects. In some cases, these reports provide little information on sample preparation, analyses or security procedures.

12.1.2 GGM Database

G Mining Services Inc.(“GMS”) was granted access to the certificates of assays for all holes in the latest drilling programs that took place between May 2014 and July 2015. Assays were verified for 2% of the drill holes from these programs.

Minor errors of the type normally encountered in a project database were identified and corrected. The final database is considered to be of good overall quality. GMS considers the GGM database for the Hardrock deposit to be valid and reliable.

12.1.3 Greenstone Gold Mines Diamond Drilling

The historical surface drill holes collars on the Hardrock deposit were either professionally surveyed or surveyed using a Trimble GPS unit without post-processing. However, the 2015 drill hole collars were surveyed using an RTK system with millimetre precision in all directions, including elevation.

Underground drill holes were compiled by Greenstone Economic Development Corporation (GEDC). However, these holes were excluded from the current resource estimate since the location data is considered unreliable, and the assay results could not be verified.

Downhole surveys were conducted on the majority of the surface holes. The Gyro and/or Reflex survey information was verified for 5% of the drill holes from the latest drilling programs. Minor errors were observed in the downhole surveys and corrections were made to the database. For the 2015 drilling program, final collar azimuths and dip measurements were collected directly on the casing using an APS system. Gyro, RTK and APS survey methods were reviewed during the site visit. Figure 12.1 and Figure 12.2 show the different survey tools and some examples of drill sites that were reviewed during the site visit.

During the GMS site visit, a total of seven drill hole collars were checked for X-Y accuracy. A handheld Garmin GPS was used to collect ground survey data, as summarized in Table 12.1. Given the accuracy of handheld GPS, the results are judged satisfactory by GMS. Figure 12.1 shows some examples of drill hole collars surveyed during the site visit.

Table 12.1: Drill Hole Collar Checks - 2016 Site Visit

Hole-ID	Check		Database		Difference	
	Easting	Northing	Easting	Northing	Easting	Northing
88-17A	504,781	5,502,825	504,781	5,502,827	0.1	2.0
EP100	504,451	5,502,970	504,450	5,502,969	-1.5	-0.8
EP120	504,400	5,502,999	504,402	5,502,998	1.6	-0.7
EP161	504,900	5,502,929	504,900	5,502,930	0.4	0.8
MM267	504,798	5,502,801	504,800	5,502,800	2.3	-1.1
MM534	504,503	5,502,963	504,501	5,502,965	-2.4	1.7
MM598	504,247	5,502,968	504,250	5,502,964	3.2	-4.0

Figure 12.1: Drill Hole Collars Surveyed during GMS 2016 Site Visit

12.1.4 GGM Logging, Sampling and Assaying Procedures

GMS reviewed several sections of mineralized core while visiting the on-site core logging and core storage facilities. All core boxes were labelled and properly stored outside. Sample tags were still present in the boxes and it was possible to validate sample numbers and confirm the presence of mineralization in witness half-core samples from mineralized zones.

Drilling was not underway in the resource area during GMS' site visit. GGM personnel explained the entire path of the drill core, from the drill rig to the logging and sampling facility and finally to the laboratory (Figure 12.2). GMS is of the opinion that the protocols in place are adequate.

Figure 12.2: Core Logging Procedures Reviewed during Site Visit

12.1.5 Independent Re-sampling

GMS re-sampled a series of intervals from the latest drilling program. During the site visit, quarter-splits of selected core intervals were cut by GGM personnel. The author collected several samples representing different types of host rocks and a wide range of gold grades were re-analyzed at Actlabs Geraldton. Samples were collected in a random order inside relevant mineralized intercepts. For each zone and drill hole, one sample was collected at around 20 m interval, when possible. Only samples grading more than 1.0 g Au/t were selected and 50 cm of quarter core splits were collected randomly in the sample interval.

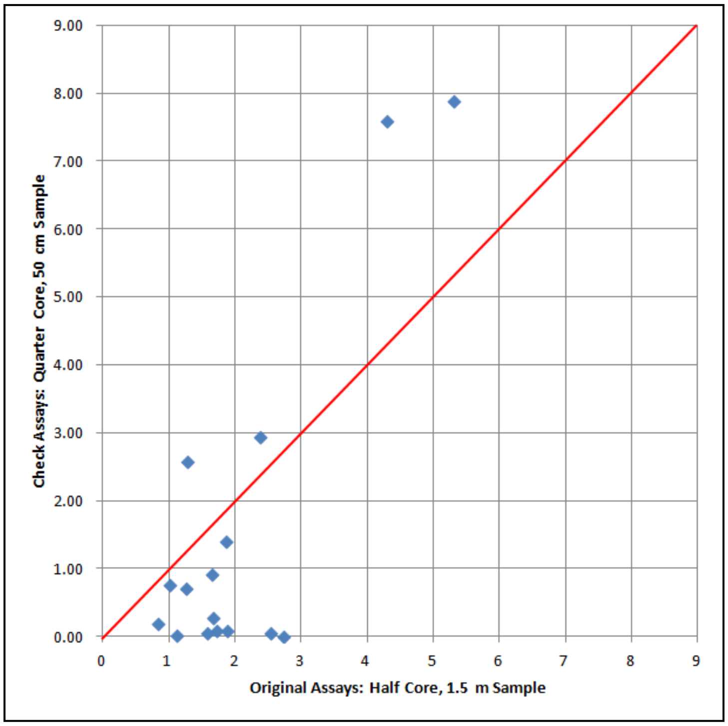
A total of 16 samples were assayed for gold using fire assay with AA finish. Samples assaying more than 5 g Au/t with AA were rerun with gravimetric finish. Table 12.2 presents the results of the field duplicate compared to the original samples.

Table 12.2: Original and Re-sampling Gold Analysis Results

DDH	Zone	From (m)	To (m)	Length (m)	Original Grade (g Au/t)	Lab Check (g Au/t)	Check Sample #
MM444	3300	481.5	482	0.5	4.29	7.59	262701
MM444	11140	514.1	514.6	0.5	2.54	0.05	262702
MM444	11140	532.9	533.4	0.5	0.82	0.21	262703
MM534	3600	314	314.5	0.5	5.31	7.90	262704
MM700	3205	333.4	333.9	0.5	1.00	0.78	262705
MM700	3205	355	355.5	0.5	1.85	1.41	262706
MM700	3205	368.6	369.1	0.5	1.28	2.58	262707
MM752	3500	333.8	334.25	0.45	1.67	0.29	262708
MM752	3105	458.2	458.7	0.5	1.58	0.07	262709
MM752	3105	479.8	480.25	0.45	1.11	0.03	262710
MM494	3105	341.3	341.8	0.5	1.26	0.72	262711
MM494	3105	361.6	362.1	0.5	2.72	0.01	262712
MM494	3105	383.2	383.7	0.5	2.38	2.95	262713
MM503	3205	481.8	482.3	0.5	1.87	0.09	262714
MM503	3205	499.3	499.8	0.5	1.72	0.10	262715
MM503	3205	528.5	529	0.5	1.64	0.93	262716

Figure 12.3 presents a linear graph comparing original samples and the field duplicate samples for all 16 samples. This graph displays that six out of 16 samples were reproduced within a 50% confidence level. Two more samples yielded a higher result compared to the original assay (+77% and +102%). The remaining samples (8) all show a significant decrease in gold grades, ranging from 75% to near 100%. Since one sixth (1/6) of core samples were randomly selected in the original sample interval (0.50 m quarter core interval versus 1.5 m half core), GMS is satisfied with the results given the mineralization style of gold and the inherent nugget effect.

Figure 12.3: Original Assays Compared to Check Assays



12.1.6 Mined-out Voids

Considerable effort has been made to improve the accuracy of the stope and drift 3D objects to provide a more accurate representation of the mined-out volumes in the historical workings. In 2015, a thorough archival search was undertaken by GGM and yielded additional historical plan views, cross sections and longitudinal views. An exhaustive compilation of breakthrough drilling was also completed by GGM. This additional information allowed the 3D model to be adjusted and corrected, and also provided additional missing stopes and drifts.

Based on the type of data used to model each void, the voids were classified as medium- or high-precision.

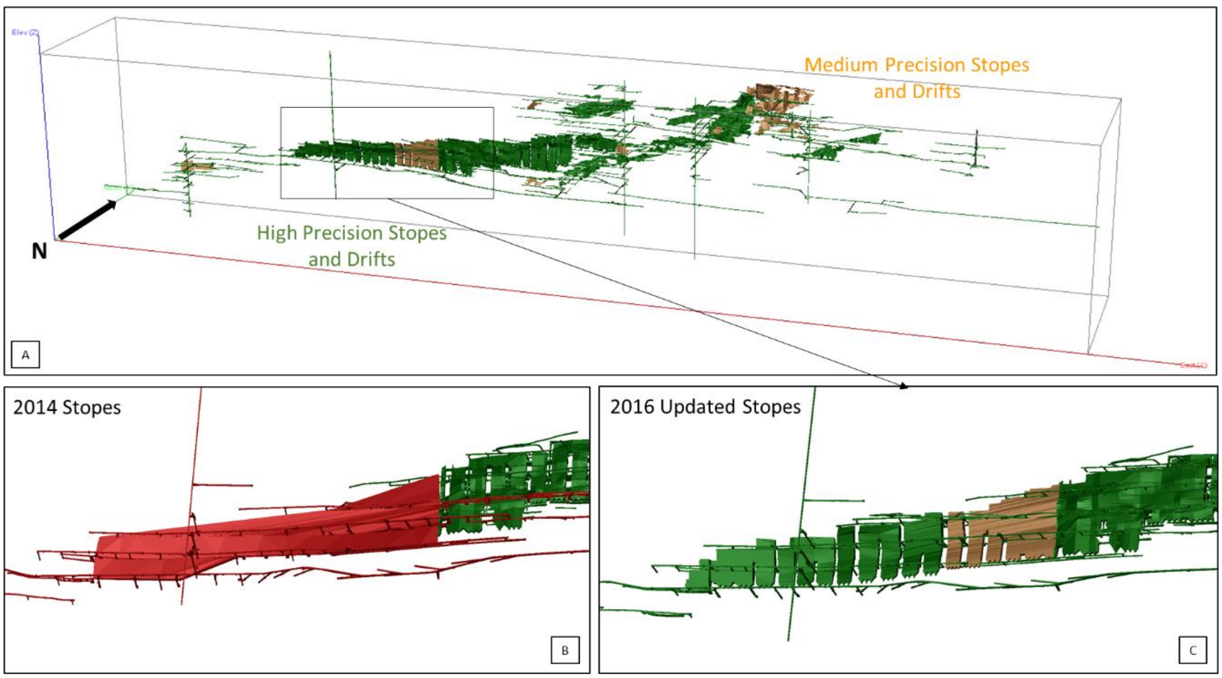
Medium-precision voids: modelled using only digitized longitudinal views combined with breakthrough drilling information.

High-precision voids: modelled using digitized plan views and/or cross sections with accurate location information for drift and stope positions.

In the end, the new information allowed all the low-precision stopes of the 2014 model to be upgraded to medium-precision in the 2016 model.

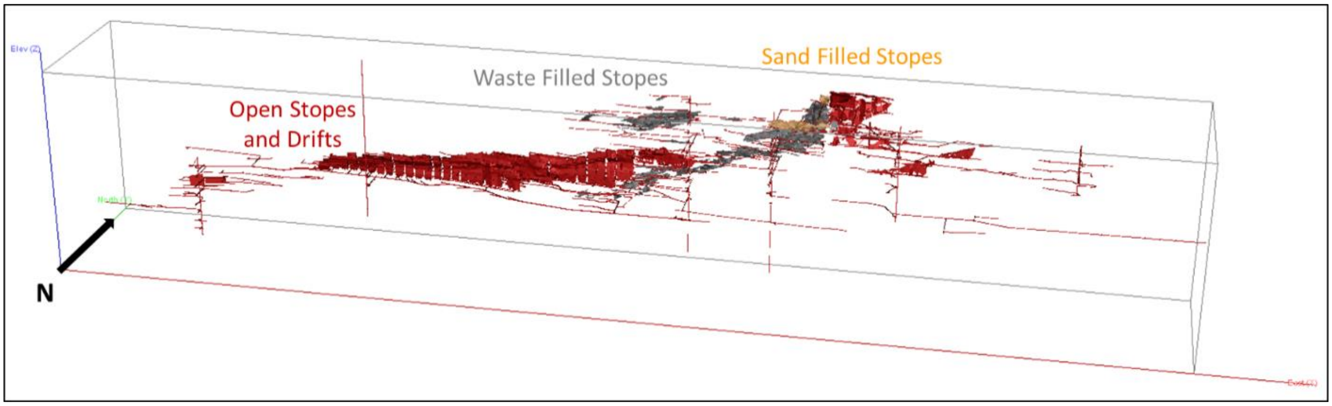
Figure 12.4 shows a compilation of the underground voids based on their level of precision as a result of the 2016 update.

Figure 12.4: Isometric View looking NNW showing a Compilation of the Mined-out Underground Voids: A) Overall View of Stopes and Drifts by Level of Precision; B) Close-up View of the Stopes Modelled in 2014; C) Close-up View of the Stopes Updated in 2016



Information on the type of backfill in the stopes was updated from the 2014 compilation and integrated into the database. The result is a classification of stopes according to three types of backfill: open (filled with water); waste (corresponding to a mix of waste and “clinker”, a reject from the process plant); and sand (corresponding to a mix of wet sand and gravel). Figure 12.5 shows a compilation of the underground voids based on backfill type. The specific gravities for each type of backfill were provided by GGM.

Figure 12.5: Isometric View looking NNW showing a Compilation of the Mined-out Underground Voids Based on their Backfill Type



For the 2016 update, the total stopes model corresponds to 89% of the total historical milled tonnes at an average density of 2.84 g/cm^3 for the Hardrock deposit, including stopes in the Hard Rock, MacLeod-Cockshutt, Mosher Long Lac and Macleod-Mosher mines.

GMS considers the refinement of the voids triangulation to be of good quality and reliable.

12.1.7 Conclusion

Overall, GMS is of the opinion that the data verification process demonstrated the validity of the data and protocols for the Hardrock Project. GMS considers the GGM database to be valid and of sufficient quality to be used for the Mineral Resource estimation.

12.2 Brookbank and Key Lake

Micon achieved data verification by visiting and inspecting the facilities of Actlabs Geraldton where the GGM samples for the Brookbank and Key Lake Projects are analyzed (effective June 2010). The laboratory visits were complemented by two site visits to the Project areas, conducting independent repeat analyses of selected sample pulps, analyzing monitoring reports on the performance of control samples and performing resource database checks.

In order to ensure that sample analyses were being performed to current industry standards, Micon visited and inspected the Actlabs Geraldton on February 15, 2011, November 9, 2011 and on March 19, 2013. The sample preparation facilities are well maintained with adequate measures in place to avoid contamination between samples. Analytical equipment and the entire complex are kept in neat condition. Records of calibration and performance parameters are kept up to date for both testing and measuring equipment. The laboratory's internal QA/QC also includes the insertion of a blank and a standard in every sample batch. Analytical procedures are satisfactory and include metallic screening analyses.

12.2.1 Site Visits

Charley Murahwi, P. Geo., conducted the Brookbank and Key Lake site visits from November 8 to 9, 2011 and on March 19, 2013. The tasks accomplished included the following:

- Review of drilling, surveying, and sampling procedures;
- Verification of some of the drill hole collars and review of the survey procedures;
- Examination of drill core/visual verification of mineralized intercepts;
- Partial validation of analytical results by comparing assays with drill core intercepts;
- Review of QA/QC protocols.

In summary, the main observations are that (a) the diamond drilling of NQ size core yields good core recoveries and representative samples; (b) the techniques used for down-hole surveys are appropriate; (c) drill core handling, logging and sampling are conducted satisfactorily; (d) QA/QC protocols in place are sound; (e) monitoring of QA/QC results is done on a real time basis; and (f) assay results generally match the mineralized intercepts observed in drill cores. Storage facilities for samples and drill cores are neat and secure - see Figure 12.6.

Figure 12.6: Brookbank Sample/drill Storage Facilities

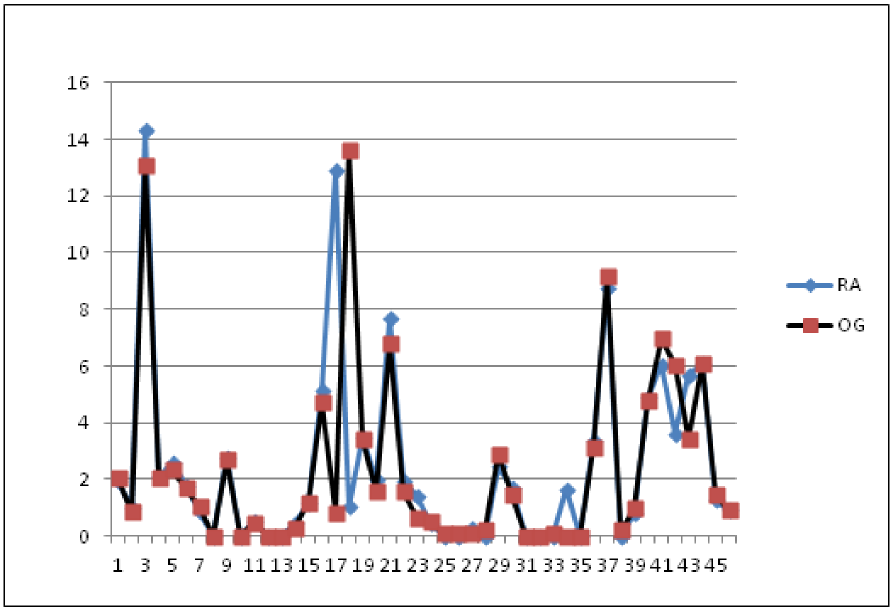
Micon, 2013

12.2.2 Independent Repeat Analyses

Micon selected 46 sample pulps encompassing a wide range of assay values (from low through medium to high) and re-numbered them in a different sequence before submitting them to Actlabs Geraldton for repeat analyses using the same method previously used.

Comparisons between original ("OG") and repeat assays ("RA") Figure 12.7 confirm the laboratory's high degree of accuracy (lack of bias) and precision with the exception of one mismatch. This mismatch is attributed to mistaken sample switch.

Figure 12.7: Comparison of Original (OG) and Repeat Analyses (RA)



12.2.3 Review of Monitoring Reports and Control Charts

Analysis of the monitoring reports reveals that adequate control samples incorporating high quality certified reference materials, blanks and duplicates were used to ensure accuracy of the analytical database. In a few instances where standards failed, appropriate investigations were conducted and re-assaying was conducted whenever it was deemed necessary. Micon did not identify any flaws in the QA/QC protocols.

12.2.4 Historical Drilling Data (2000 - 2009 Holes)

Micon reviewed the validation work that was conducted on the historical drilling data by Scott Wilson in 2009 and concurs with their findings in justifying the inclusion of the drill hole data in the resource database.

12.2.5 Database Validation

The resource database validation conducted by Micon involved the following steps:

- Checking for any non-conforming assay information such as duplicate samples and missing sample numbers;
- Verifying collar elevations against survey information for each drill hole;
- Verifying collar coordinates against survey information for each drill hole;

- Verifying the dip and azimuth against survey information for each hole;
- Comparing the database assays and intervals against the original assay certificates and drill logs.

The main issue identified is lack of sampling and/or selective (incomplete) sampling in some zones/drill intercepts falling within the mineralization envelopes for the Brookbank project. In all such zones, Micon adopted a prudent approach and assigned a detection limit assay value of 0.01 g Au/t. It is recommended that GGM completes the assaying prior to the next resource update.

Some minor discrepancies involving duplication of sample intervals where duplicate analyses had been conducted were easily corrected.

12.2.6 Data Verification Conclusions and Recommendations

On the basis of the verification procedures described above, Micon considers the database generated by GGM to be representative of the main characteristics of the Brookbank and Key Lake projects and therefore suitable for use in Mineral Resource estimation. The insertion of a detection limit value of 0.01 g Au/t for a missing assay may likely lead to an understatement of the resource grade for Brookbank but nonetheless it ensures that all intercepts are used in the estimate with no danger of over-estimating the grade.

Micon recommends that all future blanks used to monitor contamination between samples should look exactly the same as the other samples in the batch to avoid preferential attention. This, coupled with periodic check analyses of the sample pulps at an umpire laboratory, will always ensure a high quality database.

13. MINERAL PROCESSING AND METALLURGICAL TESTING

This section summarizes all the relevant testwork performed on the Hardrock deposit. It is divided into two sections, one describing testwork done before the FS and one dedicated to the testwork performed during the course of the FS. Only recent reports (from 2011 onwards) are summarized in this section.

13.1 Previous Testwork

Some mineralogy, grindability and gold recovery testwork was performed prior to the start of the FS. The key reports from 2011 to 2013 are summarized in this section. The reference documents are listed below.

- SGS Lakefield Research Limited, *An Investigation into Gold Recovery from Hardrock Project Ore, Final Report-12400-001*, March 1, 2011;
- SGS Canada Inc., *The Recovery of Gold from the Hardrock Project - Phase 2 Samples, Final Report-12400-002* -, December 11, 2012;
- McClelland Laboratories, Inc., *Whole Ore Cyanidation Testing - Project AF Drill Hole Reject Composites, MLI Job No. 3817*, September 24, 2013;
- SGS Canada Inc., *QEM Automated Rapid Mineral Scan,* Report 14117-001 – MI6000-OCT13 -, October 31, 2013.

13.1.1 Gold Recovery Testwork at SGS Lakefield (Phase 1)

Samples were sent to SGS Lakefield Research Limited in March of 2010. Two composites were prepared and were subjected to head analyses, mineralogy, Bond Work Index determination, gravity separation, gravity tailings flotation and whole ore, gravity tailings and flotation concentrate cyanidation.

13.1.1.1 Head Assays

Composites 1 and 2 were submitted for gold analysis according to the metallic sieve protocol. Two 1 kg sample of each composite were submitted for coarse gold analysis (+/- 106 µm or 150 mesh fractions). The fine fraction was assayed in duplicate (analysis “a” and “b”). The results are presented in Table 13.1.

Table 13.1: Gold Head Analyses by Metallic Sieve

Sample	Head Grade Au (g/t)	+ 106 µm			- 106 µm Au (g/t)		
		Distribution		Au			
		Mass (%)	Au (%)	(g/t)	Mean	a	b
Composite 1 (A)	3.98	3.03	6.01	7.90	3.86	3.99	3.73
Composite 1 (B)	3.92	2.22	4.37	7.72	3.84	3.62	4.05
Composite 2 (A)	3.42	2.57	9.09	12.1	3.20	3.23	3.16
Composite 2 (B)	3.13	1.77	1.92	3.41	3.13	3.21	3.05

The metallic sieve gold analyses indicate the presence of free gold in the samples but only a small concentration is found in the coarse fraction. The difference in gold content in the coarse fraction between Composite A and Composite B and the variation between the gold content in the coarse fraction of Composite 2 (A) and Composite 2 (B) suggests the presence of fine free gold.

13.1.1.2 Mineralogy

A sample of each composite underwent an analysis of the rock forming components using light microscopy, XRD, chemical analysis and SEM techniques. Table 13.2 lists Composite 1 and Composite 2 constituents. Other trace constituents include Fe-Ti oxides, amphibole, apatite and other sulphides.

Table 13.2: Constituents of Composite 1 and Composite 2

Sample	Composite 1 (wt %)	Composite 2 (wt %)
Quartz	26.2	32.5
Plagioclase	24.4	8.3
Ankerite	11.2	6.2
Chlorite	10.4	5.6
Muscovite	9.8	6.9
Pyrite	4.7	6.9
Clays	2.8	2.3
Biotite	2.7	1.8
Iron Oxides	1.8	18.8
Arsenopyrite	1.2	0.4
Siderite	1.2	7.2
Calcite	1.0	0.1
Pyrrhotite	0.7	1.8

13.1.1.3 Grindability Testwork

A standard Bond ball mill grindability test was completed on each composite (closing screen size of 150 µm). The results are shown in Table 13.3. Composite 1 falls into the moderately hard category while Composite 2 can be considered of medium hardness according to SGS Lakefield's database.

Table 13.3: Composites 1 and 2 Bond Ball Mill Grindability Tests Results

Sample	Work Index (kWh/t)	Hardness Percentile
Composite 1	16.0	65
Composite 2	14.6	51

13.1.1.4 Gravity Separation

Gravity separation tests including a Knelson Concentrator and a Mozley table were performed in order to examine the amenability of the ore to gravity concentration and produce gravity tailings for cyanidation and flotation tests.

The effect of grind size was not investigated in these tests as all the test feeds were approximately 80% passing 100 µm. Gravity gold recovery ranged from 11.3% to 23.6% for Composite 1 and between 9.2% and 16.1% for Composite 2.

13.1.1.5 Flotation

The gravity tailings were subjected to flotation testing. The objective of the initial kinetic rougher flotation tests was to evaluate the impact of grind size on gold recovery and determine the conditions to generate bulk concentrate for further testwork. The purpose of the tests was to recover gold in a sulphide rougher concentrate.

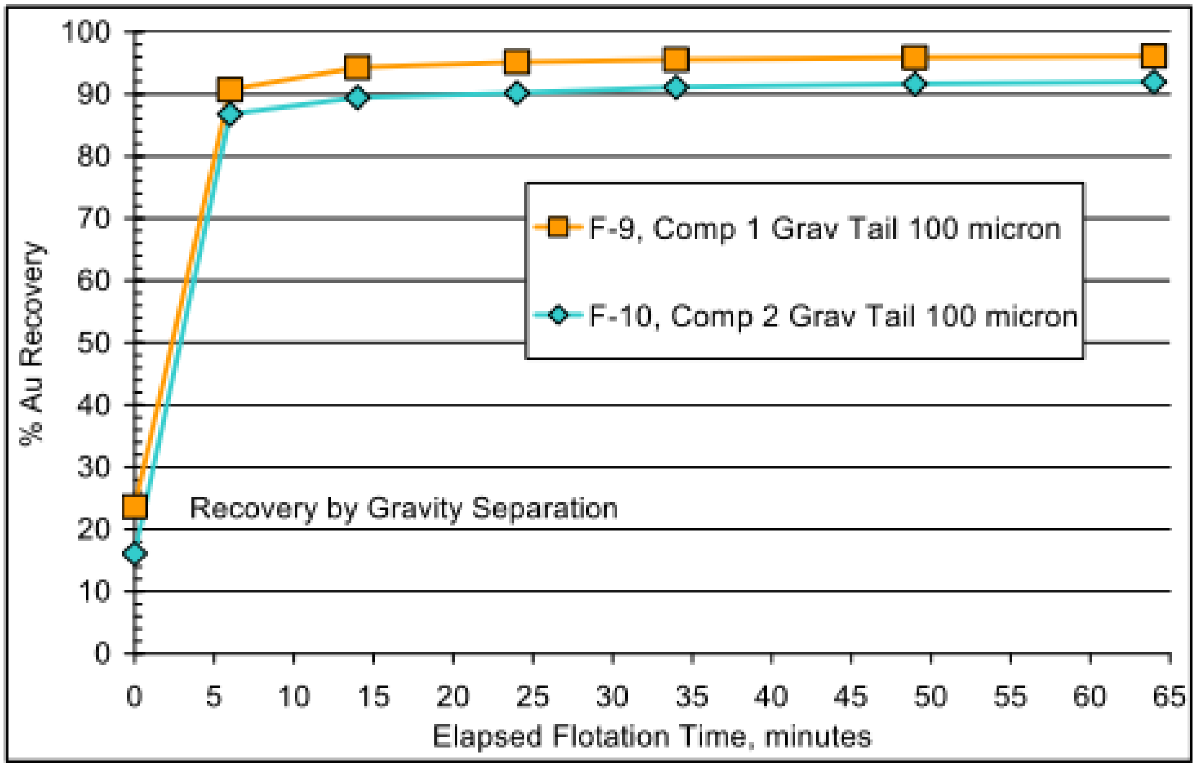
Flotation tests were carried out at grinds of 47, 70 and 95 µm (P80) for Composite 1. A concentrate mass recovery of 17% to 41% was achieved with gold grades ranging from 16.4 g Au/t at 95% overall recovery (coarsest grind) to 7.3 g Au/t at 98% overall recovery (finest grind). The tailings gold grade ranged from 0.19 g Au/t (P80 = 95 µm) to 0.15 g Au/t (P80 = 47 µm).

For Composite 2, flotation tests were performed at P80's of 51, 75 and 108 µm. Approximately 23% to 33% mass was recovered to the concentrate. Concentrate gold grades ranged from 11.3 g Au/t at 93% overall recovery (P80 =108 µm) to 9.1 g Au/t and 95% overall recovery (P80 = 51 µm). The tailings gold grade was 0.22 g Au/t for the 51 µm sample and 0.29 g Au/t for the 108 µm sample.

Bulk flotation tests were conducted to generate concentrate for cyanidation. The results of the 10 kg bulk tests on the P80 = 100 µm gravity tailings were comparable to the 2 kg flotation tests on similar feed. The correlation between gold (non-gravity recoverable) and sulphide sulfur recovery indicates an association.

Figure 13.1 presents the results of the bulk flotation tests on Composite 1 and Composite 2.

Figure 13.1: Gravity Tailings Bulk Rougher Kinetics Results



Title: Gravity Tailing Bulk Rougher Kinetics Results	Author: SGS Lakefield Date: 11/03/01

13.1.1.6 Cyanidation

Whole ore cyanidation tests were conducted to examine cyanide leach amenability. The effect of particle size on gold extraction was also investigated. Bottle roll tests were completed at three grind sizes. Gold extraction ranged from 69% to 79% for Composite 1 and from 81% to 84% for Composite 2. Increased extraction with fine grinding is more pronounced for feed P80's larger than approximately 70 µm. Below this size, the gain in recovery is less significant.

Gravity tailings cyanidation tests also aimed to determine the ore amenability to cyanide leaching but also examined the effect of regrind on gold extraction. Bottle roll tests were performed under the same conditions as the whole ore tests. Cyanide extraction ranged from 64% to 70% for Composite 1 and between 75% and 82% for Composite 2. As observed in the previous test, regrind fineness was also less beneficial for regrinds below 70 µm. Combined gold recovery from gravity concentration and cyanidation was approximately 68% to 73% for Composite 1 and 78% to 83% for Composite 2.

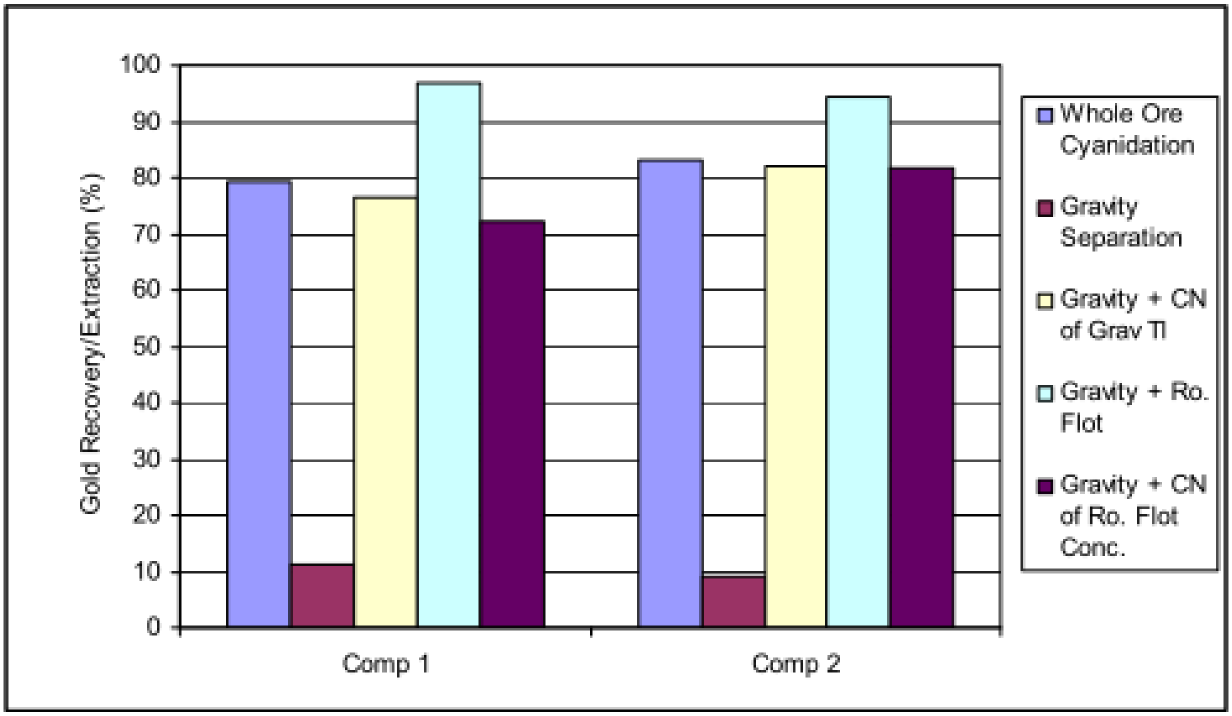
Bottle roll tests were carried out on flotation concentrates and on reground concentrates (P80 = 10 µm). Gold extraction by cyanidation increased with finer grinding for both composites. For Composite 1, recovery increased from 60.1% to 67.3% at 11 µm. For Composite 2, the recovery reached 87.2% at the finer size compared to 77.7% for the unground flotation concentrate. Combined with the gravity recovery, the overall gold recovery of the reground samples reached 72.2% for Composite 1 and 81.9% for Composite 2.

Table 13.4 below summarizes the whole ore, gravity tailings and flotation concentrate cyanidation tests results. Although recoveries vary for each process, the final tailings gold grades and calculated head grades are similar for both the Composite 1 and Composite 2 tests series. Figure 13.2 presents the results of the combined methods.

Table 13.4: Cyanidation of Whole Ore, Gravity Tailings and Flotation Concentration

Test	Feed	CN Leach				Combined Recovery (%)	Final Tailings (g Au/t)
		Reagent Consumption (kg/t of Whole Ore)		Grind (µm)	Recovery (%)		
		NaCN	CaO				
CN-2	Composite 1 Whole Ore	1.5	0.4	59	79.2		1.01
CN-8	Composite 1 Gravity Tail	0.8	0.5	68	62.1	73.3	0.99
CN-14	Composite 1 Flot. Conc.	0.9	0.7	11	48.6	72.3	1.04
CN-5	Composite 2 Whole Ore	1.3	0.5	66	83.2		0.51
CN-11	Composite 2 Gravity Tail	0.8	0.8	68	72.9	82.1	0.53
CN-16	Composite 2 Flot. Conc.	1.0	1.0	11	65.8	81.9	0.52

Figure 13.2: Comparison of Combined Results



Title: Comparison of Combined Results	Author: SGS Lakefield Date: 11/03/01

It can be seen that gravity separation followed by gravity tailings cyanidation achieved similar results to whole ore cyanidation. Gravity separation followed by flotation yielded the highest recoveries but assumes that the flotation concentrate can be sold as smelter feed.

13.1.2 Gold Recovery Testwork at SGS Lakefield (Phase 2)

Samples were sent to SGS Lakefield in May 2011. This second phase of work followed the previous testwork campaign completed on Composite 1 and Composite 2. For Phase 2, two new composites were prepared (Composite IF1 and Composite P2). Composites IF1 and P2 were subjected to gold deportment by mineralogy analysis, gravity recoverable gold determination and whole ore flotation evaluation. Moreover, gravity tailings flotation and flotation rougher concentrate cyanidation tests were included in the program.

13.1.2.1 Head Assays

Composites IF1 and P2 were submitted for gold analysis according to the metallic sieve protocol. Two 1 kg sample of each composite were submitted for coarse gold analysis (+/- 106 µm or 150 mesh fractions). The fine fraction was assayed in duplicate (analysis “a” and “b”). The results are presented in Table 13.5.

Table 13.5: Gold Head Analyses by Metallic Sieve

Sample	Head Grade (g Au/t)	+ 106 µm			- 106 µm (g Au/t)		
		Distribution		Au			
		Mass (%)	Au (%)	(g/t)	Mean	a	b
Composite IF1 (A)	4.53	1.96	3.86	8.91	4.44	4.32	4.57
Composite IF1 (B)	4.66	1.91	3.11	7.58	4.60	4.62	4.58
Composite P2 (A)	6.02	2.39	5.77	14.5	5.81	5.70	5.92
Composite P2 (B)	5.37	2.87	5.91	11.0	5.20	4.98	5.43

The metallic sieve gold analysis indicated some free gold in the samples but only a minimal concentration is found in the coarse fraction. The difference in gold content in the coarse fraction between Composite IF1 and Composite P2 and the variation between the gold content in the coarse fraction of IF1-A and IF1-B and between P2-A and P2-B suggests the presence of fine free gold.

13.1.2.2 Mineralogy

A sample of each composite underwent a gold deportment study to provide the mode and occurrence of the microscopic gold. The gold chemical composition was analyzed using SEM-EDS (Scanning Electron Microscopy-Energy Dispersive Spectroscopy). The major gold mineral in the samples was native gold. In Composite IF1, the overall gold was 22% liberated, 20% attached and 58% locked. In Composite P2, 28% of the gold was liberated, 31% was attached and 41% was locked. The study determined that gold in the samples could effectively be concentrated by gravity methods.

13.1.2.3 Grindability Testwork

A standard Bond rod mill grindability test (closing screen size of 14 mesh / 1180 µm) was completed on a separate sample made by combining ore from three zones. The composite was also subjected to a standard Bond ball mill grindability test (closing screen size of 150 µm).

With a rod mill work index ("RWI") of 17.3 kWh/t and a ball mill work index ("BWI") of 16.5 kWh/t, the sample can be considered moderately hard with respect to both parameters according to SGS Lakefield's databases.

13.1.2.4 Gravity Recoverable Gold

A gravity recoverable gold ("GRG") test was performed on a sample from each Composite IF1 and P2. The GRG tests yield the maximum amount of gold that can be recovered by gravity. Plant recoveries are typically lower.

For IF1, it was found that 8% of the gold could be recovered to a gravity concentrate at a P80 of 570 µm. A 14% recovery is reached at 241 µm and 24% at 60 µm. For P2, 9% could be recovered at 570 µm, 17% at 267 µm and 31% at 106 µm.

13.1.2.5 Gravity Separation

Gravity separation tests (Knelson/Mozley) were performed on IF1 and P2 to produce gravity tailings for flotation tests, for bulk flotation tests followed by concentrate cyanidation and for a cyanidation test. Gold recovery varied from 38% to 39% for IF1 and between 17% and 40% for P2. The grind sizes for all five tests ranged from 80 to 101 µm.

13.1.2.6 Flotation

Whole ore flotation tests were carried out to evaluate the effect of rougher concentrate cleaning on overall concentrate mass reduction and final concentrate gold grade and recovery.

For Composite IF1, the cleaning stages reduced the second cleaner concentrate mass to 12% with a grade of 30.3 g Au/t and an 85% recovery. After a coarse regrind to 45 µm (P80), the mass pull was 14%, the gold grade was 27.0 g/t and the recovery was increased to 89%.

For Composite P2, the second cleaner concentrate showed 7% mass distribution, a grade of 54.8 g Au/t and an 81% recovery. With a finer regrind to 25 µm, the mass pull increased slightly to 8.5%, the gold grade and recovery were higher at 56.6 g/t and 88%. A locked-cycle test was undertaken on Composite P2. An average grade of 27.2 g Au/t and 22.0% sulfide was achieved with a 92.5% gold recovery and a 94.8% sulfide recovery.

The cleaner flotation tests on gravity tailings also demonstrated the material could be effectively cleaned. Similar gold grade and recovery were achieved using gravity tailings as with whole ore. The results of the whole ore and gravity tailings cleaner flotation tests are summarized in Table 13.6 and the locked-cycle test projected results are presented in Table 13.7.

Table 13.6: Whole Ore and Gravity Tailings Cleaner Flotation Tests

Sample		Test No.	Regrind P_{80} (µm)	2nd Cleaner Concentrate			Gravity Conc. Grade Au (g/t)
				Recovery (wt %)	Grade Au (g/t)	Recovery (overall) Au (%)	
Composite IF1	Whole Ore	F1 F3	n/a 45	12.5 13.8	30.3 27.6	84.6 88.7	- -
	Gravity Tailing	F5 F6 F7	n/a 23 10	8.5 7.9 8.1	28.8 28.7 31.8	89.1 88.6 91.3	8.323
Composite P2	Whole Ore	F2 F4	n/a 22	7.3 8.5	54.8 56.8	81.3 87.8	- -
	Gravity Tailing	F8 F9 F10	n/a 19 9	5.9 4.9 4.8	48.9 54.2 58.6	88.6 87.4 86.2	1.208

Table 13.7: Locked-Cycle Metallurgical Projected Results

Product	Mass		Assay		Distribution	
	g	%	Au (g/t)	S^{2-} (%)	Au (%)	S^{2-} (%)
1st Cleaner Concentrate	1464.9	18.3	27.2	22.0	92.5	94.8
1st Cleaner Scavenger Tails	1291.1	16.1	0.72	1.14	2.2	4.3
Rougher Tails	5247.0	65.6	0.44	0.06	5.4	0.9
Head	8003.0	100.0	5.39	4.25	100.0	100.0

13.1.2.7 Pressure Oxidation

Assessment of pressure oxidation (“POX”) as a pre-treatment to cyanidation was performed on a rougher concentrate sample generated from Composite 1 during the previous phase of the testwork program. Only 70% of the sulfides were oxidized but it was sufficient to make the sample amenable to cyanide leaching. The results of the four tests showed that even at a coarse grind of 123 µm (P80), pressure oxidation increased gold extraction to 97% with a 94% overall gold recovery (including the flotation stage).

13.1.2.8 Cyanidation

Cyanide leach tests were performed on whole ore and flotation rougher concentrate samples. Standard bottle roll tests were conducted. The sodium cyanide concentration and aeration method were varied in the flotation concentrate cyanidation tests. The effect of regrind and lead nitrate were also evaluated.

The highest extractions were achieved at the finer grinds. A 10 µm grind resulted in a 98% extraction for IF1 while a 15 µm grind yielded 95% recovery for P2. However, cyanide consumption was also highest for these tests. The sodium cyanide concentration and aeration method did not impact gold extraction.

For Composite IF1, there was no benefit in including a flotation stage as 77% extraction was achieved after 72 hours of whole ore leaching. Cyanidation of rougher flotation concentrate achieved 75% overall recovery. For Composite P2, a flotation stage increased overall recovery to 87% compared to 75% after 72 hours of whole ore leaching. These results are summarized in Table 13.8.

Table 13.8: Whole Ore Cyanidation vs. Flotation Concentrate Cyanidation

Process	Test	K_{80} (µm)	72 hr Au Extraction (%)	Recovery		Residue/Tailings Grade		CN Consumption (kg/t)	
				Flotation (%)	Overall (%)	CN Au* (g/t)	Overall Au (g/t)	CN Unit	Overall
IF1 Whole Ore	CN-18	93	76.5	-	76.5	1.00	1.00	4.01	4.01
IF1 Flot. + CN of Flot. Conc.	CN-17	93	78.5	95.4	74.9	2.87	1.21	8.70	2.98
P2 Whole Ore	CN-20	123	75.0	-	75.0	2.23	2.23	1.08	1.08
P2 Flot. + CN of Flot. Conc.	CN-19	123	94.3	92.2	86.9	0.89	0.63	4.60	1.37

*Note:*Average of duplicate residue assays*

13.1.3 Whole Ore Cyanidation Testing at McClelland

Drill hole reject composites (13) were sent to McClelland Laboratories to undergo whole ore cyanidation tests. The objectives of the program were to confirm previous testing results and to optimize grind size and cyanide concentration for whole ore leaching.

13.1.3.1 Results

The tests consisted of standard bottle roll tests with or without carbon addition. The direct head assays of the thirteen samples ranged from 0.40 g Au/t to 7.37 g Au/t with an average of 3.20 g Au/t. The cyanidation tests were performed on three different grind sizes (P80): 125, 75 and 37 µm. The summary of the tests performed at 75 µm and without carbon addition are presented in Table 13.9.

Table 13.9: Whole Ore Cyanidation Tests Results

Sample	C_{Org} (%)	S (%)	Au Recovery (%)	g Au/t ore				Reagent Requirements kg/t ore	
				Extracted	Tail	Calc'd Head	Head Assay	NaCN Cons.	Lime Added
EP134T-A	0.03	1.25	91.7	1.76	0.16	1.92	1.85	0.58	2.9
EP134T-B	<0.01	2.37	86.5	2.43	0.38	2.81	2.75	0.42	2.2
HR124	0.03	1.08	95.4	5.62	0.27	5.89	6.87	0.28	2.6
HR133-A	0.03	3.27	85.6	2.14	0.36	2.50	2.33	0.51	2.8
HR133-B	0.01	0.84	93.8	2.57	0.17	2.74	2.47	0.32	2.8
HR142	0.05	8.09	76.6	5.44	1.66	7.10	7.37	0.99	3.9
HR145-A	0.03	0.20	86.9	0.53	0.08	0.61	0.69	0.49	2.1
HR145-B	0.01	1.42	89.1	1.56	0.19	1.75	1.30	0.38	4.1
HR148	0.01	0.23	86.0	0.43	0.07	0.50	0.40	0.39	1.8
MM005T-A	0.01	2.17	92.6	2.13	0.17	2.30	2.50	0.38	2.1
MM005T-B	0.02	0.87	86.2	1.44	0.23	1.67	1.68	0.50	1.9
MM351-A	0.04	12.70	63.8	4.43	2.51	6.94	6.69	0.90	4.1
MM351-B	0.06	5.41	77.4	3.77	1.10	4.87	4.74	1.06	3.8

All thirteen composites were amenable to cyanidation under the tested conditions. Gold recovery was between 85% and 95% for the composites with low sulfide sulfur content (less than 2.5%). Three composites showed higher sulfide sulfur levels (5.4% to 12.7%) and yielded lower gold recoveries (63.8% to 77.4%). Cyanide consumption was also higher for these three samples.

Gold recovery increased with finer grind sizes (2.1% increase between 120 and 75 µm and 4.3% between 75 and 37 µm) but was not affected by cyanide concentration. No preg-robbing characteristics were observed and recoveries were similar whether activated carbon was added or not. Gold leaching was complete in approximately eight hours and recovery rates were fast.

13.1.4 QEM Rapid Mineral Scan at SGS

A Global Composite sample was subjected to a QEM Rapid Mineral Scan at SGS Minerals in Lakefield, Ontario in October of 2013. The results are presented in Table 13.10.

Table 13.10: QEMSCAN Modals on Global Composite

Survey Project / LIMS Sample	Global Composite Mineral Mass (%)
Pyrite	2.62
Pyrrhotite	1.20
Chalcopyrite	0.01
Arsenopyrite	0.11
Quartz	28.6
K-Feldspar	0.39
Plagioclase	19.3
Sericite/Muscovite	13.8
Biotite	1.72
Chlorite	9.53
Other Micas/Clays	0.60
Magnetite	7.96
Hematite	0.69
Other Oxides	0.46
Calcite	1.33
Ankerite	9.48
Siderite	1.46
Apatite	0.28
Other	0.42
Pyrite	2.62

13.2 Feasibility Study Testwork

This section includes any testwork program that was performed during the PEA and during the FS. As the FS progressed, additional testwork was initiated and is described in this section. Primarily, high pressure grinding rolls ("HPGR") tests were required to confirm the ore amenability for high pressure grinding, to select the equipment and estimate the operating costs. The key reports from 2014 and 2015 are summarized in this section. The reference documents are listed below:

- SGS Canada Inc., *An Investigation into the Grindability Characteristics of Samples from the Hardrock Deposit, Report 1 (Grindability)-14117-001*, August 26, 2014;
- SGS Canada Inc., *An Investigation into The Hardrock Deposit, Final Report-14117-001*, October 8, 2014;
- SGS Canada Inc., *The HPGR Amenability of Samples from The Hardrock Deposit, Report 2 - Rev 1- 14117-001*, March 6, 2015;
- JKTech Pty Ltd., *Revised SMC Test Report*, April 2014;
- FLSmidth, *Thickening and Rheology Tests on Gold Ore Composite*, June 2014.

13.2.1 Grindability Testwork

Dilution samples (5), PQ core samples (3) and core interval samples (53) were submitted for comminution testing at SGS Canada Inc. in Lakefield, Ontario. In addition, nine variability composites and one Global Composite sample were prepared using the core samples. The Global Composite is considered to be the most representative of the run-of-mine during the project's life. The samples were submitted for JK drop-weight tests, SMC tests, Bond low-energy impact tests, Bond rod mill and ball mill grindability tests, ModBond tests and Bond abrasion tests.

13.2.1.1 Grindability Tests Results

The grindability tests results for the Composite samples, the PQ core samples and the Dilution samples are presented in Table 13.11.

Table 13.11: Composites, PQ Core and Dilution Samples Comminution Tests Results

Type	Name	Interval Number	CWI (kWh/t)	Relative Density	JK Parameters			BWI (kWh/t)	Mod Bond (kWh/t)	Ai (g)
					A	b	Axb			
Composites	Global							15.2		
	A							15.9		
	B							15.3		
	C							15.9		
	D							15.8		
	E							15.1		
	F							14.5		
	G							16.4		
	H							14.3		
	I							15.0		
PQ Core	PQ Iron Formation (DWT)		12.0	3.26	75.1	0.43	32.3			
	PQ Iron Formation (SMC)			3.24	84.1	0.40	33.6			
	PQ Greywacke (DWT)		10.2	3.26	59.6	0.76	45.3			
	PQ Greywacke (SMC)			3.11	75.7	0.54	40.9			
	PQ Porphyry with Minor Greywacke (DWT)		14.6	2.93	75.1	0.32	24.0			
	PQ Porphyry with Minor Greywacke (SMC)			2.76	76.3	0.34	25.9			
Dilution Samples	Greywacke			2.77	94.6	0.24	22.7	15.5	16.0	0.154
	Iron Formation			2.95	81.2	0.35	28.4	10.5	11.1	0.091
	Gabbro			2.78	65.7	0.48	31.5	14.5	14.8	0.102
	Porphyry			2.68	92.0	0.27	24.8	16.0	16.5	0.194
	Ultramafic			2.96	66.7	0.89	59.4	10.2	10.2	0.069

The results were computed for each lithology in order to calculate the 90th percentile values as presented in Table 13.12.

Table 13.12: Comminution Test Results per Lithologies

Samples	Mod Bond 90th percentile (kWh/t)	DWI 90th percentile
Greywacke (S3E) & Gabbro (I1A)	15.5	11.7
Iron formation (C2A)	15.5	12.3
Porphyry (I3P)	16.4	10.7
Overall	15.6	11.7

Fifty-three core interval samples made of material from various lithologies that represent the entire deposit were submitted to comminution testing. The samples show little variability between the samples. The summary of the results is presented in Table 13.13.

Table 13.13: Core Interval Samples Comminution Tests Results

Description	JK Parameter		RWI	BWI	Mod Bond
	Rel. Density	A x b	(kWh/t)	(kWh/t)	(kWh/t)
Average	2.98	29.2	16.5	14.9	14.4
Std. Dev.	0.21	3.4	0.2	1.0	1.2
Rel. Std. Dev.	7	12	1	7	8
Minimum	2.71	41.0	16.3	13.2	11.3
Median	2.92	28.8	16.4	15.4	14.6
Maximum	3.35	24.1	16.8	16.0	16.5

In terms of resistance to impact breakage (Axb), the samples were found to be hard to very hard. Their abrasion resistance (t_a) fell into the very hard category. The Bond low-energy indices characterize the samples as medium to moderately hard.

The rod mill work indexes were all similar and fell into the moderately hard category. The ball mill work indexes ranged from soft to moderately hard. Finally, the abrasion indices denoted a mildly to medium abrasive ore.

13.2.2 Characterization and Recovery Testwork

The samples used for the grindability tests were submitted to head grade determination, mineralogy, magnetic separation, flotation, gravity separation, cyanidation with cyanide destruction, carbon modelling, solid-liquid separation and environmental testing. The dilution samples were only assayed for direct head grade and were not submitted to any metallurgical testwork. In addition, six low grade composites and a master composite representing the lithological ratios for the first three years of operation were prepared and tested. The proportion of each lithology in the prepared samples is shown in Table 13.14.

Table 13.14: Global, Master, Variability and Low Grade Samples Composition

Composite	Lithology Constitution (%)				
	Wacke to Greywacke S3E	Iron Formation C2A	Gabbro I1A	Porphyry I3P	Quartz-Feldspar-Porphyry I3R
Global	46.2	33.5	5.3	15.1	
Master	43.8	35.1	3.6		17.5
A	100				
B		55.8	11.4	32.8	
C	96.3	3.7			
D		72.0	28.0		
E	78.3		21.7		
F		100			
G				100	
H		100			
I	100				
S3E-0.5-WCE	100				
S3E-0.7-WCE	100				
I3P-0.5-WCE				100	
I3P-0.7-WCE				100	
C2A-0.5-WCE		100			
C2A-0.7-WCE		100			

13.2.2.1 Characterization and Recovery Tests Results

13.2.2.1.1 Head Grade Determination

The head grades of the composites were determined by metallic sieve and a weighted average was calculated from the testwork (Table 13.15). The direct and calculated head grades all correlate well except for Composite C and Composite F.

Table 13.15: Composite Samples Direct and Calculated Head Grade

Sample Name	Direct Au g/t	Calculated (From Testwork) Au g/t
Composites		
Global	1.74	1.92
Master	1.94	2.08
A	2.56	2.62
B	2.04	2.19
C	1.71	2.04
D	1.68	1.58
E	1.18	1.39
F	1.36	2.01
G	1.59	1.59
H	2.65	2.59
I	2.29	2.07
Dilution Samples		
Greywacke	0.06	-
Iron Formation	<0.01	-
Gabbro	0.08	-
Porphyry	0.06	-
Ultramafic	0.04	-
Low Grade Composites		
S3E-0.5-WCE	0.55	0.50
S3E-0.7-WCE	0.67	0.72
I3P-0.5-WCE	0.46	0.49
I3P-0.7-WCE	0.75	0.67
C2A-0.5-WCE	0.34	0.38
C2A-0.7-WCE	0.85	0.82

13.2.2.1.2 Mineralogy

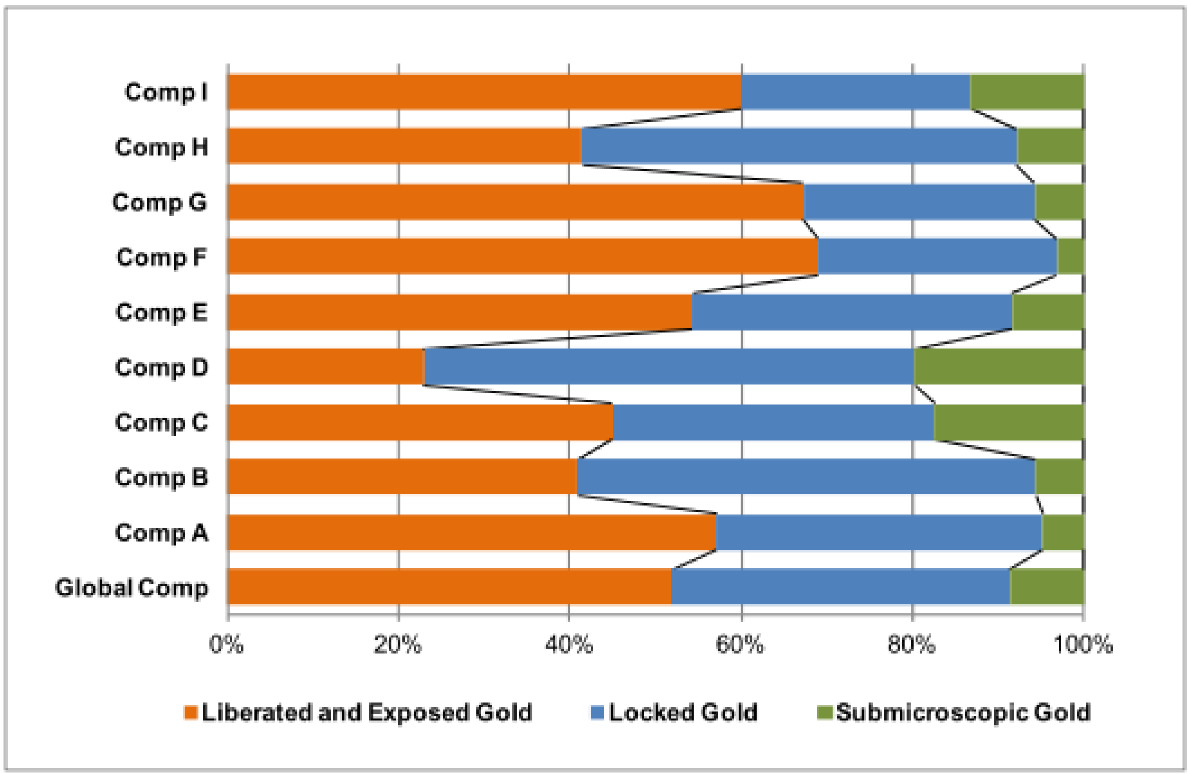
The Global and Variability composites were submitted to a microscopic (> 0.5 µm) and submicroscopic (< 0.5 µm) gold deportment study. The gold mineral association and distribution are presented in Table 13.16. The gold occurrence by distribution based on an approximate P80 of 300 µm is shown in Figure 13.3.

Table 13.16: Gold Deportment Results

Composite	Gold Distribution (%)		Gold Associated Minerals (% - Normalized to 100%)							
	Sub microscopic Au	Microscopic Au	Py	Apy	Py-Sul	FeOx	Py-Silc	Silc	Carb	Other
Global	8.6	91.4	75.8	7.75	8.94	2.97	4.16	-	-	0.33
A	4.8	95.2	58.6	14.3	5.69	3.14	14.9	-	1.38	1.96
B	5.7	94.4	58.4	4.66	1.33	8.24	20.8	1.85	1.38	3.43
C	17.4	82.6	83.4	1.43	0.64	3.81	7.52	-	2.85	0.36
D	19.7	81.0	78.7	4.58	-	13.6	-	0.58	2.27	0.28
E	8.3	92.7	34.3	-	-	17.2	23.8	22.4	-	2.25
F	3.2	96.9	74.9	3.42	-	10.6	4.62	3.53	-	2.93
G	5.7	94.3	90.3	5.38	0.99	1.18	0.59	-	1.42	0.19
H	7.8	92.2	87.9	2.54	-	0.92	0.72	7.93	-	-
I	13.2	86.8	5.45	12.61	-	-	0.23	80.79	0.43	0.51

Note: Py-pyrite (including greigite); Apy – arsenopyrite and with other sulphides; Py-Sul – Pyrite with other sulphides; FeOx – iron oxides; Py-Silc – pyrite with silicates; silc – Silicates; Carb – carbonate minerals and mixture

Figure 13.3: Gold Occurrence (by Distribution)



Title: Gold Occurrence (by Distribution)	**Author: SGS Canada Inc.** **Date: 14/07/21**

13.2.2.1.3 Magnetic Separation

Davis Tube testing was performed on the Global and Variability Composites to identify the presence of magnetic minerals. The results showed a large variation in the weight recovery to the concentrates: 0% for Composite G (100% Porphyry), up to 27% for Composite F (100% Iron Formation) and around 10% for the Global Composite. The Global Composite was also subjected to low intensity magnetic separation ("LIMS") and wet high intensity magnetic separation ("WHIMS") testing in order to evaluate the possibility of removing iron minerals without incurring gold losses. A LIMS stage was effective in removing significant amounts of iron but resulted in a 7.4% gold loss. The WHIMS stage did not significantly split the iron and gold distribution.

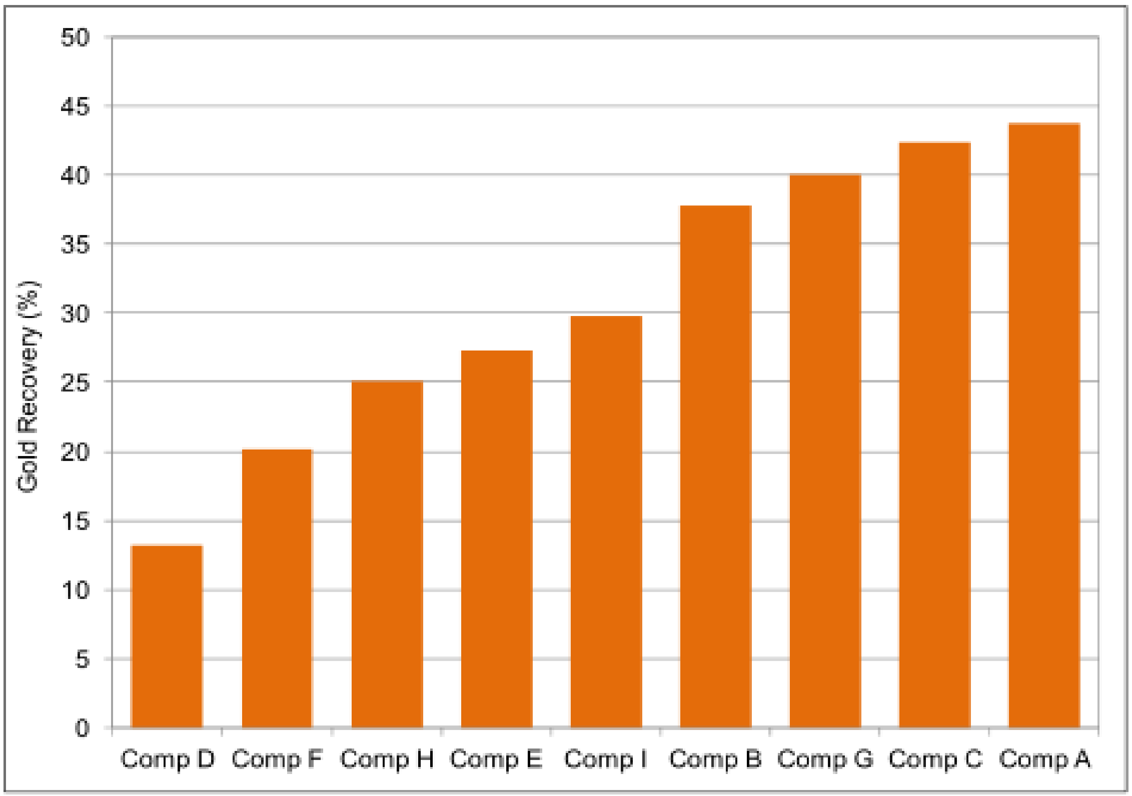
13.2.2.1.4 Gravity Recovery

All composites were subjected to gravity separation testing using a Knelson Concentrator and a Mozley table. Based on a series of gravity recovery tests, the Global Composite recovery varied from 15%

(P80 = 129 µm) to 42% (P80 = 61 µm), the Master Composite recovery from 18% (P80 = 105 µm) to 30% (P80 = 61 µm) and the Low Grade Composite recovery from 5% to 39% at a P80 of 110 µm.

The Variability Composites were submitted for a single gravity separation test at a target grind of 80 µm (P80). The gold recovery varied from 13% to 44%. The results are shown in Figure 13.4.

Figure 13.4: Variability Composites Gravity Recovery Results



Title: Variability Composites Gravity Recovery	Author: SGS Canada Inc. Date: 14/10/08

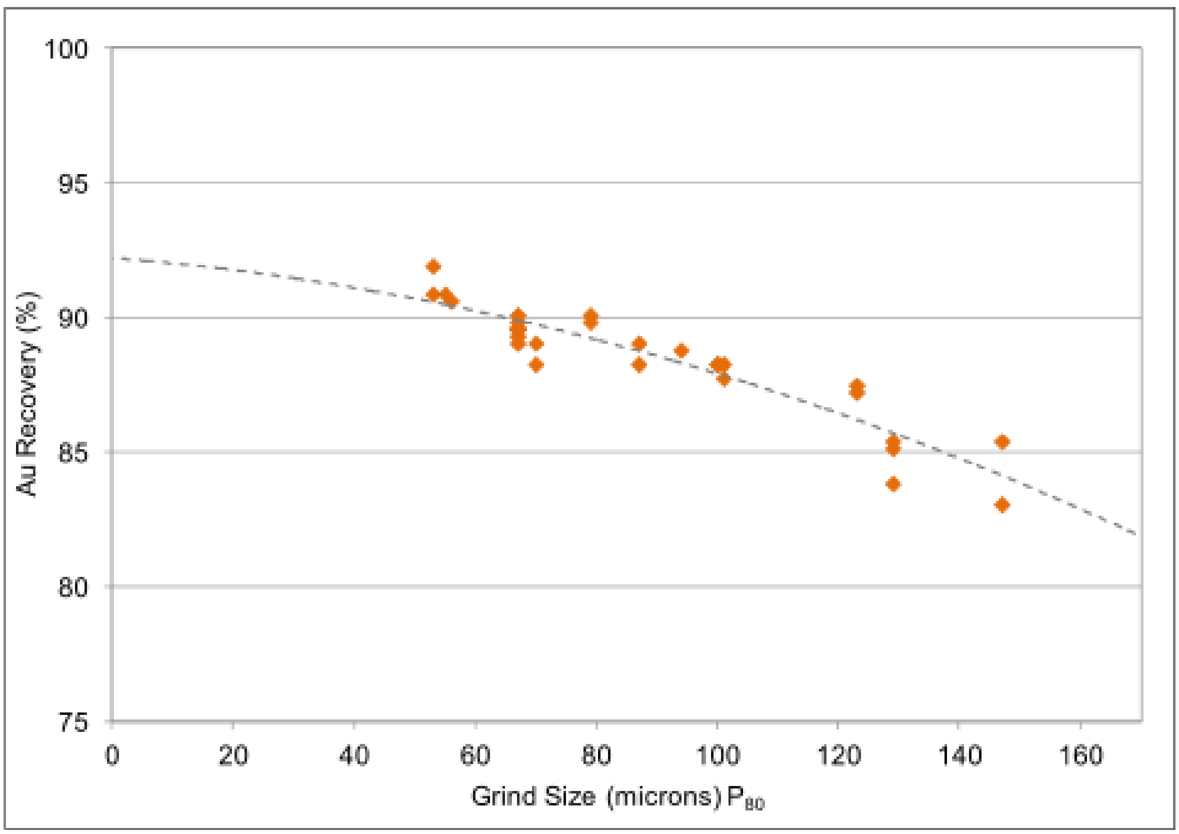
The Global Composite was also submitted for an extended Gravity Recoverable Gold ("E-GRG") test. The amount of gravity recoverable gold in the sample was assessed at 47.2%.

13.2.2.1.5 Cyanidation Testing

The Global Composite was subjected to developmental cyanidation testing. The program included whole ore versus gravity tailings leaching, effect of pre-aeration, grind size and percent solids. The results can be summarized as follows:

- Whole ore leach extraction: 85-93%;
- Gravity tailings leach recovery: 81-90%;
- Leach kinetics increased with a finer grind size and oxygen sparging;
- Oxygen sparging yielded lower cyanide consumptions compared to air sparging;
- Variations in slurry percent solids (33% to 50%) did not affect gold extraction;
- Cyanide concentration and pre-oxygenation duration did not significantly affect gold extraction;
- A finer grind improves gold recovery (Figure 13.5).

Figure 13.5: Gold Recovery as a Function of Grind Size (Global Composite)



Title: Gold Recovery vs. Grind Size	**Author: SGS Canada Inc.** **Date: 14/10/08**

Gravity tailings of the Global Composite and the Variability Composites underwent cyanidation testing at P80's of approximately 80 and 60 µm. The finer grind resulted in better gold extractions for all the samples (86 to 95% recovery versus 78 to 90%).

The Master Composite was submitted to leach optimization testing. The effects of grind size, residence time, lead nitrate addition, pH and carbon concentration were examined. The grind size had the most impact on gold extraction while a 2% gold recovery increase was observed when increasing the retention time from 32 to 72 hours. The gold recovery was between 85% and 89%.

The Low Grade Composites were also submitted to cyanidation testing. Gold recovery and leach kinetics were improved at finer grind sizes. The gold recovery ranged from 80% to 95%.

The optimized leach conditions defined during the tests are as follows:

- Slurry density: 50% w/w;
- pH: 10.5-11.0;
- Dissolved oxygen: > 15 mg/L;
- Cyanide concentration: 0.5 g/L NaCN (maintained);
- Retention time: 36 hours.

13.2.2.1.6 Carbon Circuit Modelling

SGS Canada Inc. uses the semi-empirical models developed by Mintek SA (South Africa’s national mineral research organization) to simulate Carbon-in-Leach (“CIL”) and Carbon-in-Pulp (“CIP”) circuits. The approach to CIL and CIP modelling involves conducting batch gold leaching and carbon adsorption tests with representative samples and commercially available activated carbon. The leach rate is determined through a classic bottle roll experiment by taking timed samples over a 72 hour period. The gold adsorption rate is determined by adding carbon to the slurry and again taking samples during 72 hours. Equilibrium adsorption isotherms are then established.

The Master Composite gravity tailings sample from test G-24 was used for the CIL/CIP modelling. The test revealed that leaching of the Master Composite Sample was complete after 24 hours. The sample showed relatively slow adsorption kinetics but very favorable equilibrium loading. The simulation results are presented in Table 13.17.

Table 13.17: Leach / CIP Modelling Results

Parameter	Value
Number of Leach Tanks	6 @ 4,200 m^3 each
Slurry Flow Rate	548 t/h at 55% solids
Number of Adsorption Tanks	6
Slurry Time in Each CIP Tank	0.3 h
Carbon in Each of the 6 Adsorption Tanks	12.6 t
Carbon Concentration in Adsorption Tanks	80 g/L
Gold on Carbon/Gold in Feed	1427
Carbon Advance Rate to Elution/Regeneration	7.9 t/d
Gold on Loaded Carbon	2,310 g/t
Gold on Eluted Carbon	50 g/t
Gold Locked Up on Carbon in Plant	48 kg
Ramp-up Time	11 days
Soluble Gold Losses	0.007 mg/L

13.2.2.1.7 Cyanide Destruction

The bulk leach product of the Global and Variability Composites were subjected to a single-stage cyanide destruction test to determine the samples amenability to detoxification using the SO_2/Air process. The objective of the test was to achieve weak acid dissociable cyanide (CN_{WAD}) levels below 1 mg/L.

The Global and Variability Composites A, B, G and I were the most difficult to treat. A retention time of 120 minutes, 30 to 45 mg/L of copper sulfate and more than 7.0 g of sulfur dioxide per gram of CN_{WAD} were required to meet the target. Variability Composites D, E and F also required 120 minutes of retention time but reagents addition was lower (20 to 30 mg/L of copper sulfate and 5.7 to 6.1 g of sulfur dioxide per gram of CN_{WAD}). Finally, Variability Composites C and H only required 60 to 90 minutes of retention and 5.5 g of sulfur dioxide per gram of CN_{WAD}. It was also found that there is a strong relationship between the residual iron and the total cyanide (CN_T). The residence time and copper sulfate addition can be increased to further reduce total cyanide levels.

A two-stage cyanide destruction test was carried out on the Global Composite. The CN_{WAD} and CN_T were reduced to the targeted 1 mg/L in 90 minutes by adding 45 mg/L of copper sulfate and 7.32 g of sulfur dioxide per gram of CN_{WAD}. A shorter retention time (60 minutes) during test CND12-4 led to an increased concentration of CN_T in the cyanide destruction discharge solution (refer to Table 13.18).

Table 13.18: Two Stage Cyanide Destruction Discharge Solution Analysis

Test No.	Solution	Analysis (mg/L)									
		Fe	Cu	CN_T	CN_F	CN_{WAD}	CNS	CNO	NH_3	NO_2	NO_3
CN-94	Final Barren	1.76	6.87	258	222	204	40	39	1.00	-	-
CND12-2	Final Destruction (R2)	0.26	0.11	0.63	0.08	0.08	46	120	12.2	< 0.3	< 0.6
CND12-4		2.22	0.11	6.07	0.04	0.04	55	190	5.3	< 0.3	< 0.6

Note: CND12 was a two stage cyanide destruction, the final solution is the discharge from the second reaction vessel.

13.2.2.1.8 Solid-Liquid Separation and Rheology

The Global Composite and Variability Composites C, F and G cyanide destruction discharge samples were subjected to flocculant selection, static settling, dynamic settling and underflow rheology tests.

The objective of the flocculant screening test was to identify the right type of reagent for the separation process and to find a widely available and inexpensive reagent that would suit all the samples. The flocculant performance was evaluated in terms of relative effectiveness regarding particle aggregation, floc formation, resulting structure characteristics and supernatant water clarity. All the samples responded well to a low charge density anionic flocculant.

For the static tests, standard Kynch tests were conducted at variable slurry percent solids and reagent dosages. The non-optimized static settling tests results were used to define the starting conditions (feedwell solids density and relevant flowrates) for the dynamic settling tests.

The optimized dynamic settling parameters and results (flocculant dosage, unit area, solids and hydraulic loading, rise rate and residence time) are presented in Table 13.19.

Table 13.19: Dynamic Settling Test Results

Sample I.D.	Flocculant (BASF)	Dosage (g/t dry)	Dry Solids SG	U/F[1] (% wt)	U/F Extended (% wt)	TUFUA[2] (m^2/t/d)	THUA[3] (m^2/t/d)	Net Rise Rate (m^3/m^2/day)	Solids Loading (t/m^2/day)	Net Hydraulic Loading (m^3/m^2/day)	Res. Time (h)	Overflow (Visual)	TSS[4] (mg/L)
CND-1 Global Composite	Magnafloc 10	15	2.88	64.5	63.9	0.090	0.042	61.1	0.462	2.54	1.12	Clear	27
CND-2 Variability Composite C		17	2.82	63.5	63.7	0.080	0.019	68.6	0.519	2.86	0.95	Clear	10
CND-3 Variability Composite F		15	3.19	70.0	71.5	0.080	0.026	68.8	0.520	2.87	1.04	Clear	12
CND-4 Variability Composite G		18	2.74	64.2	67.1	0.100	0.030	54.6	0.415	2.28	1.19	Clear	43

Notes: All values were calculated without a safety factor. Key underflow rheology data were included in the rheology section.

Common Test Conditions:

Autodiluted Thickener Feed % solids = 15% w/w solids

Solution S.G – 1.000

[1] Ultimate Underflow (UF) Density

[2] Thickener Underflow Unit Area

[3] Thickener Hydraulic Unit Area

[4] Total Suspended Solids of the Overflow

The rheology tests were performed on the underflow samples generated under optimized settling conditions. The critical solids density ("CSD") for each sample is presented below. The CSD is the solids density at which a small increase in density causes a significant decrease in flowability. It also predicts the maximum solids density that is achievable in an industrial thickener and practical for pumping.

All the underflow samples displayed Bingham plastic behavior and the CSD for all four samples varied between 65 to 69% solids.

Table 13.20: Underflow Rheology Test Results

Sample I.D.	CSD	Yield Stress (Pa)		Flow Behaviour & Range (wt % solids)
	(wt % solids)	Unsheared	Sheared	Thixotropy
CND-1 Global Composite	66	33	14	60.5-68.9
CND-2 Variability Composite C	65	31	15	60.0-68.0
CND-3 Variability Composite F	69	35	10	63.1-73.4
CND-4 Variability Composite G	67	40	14	61.4-70.4

Note: CSD: Rheology-determined Critical Solids Density

13.2.3 Thickening and Rheology Tests

Additional thickening and rheology testwork was carried out by FLSmidth in June 2014 in order to determine the sizing and operating parameters of a pre-leach thickener. The objective was to reach a 55% underflow density and a 50 to 75 ppm solids concentration in the overflow.

13.2.3.1 Thickening and Rheology Results

FLSmidth tested five types of flocculant and the results show that an anionic polyacrylamide flocculant with a very high molecular weight and very low charge density yielded the best settling rates and overflow clarity. The flocculant recommended dosage is between 15 and 25 grams of flocculant per tonne of dry solids.

The settling flux tests determined that a feedwell percent solids of between 8% to 11% provides the best conditions for flocculation. The continuous fill tests yielded a recommended solids loading of 25 t/d/m² or a unit area of 0.04 m²/t/d for the Composite Sample. The rheology tests determined that a 50% to 55% solids thickener underflow could be achieved in less than two hours with design yield stress lower than 50 Pa. The results are summarized in Table 13.21.

Table 13.21: Thickening and Rheology Tests Results Summary

Thickener Operating Parameters	Gold Ore Composite
Recommended Feedwell Suspended Solids Concentration (wt%)	11
Recommended Total Flocculant Dose (g/t)	25
Recommended Minimum Unit Area (m^2/t/d)	0.04
Design Overflow Clarity (ppm)	<40
Rheological Characteristics	
Est. Bed Solids at 0.5 hr Retention Time (wt%)/ Est. Yield Stress (Pa)	57/<25
Est. Bed Solids at 1 hr Retention Time (wt%)/ Est. Yield Stress (Pa)	58/<25
Est. Bed Solids at 2 hr Retention Time (wt%)/ Est. Yield Stress (Pa)	60/<25
Est. Bed Solids at 4 hr Retention Time (wt%)/ Est. Yield Stress (Pa)	61/<25
Est. Bed Solids at 6 hr Retention Time (wt%)/ Est. Yield Stress (Pa)	73/120
High Rate Thickeners Sizing Basis	
Design U/F Solids (wt%)	50 – 60
Design U/F Retention Time (hr)	2 or less
Design Yield Stress (Pa)	25

13.2.4 HPGR Testwork

The HPGR testing program was threefold. First, laboratory scale tests (batch and locked-cycle tests) were performed to determine the amenability of the ore to HPGR milling and yield data to allow a preliminary sizing to be done. Then, abrasion tests were completed to provide the data necessary to predict the service life of the rolls. Finally, a large scale pilot plant test was planned in order to adequately size the equipment. Bond grindability testing was included in the scope of work in order to evaluate the BWI reduction of the HPGR product compared to the feed.

ThyssenKrupp is affiliated with SGS Minerals for the HPGR laboratory scale tests (Labwal) and the tests could be done in Canada. The abrasion tests (Atwal) were performed at ThyssenKrupp's Resource Technologies Research Center in Germany. The pilot plant test was carried out in Germany.

Samples from each major lithology (Greywacke, Iron Formation and Porphyry) were prepared and sent to the laboratory for the Labwal tests. A representative composite sample was made from these lithology samples. The pilot plant composite sample was prepared at the same time in order to ensure the samples used for the laboratory scale tests and the future pilot scale tests would have the same characteristics. Table 13.22 below shows the sample preparation details.

Table 13.22: HPGR Tests Samples Preparation Details

Samples	Material Weight Distribution (kg)						Comp.
	Received	Stored*	HPGR Testing	ATWAL	Compositing	Left Over	Ratio (%)
Greywacke	969	594	165	210	594	0	50.5
Iron Formation	791	416	165	210	343	73	29.1
Porphyry	710	335	165	210	240	95	20.4
HPGR Comp	0	-	1,178	0	1,178	0	100
Total	**2,471**	**-**	**1,673**	**630**	**-**	**168**	**-**

*Note: *Material set aside for the composite*

13.2.4.1 Labwal Tests Results

The results of the Labwal tests are summarized below. The locked-cycle tests were performed using the optimal batch test conditions. One of the parameters used to determine the optimal conditions was the HPGR product fineness as a function of applied pressure. The tests results were used in SGS's comminution circuit simulations to size the HPGR.

Table 13.23: Labwal Tests Results

Sample Name	HPGR Batch Test						HPGR Locked-Cycle Test					
	Operating Press. (bar)	t/h	Net kWh/t	N/mm²	m_f	P_{80} (mm)	t/h	Net kWh/t	N/mm²	m_f	CL (%)[1]	P_{80}
Greywacke	35	2.9	1.04	1.75	255	5.259	-	-	-	-	-	-
Greywacke	60	2.7	1.66	2.99	239	4.321	-	-	-	-	-	-
Greywacke	72	2.7	2.02	3.59	236	3.904	1.8	2.60	3.25	230	46	2.218
Iron Formation	36	3.1	0.97	1.79	273	4.731	-	-	-	-	-	-
Iron Formation	60	3.0	1.55	3.00	263	4.074	1.9	2.06	2.76	260	52	2.226
Iron Formation	72	2.9	1.80	3.57	255	4.024	-	-	-	-	-	-
Porphyry	34	2.6	1.01	1.70	233	5.243	-	-	-	-	-	-
Porphyry	58	2.5	1.69	2.87	221	4.184	1.7	2.31	2.74	224	52	2.067
Porphyry	70	2.4	1.96	3.48	216	4.060	-	-	-	-	-	-
HPGR Comp	-	-	-	-	-	-	1.8	2.59	3.22	240	48	2112

Note:

[1]Circulating Load

13.2.4.2 Atwal Tests Results

The results of the Atwal tests are summarized in Table 13.24. The most abrasive sample was the Greywacke followed by the Porphyry and the Iron Formation that showed similar wear rates. According to these results, all the samples were classified as low to medium abrasive when dry (1% moisture) or wet (3% moisture).

Table 13.24: Labwal Tests Results

Ore	Test #	Moisture (%)	Grinding Force (N/mm²)	Wear Rate (g/t)
Greywacke	A1	1.0	4.0	17.7
Greywacke	A2	3.0	4.0	20.5
Iron Formation	A1	1.0	4.0	15.6
Iron Formation	A2	3.0	4.0	17.3
Porphyry	A1	1.0	4.0	16.6
Porphyry	A2	3.0	4.0	17.0

13.2.4.3 Bond Ball Mill Grindability Tests Results

The Bond grindability tests were performed at 106 µm on the four HPGR feed samples as well as on the four corresponding HPGR locked-cycle test products. Three additional tests were performed on the HPGR products using the particle size distribution of the HPGR feed samples (HPGR Adjusted Prod. samples).

The HPGR feed samples varied in terms of hardness from medium (Iron Formation) to moderately hard (Greywacke and Composite) to hard (Porphyry). When comparing the BWI values, the HPGR products were considerably softer and all fell into the medium hardness category, except for the Porphyry sample that went from hard to moderately hard. Results are summarized in Table 13.25.

Table 13.25: Bond Ball Mill Grindability Tests Results

Sample Name	Mesh of Grind	F_{80} (µm)	P_{80} (µm)	Gram per Revolution	Work Index (kWh/t)	% Reduction	Hardness Percentile	Feed Passing (%)
Greywacke	150	2,477	79	1.16	16.1	-	70	10.5
Greywacke HPGR Product	150	2,166	80	1.43	13.8	14	44	14.8
Greywacke – HPGR Adjusted Prod.*	150	2,520	79	1.31	14.6	10	52	10.3
Iron Formation	150	2,417	78	1.27	14.9	-	56	10.3
Iron Formation HPGR Product	150	2,256	77	1.44	13.4	10	40	15.8
Iron Formation – HPGR Adjusted Prod.*	150	2,440	80	1.36	14.3	4	49	10.3
Porphyry	150	2,392	80	1.09	17.1	-	77	7.3
Porphyry HPGR Product	150	2,173	82	1.22	15.9	7	68	13.6
Porphyry – HPGR Adjusted Prod.*	150	2,426	81	1.19	16.1	5	70	6.9
HPGR Comp	150	2,368	79	1.19	15.8	-	66	9.8
(HPGR Comp) HPGR Product	150	2,162	76	1.39	13.8	13	43	15.4

*Note: * Represent a different sample preparation approach explained in Section 13.2.4.3.*

SGS Canada Inc. developed a method that accounts for the effect of the increased amount of fines in the HPGR product to better estimate the power reduction to grind from 100% passing 6 mesh to 100% passing 150 mesh. Based on their method, the HPGR product would require 17% to 23% less power (compared to a standard feed).

A different method was suggested to SGS by an external comminution specialist at SimSAGe. The BWI test samples were prepared in order to reproduce the size distribution of the Bond ball mill grindability tests performed on the HPGR feed. Based on this modified procedure, the HPGR products required 7 to 12% less power (compared to a standard feed) to grind from 100% passing 6 mesh to 100% passing 150 mesh.

13.2.4.4 Pilot Plant Tests Results

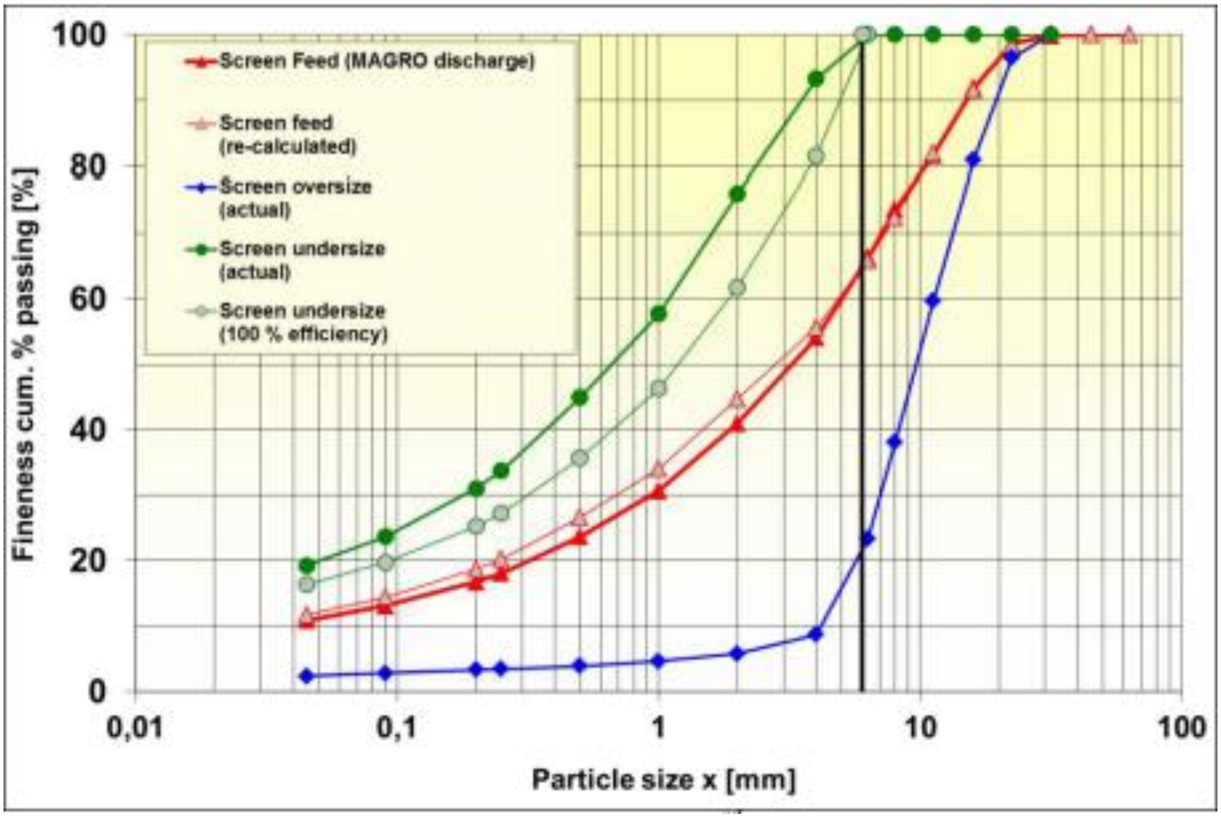
Pilot plant tests were carried out on about 950 kg of Au ore sample from the Hardrock deposit. The sample material is a composite made of 50.5% Greywacke, 29.1% Iron Formation and 20.4% Porphyry. The ore sample was provided as drill cores that had been pre-crushed to match the feed size requirements of the units.

The pilot plant tests were conducted using a semi industrial high pressure grinding roll with 0.95 m diameter rolls, 0.35 m wide. Process data obtained from testwork allow the sizing of industrial scale machines. The objectives in sizing HPGRs are to meet the throughput requirements and to achieve a certain product fineness. The key parameters are therefore the specific throughput rate and the specific energy consumption required to obtain the desired comminution result. The specific throughput varied between 306 and 320 t s/m^3·h; it was slightly dependent on the specific press force. The specific energy consumption varied between 1.4 and 2.6 kWh/t depending on the applied specific press force.

The Bond tests were conducted on a conventionally crushed fresh feed sample from the provided sample as well as on the HPGR cycle products. The Bond test was conducted using a closing mesh size of 90 µm. The Bond Work Index was 10% lower after HPGR treatment: 14.73 kWh/t on crushed material compared to 13.28 kWh/t on HPGR product.

The pilot tests allowed the prediction of the expected industrial size distribution of the HPGR discharge and of the screen undersize product for a closed circuit operation. Locked cycle tests were conducted to simulate a continuous operation. The circulation factor was fairly constant in the first three cycles indicating that the circuit was stabilized. The pilot plant test third cycle size distribution is presented in Figure 13.6.

Figure 13.6: Pilot Plant Test Third Cycle Size Distribution



Title: Pilot Plant Test Third Cycle Size Distribution	Author: ThyssenKrupp Date: 15/05/05

13.3 Conclusions and Recommendations

13.3.1 Grinding

Grindability tests have been performed on a sufficient number of samples to properly assess the comminution characteristics of the Hardrock deposit. Generally, the ore falls into the high hardness end of the spectrum. The test data from the various tests need to be manipulated to estimate values that represent the run-of-mine composition (weighted averages). These results are used as a basis for plant design.

13.3.2 High Pressure Grinding

The HPGR Labwal tests showed that the Hardrock deposit is amenable to high pressure grinding and yielded a net power consumption of 2.6 kWh/t. The abrasion tests determined that the ore falls into the low to medium abrasiveness categories. Bond ball mill grindability comparative tests done on the HPGR feed and product revealed that a 7 to 12% power reduction could be expected when grinding a HPGR product.

13.3.3 Magnetic Separation

The magnetic separation tests revealed that a variable amount of magnetic minerals is present in the different composites and that gold losses associated with the removal of the magnetic fraction can be significant. The tests also expose the fact that large amounts of gold bearing ore could potentially be rejected from the process if magnets are installed on relatively fine ore streams.

13.3.4 Gravity Recovery

Gravity recovery tests showed that gravity separation is an efficient method of recovering gold. Cyanidation of gravity tailings is an economical method of gold recovery and removal of a small portion of gold reduces cyanide consumption in the leach circuit and carbon circuit requirements.

13.3.5 Flotation

Comparing gold extraction by cyanidation of whole ore with cyanidation of flotation concentrate, there was no benefit seen by including the flotation stage because the expected recovery with the flotation process does not demonstrate improvement to the overall metallurgical performance.

13.3.6 Pressure Oxidation

Pressure oxidation as a pre-treatment ahead of cyanidation increased gold extraction to 97% (overall recovery of 94% including flotation) and compared favourably to cyanidation of finely ground rougher concentrate. Pressure oxidation, however, is a costly method for increasing gold extraction.

13.3.7 Cyanidation

The cyanidation tests revealed that overall gold recovery is improved at finer grinds but cyanide consumption is increased. The optimal leach conditions are defined as follows:

- Slurry density: 50% w/w;
- pH: 10.5-11.0;

- Dissolved oxygen: > 15 mg/L;
- Cyanide concentration: 0.5 g/L NaCN (maintained);
- Retention time: 36 hours.

13.3.8 Cyanide Destruction

The SO_2/Air process is effective at reducing cyanide levels to below 1 mg/L in the final tailings. A 90 minute retention time is required with the addition of 45 mg/L of copper sulfate and 7.32 g of sulfur dioxide per gram of CN_{WAD}.

13.3.9 Solid-Liquid Separation and Rheology

The pre-leach slurry can be thickened to 55% solids w/w by adding a low charge density anionic flocculant at a 15 g/t dosage.

13.4 Future Work

Additional tests are recommended to be conducted for the detailed engineering:

- Cyanide destruction optimisation testwork to confirm the reagents to be used and the operating conditions. Investigate the possibility of realizing the cyanide destruction and the precipitation of arsenic in two stages;
- Perform grindability and leach testing on newly discovered zones as they appear to increase data and knowledge;
- Tests to validate the oxygen consumption in the leaching tanks;
- Abrasion tests to confirm liner and steel ball consumptions in the grinding mills;
- Once the HPGR supplier is chosen, there is a potential to gain additional knowledge on the equipment and increase the confidence level from a small pilot plant using their technology;
- Perform testwork to investigate the possibility of thickening the tailings prior to cyanide destruction to increase cyanide recovery.

14. MINERAL RESOURCE ESTIMATES

14.1 Hardrock

Combined open pit and underground Indicated Mineral Resources for the Hardrock Project total 146M t at an average grade of 1.36 g/t Au for 6.4M ounces of gold.

The 2016 MRE was reviewed, updated and approved by Réjean Sirois, P. Eng., of GMS using all available information. The main objective of the mandate assigned by GGM to GMS was to update the 2014 MRE prepared by InnovExplo and published in a report titled "*Technical Report and Mineral Resource Estimate update for the Hardrock deposit (according to National Instrument 43-101 and Form 43-101F1)*", dated August 22, 2014 (Brousseau et al., 2014) and also to validate the work generated by a third-party consultant on the current resource estimates.

The Mineral Resources presented herein are not Mineral Reserves as they do not have demonstrated economic viability. A single Mineral Resource estimate was prepared for 25 structurally and lithologically defined gold-bearing zones (the "gold zones" or "mineralized zones") and a remaining undifferentiated envelope (the "envelope"; see below for details). The 2016 MRE includes Indicated and Inferred Mineral Resources for both a pit design volume and a complementary underground volume. The effective date of the 2016 MRE is August 11, 2016.

14.1.1 In-Pit and Underground Mineral Resource Estimate Methodology

14.1.1.1 Methodology

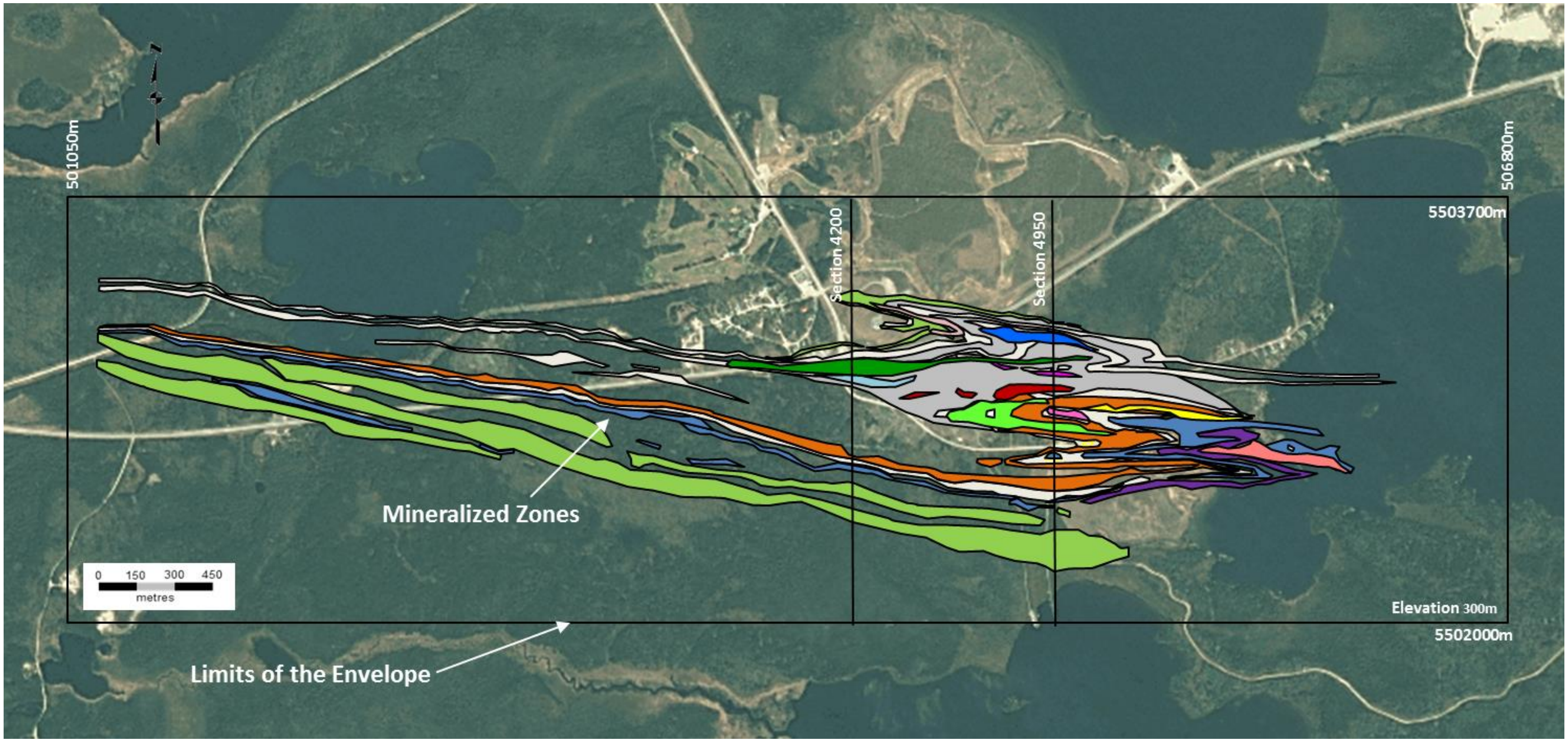
The 2016 MRE detailed in this Report was made using 3D block modelling and the inverse distance cube interpolation ("ID^3") method. The estimate covers a corridor of the Hardrock deposit with a strike-length of 5.7 km and a width of approximately 1.7 km, down to a vertical depth of 1.8 km below surface. The boundaries of the block model correspond to the limits of the envelope, as shown in Figure 14.1. The 25 mineralized zones were interpreted in 3D using GEMS and GOCAD based on a litho-structural model and the drill hole database. Premier and GGM collaborated on the update of the litho-structural model.

14.1.1.2 Drill Hole Database

The GEMS diamond drill holes database contains 1,629 surface diamond drill holes with gold assay results and specific gravity measurements, as well as coded lithological, alteration and structural data and RQD measurements taken from drill core logs. All 1,629 drill holes were used in the 2016 MRE, representing the drill holes completed and validated at the close-out date of November 18, 2015. The holes are all located within the limits of the undifferentiated envelope surrounding the 25 mineralized zones (Figure 14.1). The 1,629 drill holes cover the 5.7 km strike-length of the Project at regular 50 m drill spacing, locally tightening to 12.5 m close to the surface. This selection of 1,629 drill holes contains a total of 302,741 sampled intervals taken from 684,116.3 m of drill core.

A surface channel sample database was provided by GGM and integrated to the GEMS project. This database incorporated channel samples collected in 2014. The header table includes the channel sample number, collar location and length of each channel sample. Conventional assay grades were compiled. The database contains a total of 1,219 gold assays taken from 26 channel samples.

Figure 14.1: Surface Image of the Hardrock Mineralized Zones and the Limits of the Envelope Representing the Boundaries of the 2016 Mineral Resource Estimate Area



14.1.1.3 Interpretation of the Mineralized Model

In order to conduct an accurate resource modelling of the Hardrock deposit, GMS used a mineralized-zone wireframe model based on a litho-structural model divided into lithological domains predominantly controlled by a lithological component, and mineralized zones predominantly controlled by a structural component (see Subsection 7.4.2). This approach was adopted because gold mineralization is intimately associated with specific lithological units and specific structural elements and/or domains.

Lithological Domains

A total of 14 main lithological units were converted into mineralized zones while ensuring that all solids are valid, that they properly overlap each other, and that they honour the drill hole database. Overlaps were handled by the "precedence" system used by GEMS for coding the block model.

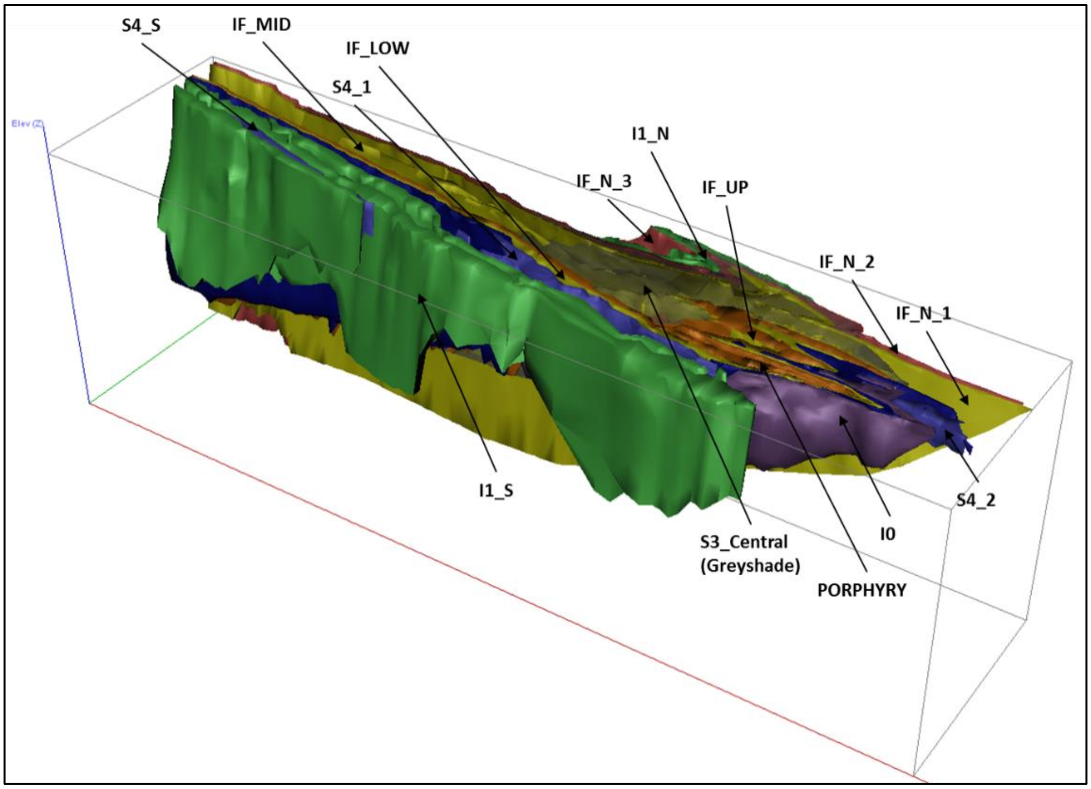
Figure 14.2 presents a 3D view of the 14 lithological units converted to mineralized zones. The nomenclature is as follows:

- Porphyry;
- Conglomerate 1 (S4_1);
- Conglomerate 2 (S4_2);
- Conglomerate South (S4_S);
- North IF 1 (IF_N_1), including the M Zone;
- North IF 2 (IF_N_2);
- North IF 3 (IF_N_3) ;
- Lower IF (IF_LOW);
- Middle IF (IF_MID);
- Upper IF (IF_UP);
- Ultramafic (IO);
- North Gabbro (I1_N);
- South Gabbro (I1_S);
- Mineralized Central Wacke (S3_Central).

The M Zone is a subdivision of the North IF 1 unit confined to an area of multiple hinge zones in the folded iron formation in order to constrain high grades.

The Mineralized Central Wacke was defined as a subdivision of the envelope based on a higher gold content and a higher concentration of gold intersects in the surrounding wacke. This mineralized zone is located in the central part of the deposit and is generally confined between the main porphyry anticline and the North Gabbro unit. The Mineralized Central Wacke has been attributed the lowest precedence of all the mineralized-zone solids, just above that of the envelope.

Figure 14.2: Isometric View of the Lithological Domains looking NW



Mineralized Zones

In 2016, eight mineralized zones were added to those of the 2014 MRE in order to address the structural control on mineralization and to impose some constraints on several of the major mineralized corridors, including the envelope. The F, Central and Tenacity zones were also revised in the 2016 MRE update.

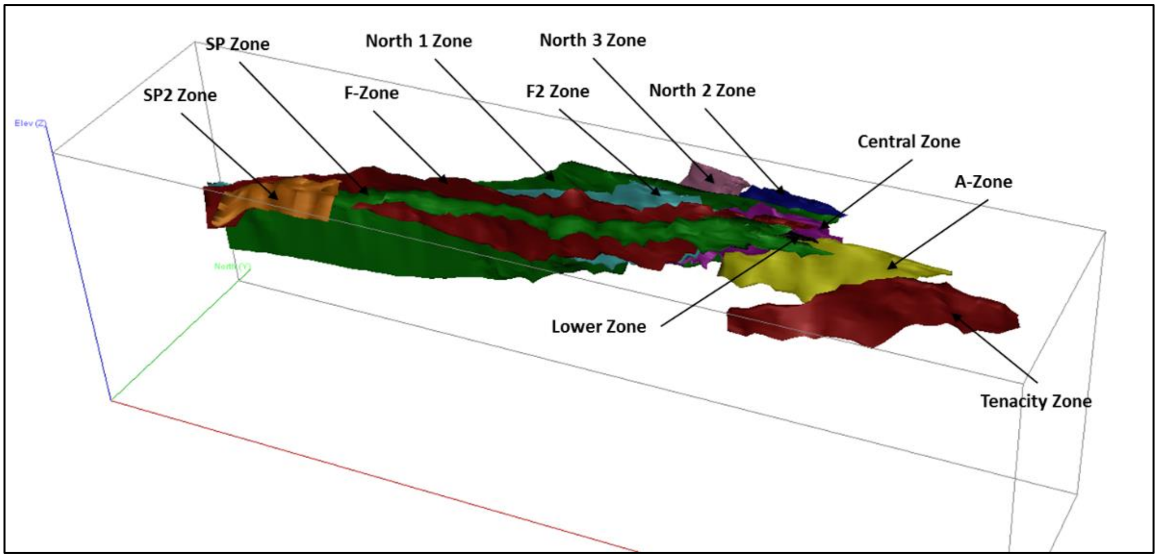
The mineralized zones were interpreted in cross section using a threshold grade of approximately 0.5 g Au/t and converted into 3D solids to ensure that they are valid and properly overlap each other. A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed. Overlaps were handled by the “precedence” system used by GEMS for coding the block model. Mineralized zones were attributed a higher precedence than lithological domains.

Figure 14.3 shows a 3D view of the 11 mineralized zones predominantly controlled by a structural component. The nomenclature is as follows:

- SP-Zone;
- SP2-Zone;
- F-Zone;
- F2-Zone;
- Central Zone;
- North 1 Zone;
- North 2 Zone;
- North 3 Zone;
- Lower Zone;
- A-Zone;
- Tenacity Zone.

The envelope was defined as the parts of a rectangular volume that are not included in any of the mineralized-zone solids. The envelope zone contains gold intersects for which continuity has not yet been demonstrated or interpreted. This envelope corresponds to an undivided greywacke country rock (or possibly other type of sedimentary rock).

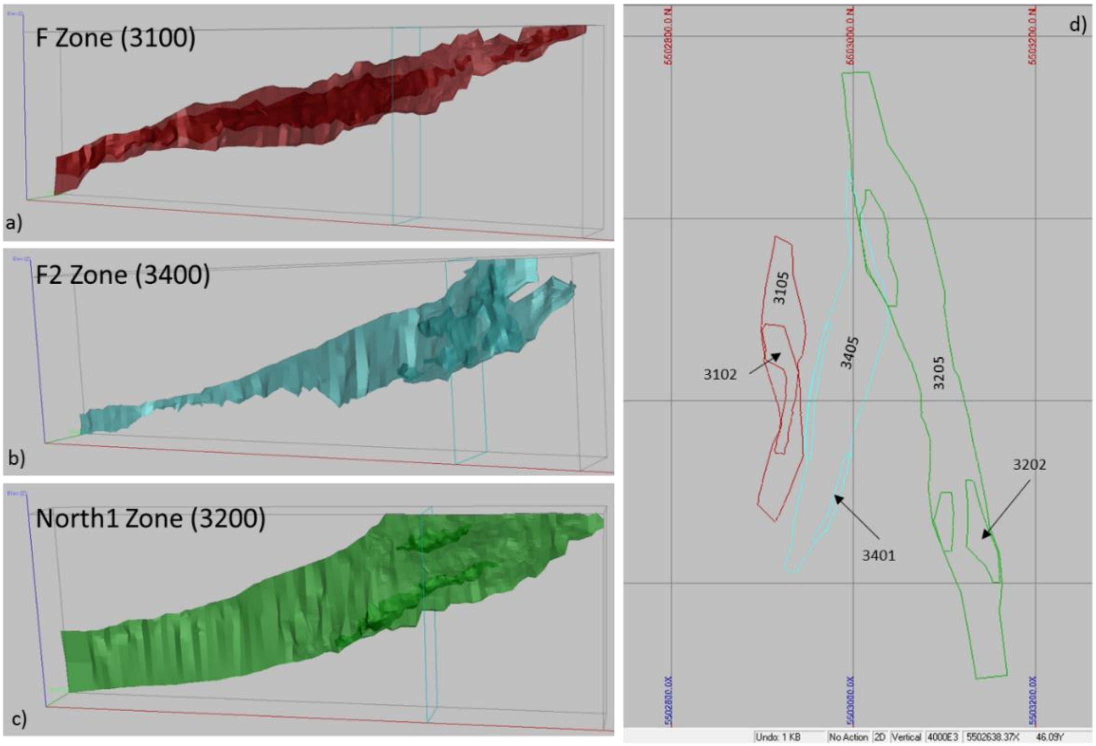
Figure 14.3: Isometric View of the Mineralized Zones Looking NW



High-grade Zones

Three of the mineralized zones (F-Zone, F2-Zone and North 1 Zone) were collecting a significant amount of low grades and possibly mixing more than one population. A higher-grade wireframe within these three mineralized zones was modelled to impose some additional constraints on the high-grade gold population of these major mineralized corridors. Figure 14.4 shows the resulting high-grade zones for the three mineralized zones.

Figure 14.4: a, b, c) Isometric Views of Mineralized Zones and their High-grade Shells, looking North; d) Cross Section 4000E showing Mineralized Zones (3105, 3405, 3205) and their High-grade Shells (3102, 3401, 3202)



Topographic and Bedrock Surfaces

A topographic surface was generated from a LIDAR survey. A bedrock surface was generated from a combination of surface drill holes and mechanical test pits completed in 2014 to evaluate the overburden thickness. Both surfaces were provided by GGM and cover the block model area and beyond.

An overburden wireframe was generated between the topographic and bedrock surfaces, corresponding to the surface limit of the mineralized-zone model. Included in the overburden wireframe, two tailings wireframes were also modelled by GGM and included in the 2016 block model for engineering purposes.

14.1.1.4 <u>High-grade Capping and Compositing</u>

For drill hole assay intervals that intersect interpreted mineralized zones, codes were automatically attributed based on the name of the 3D solids, and these coded intercepts were used to analyze sample lengths and generate statistics for high-grade capping and composites.

High-grade Capping

Basic univariate statistics were performed on raw assay datasets grouped by zone or lithology using raw analytical assay data, for a total of 302,741 drill hole samples. High-grade capping was established by mineralized zone or lithological domain based on statistical analysis, inflections of log probability plots and other considerations.

A total of 357 core samples were capped at the determined capping limits. The capping of high grade assays removed 16% of the metal content of the raw assays that will be used for the composites. Table 14.1 presents a summary of the statistical analysis for each mineralized zone and lithological domain for the drill hole raw assays.

Figure 14.5 presents the statistical analysis of the raw gold assays for the high-grade portion of the F-Zone.

The capping levels determined for the drill hole raw assay datasets were applied to the channel sample raw assays for each mineralized zone or lithological domain. A total of four samples were capped, removing 12.4% of metal content of the raw assays. Table 14.2 provides a summary of the statistics for the channel sample raw assays.

Arsenic and Sulfur Database

Basic univariate statistics were performed on raw arsenic (As) and sulfur (S) assay datasets grouped by zone or lithology using raw analytical assay data, for a total of 10,553 samples for As and 10,881 sample for S.

Table 14.3 presents a summary of the median analysis for each mineralized zone and lithological domain for the raw As and S assays.

Table 14.1: Summary Statistics-DDH Raw Au Assays by Mineralized Zones or Lithological Domain

Zone / Lithology	Block Code	Number of Samples	Max (g Au/t)	Uncut Mean (g Au/t)	High Grade Capping	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
HG F-Zone	3102	1,887	859.00	4.18	45	2.94	18	0.95%	25.34%
F-Zone	3105	6,168	427.00	1.03	35	0.86	15	0.24%	13.74%
HG North 1 Zone	3202	2,032	98.50	2.70	40	2.61	7	0.34%	3.43%
North 1 Zone	3205	14,072	402.00	0.86	35	0.73	36	0.26%	13.65%
Central Zone	3300	3,253	436.00	1.26	40	0.85	18	0.55%	30.38%
HG F2-Zone	3401	833	251.00	2.00	40	1.58	5	0.60%	18.76%
F2-Zone	3405	4,872	193.00	0.70	30	0.58	12	0.25%	15.10%
SP-Zone	3500	9,487	2,363.36	1.69	35	1.28	45	0.47%	9.94%
North 2 Zone	3600	2,761	2,000.00	2.56	30	1.79	15	0.54%	28.69%
North 3 Zone	3700	1,003	286.00	1.50	20	1.21	5	0.50%	16.69%
Lower Zone	3800	655	34.30	0.91	40	0.91	0	0.00%	0.00%
A-Zone	3900	1,338	59.28	1.10	25	1.00	8	0.60%	4.07%
Tenacity Zone	4000	2,231	1,560.00	2.18	20	1.07	13	0.58%	28.92%
SP2-Zone	4100	469	156.00	2.54	25	2.03	7	1.49%	23.10%
Porphyry	8100	24,550	114.69	0.27	15	0.25	24	0.10%	7.21%
S4_1	9100	5,408	40.20	0.15	20	0.14	1	0.02%	1.96%
S4_2	9200	523	27.80	0.14	20	0.13	1	0.19%	9.47%
S4_South	10100, 10200, 10300, 10400	651	7.21	0.05	20	0.05	0	0.00%	0.00%
IF_N_1	11100 (11110 to 11150)	11,143	199.00	0.36	20	0.32	16	0.14%	8.29%
IF_N_1 (M_Zone)	11160	2,656	133.00	0.86	20	0.73	17	0.64%	13.50%
IF_N_2	11200	12,481	251.00	0.34	20	0.30	20	0.16%	10.16%
IF_N_3	11300	1,932	6.03	0.05	20	0.05	0	0.00%	0.00%
IF_LOW	12000	6,858	200.89	0.31	20	0.25	10	0.15%	13.68%
IF_MID	13100, 13200	403	2.55	0.05	20	0.05	0	0.00%	0.00%
IF_UP	14100, 14200, 14300	5,357	101.00	0.23	20	0.20	6	0.11%	9.85%
I0	15000	2,136	3.90	0.07	20	0.07	0	0.00%	0.00%
I1_N	16000	10,840	115.00	0.16	20	0.14	11	0.10%	12.63%
I1_S	17100, 17200, 17300, 17400	2,929	13.60	0.07	20	0.07	0	0.00%	0.00%
S3_Central (Mineralized Wacke)	18000	68,516	2,870.00	0.29	40	0.18	20	0.03%	23.37%
Envelope	20000	95,297	262.00	0.07	20	0.06	27	0.03%	15.25%
Total		**302,741**	**2,870.00**	**0.41**	**Variable**	**0.32**	**357**	**0.12%**	**16.04%**

Table 14.2: Summary Statistics for Channel Sample Raw Au Assays by Mineralized Zone or Lithological Domain

Zone/ Lithology	Block Code	# Samples	Max (g Au/t)	Uncut Mean (g Au/t)	High Grade Capping	Cut Mean (g Au/t)	# Samples Cut	% Samples Capped	% Loss Metal Factor
HG F-Zone	3102	18	4.59	1.55	45	1.55	0	0.00%	0.00%
F-Zone	3105	109	30.00	2.03	35	2.03	0	0.00%	0.00%
North 1 Zone	3205	239	110.00	1.26	35	0.84	2	0.84%	29.61%
F2-Zone	3405	65	1.27	0.13	30	0.13	0	0.00%	0.00%
SP-Zone	3500	499	38.90	0.53	35	0.53	1	0.20%	1.51%
Porphyry	8100	22	0.48	0.17	15	0.17	0	0.00%	0.00%
IF_UP	14100	11	0.50	0.11	20	0.11	0	0.00%	0.00%
S3_Central (Mineralized Wacke)	18000	256	57.60	0.46	40	0.39	1	0.39%	14.84%
Total		**1,219**	**110.00**	**0.78**	**Variable**	**0.68**	**4.0**	**0.33%**	**12.41%**

Figure 14.5: Statistical Analysis for DDH Raw Au Assays of the High-grade F-Zone (3102)

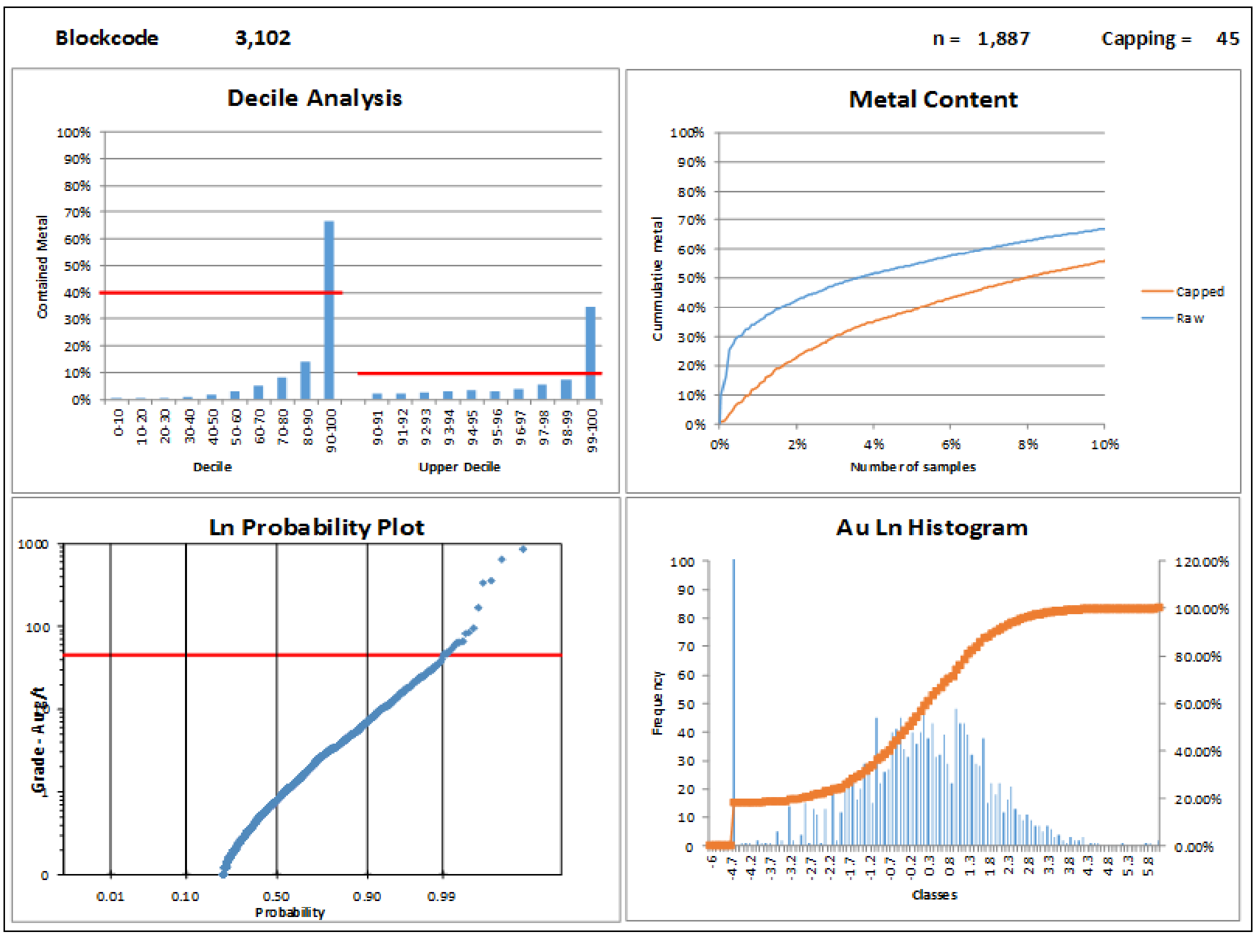


Table 14.3: Summary Statistics for As and S Sample Raw Assays by Mineralized Zone or Lithological Domain

Zone/ Lithology	Block Code	Number of Assays (As)	Median (As ppm)	Number of Assays (S)	Median (S%)
HG F-Zone	3102	46	333	46	0.73
F-Zone	3105	177	98	191	0.32
HG North 1 Zone	3202	33	6,620	51	2.79
North 1 Zone	3205	195	118	249	0.28
Central Zone	3300	103	146	121	0.51
HG F2-Zone	3401	36	276	36	0.48
F2-Zone	3405	81	131	81	0.29
SP-Zone	3500	656	84	656	0.57
North 2 Zone	3600	93	300	93	1.52
North 3 Zone	3700	91	169	91	0.37
Lower Zone	3800	70	6	70	0.18
A-Zone	3900	58	1	58	0.28
Tenacity Zone	4000	22	2,205	22	0.72
SP2-Zone	4100	-	-	-	-
Porphyry	8100	1,556	5	1,556	0.35
S4_1	9100	74	23	74	0.27
S4_2	9200	-	-	-	-
S4_South	10100, 10200, 10300, 10400	50	6	50	0.38
IF_N_1	11100 (11110 to 11150)	331	7	332	0.11
IF_N_1 (M_Zone)	11160	-	-	-	-
IF_N_2	11200	248	12	261	0.17
IF_N_3	11300	17	8	17	0.10
IF_LOW	12000	458	4	458	0.17
IF_MID	13100, 13200	-	-	-	-
IF_UP	14100, 14200, 14300	316	7	322	0.20
I0	15000	41	303	41	0.14
I1_N	16000	202	132	202	0.12
I1_S	17100, 17200, 17300, 17400	1	6	1	0.18
S3_Central (Mineralized Wacke)	18000	2,415	44	2,506	0.22
Envelope	20000	3,183	24	3,216	0.20
Total		**10,553**		**10,801**	

Compositing

In order to minimize any bias introduced by the variable sample lengths, the capped gold assays of the DDH and channel sample data were composited to equal lengths of 1.5 m (“1.5 m composites”) within all intervals that define each of the mineralized zones. The compositing parameters were fixed to allow intervals to be less than 1.5 m but not less than 0.75 m. The total number of composites used in the dataset is 386,506, combining DDH and channel sample data. A zero grade was assigned to missing sample intervals. Table 14.4 and Table 14.5 summarize the basic statistics for the gold composites for the DDH and channel sample data respectively.

Table 14.4: Summary Statistics for the 1.5 m DDH Composites

Zone/ Lithology	Block Code	Number of Composites	Max (g Au/t)	Mean (g Au/t)	Standard Deviation	Coefficient of Variation
HG F-Zone	3102	1,708	45.00	2.42	4.56	1.88
F-Zone	3105	6,046	35.00	0.74	1.84	2.47
HG North 1 Zone	3202	1,760	40.00	2.30	4.37	1.90
North 1 Zone	3205	13,067	35.00	0.64	2.05	3.22
Central Zone	3300	2,910	40.00	0.75	2.60	3.48
HG F2-Zone	3401	726	38.75	1.42	3.05	2.15
F2-Zone	3405	4,537	29.98	0.50	1.67	3.32
SP-Zone	3500	8,833	35.00	1.02	2.44	2.39
North 2 Zone	3600	2,379	28.50	1.49	3.38	2.28
North 3 Zone	3700	873	18.97	1.10	1.91	1.74
Lower Zone	3800	583	18.65	0.85	1.74	2.05
A-Zone	3900	1,288	16.21	0.72	1.38	1.91
Tenacity Zone	4000	2,107	16.33	0.82	1.55	1.89
SP2-Zone	4100	421	25.00	1.73	3.28	1.89
Porphyry	8100	28,294	13.38	0.18	0.51	2.75
S4_1	9100	8,664	13.73	0.08	0.41	5.34
S4_2	9200	611	7.84	0.08	0.42	5.14
S4_South	10100, 10200, 10300, 10400	1,896	4.84	0.02	0.16	9.45
IF_N_1	11100 (11110 to 11150)	10,414	20.00	0.26	1.03	3.89
IF_N_1 (M_Zone)	11160	2,379	19.43	0.63	1.72	2.72
IF_N_2	11200	12,026	19.98	0.23	0.90	3.84
IF_N_3	11300	2,916	5.28	0.03	0.18	5.88
IF_LOW	12000	8,227	20.00	0.16	0.79	4.93
IF_MID	13100, 13200	596	1.46	0.03	0.12	3.85
IF_UP	14100, 14200, 14300	5,714	17.26	0.15	0.61	4.10
I0	15000	5,737	3.86	0.02	0.15	6.61
I1_N	16000	14,331	19.79	0.09	0.52	5.91
I1_S	17100, 17200, 17300, 17400	15,010	8.36	0.01	0.17	13.95
S3_Central (Mineralized Wacke)	18000	68,231	38.08	0.15	0.76	5.04
Envelope	20000	154,018	19.94	0.03	0.27	8.34
Total		**385,625**				

Table 14.5: Summary Statistics for the 1.5 m Channel Composites

Zone/ Lithology	Block Code	Number of Composites	Max (g Au/t)	Mean (g Au/t)	Standard Deviation	Coefficient of Variation
HG F-Zone	3102	19	4.10	1.52	1.02	1.49
F-Zone	3105	104	15.39	2.07	2.73	0.76
North 1 Zone	3205	187	22.21	0.77	2.46	0.31
F2-Zone	3405	51	0.85	0.13	0.17	0.77
SP-Zone	3500	316	17.06	0.54	1.26	0.43
Porphyry	8100	14	0.44	0.17	0.14	1.18
IF_UP	14100	8	0.43	0.11	0.13	0.89
S3_Central (Mineralized Wacke)	18000	182	13.58	0.37	1.42	0.26
Total		**881**				

14.1.1.5 Interpolation Strategy

Structural Domain Subdivisions

Thirteen of the 14 lithological domains and one of the mineralized zones (North 3 Zone) were subdivided into subunits on the basis of their structural characteristics in order to define datasets of similar orientation for the benefit of the geostatistical analysis. Hinges were separated into antiform and synform entities and limbs were separated into north or south dipping. Cutting planes were defined in GOCAD to separate hinges from limbs in order to give each of these subunits a “subunit block code” to which specific research ellipsoids could be attributed. Figure 14.6 and Figure 14.7 illustrate the subdivisions of the Porphyry Zone as well as the cutting planes used, and thus represent the methodology used to subdivide the zones.

Variography

A 3D directional-specific variography was completed, where possible, by subunits or mineralized zones, using the 1.5 m composites of the capped gold assay data for the DDH populations.

The result of the 3D variographic investigations for the mineralized zone composites are consistent with the univariate statistics and generally correlate with geological features of the deposit. Some changes were introduced to the best-fit model to better reflect the geological model. Figure 14.8 presents directional variograms for the F-Zone.

Search Ellipsoids

Taking into consideration the subdivisions defined above, 68 distinct subunits were used to characterize the search ellipsoids. Different search ellipsoids were created for each subunit based on the variography orientations.

Figure 14.6: Isometric View showing the Division of the Porphyry Zone into Subunits as an Example of the Methodology used to Subdivide Zones on the Basis of their Structural Characteristics (looking southeast)

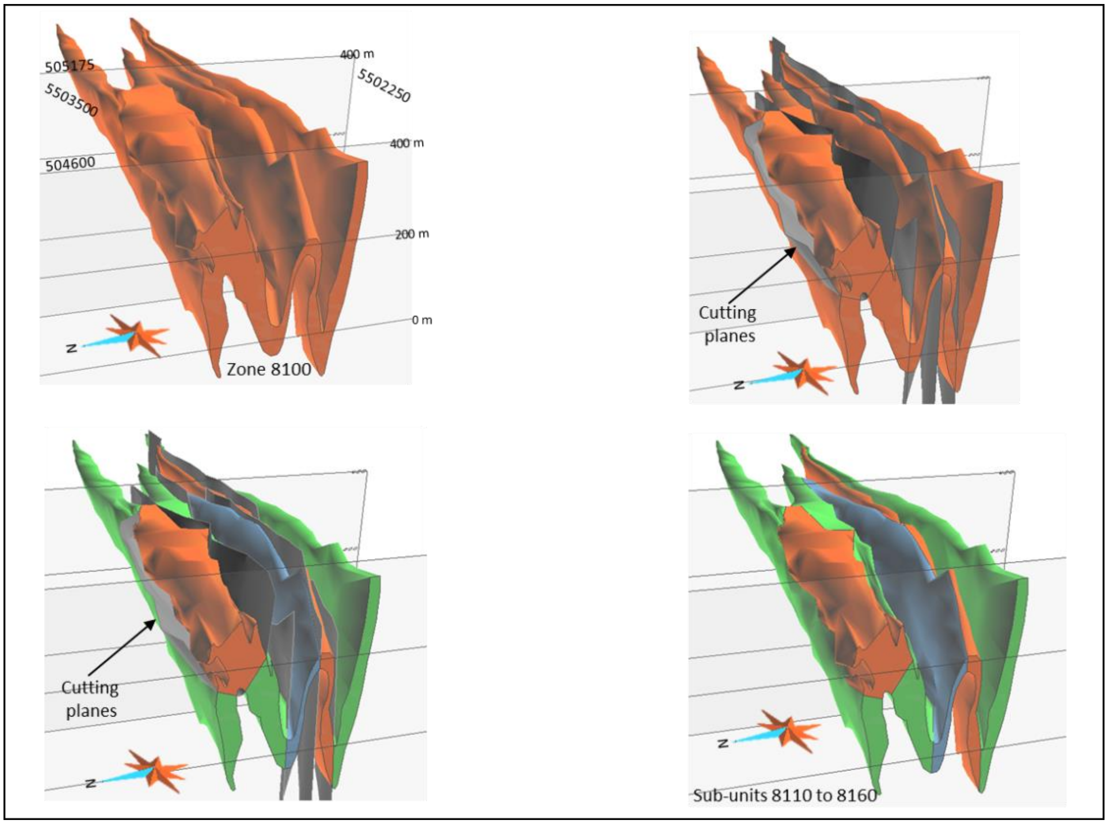


Figure 14.7: Cross-section of the Porphyry Zone Subdivided into Six Subunits (Looking east)

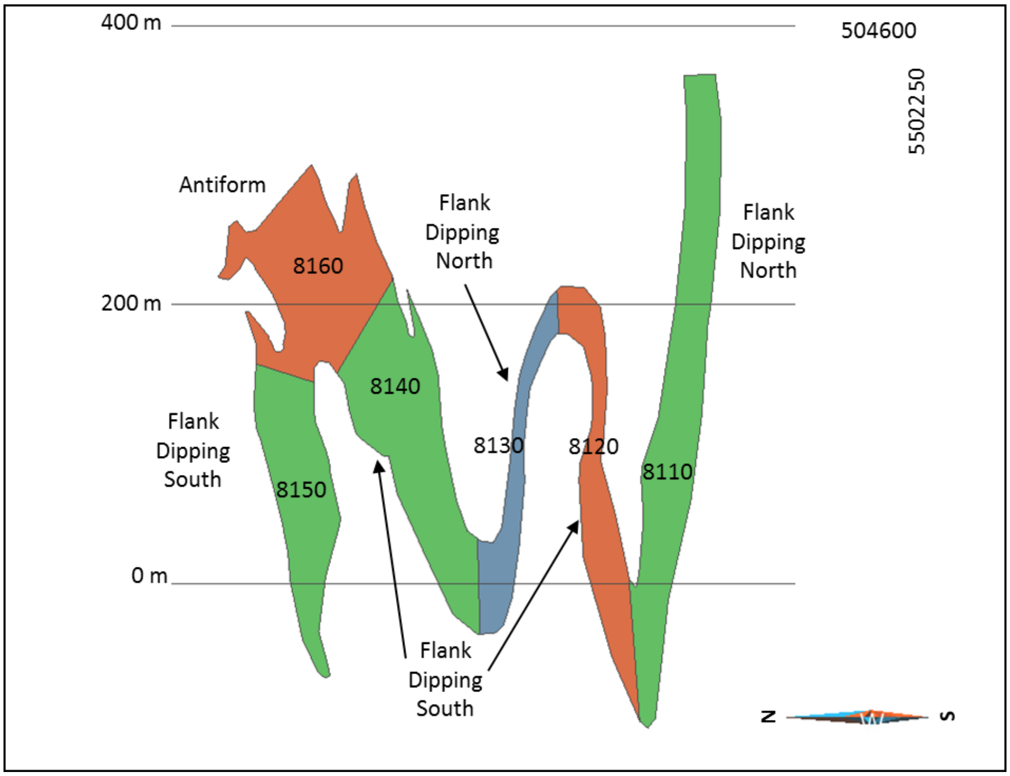
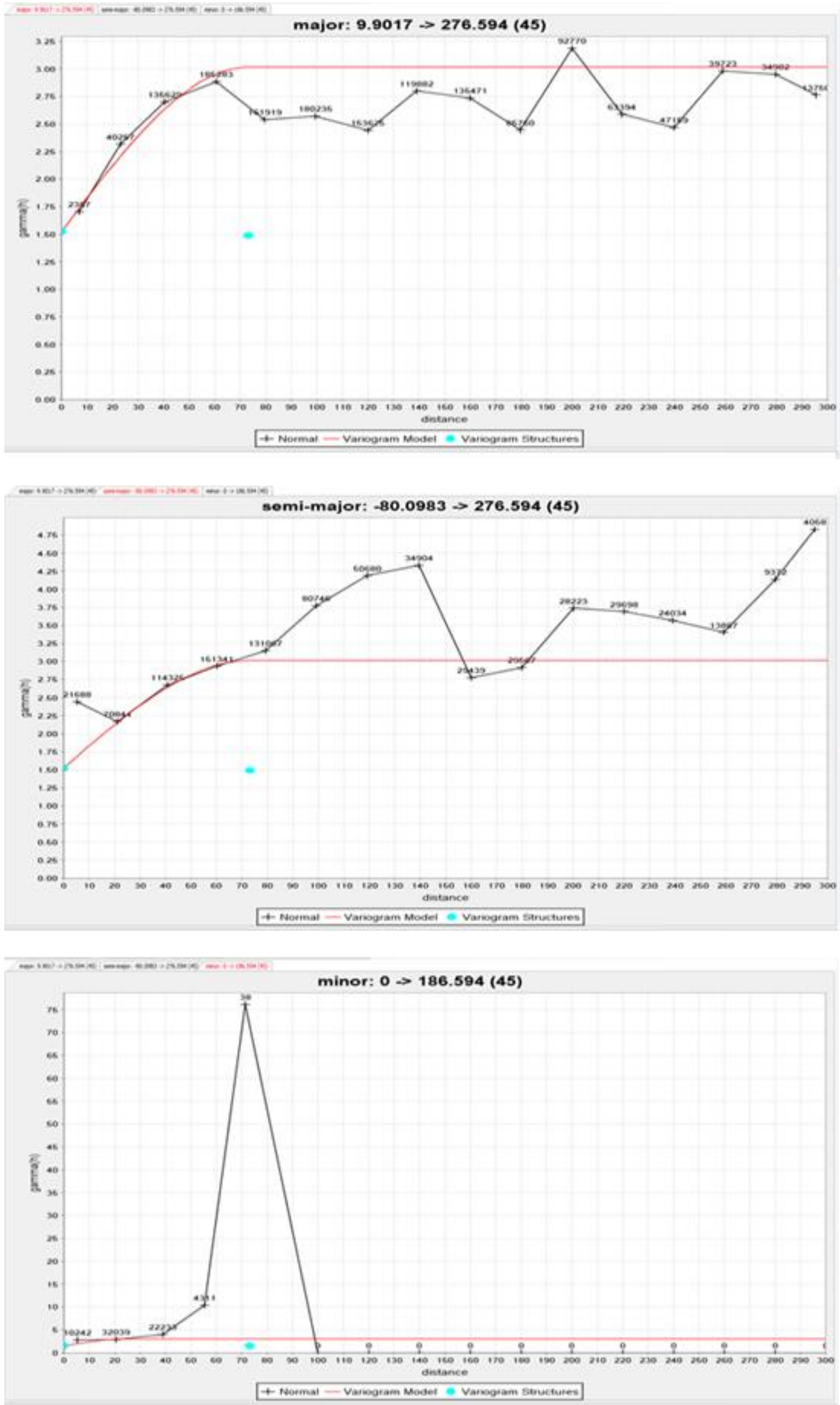


Figure 14.8: Directional Variograms for the F-Zone (3105)



Three sets of search ellipsoids were used for the final interpolation. The ellipsoid radiuses from Pass 1 were established using the ranges determined from the geostatistical analysis. The ellipsoid radiuses from Pass 2 were fixed at values equivalent to 2x and 2.5x the Pass 1 ranges for the mineralized zones and lithological domains, respectively. The ellipsoid radiuses from Pass 3 were fixed to 2.5x or 3x the Pass 1 ranges for the mineralized zones and lithological domains, respectively.

One size of search ellipse was applied to all lithological domains based on the mean range and anisotropy of the geostatistical analysis of the major units. An elongated shape was given to the ellipsoids of the hinges compared to the limbs. For the M Zone, a sphere was used for the interpolation based on the shape of the subunit, which reflects a sequence of antiform and synform features.

Table 14.6 summarizes the parameters of the final ellipsoids used for interpolation.

Boundaries

Three types of boundaries were selected for the grade interpolation of the 2016 block model. For the mineralized zones, hard boundaries were applied as described below.

The interpolation profiles specify a single target and sample rock code for each mineralized-zone solid or subunit solid, thus establishing hard boundaries between the mineralized zones and preventing block grades from being estimated using sample points with different block codes than the block being estimated.

The subdivision of the high-grade portions for the three mineralized zones (F-Zone, F2 and North 1) had a favourable impact on the resulting coefficient of variation, which confirms that the assay population was adequately separated (see Table 14.4). Contact plots were completed on the mineralized zones and their high-grade portions to evaluate the pertinence of hard boundary management for the grade interpolation. The results of the contact plots support the use of a hard boundary between the high grades and their mineralized zones as shown on Figure 14.9. The use of a hard boundary prevents the smearing of high grades in the lower grade domain and the dilution of the high-grade domain by external low grade material.

Table 14.6: Final Search Ellipsoid Parameters

Zone/ Lithology	Geometry	Block Code	Rotation*			Pass 1 -Radius			Pass 2 -Radius			Pass 3 -Radius		
			Z°	X°	Z°	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)
HG F-Zone	Dipping_South	3102	-5	90	20	60	44	26	120	88	52	150	110	65
F-Zone	Dipping_South	3105	-5	90	20	73	73	24	146	146	48	183	183	60
HG North 1 Zone	Dipping_North	3202	180	75	-55	40	33	31	80	66	62	100	83	78
North 1 Zone	Dipping_North	3205	180	75	-55	39	30	28	78	60	56	98	75	70
Central Zone	Dipping_South	3300	-10	70	30	51	51	24	102	102	48	128	128	60
HG F2-Zone	Dipping_South	3401	175	-80	35	38	38	16	76	76	32	95	95	40
F2-Zone	Dipping_South	3405	175	-80	35	50	33	30	100	66	60	125	83	75
SP-Zone	Antiform_Dipping_South	3500	0	75	15	52	35	26	104	70	52	130	88	65
North 2 Zone	Dipping_South	3600	-5	75	52.5	35	18	17	70	36	34	88	45	43
North 3 Zone	Flank_Dipping_South	3710	-15	70	45	31	23	18	62	46	36	78	58	45
	Flank_Dipping_South	3720	-15	70	45	31	23	18	62	46	36	78	58	45
Lower Zone	Antiform_Dipping_South	3800	-5	80	22.5	38	25	19	76	50	38	95	63	48
A-Zone	Dipping_South	3900	-5	65	25	38	19	18	76	38	36	95	48	45
Tenacity Zone	Dipping_South	4000	-5	70	22.5	48	27	15	96	54	30	120	68	38
SP2-Zone	Dipping_South	4100	-30	40	35	35	22	15	70	44	30	88	55	38
Porphyry	Flank_Dipping_South	8110	-5	80	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	8120	-5	80	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	8130	5	-80	-15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	8140	-5	70	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	8150	-10	80	15	30	19	13	75	47	33	90	56	39
	Antiform_Dipping_South	8160	-5	90	15	30	13	13	75	33	33	90	39	39
S4_1	Flank_Dipping_South	9110	-10	80	7.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	9120	-5	70	7.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	9130	5	-80	-7.5	30	19	13	75	47	33	90	56	39
	Synform_Dipping_South	9140	0	80	7.5	30	13	13	75	33	33	90	39	39
S4_2	Dipping_South	9200	0	65	20	30	19	13	75	47	33	90	56	39
S4_South 1	Dipping_South	10100	-5	80	15	30	19	13	75	47	33	90	56	39
S4_South 2	Dipping_South	10200	-20	65	15	30	19	13	75	47	33	90	56	39
S4_South 3	Dipping_South	10300	-10	70	15	30	19	13	75	47	33	90	56	39
S4_South 4	Dipping_South	10400	-15	80	15	30	19	13	75	47	33	90	56	39
IF_N_1	Flank_Dipping_North	11110	-5	-85	-22.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	11120	0	-75	-22.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	11130	-5	-85	-22.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	11140	-10	70	22.5	30	19	13	75	47	33	90	56	39

Zone/ Lithology	Geometry	Block Code	Rotation*			Pass 1 -Radius			Pass 2 -Radius			Pass 3 -Radius		
			Z°	X°	Z°	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)	X (m)	Y (m)	Z (m)
	Flank_Dipping_South	11150	-10	75	22.5	30	19	13	75	47	33	90	56	39
M Zone	Antiforms and Synforms	11160	90	0	0	28	28	28	70	70	70	84	84	84
IF_N_2	Flank_Dipping_South	11210	-10	70	20	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	11220	-5	90	20	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	11230	5	-75	-20	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	11240	-25	70	20	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	11250	-10	75	20	30	19	13	75	47	33	90	56	39
IF_N_3	Flank_Dipping_North	11310	0	-80	-25	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	11320	-20	70	25	30	19	13	75	47	33	90	56	39
IF_LOW	Flank_Dipping_South	12010	-5	60	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	12020	-5	80	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	12030	5	-75	-15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	12040	-5	70	15	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	12050	-5	70	15	30	19	13	75	47	33	90	56	39
IF_MID 1	Dipping_South	13100	-10	90	15	30	19	13	75	47	33	90	56	39
IF_MID 2	Dipping_South	13200	-5	85	15	30	19	13	75	47	33	90	56	39
IF_UP 1	Flank_Dipping_South	14110	0	75	20	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	14120	-15	60	20	30	19	13	75	47	33	90	56	39
IF_UP 2	Dipping_South	14200	-5	75	20	30	19	13	75	47	33	90	56	39
IF_UP 3	Dipping_South	14300	-5	70	20	30	19	13	75	47	33	90	56	39
I0	Flank_Dipping_North	15010	0	-85	-30	30	19	13	75	47	33	90	56	39
	Flank_Dipping_South	15020	5	70	30	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	15030	5	-85	-30	30	19	13	75	47	33	90	56	39
	Synform	15040	5	85	30	30	13	13	75	33	33	90	39	39
I1_N	Dipping_South	16010	-10	70	37.5	30	19	13	75	47	33	90	56	39
	Synform_Dipping_South	16020	-10	80	37.5	30	13	13	75	33	33	90	39	39
	Flank_Dipping_South	16030	-10	75	37.5	30	19	13	75	47	33	90	56	39
	Flank_Dipping_North	16040	0	-80	-37.5	30	19	13	75	47	33	90	56	39
I1_S 1	Dipping_South	17100	-10	80	15	30	19	13	75	47	33	90	56	39
I1_S 2	Dipping_South	17200	-10	80	15	30	19	13	75	47	33	90	56	39
I1_S 3	Dipping_South	17300	-10	75	15	30	19	13	75	47	33	90	56	39
I1_S 4	Dipping_South	17400	-10	80	15	30	19	13	75	47	33	90	56	39
S3_Central (Mineralized Wacke)	n.a	18000	-11	84	20	33	21	14	83	52	36	99	62	43
Envelope	n.a	20000	-11	84	20	30	19	13	75	47	33	90	56	39

*Note: *Block model system: positive rotation is counter-clockwise*

Semi-soft boundaries were applied for the F2 and Central zones at their contact with the M Zone, affecting only one fringe of blocks at the contact. Therefore, the selected fringe of blocks for each zone was interpolated using composites of the M Zone, as well as the composites of the F2 or Central zones. This methodology was suggested by the geological interpretation and style of mineralization, and was confirmed by the contact plots.

For all lithological domains, soft boundary management for the grade interpolation was applied between subdomains of a same domain. This means, for example, that the grade in each Porphyry subdomain was interpolated using composites of all other Porphyry subdomains within the limits of its own search ellipsoid.

Treatment of High Grades

In order to improve the treatment of high grades, a detailed composite coefficient of variation ("CoV") study was completed by domain. For domains where the CoV was below 2, no search restriction was applied.

For domains where the CoV was higher than 2 after removing composites below 0.1 g Au/t (LG trim), it was decided to use a search restriction. The methodology to determine the threshold consists of removing the highest grade composites until a CoV below or equal to 2 is reached (HG trim). The remaining maximum grade was selected for the threshold to apply by the search restriction.

The size of the search restriction was fixed to half the distance of the variography to make sure that the restriction is applied starting with the first pass. Table 14.7 demonstrates that the size of the search restriction varies from 15 to 25 m, which corresponds with the maximum continuity of high-grade assays (>15 g Au/t). The maximum continuity of grades up to 15 g Au/t was demonstrated using isoshells created by combining underground and surface drill holes. For grades greater than 15 g Au/t, the search restriction does not allow high grade to be used over the mid-distance of the drill spacing.

Figure 14.9: Contact Plots of the Mineralized Zone (right side) and their Highest Grade Portion a) F-Zone, b) North 1 Zone, c) F2-Zone

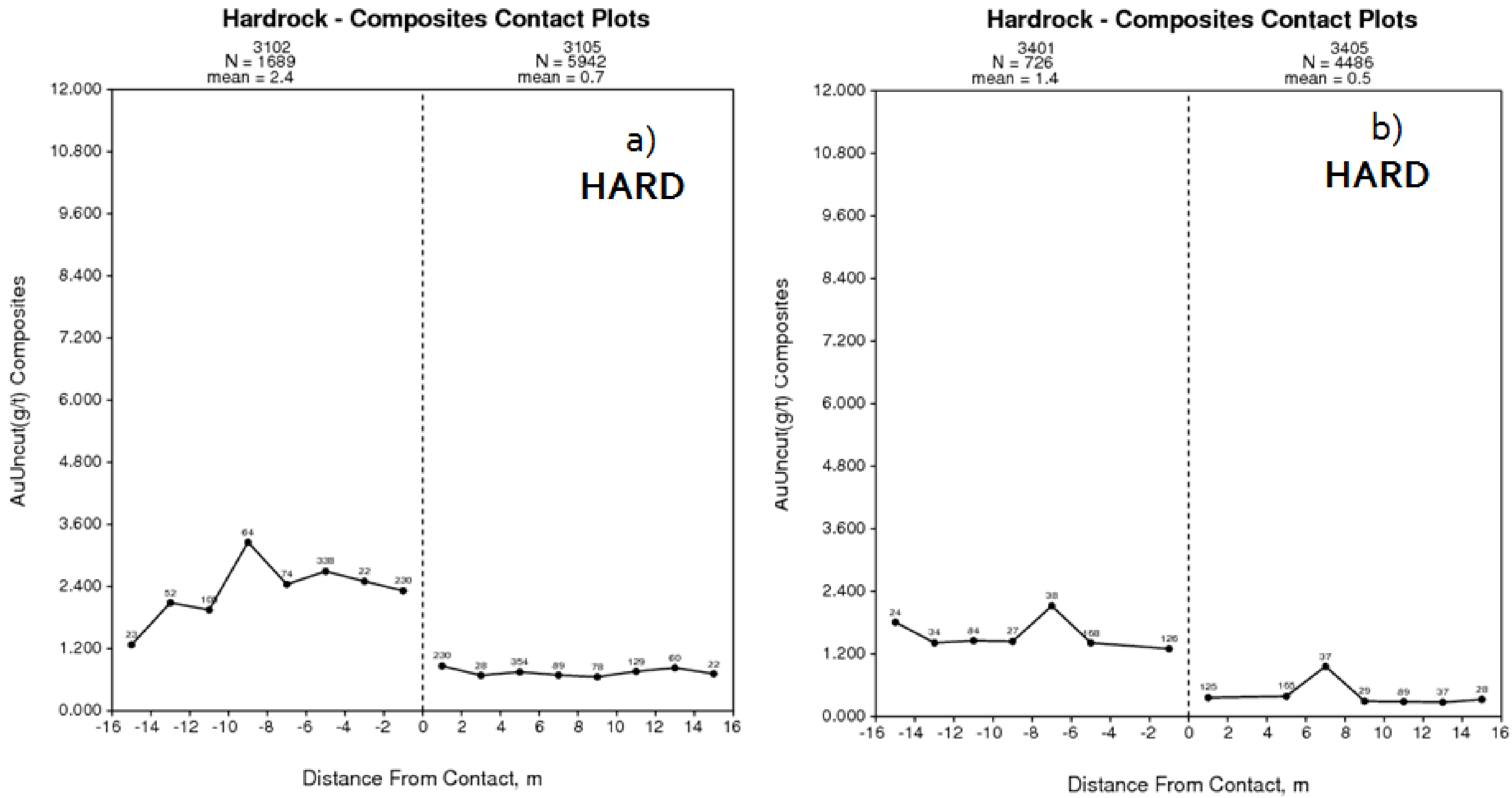


Table 14.7: Search Restrictions

Zone / Lithology	Block Code	Search Restriction Ellipsoid			
		Range 1 (m)	Range 2 (m)	Range 3 (m)	Transition Value (g Au/t)
North 1 Zone	3205	20	15	14	25
Central Zone	3300	26	26	12	25
F2-Zone	3405	25	17	15	25
I1_N	16010	15	9	7	15
	16020	15	7	7	15
	16030	15	9	7	15
	16040	15	9	7	15
S3_Central (Mineralized Wacke)	18000	17	10	7	20
Envelope	20000	15	9	7	15

14.1.1.6 Bulk Density

For the 2016 MRE, a total of 6,925 bulk density measurements were provided by GGM and integrated into the database. Lithology densities were measured at the Geraldton core shack by GGM personnel. A fixed density was applied for each mineralized zone and lithological domain based on a statistical analysis. Lithological domains S4_1 and S4_2 were grouped for statistical study because there were too few density measurements in Zone S4_2. A density of 2.00 g/cm^3 was assigned to the overburden. A density of 2.05 g/cm^3 was assigned to the tailings. For the voids, a zero density was used for drifts and stopes that are classified as “open” and assumed to be filled with water. GGM provided density measurements for stopes backfilled with sand and waste of 2.02 g/cm^3 and 2.08 g/cm^3 respectively (see details of backfilled stope classification in Subsection 12.1.6).

Bulk densities were used to calculate tonnages from the volume estimates in the resource-grade block model. Table 14.8 presents the bulk densities used for each gold zone (average values).

Table 14.8: Bulk Density by Zone

Zone/ Lithology	Block Code	Lithology	Bulk Density (g/cm3)				
			Count	Average	Min	Max	Median
F-Zone	3100 (3102, 3105)	Greywacke	104	2.75	2.58	3.43	2.73
North 1 Zone	3200 (3202, 3205)	Iron Formation and Greywacke	227	2.91	2.50	3.75	2.79
Central Zone	3300	Greywacke	80	2.77	2.25	3.44	2.75
F2-Zone	3400 (3401, 3405)	Greywacke	101	2.76	2.55	3.38	2.75
SP-Zone	3500	Porphyry and Iron Formation	95	2.76	2.50	3.33	2.73
North 2 Zone	3600	Iron Formation and Greywacke	20	3.03	2.66	3.76	2.96
North 3 Zone	3700	Iron Formation and Greywacke	17	2.84	2.66	3.27	2.80
Lower Zone	3800	Mixed (IF, Porphyry and Greywacke)	7	2.86	2.70	3.27	2.80
A-Zone	3900	Porphyry and Iron Formation	12	2.78	2.69	2.94	2.75
Tenacity Zone	4000	Greywacke	11	2.83	2.57	3.57	2.72
SP2-Zone	4100	Greywacke	6	2.72	2.58	2.78	2.75
Porphyry	8100	Porphyry	580	2.74	2.31	3.62	2.73
S4_1 and 2	9100, 9200	Conglomerate	144	2.75	2.47	3.41	2.75
S4_South	10100, 10200, 10300, 10400	Conglomerate	45	2.77	2.53	3.77	2.74
IF_N_1	11100 (11110 to 11160)	Iron Formation	264	3.01	2.54	3.80	3.04
IF_N_2	11200	Iron Formation	221	2.88	2.45	4.05	2.76
IF_N_3	11300	Iron Formation	37	2.87	2.52	3.48	2.75
IF_LOW	12000	Iron Formation	156	2.86	2.50	3.71	2.78
IF_MID	13100, 13200	Iron Formation	10	3.26	2.72	3.94	3.28
IF_UP	14100, 14200, 14300	Iron Formation	95	2.76	2.58	3.72	2.74
I0	15000	Ultramafic	88	2.90	2.50	3.79	2.89
I1_N	16000	Gabbro	225	2.78	2.55	3.52	2.75
I1_S	17100, 17200, 17300, 17400	Gabbro	385	2.77	2.50	3.50	2.75
S3_Central (Mineralized Wacke)	18000	Greywacke	1,201	2.76	2.15	3.64	2.74
Envelope	20000	Greywacke	2,794	2.75	2.27	3.92	2.74
Total			**6,925**				

14.1.1.7 Block Model

A block model was established for the 25 mineralized zones and the envelope. The block model was extended to cover an area sufficient to host an open pit. The model has been pushed down to a depth of approximately 1,800 m below surface. The block model was not rotated (Y-axis oriented along a N000 azimuth). The block dimensions reflect the sizes of the mineralized zones and could be used or any mining method. Table 14.9 presents the properties of the block model.

Table 14.9: Block Model Properties

Properties	X (Columns)	Y (Rows)	Z (Levels)
Origin Coordinates (UTM NAD83, Zone 16)	501,050	5,502,000	500
Block Extent (m)	575	340	192
Block Size	10	5	10
Rotation	Not applied		

A percent block model was generated reflecting the proportion of each block inside every solid (mineralized zones and their subunits, envelope, overburden, tailings, stopes and drifts). The 25 mineralized zones and their subunits, as well as the envelope, were coded in one folder using the 50/50 rule for the attribution of a block code. Precedence was respected during the process.

As described below, the percent block model was readjusted by script manipulation according to the level of precision of each stope and drift.

- Medium Precision: minimum percent of each block coded M_stope or M_drift fixed at 35%, all other folders adjusted accordingly;
- High Precision: exact percent for all blocks coded H_stope or H_drift and all other folders.

Table 14.10 provides details about the naming convention for the corresponding GEMS solids, as well as the rock codes and block codes assigned to each individual solid. The multi-folder percent block model thus generated was used in the Mineral Resource estimation.

Table 14.10: Hardrock Block Model

Folder	Description	GEMS Triangulation Name			Rock Code/ Block Code/ Precedence
		NAME 1	NAME 2 (Rock Type Unified)	NAME 3 (Rock Type Submit)	
Openings_All	H_STOPE_Waste	ML/HR	H_STOPE	xx	811
	H_STOPE_Sand	ML/HR	H_STOPE	xx	812
	H_STOPE_Open	ML/HR	H_STOPE	xx	813
	M_STOPE_Waste	ML/HR	M_STOPE	xx	821
	M_STOPE_Open	ML/HR	M_STOPE	xx	823
	H_DRIFT_Open	ML/HR	H_DRIFT	xx	913
	M_DRIFT_Open	ML/HR	M_DRIFT	xx	923
Overburden	OVERBURDEN	OVB	Final	Clip	1000
Tailings	TAILINGS	TAILINGS	HARDROCK/MCL EOD	20150702_F	1500
Zones_5050	HG F-Zone	HG	3100	3102F	3102
	F-Zone	Z_LG	3100	3105	3105
	HG North 1 Zone	HG	3200	3202F	3202
	North 1 Zone	Z_LG	3200	3205	3205
	Central Zone	Z_UN	3300	3300_ClipIF	3300
	HG F2-Zone	HG	3400	3401F	3401
	F2-Zone	Z_LG	3400	3405_ClipIF	3405
	SP-Zone	Z_UN	3500	3500	3500
	North 2 Zone	Z_UN	3600	3600	3600
	North 3 Zone	Z_SUB	3700	3710	3710
		Z_SUB	3700	3720	3720
	Lower Zone	Z_UN	3800	3800	3800
	A-Zone	Z_UN	3900	3900	3900
	Tenacity Zone	Z_UN	4000	4000	4000
	SP2-Zone	Z_UN	4100	4100	4100
	Porphyry	L_SUB	8100	8110	8110
		L_SUB	8100	8120	8120
		L_SUB	8100	8130	8130
		L_SUB	8100	8140	8140
		L_SUB	8100	8150	8150
		L_SUB	8100	8160	8160
	S4_1	L_SUB	9100	9110	9110
		L_SUB	9100	9120	9120
		L_SUB	9100	9130	9130
		L_SUB	9100	9140	9140
	S4_2	L_UN	9200	9200	9200
	S4_South 1	L_UN	10100	10100	10100
	S4_South 2	L_UN	10200	10200	10200
	S4_South 3	L_UN	10300	10300	10300
	S4_South 4	L_UN	10400	10400	10400

Folder	Description	GEMS Triangulation Name			Rock Code/ Block Code/ Precedence
		NAME 1	NAME 2 (Rock Type Unified)	NAME 3 (Rock Type Submit)	
Zones_5050	IF_N_1	L_SUB	11100	11110	11110
		L_SUB	11100	11120	11120
		L_SUB	11100	11130	11130
		L_SUB	11100	11140	11140
		L_SUB	11100	11150	11150
	M Zone	L_SUB	11100	11160	11160
	IF_N_2	L_SUB	11200	11210	11210
		L_SUB	11200	11220	11220
		L_SUB	11200	11230	11230
		L_SUB	11200	11240	11240
		L_SUB	11200	11250	11250
	IF_N_3	L_SUB	11300	11310	11310
		L_SUB	11300	11320	11320
	IF_LOW	L_SUB	12000	12010	12010
		L_SUB	12000	12020	12020
		L_SUB	12000	12030	12030
		L_SUB	12000	12040	12040
		L_SUB	12000	12050	12050
	IF_MID 1	L_UN	13100	13100	13100
	IF_MID 2	L_UN	13200	13200	13200
	IF_UP 1	L_SUB	14100	14110	14110
		L_SUB	14100	14120	14120
	IF_UP 2	L_UN	14200	14200	14200
	IF_UP 3	L_UN	14300	14300	14300
	I0	L_SUB	15000	15010	15010
		L_SUB	15000	15020	15020
		L_SUB	15000	15030	15030
		L_SUB	15000	15040	15040
	I1_N	L_SUB	16000	16010	16010
		L_SUB	16000	16020	16020
		L_SUB	16000	16030	16030
		L_SUB	16000	16040	16040
	I1_S 1	L_UN	17100	17100	17100
	I1_S 2	L_UN	17200	17200	17200
	I1_S 3	L_UN	17300	17300	17300
	I1_S 4	L_UN	17400	17400	17400
	S3_Central (Mineralized Wacke)	Z_UN	18000	18000	18000
	Envelope	L_UN	20000	20000	20000

14.1.1.8 Grade Block Model

The geostatistical results summarized in this section provided the parameters to interpolate a grade model using the 1.5 m composites from the capped grade data in order to produce the best possible grade estimate for the defined resources in the Hardrock deposit. The interpolation was run on a point area workspace extracted from the combined DDH and channel sample dataset.

The interpolation profiles were customized to estimate grades separately for each of the mineralized zones or lithological domains and the envelope for the DDH and channel sample composite populations. The ID^3 method was selected for the final resource estimation for all zones.

The composite points were assigned rock codes and block codes corresponding to the mineralized zone or mineralized subunit in which they occur. Hard, semi-soft or soft boundaries were applied as described in Subsection 14.1.1.5. The search/interpolation ellipse orientations and ranges defined in the interpolation profiles used for the grade estimation correspond to those developed in Subsection 14.1.1.5 (Table 14.6). Other specifications to control grade estimation are as follows:

- Pass 1
  - Minimum of seven and maximum of 15 sample points in the search ellipse for interpolation;
  - Maximum of three sample points from any one DDH;
  - Minimum of three drill holes for interpolation.
- Pass 2
  - Minimum of four and maximum of 15 sample points in the search ellipse for interpolation;
  - Maximum of three sample points from any one DDH;
  - Minimum of two drill holes for interpolation.
- Pass 3
  - Minimum of three and maximum of 15 sample points in the search ellipse for interpolation;
  - Maximum of three sample points from any one DDH;
  - Minimum of one drill hole for interpolation.

Metallurgical recoveries are impacted by the gold head grade (Head Grade g Au/t), sulfur (%S) and arsenic (%As) levels. The datasets for As and S as presented in Table 14.3 are not sampled to the same extent as the gold population. Therefore, the selected approach was to use the median value for each mineralized zone or lithological domain as defined in Table 14.3 to populate the block models.

The estimation of gold block grades is illustrated on a plan view and a cross section on Figure 14.10 and Figure 14.11.

Figure 14.10: Plan View 125 m showing Estimated Block Grades of the Hardrock Deposit

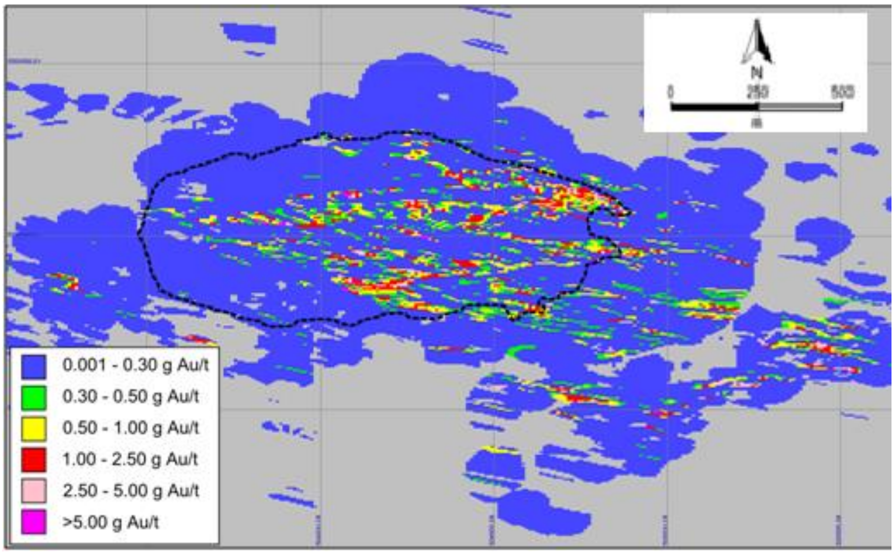
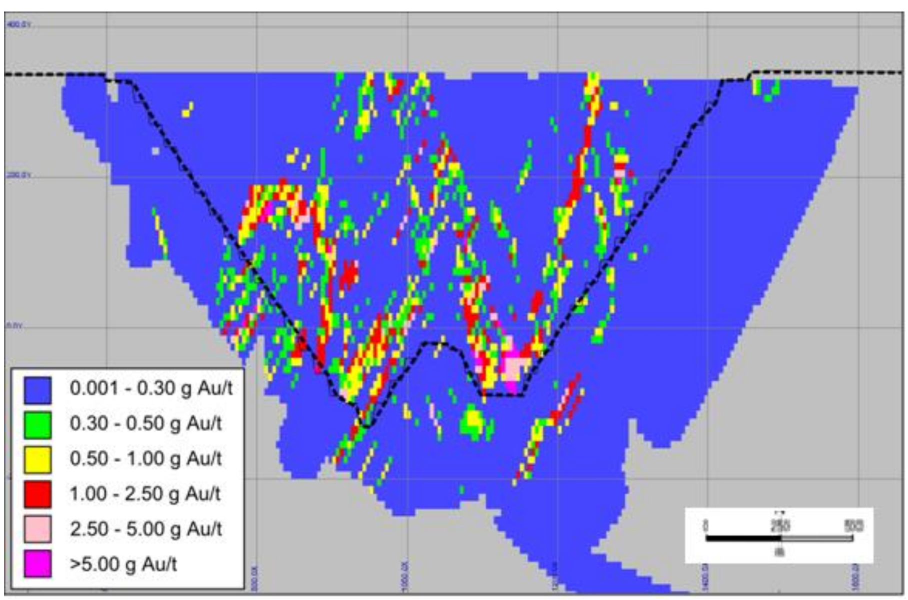


Figure 14.11: Cross Section Cross section 4325E (looking west) showing Estimated Block Grades of the Hardrock Deposit



14.1.1.9 Block Model Validation

The 2016 block model was validated throughout the process, including several steps, as presented in Table 14.11.

Table 14.11 Hardrock Block Model Validation Process

Process	Description
Database	Validation of the assays against laboratory certificates, collar surveys, deviation surveys
Database	Review of the density dataset and statistics
Voids	Review of the wireframes, rock codes and density
Voids	Validation of wireframes against DDH breakthroughs
Lithological Model	Validation of wireframes against DDH lithologies (snapped) and correspondence to the lithological description
Mineralized-Zone Model	Review of the mineralogy and composition of the different mineralized zones
Mineralized-Zone Model	Validation of the interpreted wireframes
Interpolation	Review of the interpolation parameters
Interpolation	Review of the search ellipsoid parameters (size, orientation, search restriction) against variography
Block Model	Review of the model size and orientation against the drilling density, mineralized-zone model and mining method
Block Model	Review of the block size taking into account mining units, drill hole spacing and dimensions of mineralized zones
Block Model	Validation of rock types, % and density codes
Block Model	Visual inspection in plan and section views in comparison to drill hole grades
Block Model	Comparison of the basic statistics of assay, composite and block at zero cut-off
Block Model	Comparison of the grades between the block model, composites and raw assays inside the Pit Design shell
Block Model	Visual validation of Mineral Resource classification coding
Block Model	Visual validation of grade banding, smearing of high grades, etc. on section and/or plan views
Block Model	Evaluation of sensitivity to estimation parameter changes

Summary Validation Results

Visual comparisons of block grades and composites in cross section and plan view provide a good correlation. Some high-grade blocks attributed to breakthrough drill holes ending in underground workings were observed locally. Removing intercepts ending in breakthroughs would remove valuable information given that, in most cases, the block model closely honours the grade distribution of the surrounding underground drill holes (which were not used in the Mineral Resource estimate). GMS therefore considers that the breakthrough drill holes do not add a bias to the model and have minimal impact on the 2016 MRE. Table 14.12 compares the mean block (Indicated Mineral Resource category only) and composite grades for the mineralized zones at a zero cut-off. The ratio of composite mean grade over block mean grade varies from 83 to 128%.

Table 14.12: Comparison of the Block and Composite Mean Grades at a Zero Cut-off for the Mineralized Zones

		Composites		Block Model (Indicated)		
Zone/ Lithology	**Block Code**	**Number**	**Mean (g Au/t)**	**Number**	**Mean (g Au/t)**	**Ratio Composites/ Blocks**
HG F-Zone	3102	1,708	2.42	5,426	2.08	116%
F-Zone	3105	6,046	0.74	16,395	0.67	110%
HG North 1 Zone	3202	1,760	2.30	6,746	2.78	83%
North 1 Zone	3205	13,067	0.64	48,209	0.66	97%
Central Zone	3300	2,910	0.75	11,522	0.71	106%
HG F2-Zone	3401	726	1.42	2,379	1.38	103%
F2-Zone	3405	4,537	0.50	17,884	0.59	85%
SP-Zone	3500	8,833	1.02	19,700	1.15	89%
North 2 Zone	3600	2,379	1.49	8,020	1.31	114%
North 3 Zone	3700	873	1.10	3,063	1.16	95%
Lower Zone	3800	583	0.85	644	0.74	115%
A-Zone	3900	1,288	0.72	6,341	0.61	118%
Tenacity Zone	4000	2,107	0.82	9,087	0.77	106%
SP2-Zone	4100	421	1.73	1,646	1.35	128%
		47,238	**0.93**	**157,062**	**0.92**	**101%**

Swath plots were generated to assess the correlation between composites used in the interpolation of each block versus the total gold content estimated. Swath plots were produced by vertical section and bench level. This validation method works as a visual means to identify possible bias in the interpolation (i.e., a section with significantly high gold content based on a low population of composites). In general, gold contained in each vertical section should correlate well with the amount of composites used in the interpolation. Figure 14.12 and Figure 14.13 illustrate swath plots for Indicated Mineral Resources by vertical sections and bench level respectively. Gold content is constrained by a Pit Design shell and cut-off grades were applied accordingly. Peaks and lows in gold content generally match peaks and lows in composite frequency; no bias was found in the resource estimate in this regard.

Figure 14.12: Swath Plot of Indicated Mineral Resources by North-South Vertical Sections

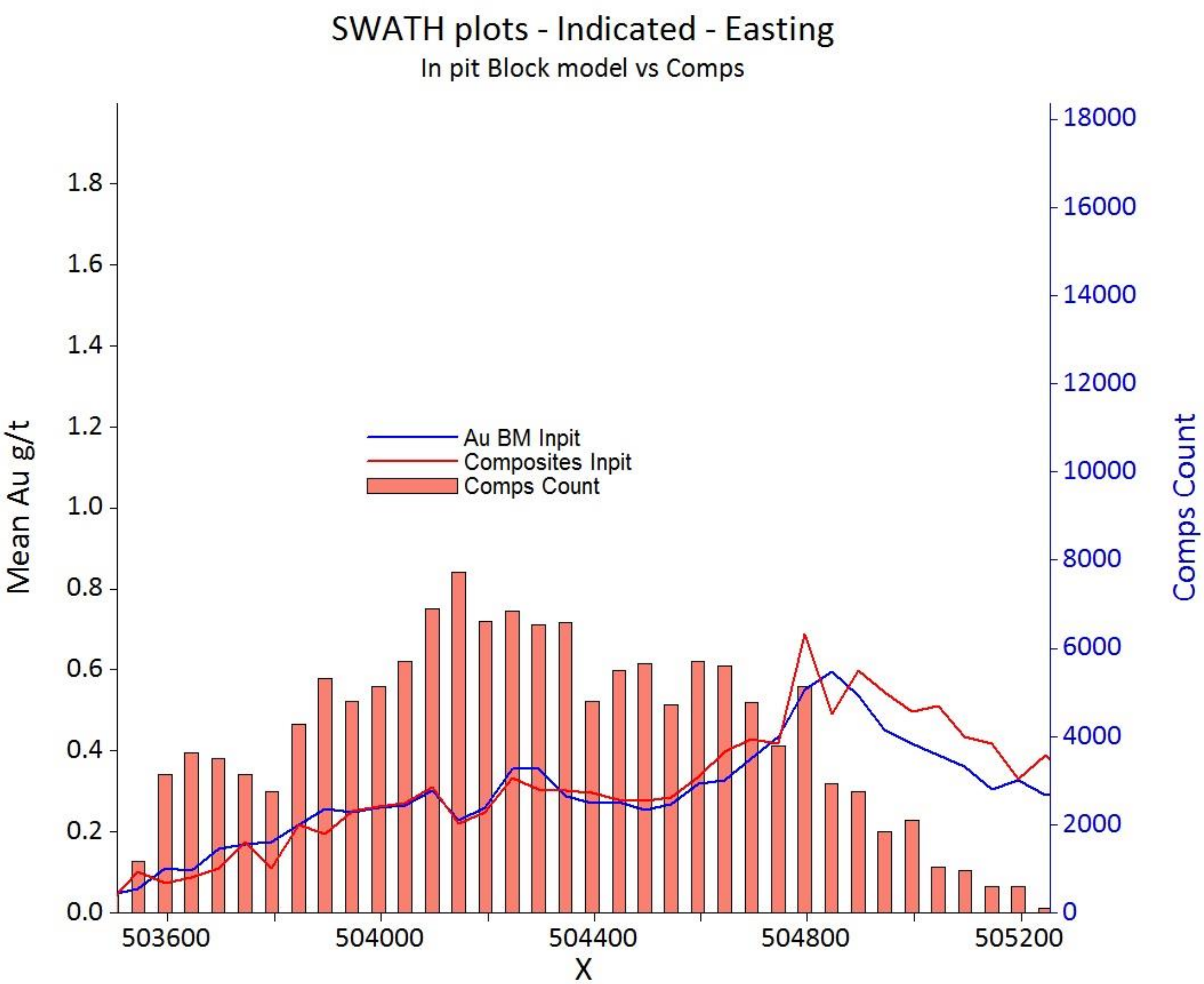
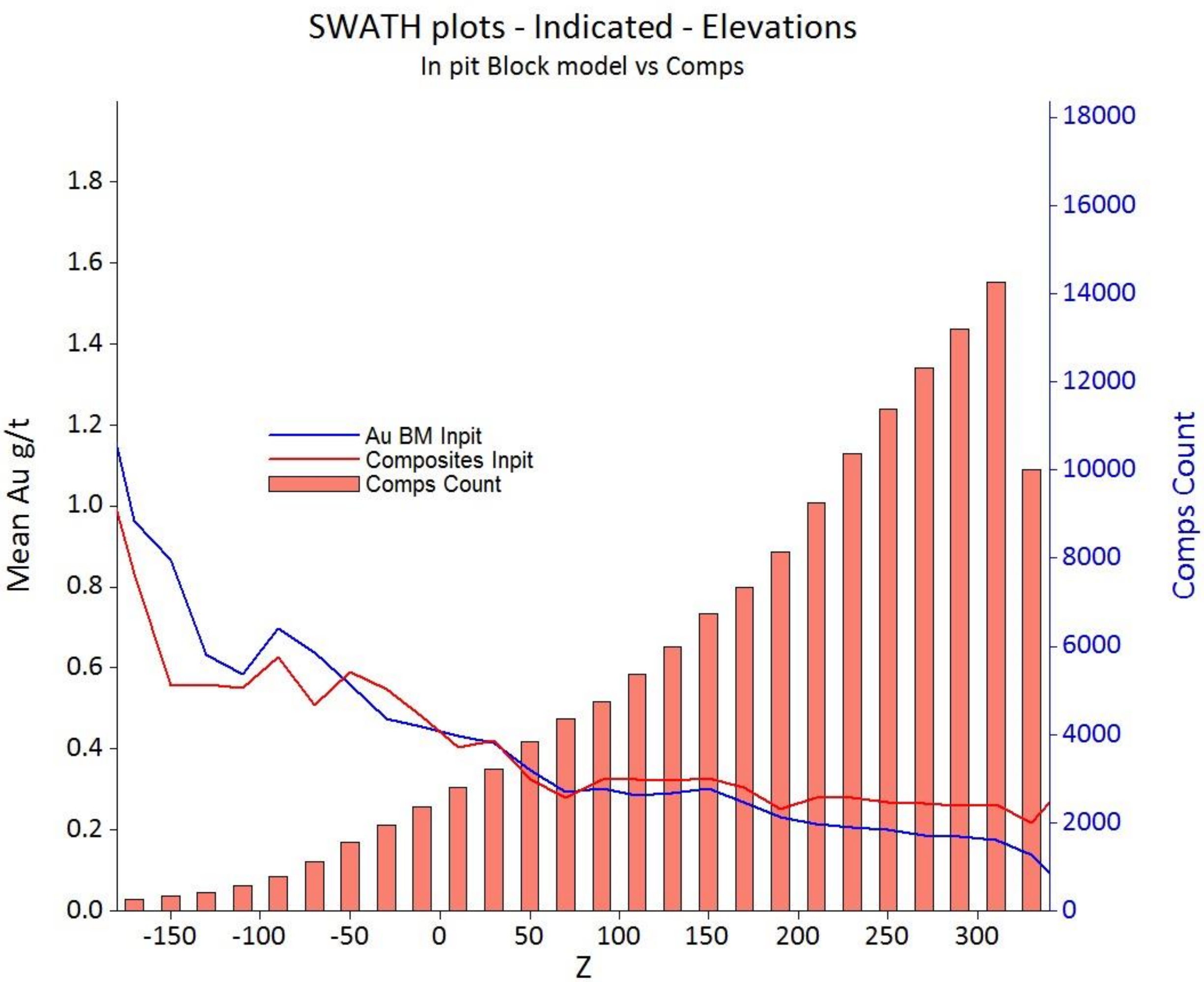


Figure 14.13: Swath Plot of Indicated Mineral Resources by Bench Levels



14.1.1.10 <u>Resource Category Block Model</u>

By default, interpolated blocks were assigned to the Inferred Mineral Resource category during the creation of the grade block model. The re-classification to an Indicated Mineral Resource category was done for any blocks meeting all the conditions below:

- Blocks interpolated with Pass 1 or Pass 2;
- Blocks interpolated with a minimum of two drill holes;
- Blocks for which the distance to the closest composite is less than 35 m.

A series of outline rings were created in plan view using the criteria described above. Indicated Mineral Resource category solids (or shell surfaces) were created from these rings, and the blocks were re-coded accordingly. Within this Indicated Mineral Resource category solid, some Inferred blocks have been upgraded to the Indicated category, whereas outside this solid, some Indicated blocks have been downgraded to the Inferred category. GMS is of the opinion that this step was necessary to homogenize (smooth out) the resource volumes in each category. A series of isolated blocks were also downgraded from the Inferred category to "exploration potential" on a visual basis, and are therefore excluded from the 2016 MRE. Rock codes from Domains 18000 and 20000 outside the Pit Design shell were systematically downgraded to the Inferred category (underground resources).

Figure 14.14 to Figure 14.16 show the Mineral Resource classification, as well as the Pit Design shell delimiting the in-pit and underground Mineral Resources.

Figure 14.14: Plan View showing the Categorized Mineral Resources and the Pit Design Shell Trace (elevation 300 m)

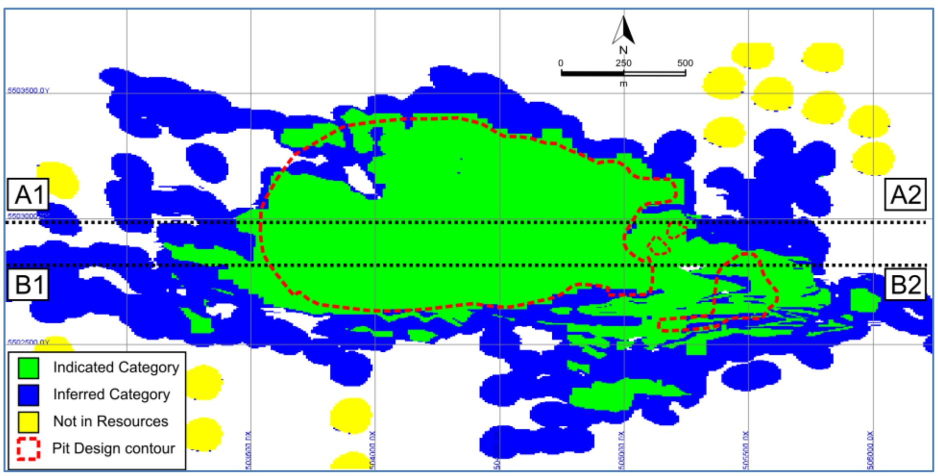


Figure 14.15: Longitudinal View showing the Categorized Mineral Resources and the Pit Design Shell Trace (Longitudinal view 5,502,987.5N)

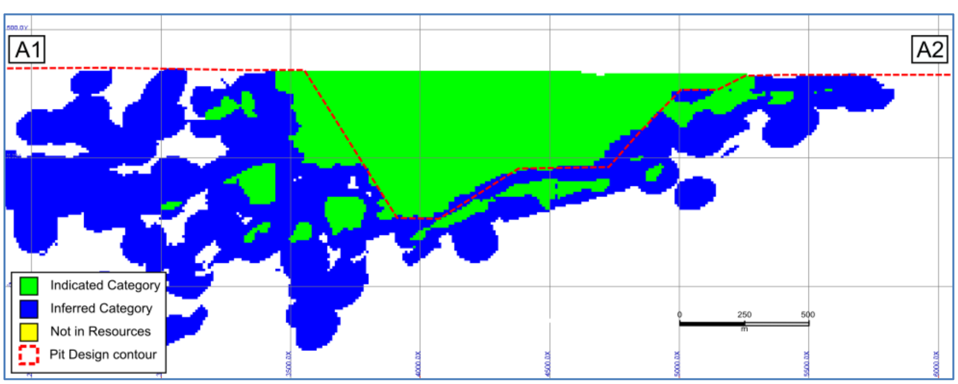
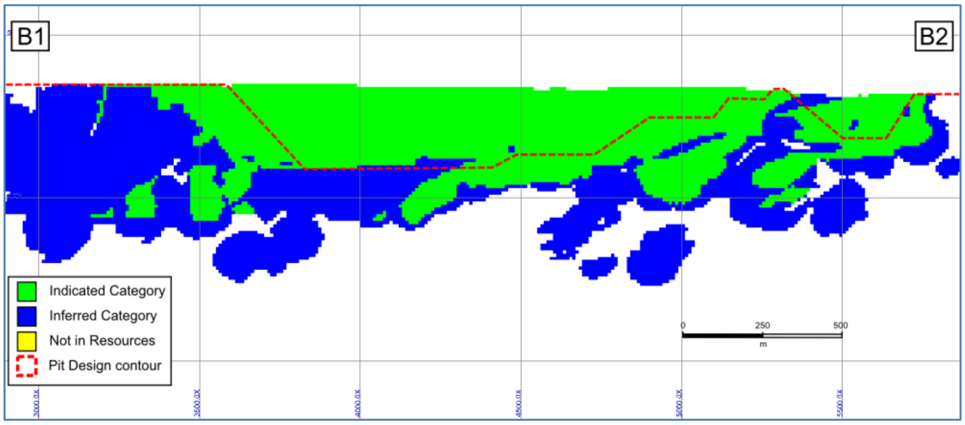


Figure 14.16: Longitudinal View showing the Categorized Mineral Resources and the Pit Design Shell Trace (Longitudinal View 5,502,762.5N)



14.1.2 <u>Mineral Resource Classification, Category and Definition</u>

The resource classification definitions used for this Report are those published by the Canadian Institute of Mining, Metallurgy and Petroleum in their document "CIM Definition Standards for Mineral Resources and Reserves" (2014).

Measured Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough to confirm both geological and grade continuity.

Indicated Mineral Resource: that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Inferred Mineral Resource: that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drill holes.

14.1.3 Estimation

Given the density of the processed data, the search ellipse criteria, and the specific interpolation parameters, GMS is of the opinion that the 2016 Hardrock in-pit MRE can be classified as Indicated Mineral Resources and Inferred Mineral Resources. The estimate is compliant with CIM standards and guidelines for reporting Mineral Resources and Mineral Reserves.

The in-pit resources were estimated using different gold cut-off grades. The selected cut-off grade of 0.30 g Au/t allowed the mineral potential of the deposit to be outlined for the in-pit mining option.

Volumetrics for the in-pit resource estimate have been constrained using the topography as the top surface and the Pit Design as the bottom surface. GMS used the Pit Design shell instead of the Whittle Shell to be able to declare Mineral Resources inclusive and exclusive of the Mineral Reserves. It would have been impossible to produce the resource breakdown between the Indicated Mineral Resource used for the Mineral Reserves (inclusive) and the remaining type of resources since the two shells are very close to each other and occasionally the Pit Design shell is larger than the Whittle Shell due to shell selections, ramp system and other considerations.

The pit parameters are presented in Table 14.13.

Table 14.13: Hardrock Pit Optimization Parameters

Hardrock Pit Optimization Parameters		
Nominal Milling Rate	t/d	27,000
Plant Throughput	kt/yr	9,855
Exchange Rate	CAD/USD	1.30
Diesel Fuel Price Delivered	CAD/l	0.80
Natural Gas Price	CAD/GJ	4.95
Electricity Cost	CAD/kWh	0.055
Gold Price	USD/oz	1250
Gold Price (local currency)	CAD/oz	1625
Transport and Refining Cost	CAD/oz	4.00
Royalty Rate	%	3.0%
Metallurgical Recovery at Cut-Off Grade	%	90%
Total Processing Cost	CAD/t milled	7.46
Re-handling	CAD/t milled	0.12
General and Administration	CAD/t milled	1.42
Rehabilitation and Closure	CAD/t milled	-
Sustaining Capital	CAD/t milled	0.60
Total Ore-based Cost	CAD/t milled	**9.60**
Marginal Cut-Off Grade	g Au/t	0.24
Mining Rate	kt/y	56,000
Mining Dilution	%	14.0%
Mining Loss	%	3.0%
Total Mining Reference Cost	CAD/t mined	**1.80**
Incr. Bench Cost (CAD /10 m bench)	CAD/10 m bench	0.030
Overall Slope Angle in Fresh Rock	degrees	55
Overall Slope Angle in Overburden	degrees	26

The shell selection for the various cases is presented in Section 15 in Table 15.6. Table 14.14 displays the results of the 2016 Mineral Resource Estimate for the in-situ in-pit portion of the Hardrock deposit.

Table 14.14: 2016 In-pit (Inclusive) Mineral Resource Estimate at Different Cut-off Grades - Hardrock Deposit

Indicated Resource				
Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz
All Zones	> 0.90	46,092,000	2.19	3,241,700
	> 0.80	52,699,000	2.02	3,421,800
	> 0.70	61,244,000	1.84	3,627,500
	> 0.60	71,763,000	1.67	3,846,500
	> 0.50	85,580,000	1.49	4,089,900
	> 0.40	104,577,000	1.30	4,363,200
	> 0.30	131,870,000	1.10	4,667,300
	> 0.20	175,249,000	0.89	5,010,200

Inferred Resource				
Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz
All Zones	> 0.90	51,000	1.79	2,900
	> 0.80	55,000	1.73	3,100
	> 0.70	64,000	1.58	3,300
	> 0.60	75,000	1.45	3,500
	> 0.50	99,000	1.23	3,900
	> 0.40	122,000	1.08	4,200
	> 0.30	170,000	0.87	4,800
	> 0.20	259,000	0.66	5,500

Notes:

- *The Mineral Resources are inclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng. from GMS;*
- *The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *In-pit results are presented undiluted within the Pit Design shell, designed with a 30 m buffer around lakes;*
- *The estimate includes 25 zones and a remaining undifferentiated envelope;*
- *In-pit resources were compiled at cut-off grades of 0.20, 0.30, 0.40, 0.50, 0.60, 0.70, 0.80 and 0.90 g Au/t. The official in-pit resource is reported at a cut-off grade of 0.30 g Au/t;*
- *Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost);*
- *Density (g/cm^3) data was established on a per zone basis and ranges from 2.72 to 3.26 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High-grade capping (g Au/t) on raw assay data was established on a per zone basis and ranges from 15 to 45 g Au/t*
- *Compositing was done on drill hole sections falling within the mineralized zones (composite = 1.5 m);*
- *Resources were estimated using GEOVIA GEMS 6.7 software from drill hole and surface channel sampling, using a 3-pass ID^3 interpolation method in a block model (block size = 10 m x 5 m x 10 m);*
- *The Inferred category is only defined within areas where blocks were interpolated during Pass 1 to Pass 3, and isolated blocks were reclassified as “exploration potential” on a visual basis;*
- *The Indicated category is only defined in areas where the maximum distance to drill hole composites is less than 35 m for blocks interpolated in Pass 1 and Pass 2 (using a minimum of 2 drill holes);*
- *Ounce (troy) = metric tons x grade / 31.10348. Calculations used metric units (metres, tonnes, g/t);*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate;*
- *Numbers may not add due to rounding.*

14.1.4 Underground Mineral Resource Estimation

Given the density of the processed data, the search ellipse criteria, and the specific interpolation parameters, GMS is of the opinion that the 2016 underground MRE can be classified as Indicated Mineral Resources and Inferred Mineral Resources. The estimate is compliant with CIM standards and guidelines for reporting Mineral Resources and Mineral Reserves.

The underground Mineral Resources were reported using a different gold cut-off grade. The gold selling and processing costs, mining dilution, and processing and mining recoveries were provided by GGM and validated by GMS. The selected underground cut-off grade of 2.0 g Au/t allowed the mineral potential of the deposit to be outlined for the underground mining option, outside the Pit Design shell. The estimation of the underground cut-off grade was based on the parameters presented in Table 14.15.

Table 14.15: Input Parameters used for the Underground Cut-off Grade Estimation - Hardrock Deposit

Input Parameter	Value
Exchange Rate (USD/CAD)	USD 1.00/CAD 1.30
Gold Price (CAD/oz)	1,625
Gold Selling Costs (CAD/oz)	4.00
Royalty (%)	3
Net Gold Price (CAD/oz)	1,572
Mining Costs (CAD/t)	55.54
G&A Costs (CAD/t)	7.00
Sustaining Costs (CAD/t)	10.00
Milling Costs (CAD/t)	7.46
Total Costs	80.00
Processing Recovery (%)	90
Mining Dilution (%)	20
Dilution Grade (g Au/t)	0.5
Marginal Cut-Off Grade (g Au/t)	2.01

The underground MRE presented herein uses a rounded value of 2.00 g Au/t for the underground cut-off grade.

A volumetric analysis of the underground resource estimate was carried out using an attribute for the blocks coded outside the Pit Design shell in order to calculate the volume of any mineralized zone or envelope zone material contained within the bedrock but extending beyond the pit boundaries.

Table 14.16 displays the results of the MRE for the in situ[1] underground portion of the Hardrock deposit.

Table 14.16: Underground (Inclusive) Mineral Resource Estimate at Different Cut-off Grades - Hardrock Deposit

Indicated Resource					Inferred Resource				
Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz	Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz
All Zones	> 4.50	3,362,000	7.11	768,400	All Zones	> 4.50	3,799,000	6.93	843,700
	> 4.00	4,301,000	6.48	896,500		> 4.00	5,247,000	6.19	1,043,700
	> 3.50	5,495,000	5.89	1,040,000		> 3.50	7,177,000	5.53	1,276,800
	> 3.00	7,139,000	5.28	1,211,300		> 3.00	10,089,000	4.87	1,579,900
	> 2.50	9,556,000	4.63	1,423,300		> 2.50	14,226,000	4.25	1,945,100
	> 2.00	13,692,000	3.91	1,719,900		**> 2.00**	21,507,000	3.57	2,470,400
	> 1.50	21,081,000	3.14	2,128,700		> 1.50	33,245,000	2.92	3,120,100

Notes:

- *The Mineral Resources are inclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 underground MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng. from GMS;*
- *The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *Underground results are presented undiluted outside the Pit Design shell;*
- *The estimate includes 25 zones and a remaining undifferentiated envelope;*
- *Underground resources were compiled at cut-off grades of 1.50, 2.00, 2.50, 3.00, 3.50, 4.00 and 4.50 g Au/t. The official Underground resource is reported at a cut-off grade of 2.00 g Au/t;*
- *Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost);*
- *Density (g/cm3) data was established on a per zone basis and ranges from 2.72 to 3.26 g/cm3;*
- *A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High-grade capping (g Au/t) on raw assay data was established on a per zone basis and ranges from 15 to 45 g Au/t;*
- *Compositing was done on drill hole sections falling within the mineralized zones (composite = 1.5 m);*
- *Resources were estimated using GEOVIA GEMS 6.7 software from drill hole and surface channel sampling, using a 3-pass ID3 interpolation method in a block model (block size = 10 m x 5 m x 10 m);*
- *The Inferred category is only defined within areas where blocks were interpolated during Pass 1 to Pass 3, and isolated blocks were reclassified as "exploration potential" on a visual basis;*
- *The Indicated category is only defined in areas where the maximum distance to drill hole composites is less than 35 m for blocks interpolated in Pass 1 and Pass 2 (using a minimum of two drill holes);*
- *Ounce (troy) = metric tons x grade / 31.10348. Calculations used metric units (metres, tonnes, g/t);*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate;*
- *Underground parameters used (all amounts in Canadian dollars): Mining costs=CAD 55.54/t, Milling cost=CAD 7.46/t, Royalty=3%, G&A costs=CAD 7.00/t, Sustaining Capital and Surface costs=CAD 10.00/t, Gold price=CAD 1,625/oz, milling recovery=90%, Mining Dilution = 20%, Estimated dilution grade = 0.5 g Au/t;*
- *Numbers may not add due to rounding.*

[1] The term "in situ" is used to represent all the remaining Mineral Resources in place at the time of the 2016 estimate.

14.1.5 Summary of the Hardrock Mineral Resource Estimate

GMS has produced an updated MRE for the Hardrock deposit. The overall 2016 MRE presented in Table 14.17 includes:

- An in-pit resource estimate, within the Pit Design shell (Table 14.14);
- An underground resource estimate, outside the Pit Design shell (Table 14.16).

Table 14.17 presents the combined Mineral Resources by category for the Hardrock deposit.

Table 14.17: Mineral Resources Estimate Inclusive of Mineral Reserves for the Hardrock Project

Resource Type	Cut-off (g/t)	In-Pit	Underground	Total
		> 0.30 g Au/t	> 2.00 g Au/t	
Indicated	Tonnes (t)	131,870,000	13,692,000	145,562,000
	Grade (g Au/t)	1.10	3.91	1.36
	Au (oz)	4,667,300	1,719,900	6,387,200
Inferred	Tonnes (t)	170,000	21,507,000	21,677,000
	Grade (g Au/t)	0.87	3.57	3.55
	Au (oz)	4,800	2,470,400	2,475,200

Notes:

- *The Mineral Resources are inclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng., from GMS;*
- *The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *In-pit results are presented undiluted within the Pit Design shell, designed with a 30 m buffer around lakes;*
- *Underground results are presented undiluted outside the Pit Design shell;*
- *The estimate includes twenty-five zones and a remaining undifferentiated envelope;*
- *In-pit resources were compiled at cut-off grades of 0.20, 0.30, 0.40, 0.50, 0.60, 0.70, 0.80 and 0.90 g Au/t. The official in-pit resource is reported at a cut-off grade of 0.30 g Au/t;*
- *Underground resources were compiled at cut-off grades of 1.50, 2.00, 2.50, 3.00, 3.50, 4.00 and 4.50 g Au/t. The official Underground resource is reported at a cut-off grade of 2.00 g Au/t;*
- *Cut-off grades must be re-evaluated in light of prevailing market conditions (gold price, exchange rate and mining cost);*
- *Density (g/cm^3) data was established on a per zone basis and ranges from 2.72 to 3.26 g/cm^3;*
- *A minimum true thickness of 3.0 m was applied, using the grade of the adjacent material when assayed, or a value of zero when not assayed;*
- *High-grade capping (g Au/t) on raw assay data was established on a per zone basis and ranges from 15 to 45 g Au/t;*
- *Compositing was done on drill hole sections falling within the mineralized zones (composite = 1.5 m);*
- *Resources were estimated using GEOVIA GEMS 6.7 software from drill hole and surface channel sampling, using a 3-pass ID^3 interpolation method in a block model (block size = 10 m x 5 m x 10 m);*
- *The Inferred category is only defined within areas where blocks were interpolated during Pass 1 to Pass 3, and isolated blocks were reclassified as "exploration potential" on a visual basis;*
- *The Indicated category is only defined in areas where the maximum distance to drill hole composites is less than 35 m for blocks interpolated in Pass 1 and Pass 2 (using a minimum of two drill holes);*
- *Ounce (troy) = metric tons x grade / 31.10348. Calculations used metric units (metres, tonnes, g/t);*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *GMS is not aware of any known environmental, permitting, legal, title-related, taxation, socio-political, marketing or other relevant issue that could materially affect the Mineral Resource estimate;*
- *Underground parameters used (all amounts in Canadian dollars): Mining costs=CAD 55.54/t, Milling cost=CAD 7.46/t, Royalty=3%, G&A costs=CAD 7.00/t, Sustaining Capital and Surface costs=CAD 10.00/t, Gold price=CAD 1,625/oz, Milling Recovery=90%, Mining Dilution = 20%, Estimated dilution grade = 0.5 g Au/t;*
- *Numbers may not add due to rounding.*

14.1.6 Comparison with Previous Estimate

The overall 2016 Indicated Mineral Resources represent a 31% increase in ounces compared to the 2014 MRE presented in Table 14.18 (at their respective cut-offs). The 2016 Inferred Mineral Resources represent a 10% decrease in total ounces compared to the 2014 MRE (at their respective cut-offs).

Table 14.18 presents the 2014 MRE by category for the Hardrock deposit.

Table 14.18: 2014 Mineral Resources Estimate (Indicated and Inferred Mineral Resources) for the Hardrock Project

Resource Type	Cut-off (g Au/t)	In-Pit	Underground	Total
		> 0.50 g Au/t	> 3.00 g Au/t	
Indicated	Tonnes (t)	83,867,800	5,169,300	89,037,100
	Grade (g Au/t)	1.47	5.40	1.70
	Au (oz)	3,972,500	897,800	4,870,300
Inferred	Tonnes (t)	10,225,000	12,921,700	23,146,700
	Grade (g Au/t)	1.53	5.40	3.69
	Au (oz)	501,300	2,242,300	2,743,600

The major difference with the previous estimate is the change in the selected cut-off grades. In 2016, a lower cut-off grade was selected for the in-pit (2016: 0.3 g Au/t vs. 2014: 0.5 g Au/t) and underground (2016: 2.0 g Au/t vs. 2014: 3.0 g Au/t) MRE based on revised economic parameters for both scenarios (see details in Subsections 14.1.3 and 14.1.4, respectively).

Using the same 2016 cut-off grades for both estimates, as presented in Table 14.19 and Table 14.20, the change from 2014 to 2016 can be broken down as follows:

In-pit (cut-off of 0.3 g Au/t):

- Indicated Mineral Resources represent a 2% increase in ounces;
- Inferred Mineral Resources represent a 99% decrease in ounces.

Underground (cut-off of 2.0 g Au/t):

- Indicated Mineral Resources represent a 34% increase in ounces;
- Inferred Mineral Resources represent a 24% decrease in ounces

Table 14.19: Comparison of the In-Pit Mineral Resources Estimate (Indicated and Inferred Mineral Resources) between 2014 and 2016 for the Hardrock Project

Resource Type	Cut-off (g Au/t)	In-Pit		Variation
		2014	2016	
		> 0.3 g Au/t	> 0.3 g Au/t	
Indicated	Au (koz)	4,568	4,667	+ 99 (+2%)
Inferred	Au (koz)	562	5	-557 (-99%)

Table 14.20: Comparison of the Underground Mineral Resources Estimate (Indicated and Inferred Mineral Resources) between 2014 and 2016 for the Hardrock Project

Resource Type	Cut-off (g Au/t)	Underground		Variation
		2014	2016	
		> 2.0 g Au/t	> 2.0 g Au/t	
Indicated	Au (koz)	1,285	1,720	+ 435 (+34%)
Inferred	Au (koz)	3,263	2,470	-793 (-24%)

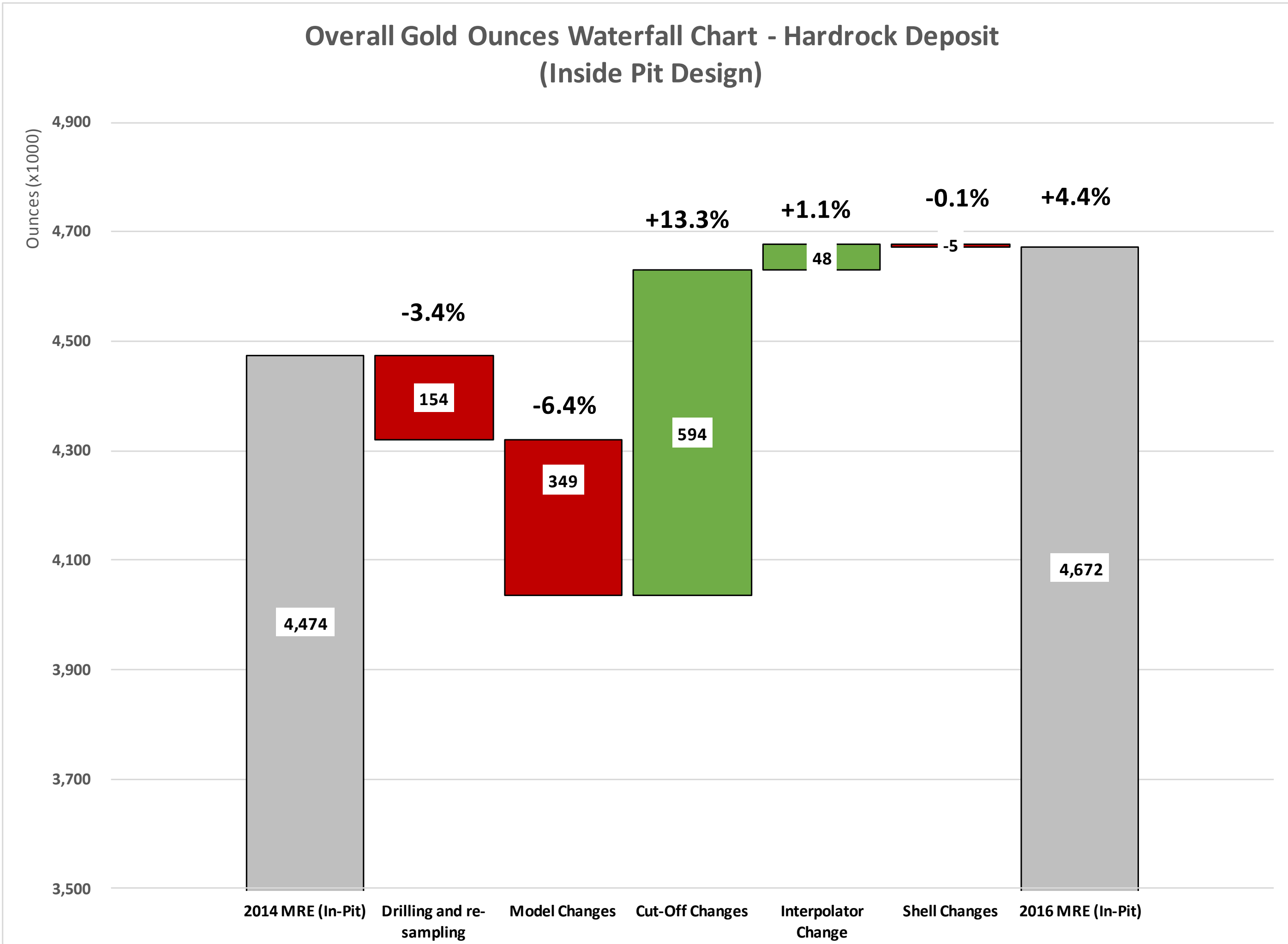
Figure 14.17 and Figure 14.18 show waterfall charts of the uncategorized in-pit and underground Mineral Resources, respectively. These waterfall charts are provided for visual support only, and the ratios presented should not be relied upon since they are approximate. These graphs show the impact, positive or negative, of the major changes to the block model from the previous 2014 MRE (left column) to the 2016 MRE (right column). The columns represent new information, a single modified parameter or group of parameters, or an additional restriction since the 2014 MRE.

It can be seen that the new drilling and re-sampling program had a small negative impact on the update of the 2016 MRE, which is common when drill holes are designed for infill and ungrade of resource category.

The effort to constrain high grades in the 2016 model, which led to a new mineralized model and revised capping levels, resulted in a considerably negative impact. It is particularly significant for the underground Mineral Resources where the most constraints were applied, compared to 2014. In the new mineralized model and revised capping levels, all the modifications to the interpolation strategy were also included, which simultaneously improved the level of local accuracy and the visual match between block grades and drill hole assays. The additional high grade restriction applied to the search ellipsoids had a very low negative impact on both scenarios, suggesting that high grades are mostly controlled by the new mineralized zones and revised capping levels.

On the other hand, the positive impacts on the 2016 MRE are mainly attributed to the revised economic parameters, which led to lower cut-off grades, as discussed above.

Figure 14.17: In-pit Waterfall Chart of the Changes since Previous 2014 MRE



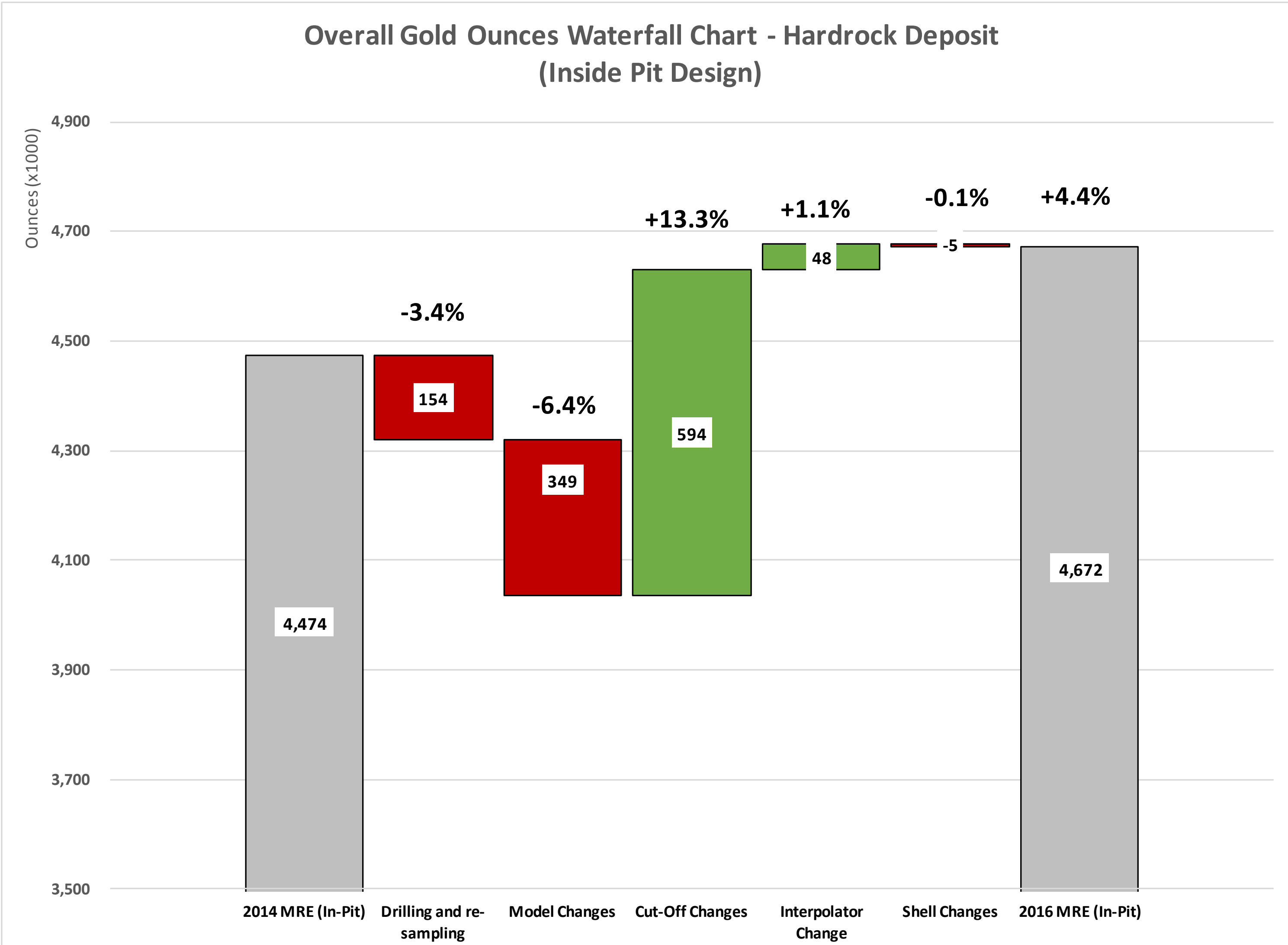
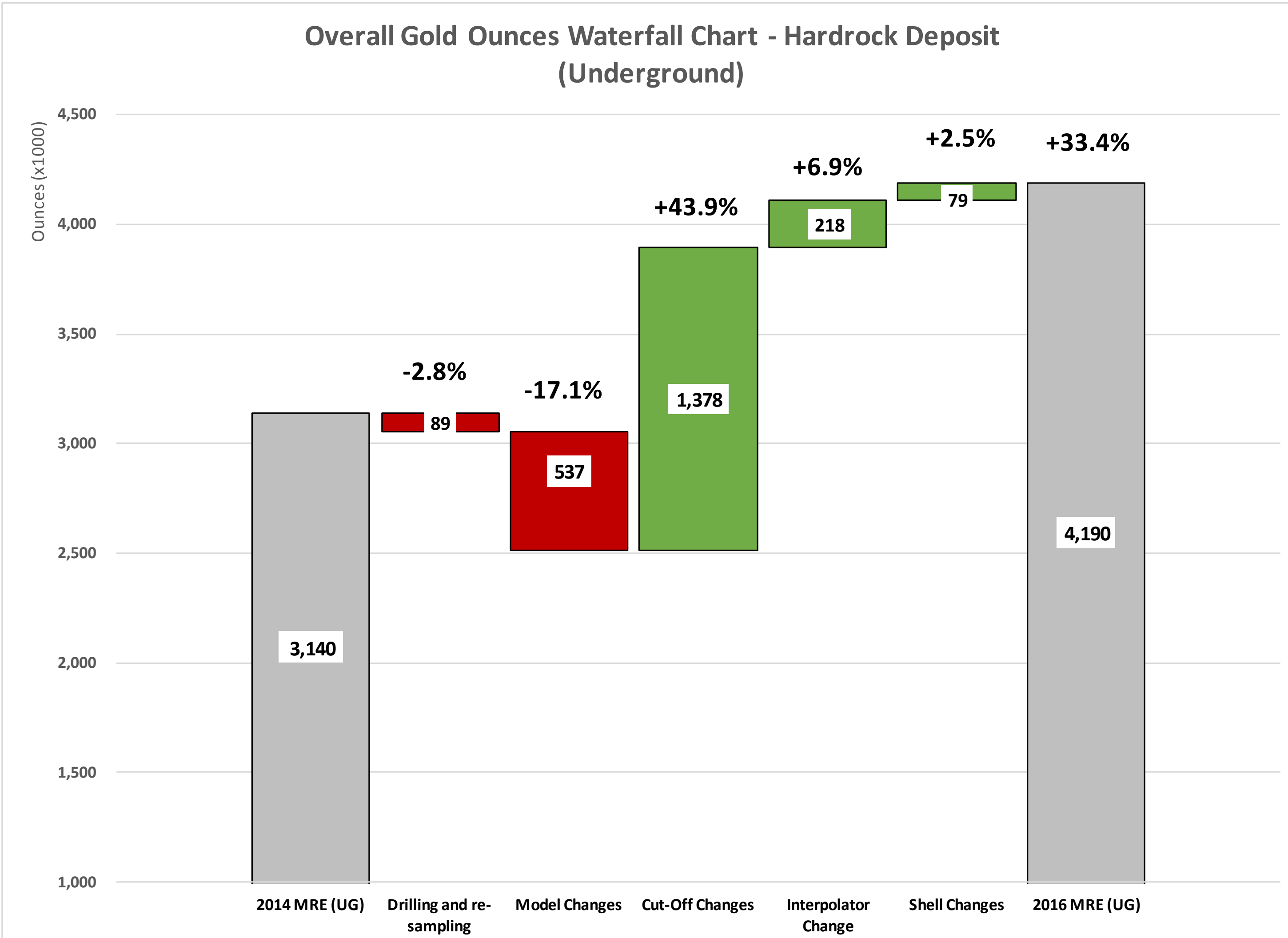
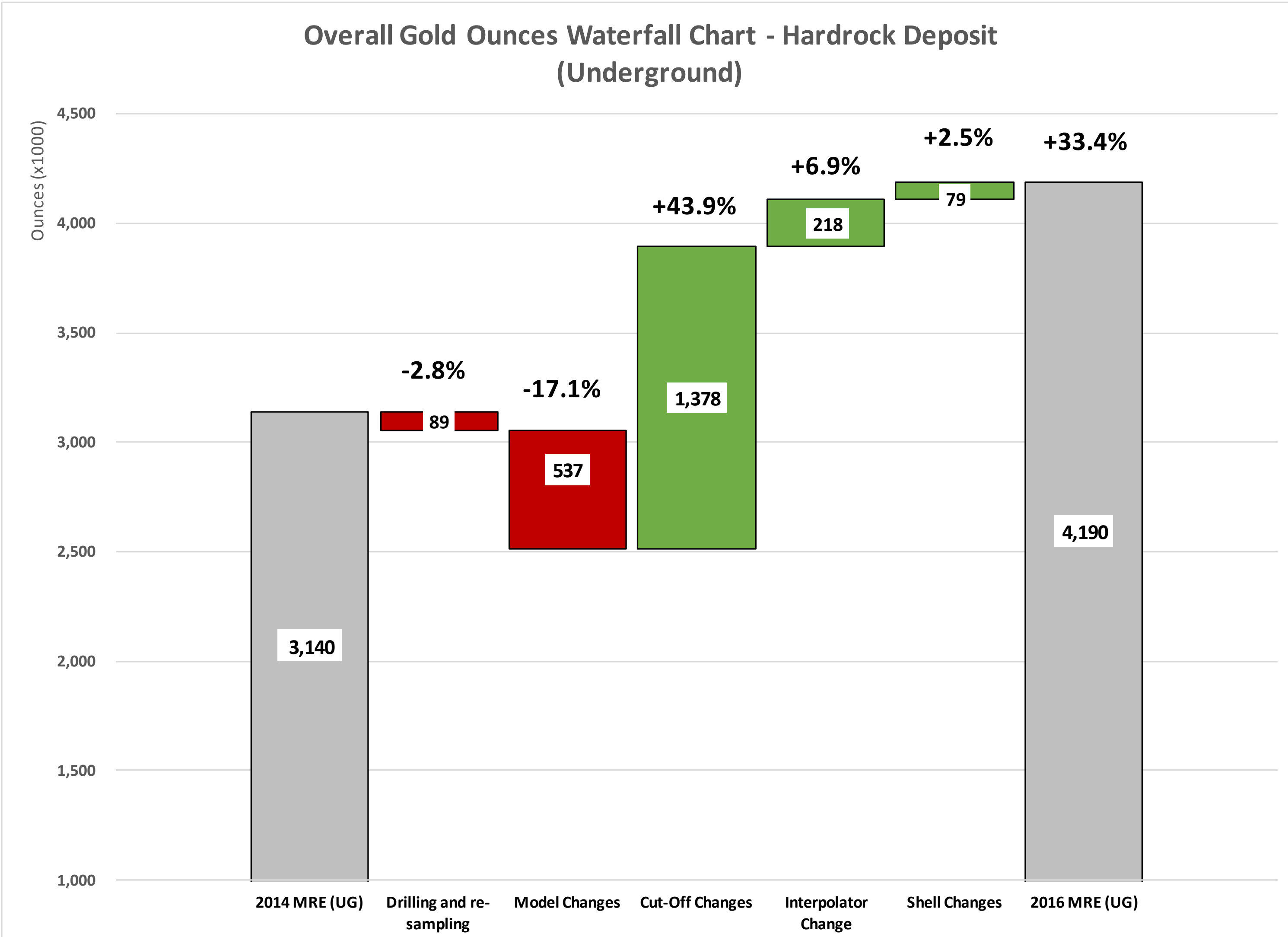

Overall Gold Ounces Waterfall Chart - Hardrock Deposit
(Inside Pit Design)
Ounces (x1000)
4,900
4,700
4,500
4,300
4,100
3,900
3,700
3,500
-3.4%
-6.4%
+13.3%
+1.1%
-0.1%
+4.4%
4,474
154
349
594
48
-5
4,672
2014 MRE (In-Pit)
Drilling and re-sampling
Model Changes
Cut-Off Changes
Interpolator Change
Shell Changes
2016 MRE (In-Pit)

Figure 14.18: Underground Waterfall Chart of the Changes since the Previous 2014 MRE




Overall Gold Ounces Waterfall Chart - Hardrock Deposit (Underground)
Ounces (x1000)
4,500
4,000
3,500
3,000
2,500
2,000
1,500
1,000
3,140
-2.8%
89
-17.1%
537
+43.9%
1,378
+6.9%
218
+2.5%
79
+33.4%
4,190
2014 MRE (UG)
Drilling and re-sampling
Model Changes
Cut-Off Changes
Interpolator Change
Shell Changes
2016 MRE (UG)

14.1.7 In-pit Mineral Resource Estimation (Exclusive of Mineral Reserves)

Premier and Centerra usually publish their Mineral Resources exclusive of the Mineral Reserves. The Mineral Resources could then be added to the Mineral Reserves since they are removed from the inclusive Mineral Resources.

Table 14.21 displays the results of the 2016 In- Situ Mineral Resources Estimate for the in-pit portion of the Hardrock deposit that are exclusive of the Mineral Reserves.

Table 14.21: 2016 In-pit Mineral Resources Estimate Exclusive of Mineral Reserves at Different Cut-off Grades - Hardrock Deposit

Indicated Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 0.90	158,000	2.04	10,300
	> 0.80	199,000	1.79	11,500
	> 0.70	244,000	1.60	12,500
	> 0.60	308,000	1.40	13,900
	> 0.50	470,000	1.11	16,700
	> 0.40	966,000	0.76	23,700
	> 0.30	11,444,000	0.36	131,200
	> 0.20	54,822,000	0.27	474,100

Inferred Resource				
Zone	**Cut-off g Au/t**	**Tonnes**	**Grade g Au/t**	**Au Oz**
All Zones	> 0.90	51,000	1.79	2,900
	> 0.80	55,000	1.73	3,100
	> 0.70	64,000	1.58	3,300
	> 0.60	75,000	1.45	3,500
	> 0.50	99,000	1.23	3,900
	> 0.40	122,000	1.08	4,200
	> 0.30	170,000	0.87	4,800
	> 0.20	259,000	0.66	5,500

Notes:

- *The Mineral Resources are exclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng. from GMS; The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *Numbers may not add due to rounding.*

14.1.8 Underground Mineral Resource Estimation (Exclusive of Mineral Reserves)

Table 14.22 displays the results of the In- Situ[2] Mineral Resources Estimate for the underground portion of the Hardrock deposit located outside of the Pit Design shell. Since no Mineral Reserves are estimated for the underground portion of the deposit, Table 14.22 is identical to Table 14.16.

Table 14.22: 2016 Underground Mineral Resources Estimate Exclusive of Mineral Reserves at Different Cut-off Grades - Hardrock Deposit

Indicated Resource					Inferred Resource				
Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz	Zone	Cut-off g Au/t	Tonnes	Grade g Au/t	Au Oz
All Zones	> 4.50	3,362,000	7.11	768,400	All Zones	> 4.50	3,787,000	6.93	843,700
	> 4.00	4,301,000	6.48	896,500		> 4.00	5,247,000	6.19	1,043,700
	> 3.50	5,495,000	5.89	1,040,000		> 3.50	7,177,000	5.53	1,276,800
	> 3.00	7,139,000	5.28	1,211,300		> 3.00	10,088,000	4.87	1,579,900
	> 2.50	9,556,000	4.63	1,423,300		> 2.50	14,226,000	4.25	1,945,100
	> 2.00	13,692,000	3.91	1,719,900		**> 2.00**	21,507,000	3.57	2,470,400
	> 1.50	21,081,000	3.14	2,128,700		> 1.50	33,245,000	2.92	3,120,100

Notes:

- *The Mineral Resources are exclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng. from GMS;*
- *The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *Numbers may not add due to rounding.*

14.1.9 Summary of the Hardrock Mineral Resource Estimate Exclusive of Mineral Reserves

GMS produced an updated Mineral Resource estimate exclusive of the Mineral Reserves for the Hardrock deposit. The overall 2016 MRE presented below includes:

[2] The term “in situ” is used to represent all remaining Mineral Resources in place at the time of the 2016 estimate.

- An in-pit resource estimate, within the Pit Design shell (Table 14.21);
- An underground resource estimate, outside the Pit Design shell (Table 14.22).

Table 14.23 presents the combined resources by category excluding the Indicated Mineral Resources used in Mineral Reserves for the Hardrock deposit.

Table 14.23: Mineral Resources Estimate Exclusive of Mineral Reserves for the Hardrock Project

Resource Type	Cut-off (g Au/t)	In-Pit	Underground	Total
		> 0.30 g Au/t	> 2.00 g Au/t	
Indicated	Tonnes (t)	11,444,000	13,692,000	25,136,000
	Grade (g/t)	0.36	3.91	2.29
	Au (oz)	131,200	1,719,900	1,851,100
Inferred	Tonnes (t)	170,000	21,507,000	21,677,000
	Grade (g/t)	0.87	3.57	3.55
	Au (oz)	4,800	2,470,400	2,475,200

Notes:

- *The Mineral Resources are exclusive of Mineral Reserves;*
- *The independent and qualified person for the 2016 MRE, as defined by NI 43-101, is Réjean Sirois, P.Eng. from GMS The effective date of the estimate is August 11, 2016;*
- *These Mineral Resources are not Mineral Reserves as they do not have demonstrated economic viability;*
- *The number of metric tons was rounded to the nearest thousand and ounces of gold to the nearest hundred. Any discrepancies in the totals are due to rounding effects; rounding followed the recommendations in NI 43-101;*
- *Numbers may not add due to rounding.*

14.2 Other Greenstone Gold Property Deposits

The Greenstone Gold Property (formerly the Trans-Canada Property) includes, in addition to Hardrock, the Brookbank, Key Lake and Kailey deposits. The Mineral Resources of these deposits have been estimated by Micon and are presented for completeness.

14.2.1 Brookbank

14.2.1.1 Database Description

The Brookbank deposit (main zone) has been tested by diamond drilling over a strike length of 1,150 m and down to a vertical depth of 650 m. The database consists of a total of 376 drill holes, of which the majority are concentrated in the central part of the deposit. The drill holes are on a grid varying from 25 m (close to surface) to 200 m at depth. The main components of the database are the collar, survey, assay and lithology tables which were validated as described in Subsection 12.3.5.

14.2.1.2 Topography

The landscape in the Brookbank deposit area is generally flat with occasional slight or barely noticeable undulations; therefore, a digital terrain model was not considered critical to the Mineral Resource estimate. The topographic surface was generated using drill hole collar elevations.

14.2.1.3 Specific Gravity

Specific gravity for the different lithologies was determined at Actlabs for several samples of each rock type. The average specific gravity value of 2.87 was used to calculate the tonnages.

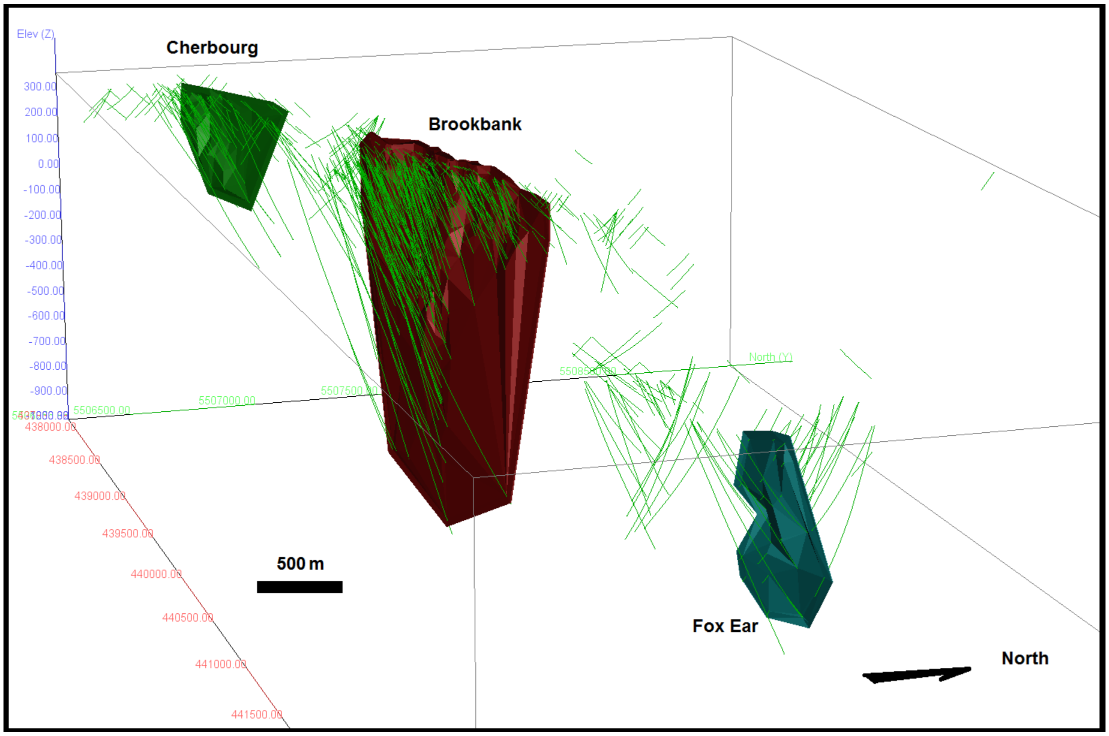
14.2.1.4 Estimation Methodology Overview

The Brookbank project Mineral Resource estimate was conducted using a systematic and logical approach involving geological interpretation, conventional statistical analysis on raw data, dynamic modelling, solid creation, statistical analysis on composite sample data and grade capping, geostatistical analysis, creation of interpolation parameters, block modelling, grade interpolation, block model validation, and finally, determination of the Mineral Resource and classification.

14.2.1.5 Geological Interpretation

The Brookbank gold deposits comprise the Brookbank Main, the Cherbourg and the Fox Ear deposits (Figure 14.19) that occur at three different localities within the 6.5 km long Brookbank shear zone. The deposits are located at and/or near the contacts between mafic volcanics and meta-sediments. Thus the deposits appear to have both structural and lithological controls. However, the fact that gold mineralization is not continuous along the entire shear zone suggests that the deposits are related to second order structures rather than the primary shear zone.

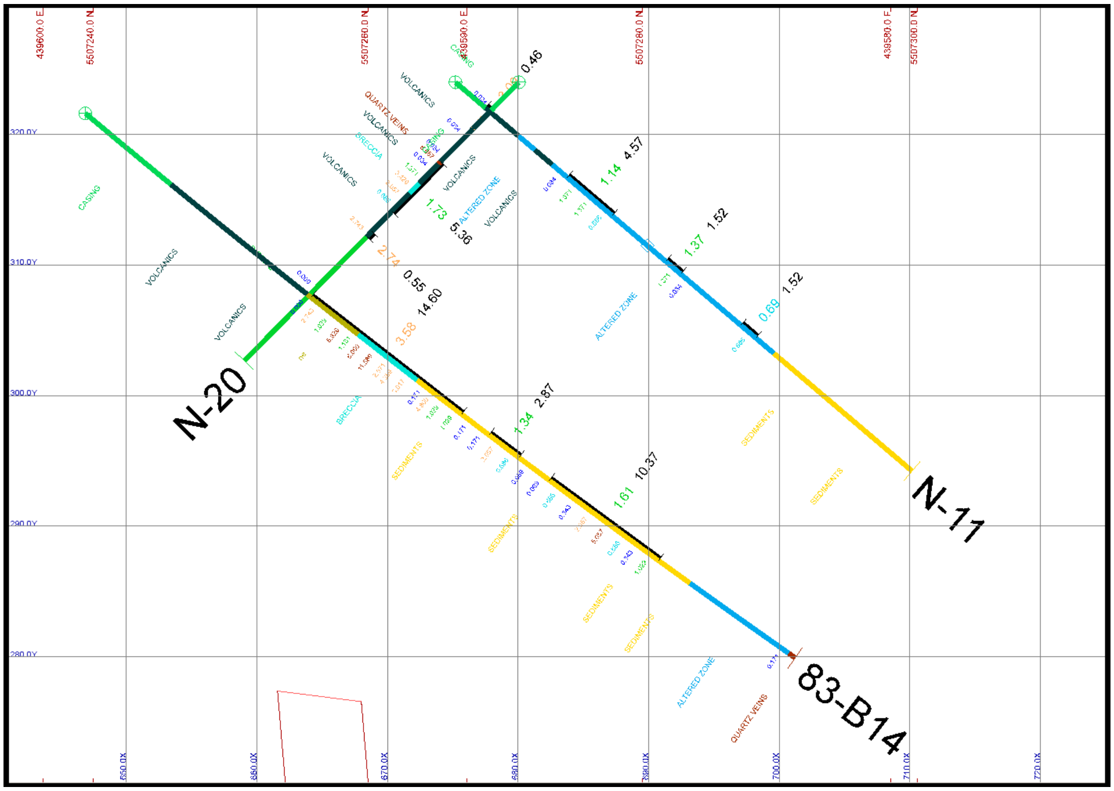
Figure 14.19: Modelled Solids of the Brookbank Deposits



Gold mineralization occurs within multiple quartz-carbonate stringers, veinlets and/or stockworks that give rise to broad zones of mineralization varying in width from 1 to 2 m at a depth of about 700 m to up to 20 to 50 m wide at or close to surface. This makes the deposit amenable to both selective and bulk mining methods. In section, the deposit is sub-vertical and cone shaped.

Analysis of drill hole profiles and sections shows that a large number of drill holes (about 40%) were selectively sampled only in those parts perceived to contain mineralization. This is demonstrated in Figure 14.20 where only a few samples were taken from drill hole N-11 in contrast to drill hole 83-B14.

Figure 14.20: Brookbank Profile showing Selective Sampling

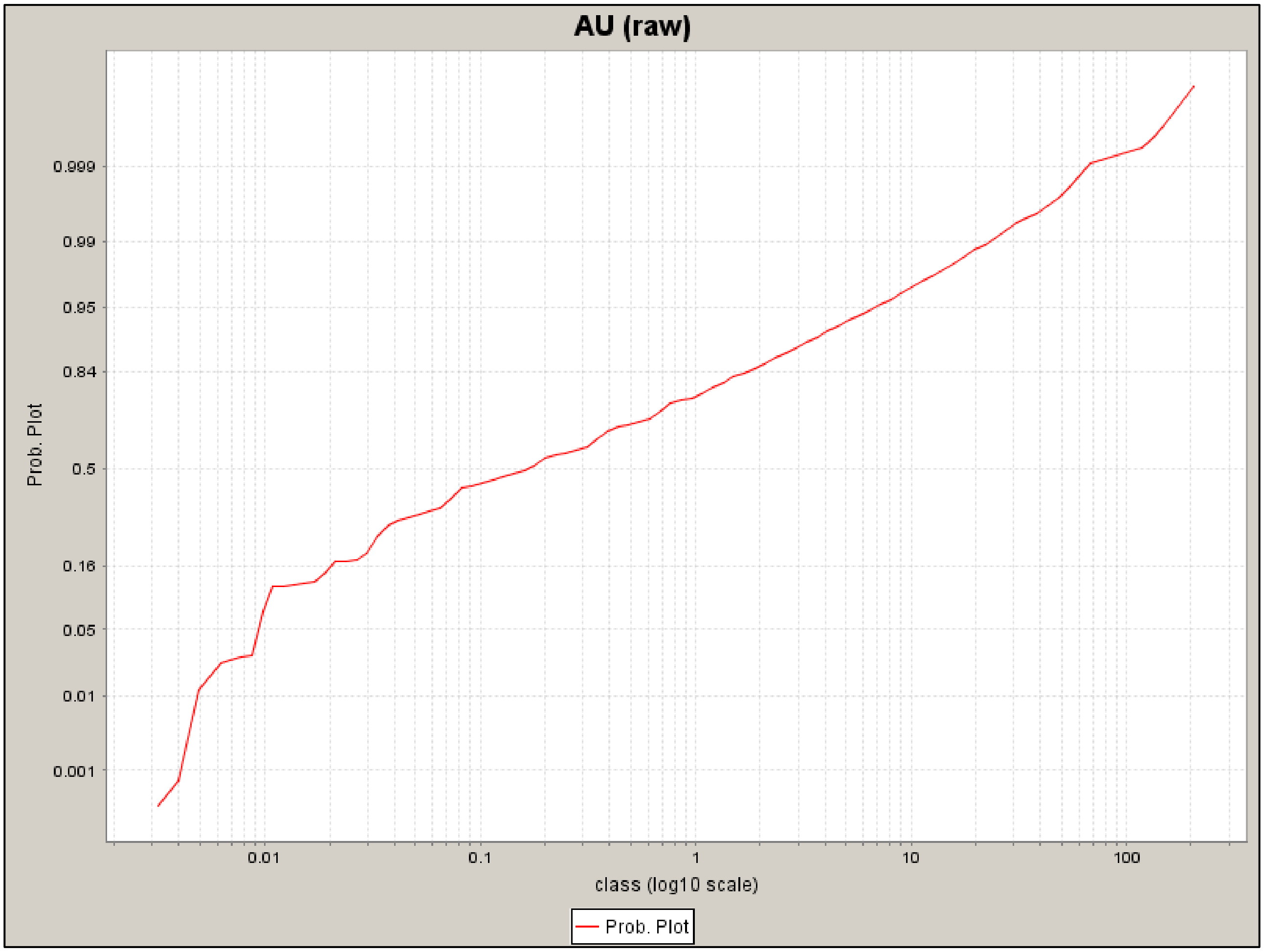


It is more than likely that some mineralized zones may have been missed as a consequence of selective sampling. The net result is a likely understatement of the resource.

14.2.1.6 Statistics/Determination of Mineral Envelope

Statistical analysis of raw assay data was conducted for the Brookbank Main deposit primarily to determine the mineralization indicator grade defining the envelope of the resource zone. Based on an interpretation of the log-probability curve obtained (Figure 14.21), the mineral envelope was established at a cut-off grade of 0.1 g Au/t.

Figure 14.21: Log Probability Plot for the Brookbank Main Deposit Assays



14.2.1.7 <u>Solid Modelling and Compositing</u>

Using the mineralization envelope cut-off grade of 0.1 g Au/t, the solid representing the deposit was modelled. The wireframe was constructed using a 3D interactive methodology. The triangulation vertices were snapped to the end points of the defined drill hole intervals to ensure proper sample capture. Snapped points were validated through visual checks. Waste zones within the envelope were modelled and discounted from the resource.

The composite length selected was 1 m based on the mode of the sample lengths. This short composite length was deemed necessary to match the selective sampling pattern and to differentiate the internal waste zones more precisely. Composites were created within the solid/mineralized envelope (lengths downhole) and adjusting lengths to avoid rejecting the last composite at the bottom limit of the solid. Composites were generated without applying grade capping so that legitimate high grade assays were honoured during interpolation, but spatially restricted to prevent grade smearing.

14.2.1.8 Composite Statistics

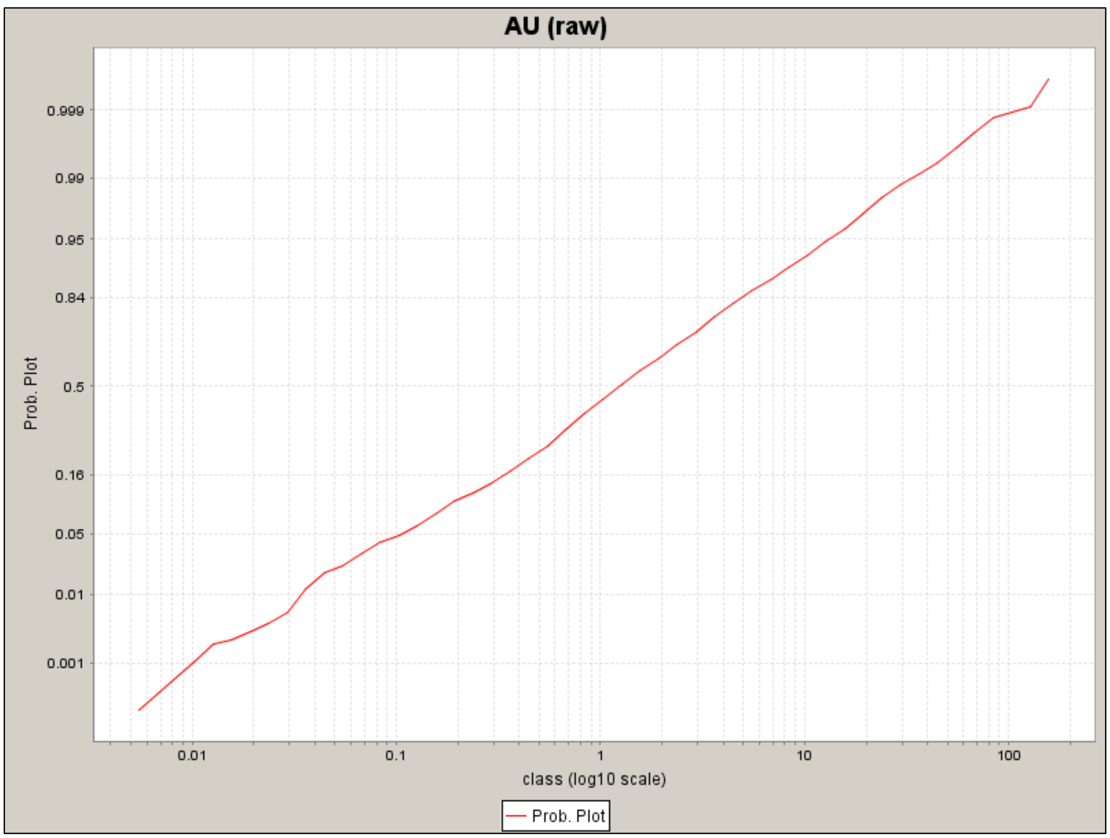
Statistical analysis of composite samples within the solid/mineralized envelope was performed to determine the population pattern, global mean and grade capping/restriction values. A summary of the statistics is presented in Table 14.24. The log-probability plot is shown on Figure 14.22. The spatial restriction and capping grades are based on significant inflexion points on the log-probability plots at or above the 95th and 99th percentile, respectively.

Table 14.24: Summary Statistics on Gold Composite Samples - Brookbank

No. of Samples	Min g Au/t	Max g Au/t	Mean g Au/t	Var g Au/t	SD g Au/t	CV g Au/t	GC g Au/t	RG g Au/t
3,319	0.01	173.57	3.37	61.75	7.86	2.33	70	30

Notes: Min = minimum; Max = maximum; Var = variance; SD = standard deviation; CV = coefficient of variation; GC = grade capping; RG = restriction grade

Figure 14.22: Composites Log Probability Plot for the Brookbank Main Mineralize Solid



14.2.1.9 Spatial Analysis

Variography was conducted using composite samples in order to define the continuity of the mineralization to establish the maximum range/distance over which samples/drill hole intercepts may be correlated, and the optimum parameters for the search ellipse to be used in the interpolation of grades.

Initially, a downhole variogram was computed in order to establish the nugget effect; thereafter, three variograms to cover the principal geometrical directions were computed and modelled using the nugget effect established from the downhole variogram. The principal results are summarized in Table 14.25.

Table 14.25: Variography Results for the Brookbank Deposit

Variogram Model	Nugget	Range Major Axis	Range Semi-major Axis	Range Minor Axis	Bearing	Plunge	Dip
Spherical	0.11	63	63	3	80	0	-85

The ranges of influence of 63 m along strike (major axis) and down dip (semi-major axis) indicate reasonable mineralization continuity in those two directions. However, the range of influence of 3 m across the width (minor axis) of the deposit indicates pronounced variability. This variability is also partly attributable to selective sampling.

14.2.1.10 Block Model Definition and Search Parameters

The block model definition is presented in Table 14.26. The upper limit representing surface topography was generated from drill hole collars. The parent block size was based on drill hole spacing, envisaged selective mining unit ("SMU") and geometry of the deposit. Partial percents were used at the solid/mineralization envelope boundary to get an accurate volume representation. A volume check of the block model versus the mineralization envelope revealed a good representation of the volume of the solid.

Table 14.26: Brookbank Main Deposit Block Model Attributes

Item	X	Y	Z
Origin Coordinates	438,940.00	5,506,300.00	500
Block Extents (m)	2,200	1,500	1,700
Parent Block Size	10	1.5	10
Rotation	15 degrees anti-clockwise		

The search ellipse configurations were defined using variography as a guide combined with the geometry of the deposit. A four-pass estimation procedure was used for the interpolation. For all passes, the maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, the minimum number of samples and the maximum samples per drill hole for interpolation were designed to ensure that the nearest samples are accorded the highest weighting and that a minimum of the three closest holes are used in the interpolation.

For Pass 2, the maximum number of samples per drill hole was designed to ensure a minimum of two drill holes in the interpolation, to go beyond the limits of Pass 1.

For Pass 3, the minimum number of samples and the maximum number of samples per drill hole allowed the bigger ellipse to interpolate grades into the remaining blocks not covered by Passes 1 and 2.

Pass 4 parameters were used to ensure that all blocks of the block model were filled.

The search parameters adopted for grade interpolation are summarized in Table 14.27.

Table 14.27: Summary of Search Parameters - Brookbank Deposit

Pass	X	Y	Z	Min. S	Max. S	Max. S/DH
1	60	3	60	6	12	2
2	120	6	120	6	12	2
3	480	12	480	2	12	2
4	600	24	600	2	12	2

Notes: Min. S = minimum samples; Max. S = maximum samples; S/DH = samples/drill hole

14.2.1.11 Grade Interpolation and Validation

Block grades were estimated using the ID^3 function of the GEMS mining software. Grade smearing was minimized by restricting the influence of values exceeding the restriction grade ("RG") of 30 g Au/t shown in Table 14.24.

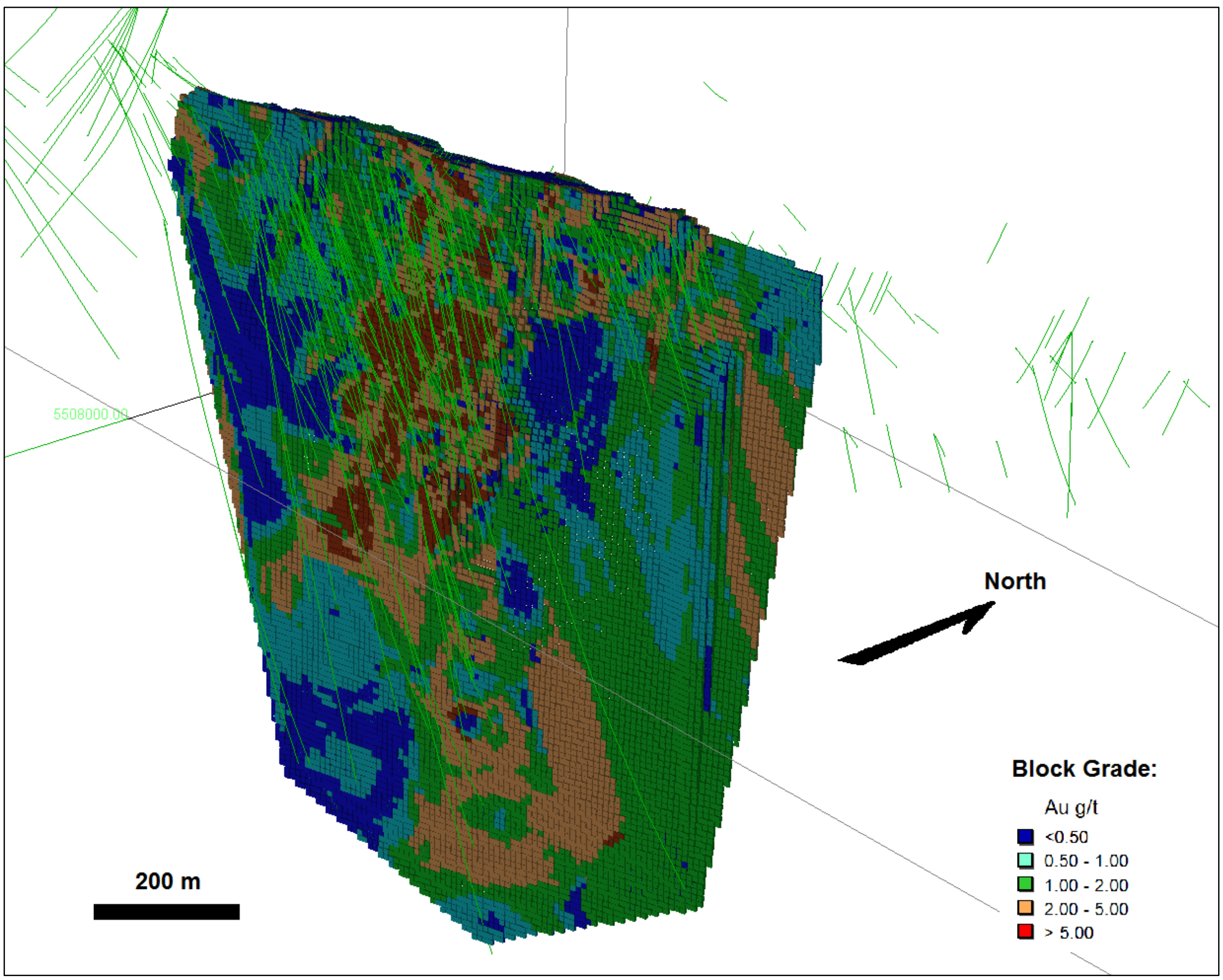
Table 14.28 presents a summary of the parameters and assumptions used for the grade interpolations.

Table 14.28: Summary of Parameters/Assumptions for Grade Interpolation - Brookbank Deposit

Parameter	Assumption
Date of Data Used	30-Apr-12
Number of Drill Holes	376
Specific Gravity (SG)	2.87
Block Model and Interpolation Software	GEMS
Interpolation Method	ID^3
Block Size (X, Y, Z)	10 x 1.5 x 10
Restricted Search Radius (X, Y, Z)	10 x 3 x 10

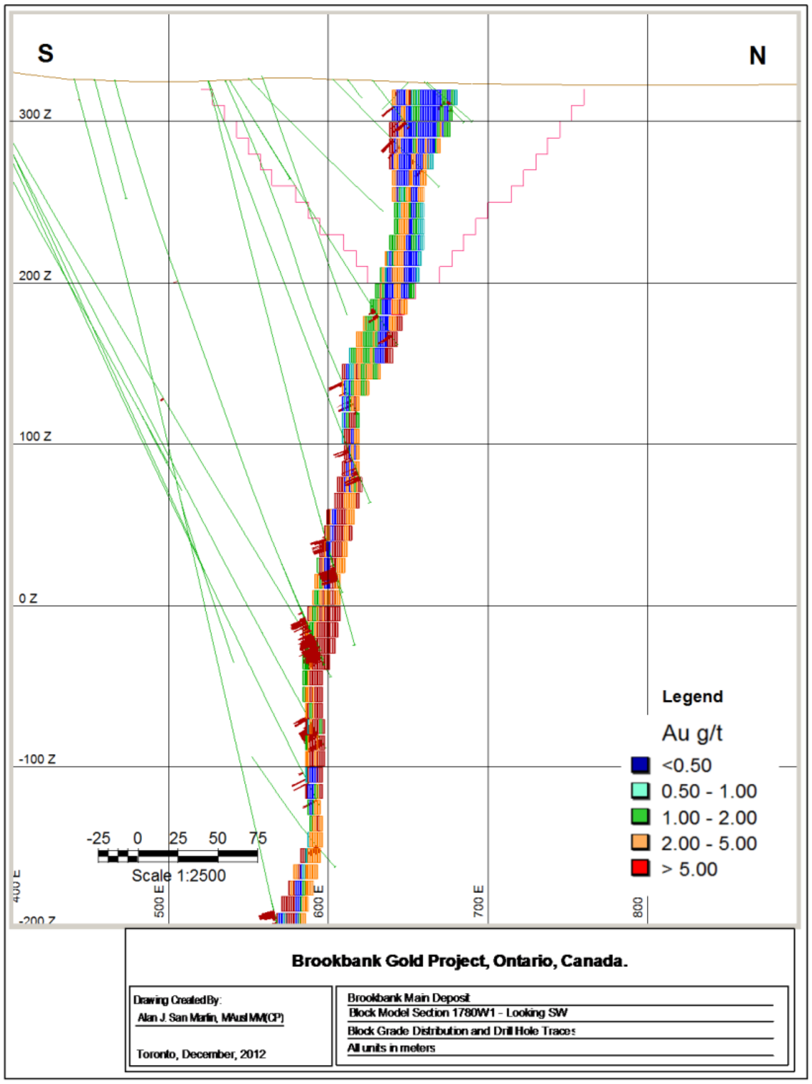
The resource block model showing distribution of gold grades is shown in Figure 14.23.

Figure 14.23: Brookbank Gold Grade Distribution in Resource Block Model



The block model was validated by visual inspection in plan and section to ensure that block grade estimates reflect the grades seen in intersecting drill holes. A typical section is shown in Figure 14.24.

Figure 14.24: Typical Section through the Brookbank Deposit



14.2.1.12 Determination of Mineral Resources (Open Pit Shell vs. Underground)

The resource block model was examined for open pit and underground economic potential at various cut-off grades. To do this, the block model was subjected to an analysis using a conventional Lerchs-Grossmann algorithm with computer software, to define a series of potentially economic open pit shells.

In order to run the Whittle economic pit optimization, assumed operating costs, gold recovery and gold price were required. The metallurgical recovery adopted by Micon was based on the historical recoveries in the

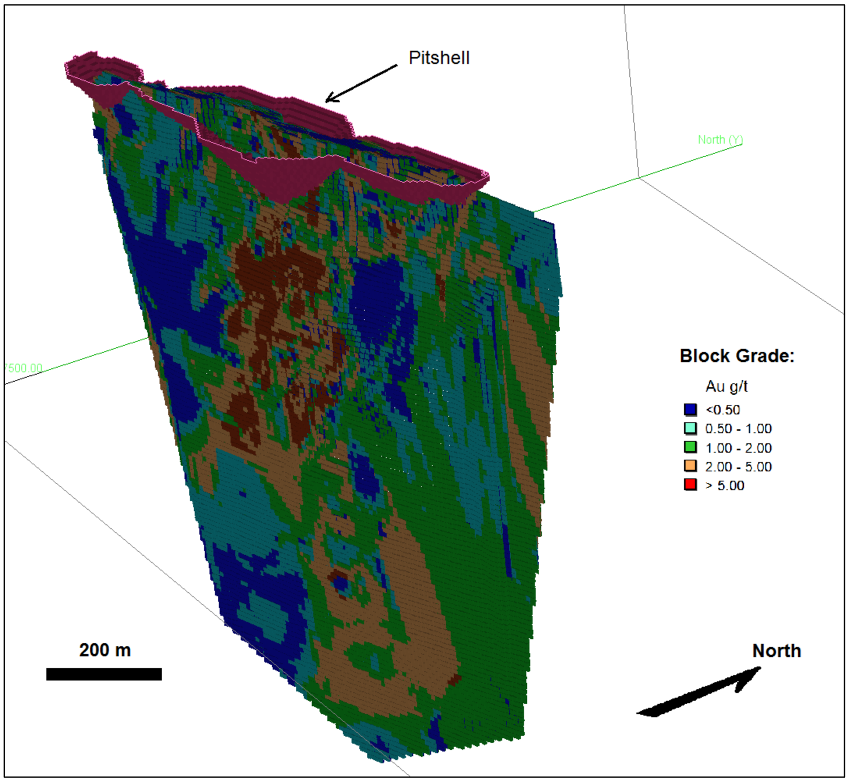
Brookbank-Hardrock district, updated for a modern process plant, and the gold price was approximately the three-year trailing average. The rest of the economic parameters estimates were based on Micon's experience with similar operations. Table 14.29 shows the various parameters/assumptions used in the open pit analysis as well as the gold cut-off grades used for reporting the Mineral Resource estimate.

Table 14.29: Economic Parameters used in the Open Pit Analysis - Brookbank Deposit

Item	Unit	Value
Open Pit Mining Cost	CAD/All Material Tonne	3.00
Underground Mining Cost	CAD/Ore Tonne	60.00
Processing Cost	CAD/Ore Tonne	20.00
G&A Cost	CAD/Ore Tonne	1.00
Gold Price	USD/Troy Ounce	1,455.00
Pit Slope	Degrees	50
Mill Recovery	Percent	90
Exchange Rate	USD to CAD	1.000
Open Pit Calculated Gold Cut-off Grade	**g Au/t**	**0.50**
Underground Calculated Gold Cut-off Grade	**g Au/t**	**2.80**

After completing the Whittle pit optimization, the results were re-imported back into GEMS where the block model was flagged for the material in the economic pit-shell, with the material outside of the shell being flagged as potential underground material. The resulting pit is shown in Figure 14.25.

Figure 14.25: Brookbank Project Open Pit



14.2.1.13 Resource Categorization

Micon has classified resource blocks in the block model based largely upon the drilling density and the passes criteria described in Subsection 14.2.1.11, while also accounting for variography results and deposit geometry. The resource categories are shown on Figure 14.26. At this stage, there are no Measured Mineral Resources for the Brookbank Project.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit covered by Pass 2 of the search ellipsoid, including islands of Pass 1 encompassed within. Good visual evidence of adequate sample/drill hole coverage was also considered.

The Inferred Mineral Resource category was assigned to Pass 3 and Pass 4 areas, including islands of Pass 2. These areas have very limited drill hole information.

The estimated Mineral Resources at various cut-off grades are presented in Table 14.30 and Table 14.31.

Figure 14.26: Brookbank Mineral Resource Classification

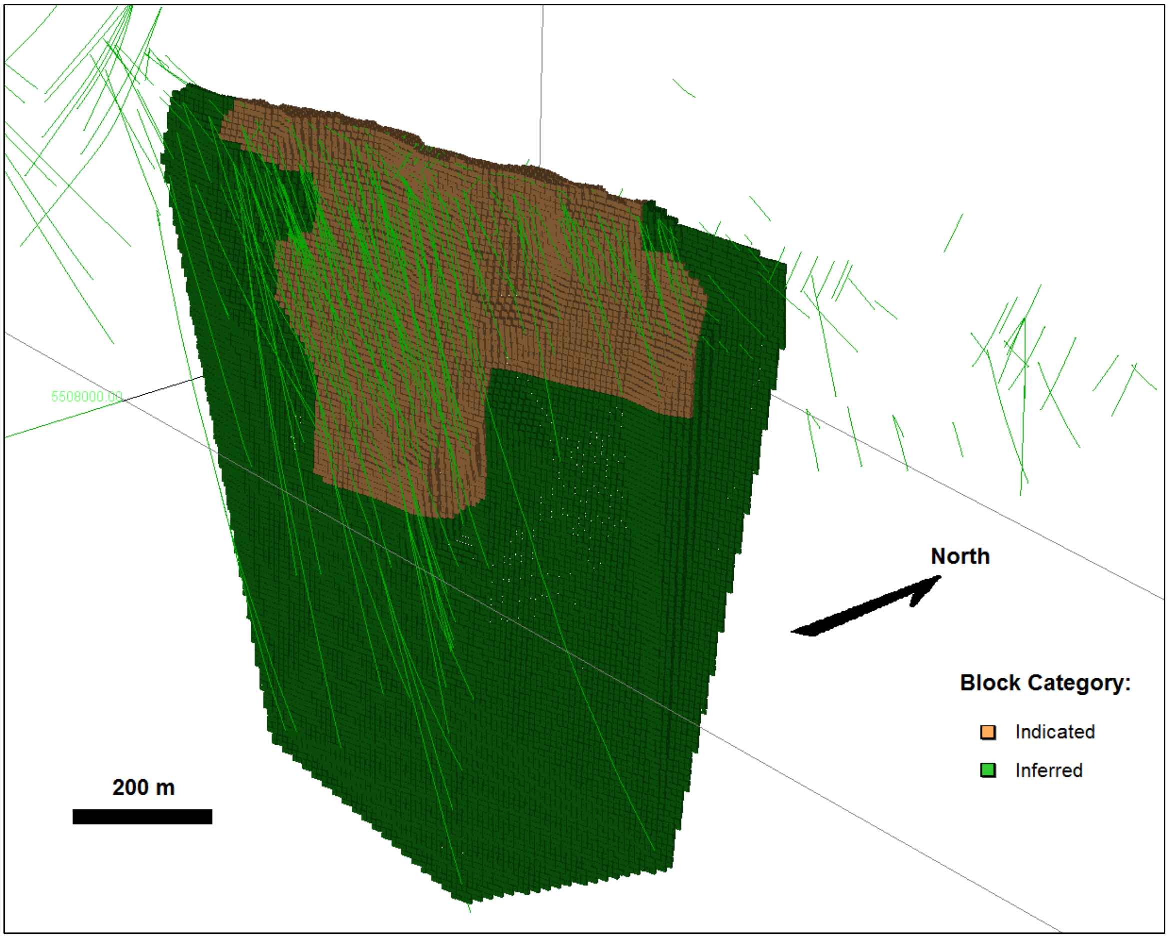


Table 14.30: Brookbank Project In-pit Mineral Resources

Resource Category	Cut-off Grade (g Au/t)	Tonnes	Avg. Grade (g Au/t)	Contained Gold (oz)
Measured	-	-	-	-
Indicated	>3.5	314,000	5.65	57,000
	3.0	434,000	4.98	70,000
	2.8	491,000	4.74	75,000
	2.5	595,000	4.37	84,000
	2.0	872,000	3.69	103,000
	1.5	1,311,000	3.04	128,000
	1.0	1,967,000	2.44	154,000
	0.83	2,230,000	2.26	162,000
	0.5	**2,638,000**	**2.01**	**171,000**
	NC	4,914,000	1.14	180,000
Total M & I	**0.5**	**2,638,000**	**2.01**	**171,000**
Inferred	>3.5	55,000	3.90	7,000
	3.0	72,000	3.74	9,000
	2.8	79,000	3.67	9,000
	2.5	88,000	3.57	10,000
	2.0	96,000	3.45	11,000
	1.5	106,000	3.29	11,000
	1.0	128,000	2.94	12,000
	0.83	139,000	2.77	12,000
	0.5	**171,000**	**2.38**	**13,000**
	NC	238,000	1.73	13,000

Note:

1. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
2. *Totals may not add correctly due to rounding.*
3. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
4. *The effective date of the estimate is December 31, 2012.*

Table 14.31: Brookbank Project Underground Mineral Resources

Resource Category	Cut-off Grade (g Au/t)	Tonnes	Avg. Grade (g Au/t)	Contained Gold (oz)
Measured	-	-	-	-
Indicated	>3.5	1,505,000	8.15	394,000
	3.0	1,732,000	7.51	418,000
	2.8	1,851,000	7.21	429,000
Total M & I	**2.8**	**1,851,000**	**7.21**	**429,000**
Inferred	>3.5	232,000	4.76	36,000
	3.0	352,000	4.24	48,000
	2.8	**403,000**	**4.07**	**53,000**

Notes:

1. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
2. *Totals may not add correctly due to rounding.*
3. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
4. *The effective date of the estimate is December 31, 2012.*

14.2.1.14 Mineral Resource Statement/Summary

The Mineral Resources are summarized in Table 14.32 at cut-off grades of 0.5 g Au/t and 2.8 g Au/t for open pit and underground resources, respectively. The cut-off grades adopted offer the deposit reasonable prospects for eventual economic extraction on the assumptions summarized in Table 14.29.

The estimated Mineral Resources conform to the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101.

Table 14.32: Summary of Brookbank Mineral Resource

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Brookbank Project	Open Pit	Measured (M)	-	-	-
		Indicated (I)	2.638	2.01	171
		Subtotal M & I	**2.638**	**2.02**	**171**
		Inferred	0.171	2.38	13
	Underground	Measured (M)	-	-	-
		Indicated (I)	1.851	7.21	429
		Subtotal M & I	**1.851**	**7.21**	**429**
		Inferred	0.403	4.02	53

Notes:

1. *CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.*
2. *Totals may not add correctly due to rounding.*
3. *Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.*
4. *The effective date of the estimate is December 31, 2012.*

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues. It is uncertain if further exploration will result in upgrading the Inferred Mineral Resources to an Indicated or Measured Mineral Resource category.

14.2.1.15 Mineral Resource Conclusions

Based on the information available at December 31, 2012, the combined Mineral Resource (open pit and underground) for the Brookbank deposit main zone, excluding the Cherbourg and Fox Ear areas, is 600,000 oz Au in the Indicated Mineral Resource category and 66,000 oz Au in the Inferred Mineral Resource category. Micon believes that selective sampling may have resulted in the exclusion of some mineralized material from the resource.

A careful analysis of drill hole information demonstrates that there is potential to increase the resource along strike as the deposit is still open ended in both directions (east and west). This is evident from Figure 14.27 and Figure 14.28 which show selectively sampled reconnaissance drill holes with high grade intercepts likely following the trend of the main zone. Note that these reconnaissance holes are single and rather too isolated to be incorporated into the current 3D model.

In section, the Brookbank Main deposit is cone-shaped and is restricted to a vertical depth of about 700 to 750 m. This apparent limitation at depth may be an artifact of selective sampling and needs to be investigated.

The Cherbourg and Fox Ear deposits are too isolated and far too small to have reasonable prospects for economic extraction. Accordingly, they are excluded from the resource estimate of the Brookbank Project. However, further exploration may increase the size of these deposits.

14.2.1.16 Mineral Resource Recommendations

In Micon's opinion, the deposit limits should be established prior to embarking on detailed economic studies. Thus, in the short term, defining the overall size of the deposit and its characteristics should be prioritized. Accordingly, Micon recommends a detailed exploration program east and west of the Brookbank deposit main zone along the main volcanic-sedimentary contact. The program should include a re-examination of existing drill holes located along strike on either side of the deposit and re-sampling the entire drill hole lengths.

New drilling should be carefully and systematically planned to take into account the possibility that the continuity of the deposit along strike may be in an echelon pattern. This program will also establish whether the Cherbourg and Fox Ear deposits are extensions of the Brookbank deposit.

Figure 14.27: East Side Plan for the Brookbank Deposit showing Mineralized Intercepts further East of the Current Deposit Solid Limit

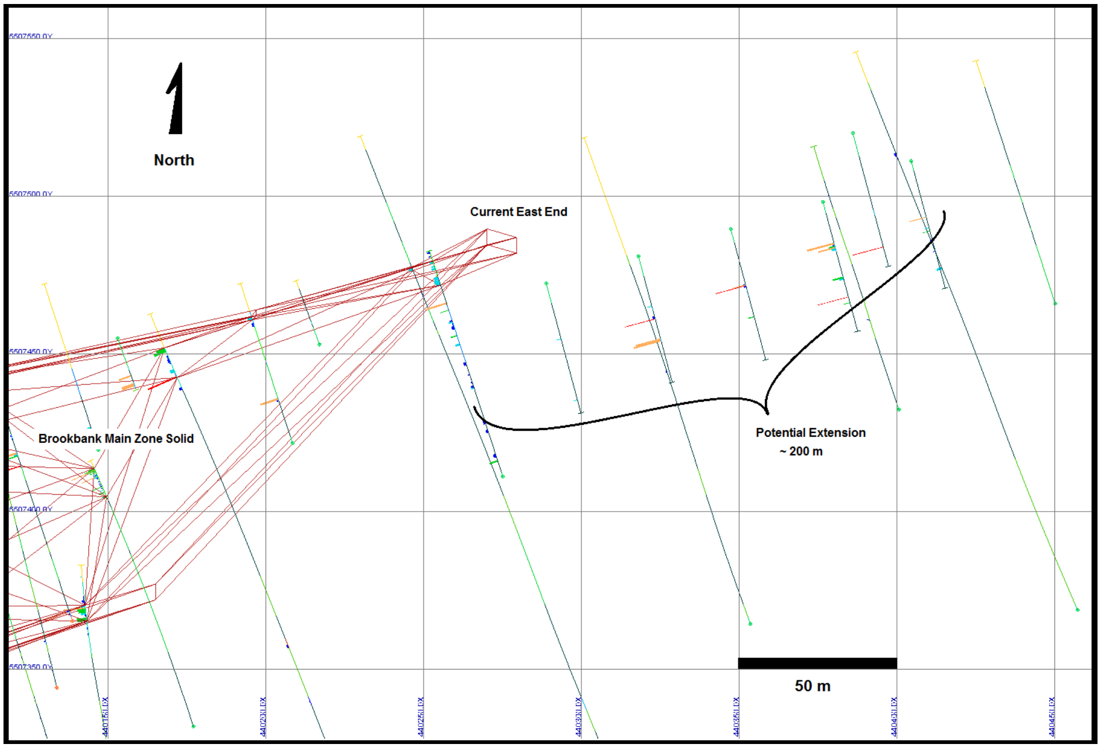
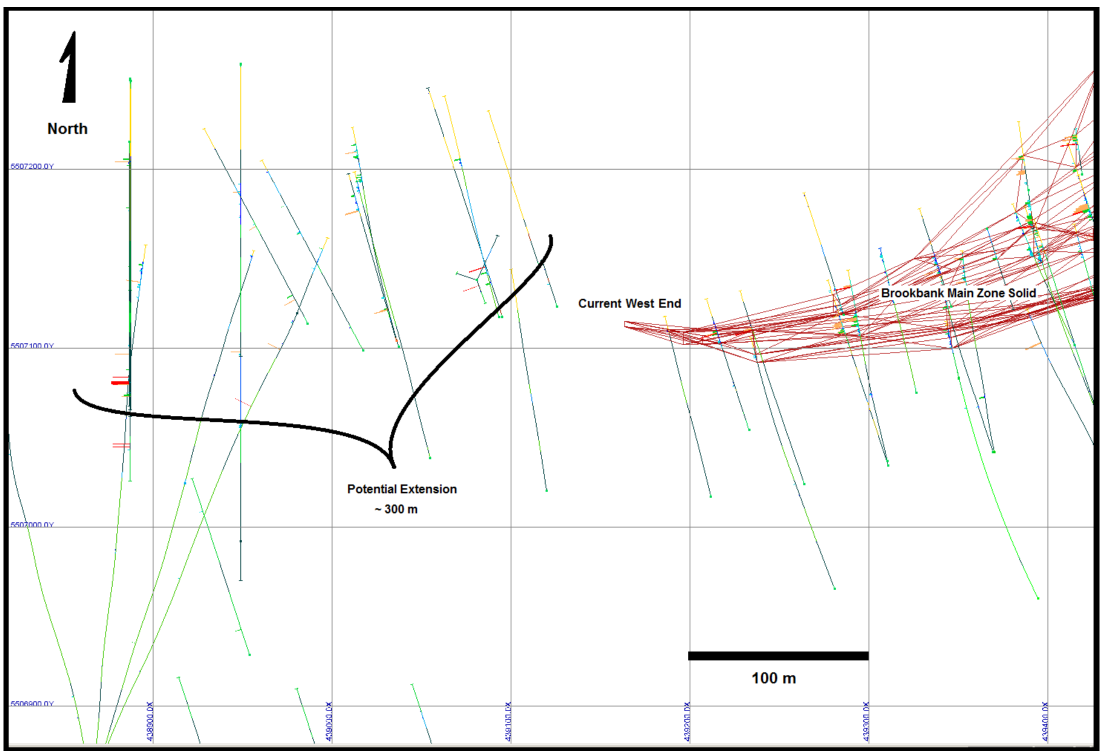


Figure 14.28: Western Side Plan for the Brookbank Deposit showing Mineralized Intercepts further West of the Current Deposit Limit



14.2.2 Key Lake

14.2.2.1 Geological Interpretation

The Key Lake deposit comprises a series of 12 domains in an echelon arrangement in a northwesterly direction. The mineralization is generally of a low-grade nature in the northwest end and medium grade in the southeast end. It has a volcanoclastic-exhalative nature. Post mineralization processes have concentrated the mineralization into isolated high-grade patches/pockets.

Statistical analysis of raw assay data was conducted for the Key Lake project primarily to determine the mineralization indicator grade defining the envelopes of the resource zones. Interpretation of the log-probability curves obtained established 0.3 g Au/t as the envelope cut-off grade.

14.2.2.2 Solid Modelling/Domain Definition/Compositing

Using the envelope cut-off grade of 0.3 g Au/t, solids representing the mineralized zones were modelled. The wireframes were constructed using a 3D interactive methodology. The triangulation vertices were snapped to the end points of the defined drill hole intervals to ensure proper sample capture. Snapped points were validated through visual checks.

The composite length selected is 3 m, based on the minimum width of a mineralized interval. Composites were created downhole within the solids/mineralized envelopes. Where necessary, the composite length was adjusted to accommodate the entire interval within the solid. Composites were generated without applying grade capping so that legitimate high grade assays were honoured during interpolation, but spatially restricted to prevent grade smearing.

14.2.2.3 Composite Statistics

Statistical analysis of composite samples within the solids/mineralized envelopes was performed to determine population patterns, global mean, restriction and top-cut values. Log-probability plots were used to establish the restriction and top-cut grades. A summary of the statistics is presented in Table 14.33.

Table 14.33: Summary of Statistics on Gold Composite Samples - Key Lake Deposit

Mineralized Domain (Code)	No. of Samples	Min	Max	Mean	Var	SD	CV	GC	RG
KL01 (31)	71	0.06	36.82	3.30	47.63	6.90	2.09	N/A	2.8
KL02 (32)	29	0.01	11.52	1.18	5.01	2.24	1.90	N/A	1.6
KL03 (33)	14	0.35	12.63	1.74	9.95	3.15	1.81	N/A	1.6
KL04 (34)	112	0.01	13.73	1.33	3.10	1.76	1.32	N/A	4.0
KL05 (35)	862	0.00	53.97	1.16	8.52	2.92	2.52	N/A	6.0
KL06 (36)	1200	0.00	40.04	0.91	3.32	1.82	2.01	N/A	7.5
KL07 (37)	83	0.01	5.41	1.34	1.37	1.17	0.88	N/A	N/A
KL08 (38)	133	0.00	7.29	0.91	1.12	1.06	1.16	N/A	N/A
KL09 (39)	11	0.02	8.49	2.99	7.56	2.75	0.92	N/A	3.3
KL10 (40)	24	0.01	5.83	1.60	2.29	1.51	0.95	N/A	N/A
KL11 (41)	8	1.21	5.79	3.20	2.90	1.70	0.53	N/A	N/A
KL12 (42)	11	0.88	3.72	2.53	1.15	1.07	0.42	N/A	N/A

Notes: Min = Minimum; Max = Maximum; Var = Variance; SD = Standard Deviation; CV = Coefficient of Variation; GC = Grade Capping; RG = Restriction Grade.

14.2.2.4 Spatial Analysis/Variography

Variography was conducted (using composite samples) in order to define the continuity of the mineralization to establish the maximum range/distance over which samples/drill hole intercepts may be correlated.

The results indicate a pure nugget effect. The pure nugget effect is consistent with the results of physical examination of drill hole intercepts and the haphazard spotted nature of the high values throughout the deposit.

14.2.2.5 Block Model Definition and Search Parameters

One 3D block model was constructed using the GEMS version 6.4 mining software. The block model details are presented in Table 14.34.

Table 14.34: Key Lake Deposit Block Model Definition

Item	X	Y	Z
Origin Coordinates	489,600.00	5,507,040.00	360
Block Extents (m)	3,400	420	550
Parent Block Size	10	3	10
Rotation	20 degrees clockwise		

The model was constrained by the 13 domain solids of the deposit. The upper limits, representing the surfaces/topographies, were generated from drill hole collars. Parent block sizes were based on the drill hole spacing and geometry of the deposit. Partial percents were used at the solid/mineralization envelope boundary to get an accurate volume representation. Volume checks of the deposit block models versus the mineralization envelopes revealed a good representation of the volumes of the solids.

The search ellipse configurations were defined using drill hole spacing combined with the geometry of the deposit. A three pass estimation procedure was used for grade interpolation. For each pass, the maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, the minimum number of samples and the maximum samples per drill hole for interpolation were designed to ensure that the nearest samples are accorded the highest weighting and that a minimum of the three closest holes are used in the interpolation.

For Pass 2, the maximum number of samples per drill hole was designed to ensure a minimum of two drill holes in the interpolation, to go beyond the limits of Pass 1.

For Pass 3, the minimum number of samples and the maximum number of samples per drill hole allowed the bigger ellipse to interpolate grades into the remaining blocks not covered by Passes 1 and 2.

The search parameters for the grade interpolation are summarized in Table 14.35.

Table 14.35: Summary of Search Parameters - Key Lake Deposit

Domain	Pass	X	Y	Z	Min. S	Max. S	Max. S/DH
KL to KL12	1	1.50	20	50	4	8	2
	2	100	20	100	4	8	2
	3	200	40	200	2	8	2

14.2.2.6 Grade Interpolation and Validation

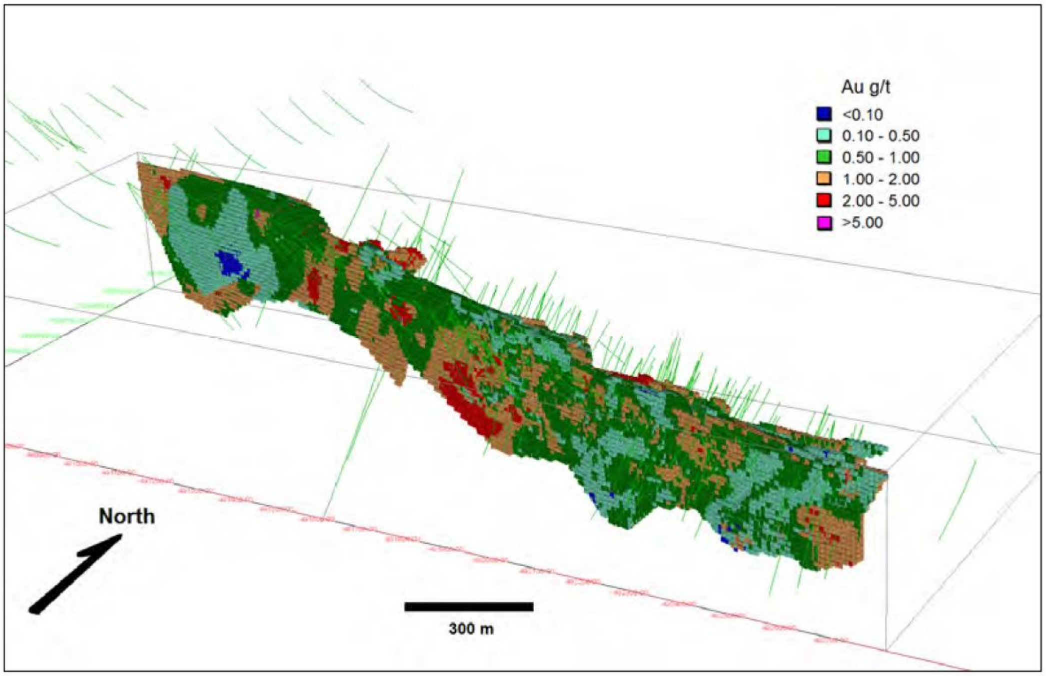
Grade interpolation for gold was performed using the ID^3 function of the GEMS mining software. Grade smearing was minimized by restricting the influence of values exceeding the RG shown in Table 14.33. The parameters and assumptions used for grade interpolations are summarized in Table 14.36.

Table 14.36: Summary of Parameters / Assumptions for Grade Interpolation - Key Lake Deposit

Parameter	Assumption
Date of Data Used	31-Oct-12
Number of Drill Holes	245
Specific Gravity (SG)	2.87
Block Model & Interpolation Software	GEMS
Interpolation Method	ID^3
Block Sizes (X, Y, Z)	10 x 3 x 10
Restricted Search Radius (X, Y, Z)	10 x 2 x 10

The resource block model showing distribution of gold grades is shown Figure 14.29. The block model was validated by visual inspection in plan and section to ensure that block grades estimates reflect the grades seen in intersecting drill holes.

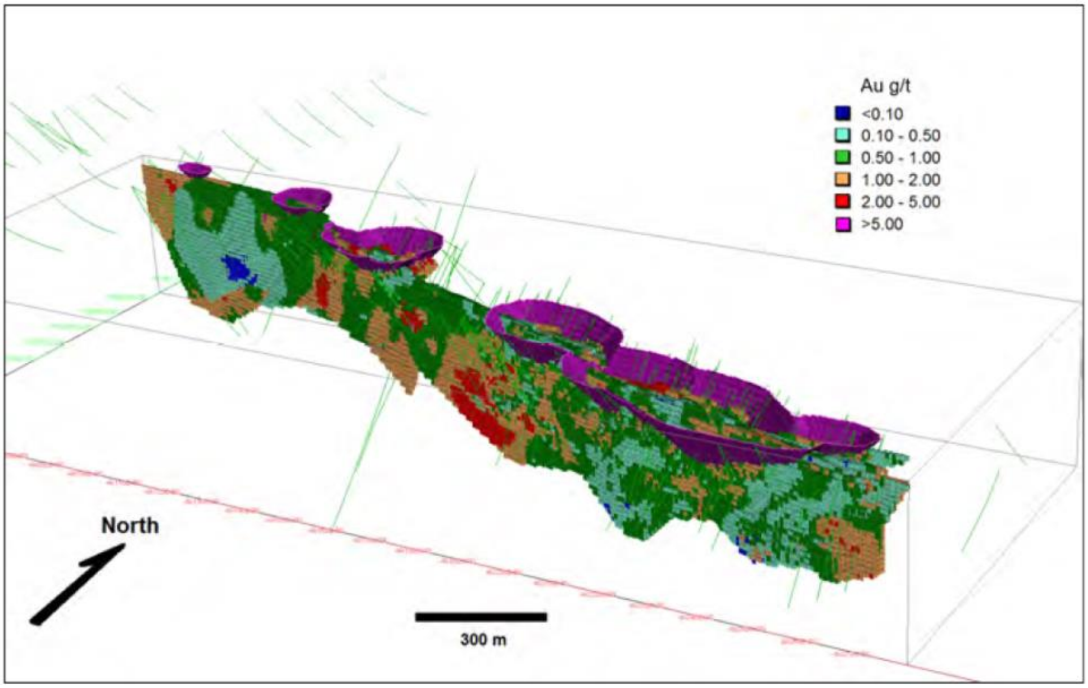
Figure 14.29: Key Lake Gold Distribution in Block Model



14.2.2.7 <u>Determination of Mineral Resources (Open Pit Shell vs. Underground)</u>

The resource block model was examined for open pit and underground economic potential at various cut cut-off grades using the same methodology and economic parameters as that adopted for the Brookbank Project - see Subsection 14.2.1.12 above. The resulting pit is shown in Figure 14.30.

Figure 14.30: Key Lake Deposit Open Pit Layout



14.2.2.8 Resource Categorization

Micon has classified resource blocks in the block model based largely upon the drilling density and the pass criteria described in Subsection 14.2.2.5.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit covered by Pass 2 of the search ellipsoid, including islands of Pass 1. Good visual evidence of adequate sample/drill hole coverage was also considered.

The Inferred Mineral Resource category was assigned to Pass 3 areas, including islands of Pass 2. These areas have very limited drill hole information.

The estimated resources at various cut-off grades are presented in Table 14.37 and Table 14.38 and the resource block model coloured by classification is shown in Figure 14.31. The tonnes and ounces have been rounded to the nearest 1,000.

Table 14.37: Key Lake In Pit Mineral Resources

Resource Category	Cut-off Grade (g Au/t)	Tonnes	Avg. Grade (g Au/t)	Contained Gold (oz)
Measured	-	-	-	-
Indicated	>3.5	24,000	4.97	4,000
	3.0	46,000	4.16	6,000
	2.8	54,000	3.97	7,000
	2.5	71,000	3.64	8,000
	2.0	178,000	2.78	16,000
	1.5	533,000	2.07	35,000
	1.0	1,364,000	1.55	68,000
	0.83	1,775,000	1.41	80,000
	0.5	**2,572,000**	**1.17**	**97,000**
	No cut-off	3,485,000	0.95	106,000
Total M & I	**0.5**	**2,572,000**	**1.17**	**97,000**
Inferred	>3.5	18,000	5.30	3,000
	3.0	39,000	4.18	5,000
	2.8	53,000	3.82	7,000
	2.5	81,000	3.42	9,000
	2.0	143,000	2.89	13,000
	1.5	368,000	2.16	26,000
	1.0	805,000	1.66	43,000
	0.83	965,000	1.54	48,000
	0.5	**1,345,000**	**1.29**	**56,000**
	No Cut-off	1,609,000	1.13	58,000

Notes:

1. CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.
2. Totals may not add correctly due to rounding.
3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.
4. The effective date of the estimate is December 31, 2012

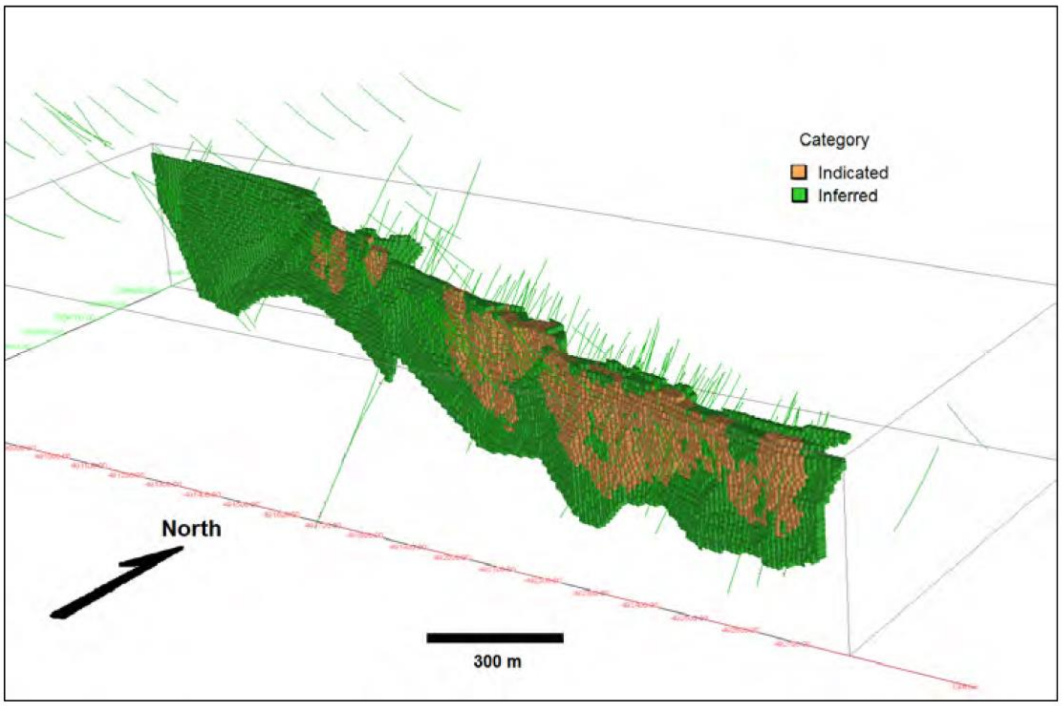
Table 14.38: Key Lake Underground Mineral Resources

Resource Category	Cut-off Grade (g Au/t)	Tonnes	Avg. Grade (g Au/t)	Contained Gold (oz)
Measured	-	-	-	-
Indicated	>3.5	14,000	10.40	5,000
	3.0	25,000	7.33	6,000
	2.8	**31,000**	**6.48**	**6,000**
Total M & I	**2.8**	**31,000**	**6.48**	**6,000**
Inferred	>3.5	21,000	4.51	3,000
	3.0	39,000	3.92	5,000
	2.8	**58,000**	**3.57**	**7,000**

Notes:

1. CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.
2. Totals may not add correctly due to rounding.
3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.
4. *The effective date of the estimate is December 31, 2012.*

Figure 14.31: Key Lake Mineral Resource Categories



14.2.2.9 Mineral Resource Statement / Summary

Based on the results of the Whittle pit optimizations, the Key Lake Project Mineral Resources are reported at cut-off grades of 0.5 g Au/t and 2.8 g Au/t for open pit and underground resources, respectively. The resources are summarized in Table 14.39. The estimated Mineral Resources conform to the current CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101.

Table 14.39: Summary of Key Lake Mineral Resources

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Key Lake Project	Open Pit	Measured (M)	0.000		0
		Indicated (I)	2.572	1.17	97
		Subtotal M & I	**2.572**	**1.17**	**97**
		Inferred	1.345	1.29	56
	Underground	Measured (M)	0.000		0
		Indicated (I)	0.031	6.48	6
		Subtotal M & I	**0.031**	**6.48**	**6**
		Inferred	0.058	3.57	7

Notes:

1. CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.
2. Totals may not add correctly due to rounding.
3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.
4. *The effective date of the estimate is December 31, 2012.*

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues. Micon considers the resources to be insufficient to support a stand-alone operation and therefore, the Key Lake deposit should be considered as a satellite to the Hardrock deposit.

14.2.3 Kailey Deposit

The Kailey deposit is a separate mineral deposit located within the confines of the Hardrock Project. It is located 1.7 km north of the Hardrock deposit.

14.2.3.1 Solid Modelling/Domain Definition and Compositing

A mineral envelope cut-off grade of 0.1 g Au/t was determined from a statistical analysis and used to model the solid representing the deposit outline. The wireframe was constructed using a 3D interactive methodology. The triangulation vertices were snapped to the end points of the defined drill hole intervals to ensure proper sample capture. Snapped points were validated through visual checks.

A composite length of 3.0 m was adopted, based on the large width of the deposit. Composites were created within the solid/mineralized envelope downhole and adjusting lengths to avoid rejecting the last composite at the bottom limit of the solid model. Composites were generated without applying grade capping so that legitimate high grade assays were honoured during interpolation, but spatially restricted to prevent grade smearing.

14.2.3.2 Composites Statistics

Statistical analysis of composite samples within the solid/mineralized envelope was performed to determine population patterns, global mean, grade capping and RG values. A summary of the statistics is presented in Table 14.40.

Table 14.40: Summary Statistics on Gold Composite Samples Kailey Deposit

No. of Samples	Min g Au/t	Max g Au/t	Mean g Au/t	Var g Au/t	SD g Au/t	CV g Au/t	GC g Au/t	RG g Au/t
3,313	0.01	25.87	0.53	1.25	1.12	2.12	N/A	7.0

Notes: Min = minimum; Max = maximum; Var = variance; SD = standard deviation; CV = coefficient of variation; GC = grade capping; RG = restriction grade

14.2.3.3 Spatial Analysis / Variography

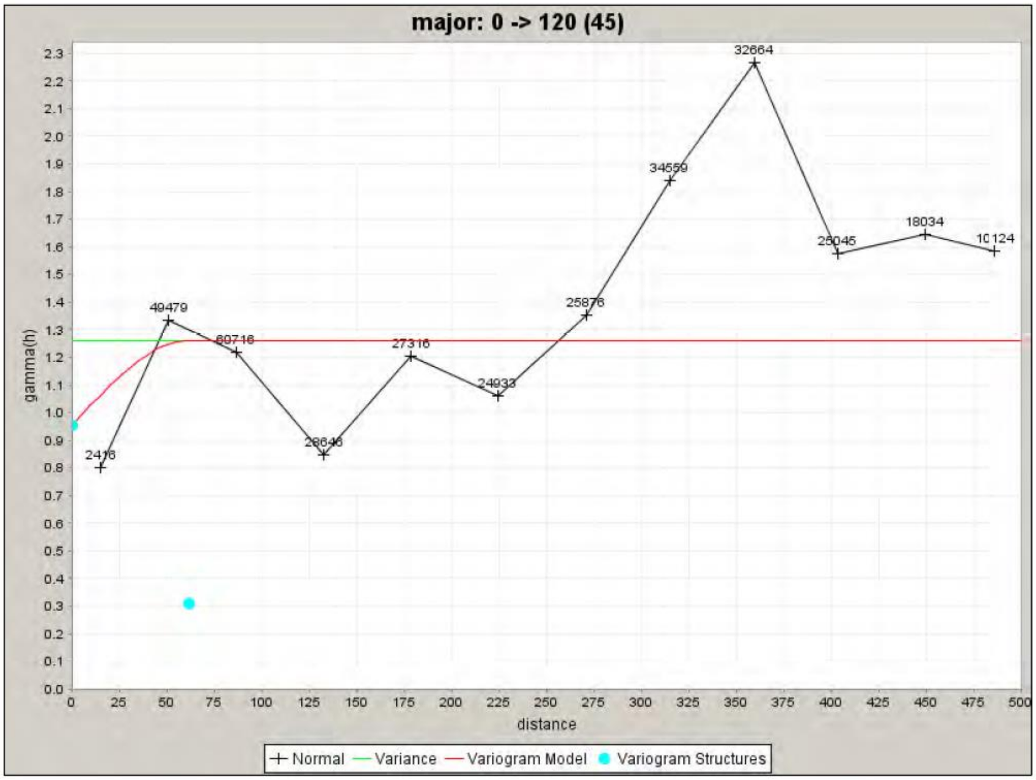
Variography was conducted using composite samples in order to define the continuity of the mineralization to establish the maximum range/distance over which samples/drill hole intercepts may be correlated, and the optimum parameters for the search ellipse to be used in the interpolation of grades.

Initially, a downhole variogram was computed in order to establish the nugget effect; thereafter, three variograms to cover the principal geometrical directions were computed and modelled using the nugget effect established from the downhole variogram. The variographic analysis results demonstrate reasonable continuity in all directions as shown in Table 14.41. The variogram model is presented in Figure 14.32.

Table 14.41: Variography Results for the Kailey Deposit

Variogram Model	Nugget	Range Major Axis	Range Semi-major Axis	Range Minor Axis	Bearing	Plunge	Dip
Spherical	0.95	62	62	62	120	0	-75

Figure 14.32: Variogram Model - Kailey Deposit



14.2.3.4 Block Model Definition and Search Parameters

A 3D block model was constructed using the GEMS version 6.4 mining software. The block model details are presented in Table 14.42.

Table 14.42: Kailey Deposit Model Definition

Item	X	Y	Z
Origin Coordinates	502,170.00	5,504,700.00	370
Block Extents (m)	1,800	1,200	650
Parent Block Size	10	10	10
Rotation	20 degrees clockwise		

The search ellipse configurations were defined using variography results as a guide and are summarized in Table 14.41. A three-pass estimation procedure was used for the interpolation. The maximum number of samples per drill hole was set to control the number of drill holes in the interpolation.

For Pass 1, the minimum number of samples and the maximum samples per drill hole for interpolation were designed to ensure that the nearest samples are accorded the highest weighting and that a minimum of the three closest holes are used in the interpolation.

For Pass 2, the maximum number of samples per drill hole was designed to ensure a minimum of two drill holes in the interpolation, to go beyond the limits of Pass 1.

For Pass 3, the minimum number of samples and the maximum number of samples per drill hole allowed the bigger ellipse to interpolate grades into the remaining blocks not covered by Passes 1 and 2.

The search parameters and assumptions are presented in Table 14.43.

Table 14.43: Summary of Search Parameters - Kailey Deposit

Pass	X	Y	Z	Min. S	Max. S	Max. S/DH
1	60	60	12	6	12	2
2	120	120	24	4	12	2
3	240	240	60	3	15	2

Notes: Min = minimum; Max = maximum; S = samples; DH = drill hole

14.2.3.5 Grade Interpolation and Validation

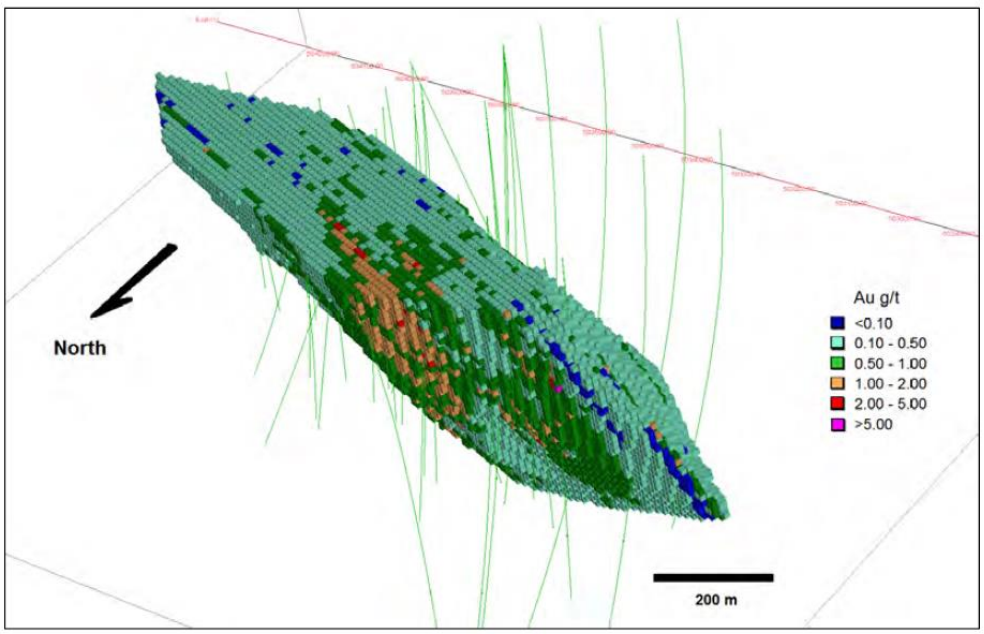
Grade interpolation for gold was performed using the Ordinary Kriging function of the GEMS mining software. Grade smearing was minimized by restricting the influence of values exceeding the RG shown in Table 14.40. A summary of the parameters and assumptions used for the interpolation is presented in Table 14.44.

Table 14.44: Grade Interpolation Parameters / Assumptions - Kailey Deposit

Parameter	Assumption
Date of Data Used	31-Oct-12
Number of Drill Holes	59
Specific Gravity (SG)	2.87
Block Model and Interpolation Software	GEMS
Interpolation Method	OK
Block Sizes (X, Y, Z) m	10 x 10 x 10
Restricted Search Radius (X, Y, Z) m	12 x 3 x 12

The representative resource block model showing distribution of gold grades is shown in Figure 14.33

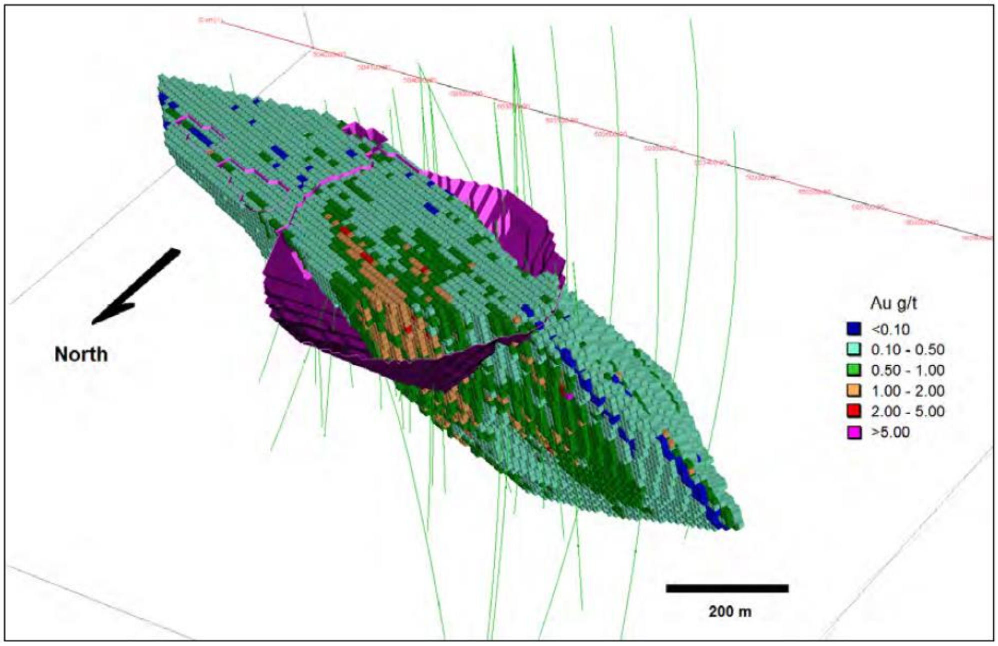
Figure 14.33: Kailey Deposit - Gold Grade Distribution in Block Model



14.2.3.6 <u>Determination of Mineral Resources (Open Pit Shell vs. Underground)</u>

The resource block model was examined for open pit and underground economic potential at various cut-off grades using the same methodology and economic parameters as that adopted for the Brookbank project - see Subsection 14.2.1.12 above. The resulting pit is shown in Figure 14.34.

Figure 14.34: Kailey Deposit Pit Layout



14.2.3.7 Resource Categorization

Micon has classified resource blocks in the block model based largely upon the drilling density and the pass criteria described in Subsection 14.2.3.4, while also accounting for variography results and geological structures.

The Measured Mineral Resource category was assigned to the coherent portions of the deposit covered by Pass 1 of the search ellipsoid, excluding islands or sporadic small volumes. Adequacy of sample/drill hole coverage was confirmed visually.

The Indicated Mineral Resource category was assigned to coherent portions of the deposit covered by Pass 2 of the search ellipsoid, including islands of Pass 1. Good visual evidence of adequate sample/drill hole coverage was also considered.

The Inferred Mineral Resource category was assigned to Pass 3 areas, including islands of Pass 2. These areas have very limited drill hole information.

The estimated resources at various cut-off grades are presented in Table 14.45 and the resource block model (coloured by classification) is shown in Figure 14.35. The tonnes and ounces have been rounded to the nearest 1,000.

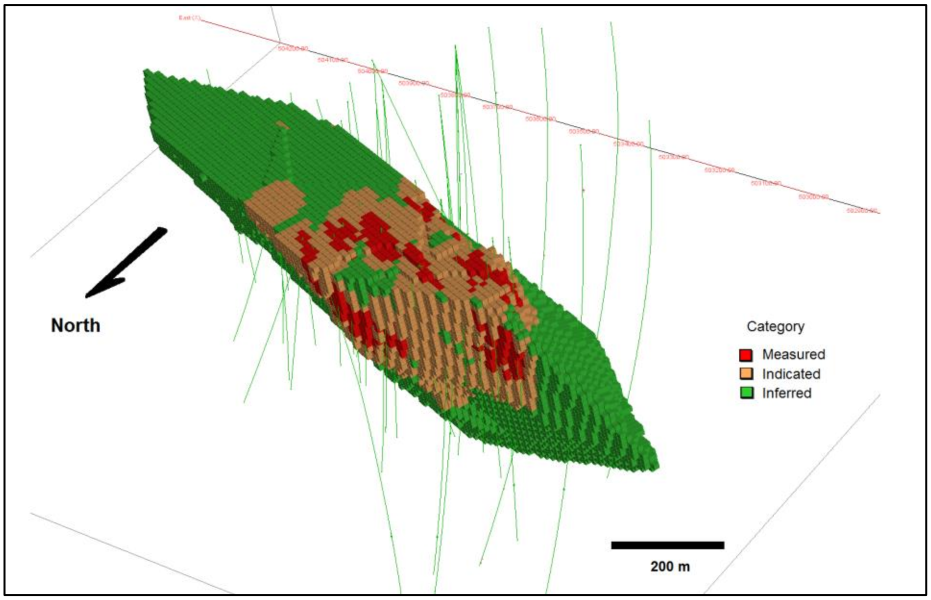
Table 14.45: Kailey Deposit In-pit Mineral Resources

Resource Category	Cut-off Grade (g Au/t)	Tonnes	Avg. Grade (g Au/t)	Contained Gold (oz)
Measured	>3.5	11,000	4.96	2,000
	3.0	11,000	4.96	2,000
	2.8	11,000	4.96	2,000
	2.5	20,000	3.95	3,000
	2.0	138,000	2.45	11,000
	1.5	617,000	1.86	37,000
	1.0	1,980,000	1.42	91,000
	0.83	2,566,000	1.31	108,000
	0.5	**4,052,000**	**1.06**	**139,000**
	No Cut-off	6,059,000	0.81	158,000
Indicated	>3.5	3,000	5.06	-
	3.0	3,000	5.06	-
	2.8	3,000	5.06	-
	2.5	3,000	5.06	-
	2.0	20,000	2.48	2,000
	1.5	232,000	1.73	13,000
	1.0	1,197,000	1.28	49,000
	0.83	1,998,000	1.14	73,000
	0.5	**4,578,000**	**0.86**	**126,000**
	NC	8,289,000	0.62	165,000
Total M & I	**0.5**	**8,630,000**	**0.95**	**265,000**
Inferred	>3.5	-		-
	3.0	-		-
	2.8	3,000	2.88	-
	2.5	9,000	2.68	1,000
	2.0	60,000	2.21	4,000
	1.5	405,000	1.76	23,000
	1.0	1,398,000	1.37	62,000
	0.83	1,992,000	1.23	79,000
	0.5	**3,688,000**	**0.97**	**115,000**
	No Cut-off	6,868,000	0.67	148,000

Notes:

1. CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.
2. Totals may not add correctly due to rounding.
3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.
4. *The effective date of the estimate is December 31, 2012.*

Figure 14.35: Kailey Deposit - Block Model Resource Categories



14.2.3.8 Mineral Resource Statement / Summary

The Kailey deposit Mineral Resources are summarized in Table 14.46 at cut-off grades of 0.5 g Au/t and 2.8 g Au/t for open pit and underground resources, respectively. The Mineral Resources estimated conform to the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves, as required by NI 43-101.

Table 14.46: Summary of Kailey Mineral Resources as at December 31, 2012

Deposit	Cut-off Category	Mineral Resource Category	Tonnes (Mt)	Gold Grade (g Au/t)	Gold Ounces (000's)
Kailey Deposit	Open Pit	Measured (M)	4.052	1.06	139
		Indicated (I)	4.578	0.86	126
		Subtotal M & I	**8.630**	**0.95**	**265**
		Inferred	3.688	0.97	115
	Underground	Measured (M)	0.000		0
		Indicated (I)	0.000		0
		Subtotal M & I	**0.000**	**0**	**0**
		Inferred			0

Notes:

1. CIM Definition Standards of November 27, 2010, were followed for mineral resource estimation.
2. Totals may not add correctly due to rounding.
3. Mineral resources, which are not mineral reserves, do not have demonstrated economic viability.
4. *The effective date of the estimate is December 31, 2012.*

Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability. Environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues may materially affect the estimate of Mineral Resources.

14.2.4 Brookbank, Key Lake and Kailey Resources Discussion

Micon considers that the resource estimates for the Brookbank, Key Lake and Kailey deposits have been reasonably prepared and conform to the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The process of Mineral Resource estimation includes technical information which requires subsequent calculations or estimates to derive subtotals, totals and weighted averages. Such calculations or estimations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, Micon does not consider them to be material.

Mineral Resources which are not Mineral Reserves do not have demonstrated economic viability. The estimate of Mineral Resources may be materially affected by environmental, permitting, legal, title, taxation, socio-political, marketing or other relevant issues.

The Mineral Resources estimated will always be sensitive and vulnerable to fluctuations in the price of gold. Other than this, Micon believes that at present there are no known environmental, permitting, legal, title, taxation, socio-economic, marketing or political issues which could adversely affect the Mineral Resources estimated above.

There has been insufficient exploration to define the Inferred Mineral Resource as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in upgrading them to an Indicated or Measured Mineral Resource category.

15. MINERAL RESERVE ESTIMATES

15.1 Summary

The Mineral Reserve for the Hardrock Project is estimated at 141.7Mt an average grade of 1.02 g Au/t for 4.65M ounces of gold as summarized in Table 15.1. The Mineral Reserve estimate was prepared by G Mining Services Inc. (“GMS”). The resource block model was also generated by GMS.

The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies. The Mineral Reserve estimate stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources. There are only Indicated Mineral Resources and no Measured Mineral Resources. Therefore, all of the Mineral Reserve classifies as Probable Mineral Reserves. The Inferred Mineral Resources contained within the mine design are classified as waste.

Table 15.1: Hardrock Open Pit Mineral Reserve Estimate

Category	Diluted Ore Tonnage (kt)	Gold Grade (g Au/t)	Contained Gold (koz Au)
Proven	-	-	-
Probable	141,715	1.02	4,647
Total P&P	**141,715**	**1.02**	**4,647**

Notes:

1. *CIM definitions were followed for Mineral Reserves*
2. *Effective date of the estimate is October 1, 2016*
3. *Mineral Reserves are estimated at a cut-off grade of 0.33 g Au/t*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30.*
5. *A minimum mining width of 5 m was used*
6. *Bulk density of ore is variable but averages 2.83 t/m^3*
7. *The average strip ratio is 3.87:1*
8. *Dilution factor is 17.3%*
9. *Numbers may not add due to rounding*

15.2 Resource Block Model

The block model consists of four folders with block percent attributes for overburden, tailings, historical underground openings and intact rock mass. The historical underground openings have been modelled and depleted in the block model with backfill densities assigned for stopes backfilled with sand or rock. Some tailings overlay the pit footprint and have been modelled to allow for their tracking and management in the material movement plan.

15.3 Pit Optimization

Open pit optimization was conducted to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using the Whittle software which is based on the Lerchs-Grossmann algorithm. The method works on a block model of the ore body, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes all the parameters used to calculate block values in Whittle.

For this Report Measured and Indicated Mineral Resource blocks were considered for optimization purposes and for mineable resource calculations. However, sensitivities were run using the complete resource.

15.3.1 Pit Slope Geotechnical Assessment

Golder was mandated to produce a feasibility level pit slope design study to support the mine designs. The conclusions of this study have been used as an input to the pit optimization and design process.

The Golder scope included reviewing geotechnical field investigations carried out by mine design engineering, carrying out follow-up field investigations and providing feasibility level slope designs for the open pit.

It has been assessed that the open pit will be developed in a good to very good rock mass where rock mass failure is not a concern. Historical underground long wall mining has proven the quality of the rock mass. The mineralization is found in upright sub-vertical axial planes that trend roughly east-west. The fold axes are shallowly west-plunging.

Rock mass failure has not been identified as a major concern. Rather, potential instability will involve structural controls, the most significant being the foliation control on the bench face angle and the potential control of flat sets on the bench crest back-break angles. No major faults have been identified that will adversely daylight on the final pit walls. The locations of the underground workings and whether they are filled or unfilled are well understood. Risks to safety and pit access due to these stopes must be mitigated through design and planning at all stages of the Project.

While there are localized differences in the orientations of the discontinuity populations, they do not justify distinctly different slope designs. The slope configuration options are presented in Table 15.2. Double benching will have to be done with vertical pre-split, no sub-grade drilling and well controlled blasting practices are required.

The final pit was designed using a double benching configuration to a final height of 20 m. The pit slope profile is based on recommendations by Golder as presented in Section 15.3.1. The slope profile is based on vertical batter angles with a 10 m catch bench width for an inter-ramp angle of 63.4 degrees. A 16 m geotechnical berm is introduced every 100 m, where ramp segments do not pass in the slope to reduce the vertical stack height.

At the bedrock-overburden contact, a 10 m catch bench is introduced and the overburden is sloped at a 2H:1V angle. The overburden slopes will be comprised of fluvial or glacial cohesionless or cohesive material of sufficient strength. On the east side of the pit the overburden thickness averages 15 m with a maximum depth of 25 m. On the north side, the average depth is approximately 10 m with a maximum of 30 m when including the historical MacLeod Mine tailings.

As reported by Golder, the rock mass is assumed to have a very low permeability. It is also unknown at which rate the historical underground workings were filled with water. The water table is observed to be close to surface in fenced glory holes. For slope stability assessments, it has been assumed that slopes will be partially saturated with drawdown cones similar to another open pit in the region.

Table 15.2: Hardrock Final Wall Geotechnical Recommendations

Slope Parameters	
Final Bench Height (m)	20.0
Bench Face Angle (^{0})	90
Avg. Design Catch Bench Width (m)	10.0
Inter-ramp Angle (^{0})	63.4
Overall Slope Angle (^{0})	60.8
Geotechnical benches (m)	16.0[1]

Note 1: Geotechnical Catch berm will be of 16.0 m at every 100 m

15.3.2 Mining Dilution and Ore Loss

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade ("COG"). The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining. The dilution skin consists of 0.75 m of material in a north-south direction (across strike) and 1.0 m in an east-west direction (along strike). The dilution is therefore specific to the geometry of the ore body and the number of contacts between ore and waste.

Each mineralized block in the resource model is assigned a dilution case which corresponds to the number of dilution contacts for the block. There are nine possible cases. Isolated blocks tagged as Case 8 were treated as an ore loss and were excluded from the optimization process. This dilution estimate was based on evaluations at a COG of 0.33 g Au/t. The estimate evaluates a dilution skin around blocks with a minimum COG of 0.33 g Au/t.

15.3.3 Pit Optimization Parameters

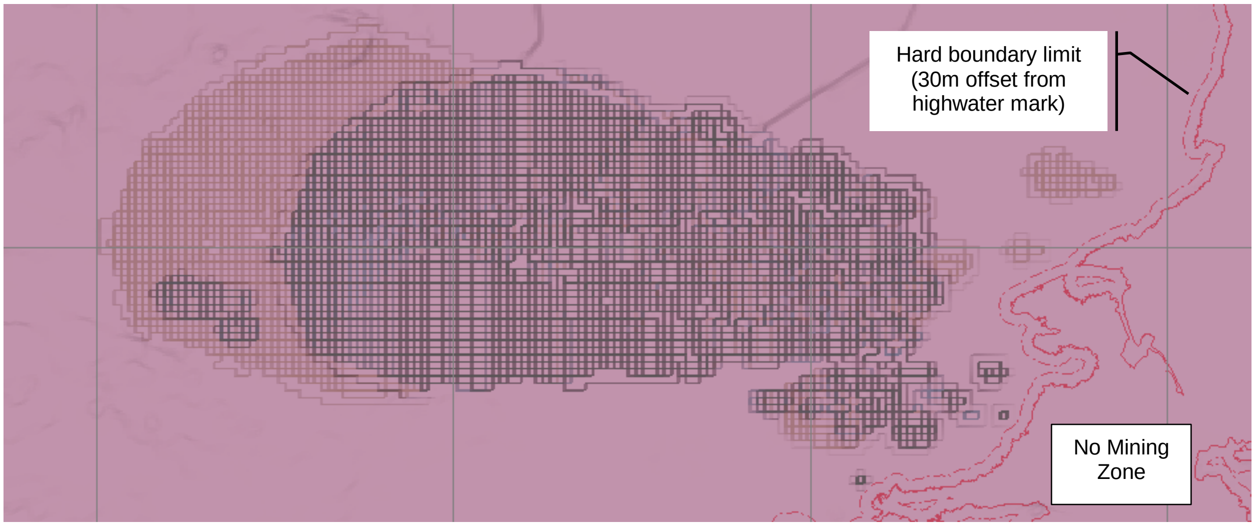
A summary of the pit optimization parameters is presented in Table 15.3 for a milling rate of 27 kt/d based on a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30.

The gold selling cost includes a 3% royalty fee plus a transport and refining cost of CAD 4.00/oz. The cost parameters were estimated based on first principles. The total ore based cost is estimated at CAD 9.60/t which includes processing, general and administration costs and a sustaining capital provision.

Unit reference mining costs are used for a "reference mining block" usually located near the pit crest or surface and are incremented with depth which corresponds to the additional cycle time and thus hauling cost. The reference mining cost is estimated at CAD 1.80/t with an incremental depth factor of CAD 0.03/t per 10 m bench.

A physical hard boundary was imposed in the optimization process to prevent the pit from encroaching into nearby lakes (Figure 15.1). The hard boundary was established to maintain a 30 m buffer zone between the pit and the lake high water limit which corresponds to the 330 m level contour.

Figure 15.1: Pit Limit Hard Boundary Constraint



MINING SERVICES INC.	**Pit Limit Hard Boundary**	**Author: LP Gignac** **Date: Aug 2016**

The overall slope angles utilized in Whittle are based on the inter-ramp angles recommended in the Golder pit slope study with provisions for ramps and geotechnical berms. The overall slope angle in competent rock is 55 degrees based on a designed inter-ramp angle of 63.4 degrees. The overall slope angle in overburden is 26 degrees.

15.3.4 Cut-Off Grades

The cut-off grade resulting from the optimization parameters is estimated at 0.24 g Au/t which assumes an average metallurgical recovery of 90% and an average mining dilution of 14%.

The cut-off grade is the breakeven grade where revenue equals costs to carry the full operation while excluding direct mining costs:

$$COG(g/t) = \frac{Cp + Ca + Cr + Com + Csibc + Cmc}{r*(P - Cs)}$$

Where:

r is the metallurgical recovery (%)

P is the gold price in CAD/oz

Cs is the cost of selling gold (refining and royalties) in CAD/oz

Cp is the total Processing Costs (Fixed and Variable) in CAD/t treated

Ca is Administration and General cost in CAD/t treated

Cr is the cost of rehandle in CAD/t treated

Com is the difference between ore and waste mining cost in CAD/t treated

Csibc is Non-mining sustaining capital in CAD/t treated over life of mine

Cmc is Mine Closure cost incurred during the life of mine in CAD/t treated

Table 15.3: Optimization Parameters

Hardrock Pit Optimization Parameters		
Nominal Milling Rate	t/d	27,000
Plant Throughput	kt/y	9,855
Exchange Rate	CAD/USD	1.30
Diesel Fuel Price Delivered	CAD/litre	0.80
Natural Gas Price	CAD/GJ	4.95
Electricity Cost	CAD/kWh	0.055
Gold Price	USD/oz	1250
Gold Price (local currency)	CAD/oz	1625
Transport and Refining Cost	CAD/oz	4.00
Royalty Rate	%	3.0%
Metallurgical Recovery at Cut-Off Grade	%	90%
Total Processing Cost	CAD/t milled	7.46
Re-handling	CAD/t milled	0.12
General and Administration	CAD/t milled	1.42
Rehabilitation and Closure	CAD/t milled	-
Sustaining Capital	CAD/t milled	0.60
Total Ore-based Cost	CAD/t milled	**9.60**
Marginal Cut-Off Grade	g Au/t	0.24
Mining Rate	kt/y	56,000
Mining Dilution	%	14.0%
Mining Loss	%	3.0%
Total Mining Reference Cost	CAD/t mined	**1.80**
Incr. Bench Cost (CAD /10 m bench)	CAD/10 m bench	0.030
Overall Slope Angle in Fresh Rock	degrees	55
Overall Slope Angle in Overburden	degrees	26

15.3.5 Open Pit Optimization Results

The Whittle nested shell results are presented in Table 15.4 using only the Measured and Indicated Mineral Resource. The nested shells are generated by using revenue factors to scale up and down from the base case selling price.

Table 15.4: Measured and Indicated Mineral Resource Whittle Shell Results

Pit Shell	Rev. Factor	Price USD/oz	Total (kt)	Ore (kt)	Strip Ratio	Metal koz Au	Grade g Au/t
10	0.4000	500	489,161	61,857	6.91	3,050	1.53
11	0.4167	521	537,013	70,152	6.65	3,331	1.48
12	0.4333	542	577,588	77,596	6.44	3,557	1.43
13	0.4500	562	639,454	86,933	6.36	3,867	1.38
14	0.4667	583	656,008	91,602	6.16	3,967	1.35
15	0.4833	604	685,654	97,441	6.04	4,119	1.31
16	0.5000	625	709,496	102,824	5.90	4,247	1.28
17	0.5167	646	712,013	106,188	5.71	4,297	1.26
18	0.5333	667	743,472	113,029	5.58	4,454	1.23
19	0.5500	687	751,189	117,059	5.42	4,521	1.20
20	0.5667	708	790,045	123,458	5.40	4,676	1.18
21	0.5833	729	842,336	133,429	5.31	4,889	1.14
22	0.6000	750	972,078	145,800	5.67	5,267	1.12
23	0.6167	771	1,011,721	152,551	5.63	5,413	1.10
24	0.6333	792	1,019,394	157,198	5.48	5,480	1.08
25	0.6500	812	1,030,700	161,881	5.37	5,550	1.07
26	0.6667	833	1,050,070	167,241	5.28	5,632	1.05
27	0.6833	854	1,231,162	179,379	5.86	6,076	1.05
28	0.7000	875	1,260,602	183,894	5.86	6,165	1.04
29	0.7167	896	1,274,878	185,765	5.86	6,205	1.04
30	0.7333	917	1,278,478	186,327	5.86	6,217	1.04
31	0.7500	937	1,288,945	187,394	5.88	6,244	1.04
32	0.7667	958	1,294,720	187,934	5.89	6,258	1.04
33	0.7833	979	1,296,813	188,256	5.89	6,264	1.04
34	0.8000	1,000	1,310,901	189,025	5.94	6,293	1.04
35	0.8167	1,021	1,341,410	190,882	6.03	6,344	1.03
36	0.8333	1,042	1,346,707	191,318	6.04	6,354	1.03
37	0.8500	1,062	1,531,189	197,346	6.76	6,669	1.05
38	0.8667	1,083	1,557,013	198,719	6.84	6,721	1.05
39	0.8833	1,104	1,560,991	199,127	6.84	6,730	1.05
40	0.9000	1,125	1,573,956	199,978	6.87	6,752	1.05
41	0.9167	1,146	1,577,869	200,261	6.88	6,760	1.05
42	0.9333	1,167	1,583,724	200,765	6.89	6,770	1.05
43	0.9500	1,187	1,588,280	200,975	6.90	6,777	1.05
44	0.9667	1,208	1,590,584	201,220	6.90	6,782	1.05
45	0.9833	1,229	1,598,193	201,644	6.93	6,794	1.05
46	1.0000	1,250	1,603,975	202,059	6.94	6,805	1.05

The shell selection is presented in Table 15.5. Pit shell 19 is selected as the optimum final pit shell which corresponds to a USD 687/oz pit shell (Revenue Factor 0.55) which is a conservative selection that minimizes the strip ratio. This shell has a total tonnage of 751M t including 141.6M t of ore at an average grade of 1.05 g Au/t for 4.79M in-situ ounces of gold. The average strip ratio is 4.3:1. This is the smallest shell that achieves close to maximum value using a practical phasing approach.

Table 15.5: Measured and Indicated Mineral Resource Pit Shell Selection

Shell Selection	
Shell Number	19
Shell RF	0.550
Shell Price (USD/oz)	687
Total Tonnage (kt)	751,189
Waste Tonnage (kt)	609,619
Strip Ratio (W:O)	4.31
Ore Tonnage (kt)	141,570
Grade (g Au/t)	1.05
In-situ Gold (koz)	4,792

15.4 Mine Design

15.4.1 Underground Voids

The presence of underground stopes was considered when designing the pits mainly for the void in the F-Zone which is 150 m high and 30 m wide. Most of the other underground openings are backfilled with sand fill or rock fill.

Three permanent accesses have been maintained in the West Wall in order to have accesses where the F-Zone stope intersects the pit wall. An access is established above the stope (level 50), near the middle (level -30) and below at the bottom of the pit, in order to do wall maintenance.

The C-Zone underground stope, which is backfilled, was considered by avoiding intersecting it with the final ramp. To achieve this, the ramp circles around the opening at the 240 and 130 levels.

15.4.2 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being a 181-tonne class haul truck with a canopy width of 7.6 m. For double lane traffic, industry best-practice is to design a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.7 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned on the highwall will capture run-off from the pit wall surface and assure proper drainage of the running surface. The ditch will be 1.2 m wide. To facilitate drainage of the roadway a 2% cross slope on the ramp is planned.

The double lane ramp width is 28.5 m wide and the single lane ramp is 16.6 m wide. Single lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced.

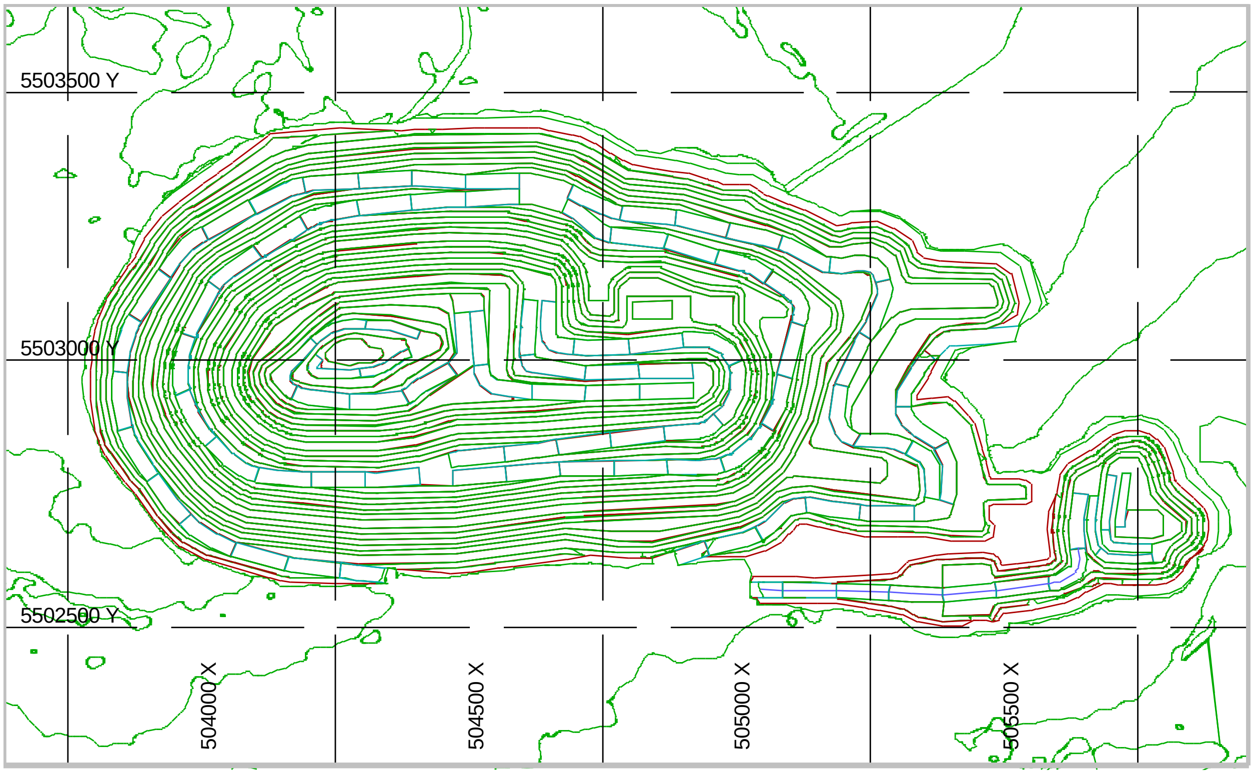
15.4.3 Open Pit Mine Design Results

The final pit design is presented in Figure 15.2. The final pit is 1,800 m along strike and 875 m wide and reaches a depth of 570 m. The final pit design has three exits; two to the east and one to the west, to provide access to the pushbacks and to shorten haul distances to the crusher and waste dumps. The west ramp system connects with the east ramp system at a plateau at a depth of 160 m (on level 180). The ramp system introduces several switchbacks in several instances to avoid ramps passing through underground openings.

The west wall is steepest to access ore at depth. The ore is located mostly in the F-Zone. The pit is shallower on the east side as the mineralization plunges to the West.

There is a satellite pit to the east that is separated from the main pit. This satellite pit is 125 m deep and is limited to the east by the high-level water mark of the Kenogamisis Lake.

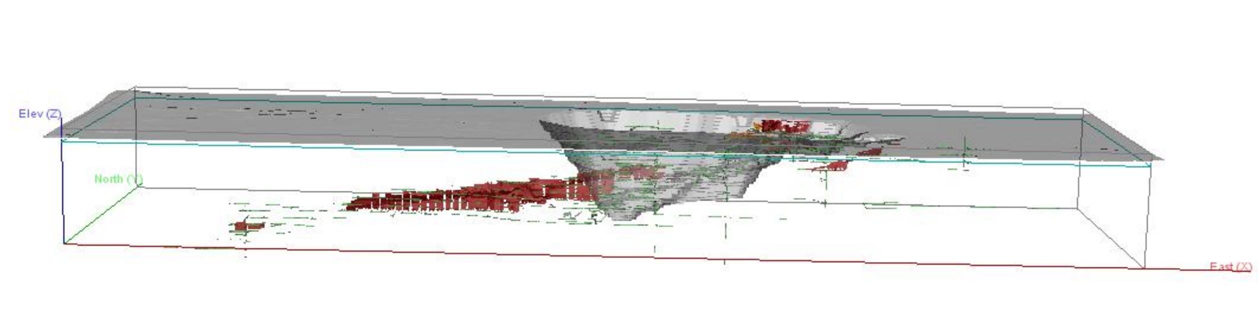
Figure 15.2: Final Pit Design



G MINING SERVICES INC.	**Final Pit Design**	**Author: LP Gignac** **Date: Aug 2016**

A 3D view and longitudinal section is presented in Figure 15.3. Several of the underground voids will be entirely mined by the final open pit but certain voids will remain in the wall such as those from the F-Zone at depth.

Figure 15.3: 3D View of Final Open Pit with Historical Underground Voids



G MINING SERVICES INC.	**3D View of Final Pit with Historical Underground Voids**	**Author: LP Gignac** **Date: Aug 2016**

15.5 Mineral Reserve Statement

The Mineral Reserve and stripping estimates are based on the final pit design presented in the previous section.

The Proven and Probable Mineral Reserves are inclusive of mining dilution and ore loss. The total ore tonnage before dilution and ore loss is estimated at 122.5M t at an average grade of 1.16 g Au/t for 4,575 koz. Isolated ore blocks are treated as an ore loss and represent 1.76M t or 1.4% in terms of ore tonnage. The dilution envelope around the remaining ore blocks (>0.33 g Au/t) results in a dilution tonnage of 20.9M t at an average grade of 0.15 g Au/t for 100 koz. The dilution tonnage represents 17.3% of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15.6 presents a Resource to Reserve reconciliation.

Table 15.6: Resource to Reserve Reconciliation

Resource to Reserve Reconciliation	Tonnage (kt)	Grade (g Au/t)	Contained gold (koz)
Ore before ore loss and dilution	122,527	1.16	4,575
Less: Ore loss (isolated blocks)	1,758	0.49	28
Ore before mining dilution	120,769	1.17	4,547
Add: Mining dilution	20,945	0.15	100
Proven & Probable Mineral Reserve	141,715	1.02	4,647

The Proven and Probable Mineral Reserves total 141.7M t at an average grade of 1.02 g Au/t for 4,647 k in-situ ounces of gold. The total tonnage to be mined is estimated at 690.7M t for an average strip ratio of 3.87 which includes overburden, historical tailings and underground backfill (Table 15.7).

Table 15.7: Hardrock Open Pit Mineral Reserves and Quantities

Final Pit Quantities		
Proven and Probable Mineral Reserve		
Proven Ore Tonnage	kt	-
Gold Grade	g Au/t	-
Contained Gold	koz	-
Probable Ore Tonnage	kt	141,715
Gold Grade	g Au/t	1.02
Contained Gold	koz	4,647
Proven & Probable Ore Tonnage	kt	141,715
Gold Grade	g Au/t	1.02
Contained Gold	koz	4,647
Waste Material (including Inferred)		
Overburden	kt	17,801
Historical Tailings	kt	2,137
UG Backfill	kt	1,324
Waste Tonnage	kt	527,674
Total Tonnage	**kt**	**690,650**

Notes:

1. *CIM definitions were followed for Mineral Reserves*
2. *Effective date of the estimate is October 1, 2016*
3. *Mineral Reserves are estimated at a cut-off grade of 0.33 g Au/t*
4. *Mineral Reserves are estimated using a long-term gold price of USD 1,250/oz and an exchange rate of CAD/USD 1.30.*
5. *A minimum mining width of 5 m was used*
6. *Bulk density of ore is variable but averages 2.83 t/m^3*
7. *The average strip ratio is 3.87:1*
8. *Dilution factor is 17.3%*
9. *Numbers may not add due to rounding*

16. MINING METHODS

16.1 Introduction

The Project consists of developing an open pit that will mine through the historical underground workings of the MacLeod-Cockshutt and Hard Rock mines. Furthermore, the proposed open pit location is bisected by the Trans-Canada Highway 11 and requires a new by-pass and the relocation of various surface infrastructures.

16.2 Mine Designs

16.2.1 Open Pit Phases

Mining of the Hardrock main pit will occur in four phases (including a borrow pit) and a single phase for the smaller satellite pit to the East. The content of each mining phase is summarized in Table 16.1. The objective of pit phasing is to improve the economics of the Project by feeding the highest grade during the earlier years and/or delaying waste stripping until later years. With the mineralization plunging westward, the pit phases progressively expand to the west.

Table 16.1: Pit Phase Design Summary

Phase Design Content		Borrow Pit	Phase 1	Phase 2	Phase 3	Total Main Pit	SAT 1	Total All Pits
Total Tonnage	kt	44,736	221,087	149,705	259,883	**675,411**	15,239	**690,650**
Overburden	kt	4,253	7,031	1,841	1,798	**14,923**	2,877	**17,801**
Tailings	kt	0	1,181	640	90	**1,911**	226	**2,137**
UG Backfill	kt	98	656	183	387	**1,324**	0	**1,324**
Waste Rock	kt	27,173	169,154	119,317	201,528	**517,172**	10,501	**527,674**
Diluted Ore	kt	13,212	43,065	27,723	56,081	**140,080**	1,634	**141,715**
Diluted Grade	g Au/t	1.01	1.09	0.91	1.02	**1.02**	1.03	**1.02**
In-situ Gold	koz	428	1,507	812	1,847	**4,593**	54	**4,647**
Strip Ratio	W:O	2.39	4.13	4.40	3.63	**3.82**	8.32	**3.87**
% of Gold	%	9.2%	32.4%	17.5%	39.7%	**98.8%**	1.2%	**100.0%**

The phase designs introduce different geotechnical slope profiles for temporary pit walls. The temporary wall slope profile allows for wider catch benches to allow for overbank hazard management on pit walls. Overbank hazard results from muck from one phase spilling down the slope of the previous pit phase, filling the catch benches. This creates an increased rockfall hazard for workers and equipment at the bottom of the previous pit phase. The temporary wall design allows the catch bench to be accessed to remove debris. The maximum double bench inter-ramp angle for temporary walls is 52 degrees. Table 16.2 presents the different slopes.

Table 16.2: Pit Phase Design Criteria

Slope Parameters	Temporary Pit Walls	Final Pit Walls
Final Bench Height (m)	20.0	20.0
Bench Face Angle (º)	90	90
Avg. Design Catch Bench Width (m)	15.5	10.0
Inter-ramp Angle (º)	52.2	63.4
Overall Slope Angle (º)	50.1	60.8
Geotechnical Benches (m)	16.0[1]	16.0[1]

Note 1: Geotechnical catch berm will be of 16.0 m at every 100 m

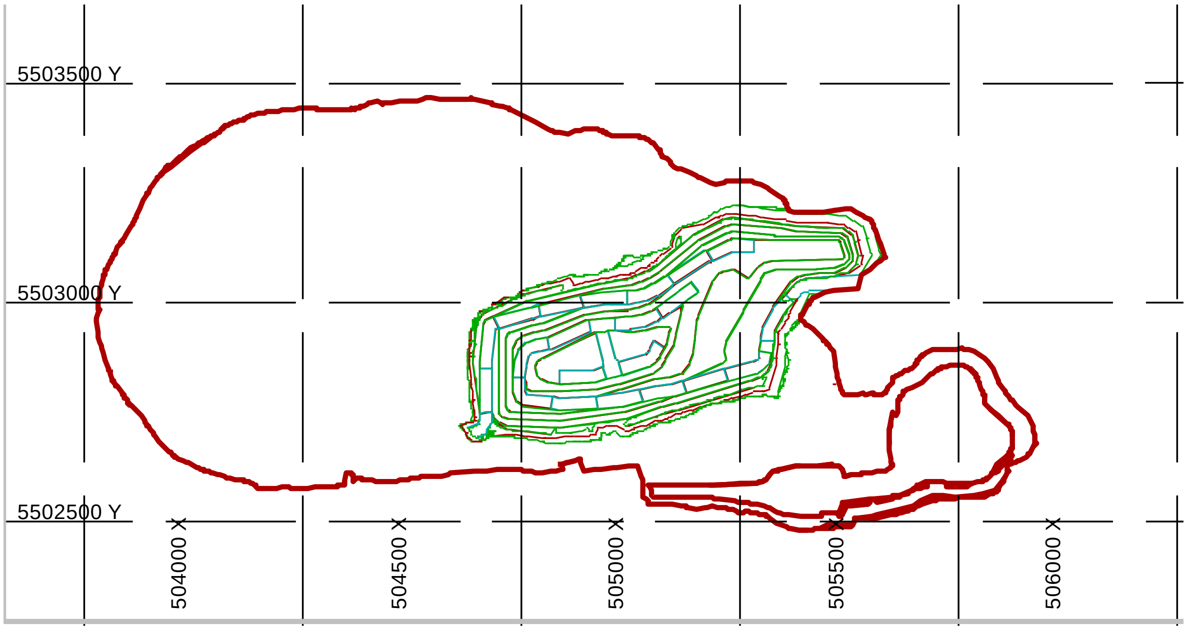
The borrow pit phase is designed to avoid various surface constraints such as the Trans-Canada Highway 11 and to avoid mining historical tailings during pre-production. Mining during the pre-production period is concentrated in the borrow pit which provides for a 60 m buffer with the Trans-Canada Highway 11. The highway will be relocated to the north during the initial construction period. This borrow pit phase reduces risk with respect to the timing of the highway relocation.

In the first phase of mining, the mining is done to the final pit limit on the eat side. Two ramp exits are established to shorten the ore and waste hauls. The Phase 1 pit is 1,500 m long by 800 m wide and reaches a depth of 280 m.

For Phase 2, an intermediate ramp is built on the west wall which will be completely mined with the final phase. The east ramp is kept on the east wall, where possible, to take advantage of the westward plunge of the mineralization. The west ramp, built for the early stripping activities, connects to the final ramp system at a depth of 160 m on level 180 on the north pit wall. The Phase 2 pit is 1,700 m long by 875 m wide and reaches a depth of 370 m.

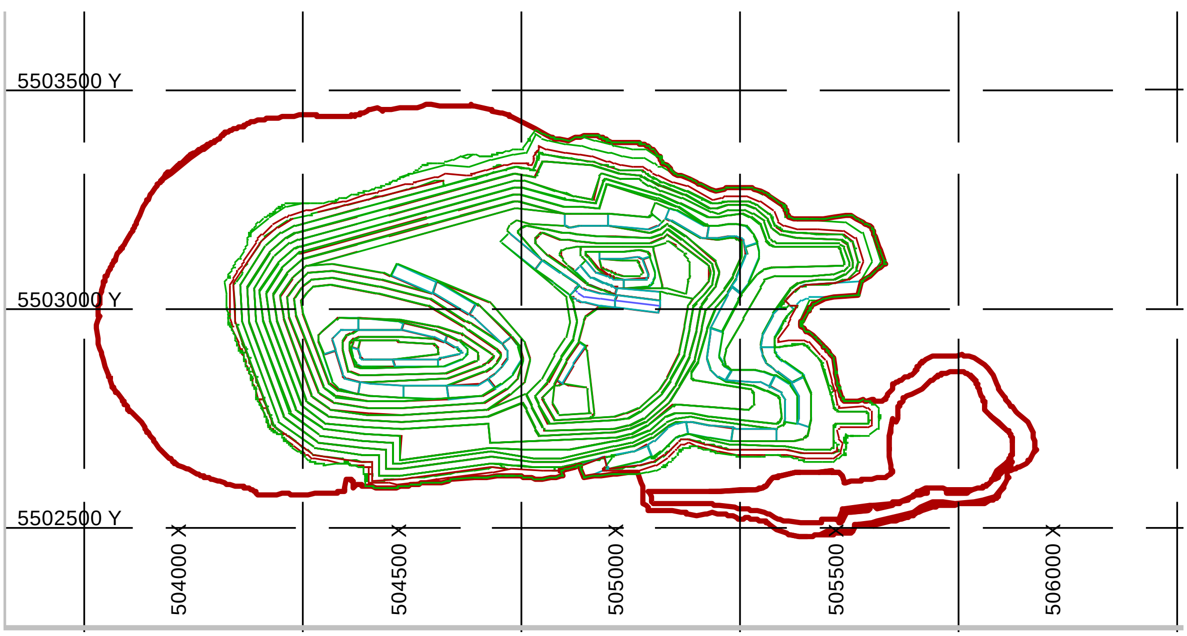
The satellite pit is less economic compared to the main pit phases with a strip ratio of 8.32 compared to a strip ratio 3.82 for the main pit as a whole.

Figure 16.1: Borrow Pit Phase Design



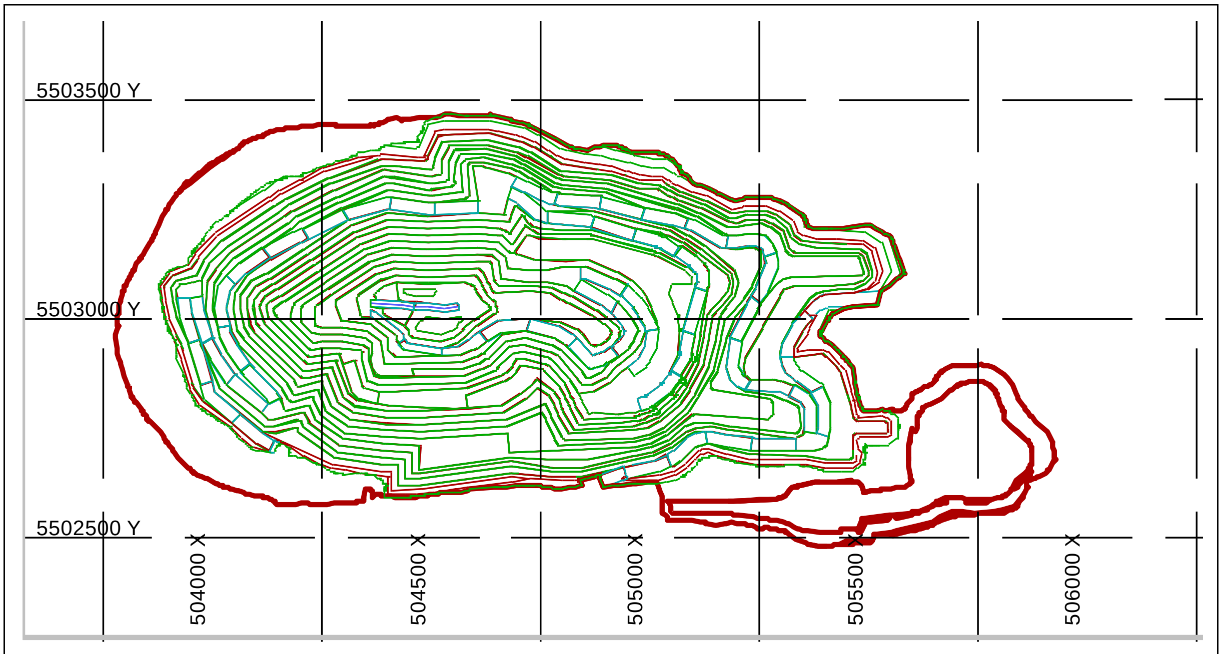
MINING SERVICES INC.	**Borrow Pit Phase Design**	**Source: LP Gignac** **Date: Aug. 2016**

Figure 16.2: Phase 1 Design



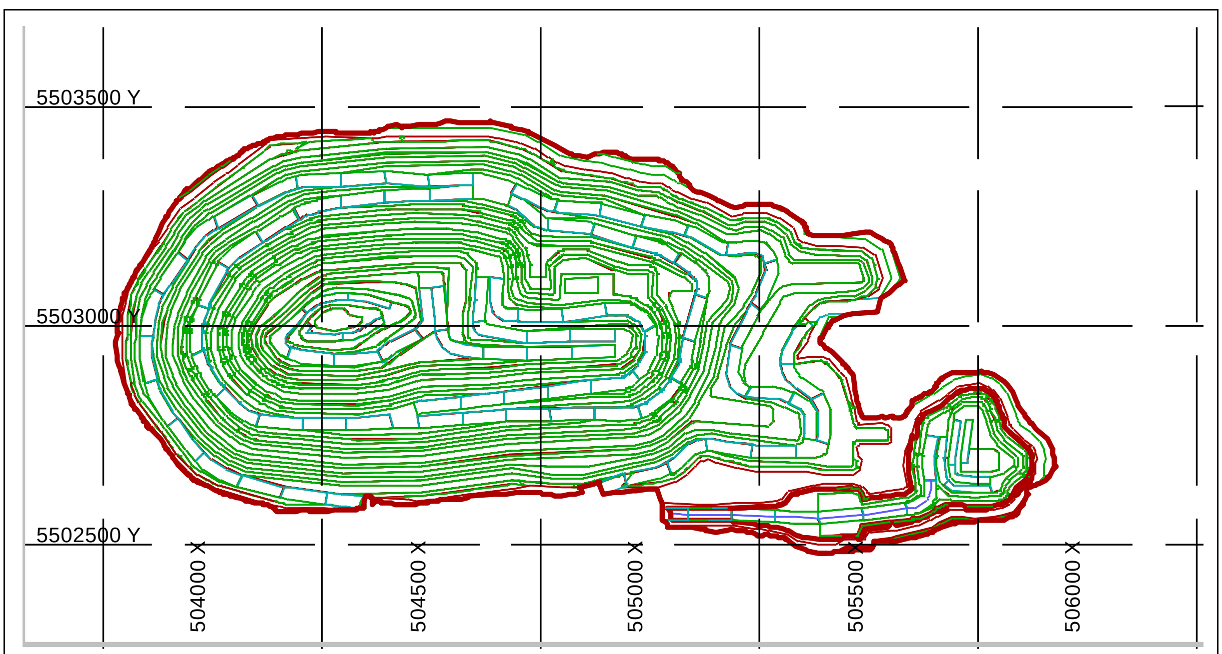
MINING SERVICES INC.	**Phase 1 Pit Design**	**Source: LP Gignac** **Date: Aug. 2016**

Figure 16.3: Phase 2 Design



MINING SERVICES INC.	Phase 2 Pit Design	Source: LP Gignac Date: Aug. 2016

Figure 16.4: Phase 3 and Satellite Design



MINING SERVICES INC.	Final (Phase 3) Pit Design & Satellite Pit	Source: LP Gignac Date: Aug. 2016

16.2.2 Overburden and Waste Rock Storage

Waste rock will be disposed of in four distinct waste rock storage areas ("WRSA") of which three are located around the pit and one further to the south. The open pit generates 527.7Mt of waste rock, 1.3Mt of backfill, 2.14Mt of historical tailings and 17.8Mt of overburden that require storage. The tailings material will be transported for disposal within the TMF.

The design criteria of each waste dump has been adjusted based on foundation stability assessments by Amec. Amec has recommended various waste dump design profiles which are shallower than the typical 2:1 in certain specific areas as presented in Table 16.4 to assure adequate safety factors. All waste dumps have 20 m high lifts to allow for wider catch benches to facilitate reclamation. All waste dump capacities are shown in Table 16.3.

Table 16.3: Waste Storage Capacities

Waste Dump	Capacity (Mt)	Capacity (Mm^3)	Surface Area (ha)	% Filled
Construction	39.1	18.2	n/a	100%
Waste Dump A	43.6	20.5	50.2	100%
Waste Dump B	12.9	6.1	22.5	100%
Waste Dump C	114.3	53.8	115.2	98%
Waste Dump D	276.8	130.4	183.2	91%
In Pit Dumping + A extension	73.5	34.6	52.3	100%
Overburden Pile	27.7	16.2	57.6	69%
Total	**587.9**	**279.9**	**481.1**	**94%**

Table 16.4: Waste Pile Design Criteria

Waste Dump	Avg. Catch Bench Width (m)	Pile Face Angle (deg)	Overall Slope angle (H:V)	Maximum Elevation (m)	Approximate Height (m)
Waste Dump A	18.0	37	2:1 / 2.5:1/ 4:1	435	100
Waste Dump B	18.0	37	2:1 / 3:1	395	60
Waste Dump C	16.0	37	2:1 / 3:1	430	85
Waste Dump D	18.5	37	2:1 / 2.5:1/ 3:1	440	105
In Pit Dumping + A extension	14.0	37	2:1	435	200*
Overburden Pile	36.0	37	2:1 / 2.5:1/ 3:1 / 4:1	405	65

Note: (*) height is from the bottom of the pit

16.2.3 Ore Stockpiles

The low grade ("LG") stockpile is designed to store a maximum of 20.8M t of ore, the medium grade ("MG") stockpile will have a maximum capacity for 4.1M t and the high grade ("HG") stockpile will reach 3.9M t in Year 1, just before commercial production. These maximum stockpile levels are not reached at the same time; the HG & MG stockpiles will be reclaimed faster.

The stockpile pad has been designed to connect to the crusher pad, thus decreasing cycle time for ore rehandling when the stockpile is higher or at the same height as the crusher pad. The ore stockpile capacity is greater than required allowing for additional capacity in the future or the ore stockpile capacity could be reduced to add space for waste rock, if required.

The stockpile design criteria are presented in Table 16.5. The capacity found below is for one stockpile grade for all the ore. There is adequate space to have multiple grade stockpiles on the designated ore stockpile pad.

Table 16.5: Stockpile Design Criteria

Ore Stockpiles	Catch Bench Width (m)	Overall Slope Angle (H:V)	Maximum Elevation (m)	Approximate Height (m)	Max Capacity (Mt)
Stockpile Max Capacity	7	2:1	410	70	28.2

16.2.4 Mine Haul Roads

The haul roads, from the pit to the dumps, the crusher and the tailing storage facility, will be constructed mostly during the construction period. However, some haul roads will be constructed during operations as the pit evolves. Over the LOM a total of 5.2km mine haul roads will be constructed. In addition, the TMF access road is 3.8 km in length and will accommodate mine trucks.

16.3 Production Schedule

The mine production schedule is completed on a monthly basis during the pre-production period and first three months of commercial production. The first full year of operations is developed on a quarterly basis and on an annual basis thereafter. The mine pre-production is initiated in Year -1 and transitions to commercial operations in Year 1 after commissioning and achieving 60% of nameplate capacity for a period of 30 days. The mine pre-production period lasts a total of 17 months which is planned to allow for a gradual assembly and commissioning of mining equipment, training and timely delivery of waste rock for civil works.

The objectives of the LOM plan are to maximize discounted operating cash flow of the Project, subject to various constraints:

- Limit mining during pre-production within the borrow pit phase;
- Supply best grade ore to the plant and feed to a nominal capacity which ultimately reaches 27,000 t/d (9.86Mt/y);
- Limit the mining rate to approximately 68.2Mt/y;
- Limit the vertical drop down rate to approximately 7 benches, per phase, per year;
- Limit peak truck requirements;
- Utilize a grade segregation and stockpiling strategy with a maximum stockpile of 21Mt which is roughly equivalent to two years of processing.

The mining schedule pre-production tonnage is 42.2Mt over a period of 17 months. Mining will be conducted on day shift only for a period of three months and on two shifts by the 4th month. The remaining fleet is commissioned two months prior to commercial production. The peak mining rate of approximately 68Mt is maintained for four years (Year 2 to Year 5) and then gradually declines as either sufficient ore for the process plant is available or to limit peak truck requirements. The low-grade ore is stockpiled and reaches 20.8Mt by Year 12. The annual mine production, stockpile inventory, process plant production and gold production are presented in Figure 16.5 to Figure 16.8. The end of period mine infrastructure status at different dates are presented in Figure 16.9 to Figure 16.12.

The operating strategy is to process at a finer grind size of P80 of 72µm when the grade is high in the early years at a rate of 24 kt/d and to increase the throughput to the targeted rate of 27 kt/d by relaxing the grind size to P80 of 90µm. These operating regimes are expected to impact the gold recovery. Metallurgical recovery equations were established for these two regimes which are also impacted by the gold head grade (Au in g Au/t), sulfur (S in %) and arsenic (As in %) levels. With these recovery equations two recovered gold grade attributes were estimated for the two grind sizes.

The metallurgical recovery equations are as follows:

- Tails Grade (g Au/t) @ 72 µm = 0.001904 + 0.0733*(Head Grade g Au/t) + 0.301*(As)+0.0269*(S);
- Tails Grade (g Au/t) @ 90 µm = 0.01328+0.0733*(Head Grade g Au/t) + 0.301*(As) + 0.0269*(S);
- Metallurgical Recovery = 1 – (Tails Grade / Head Grade);
- Recovered Gold Grade = Metallurgical Recovery x Head Grade (g Au/t).

Figure 16.5: Annual Mine Production

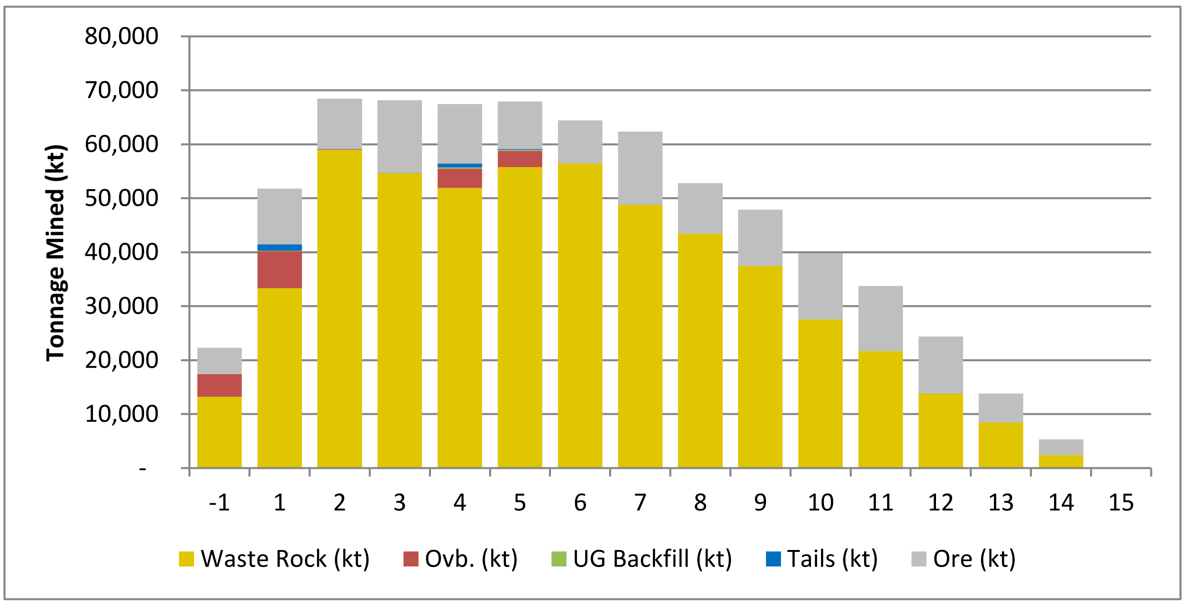


Figure 16.6: Annual Stockpile Inventory

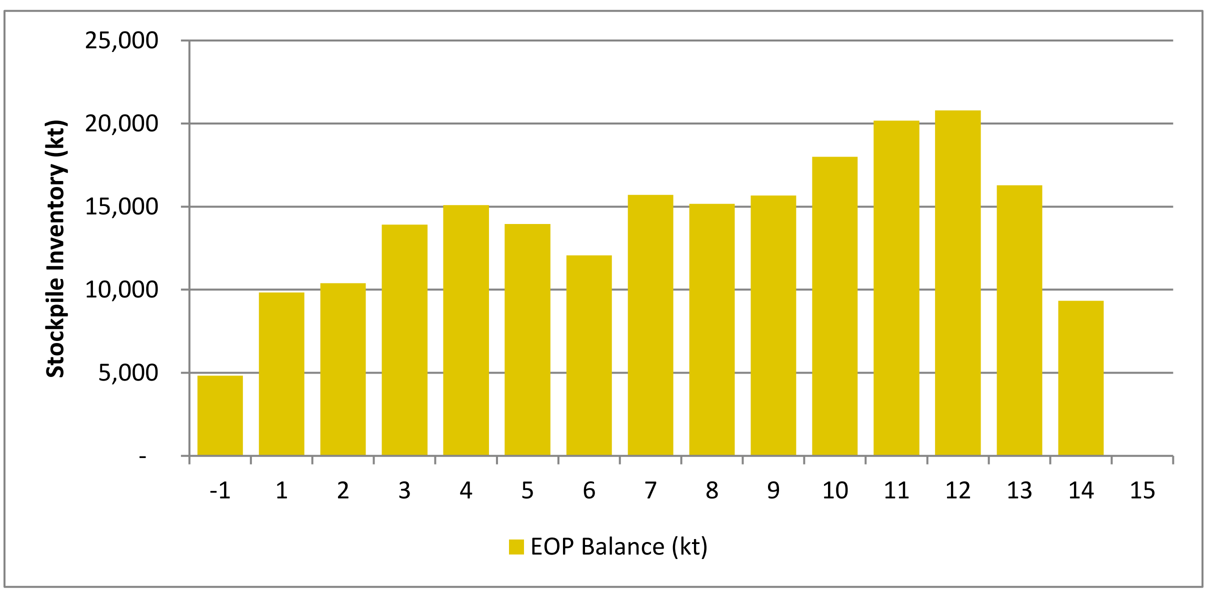


Figure 16.7: Annual Mill Production

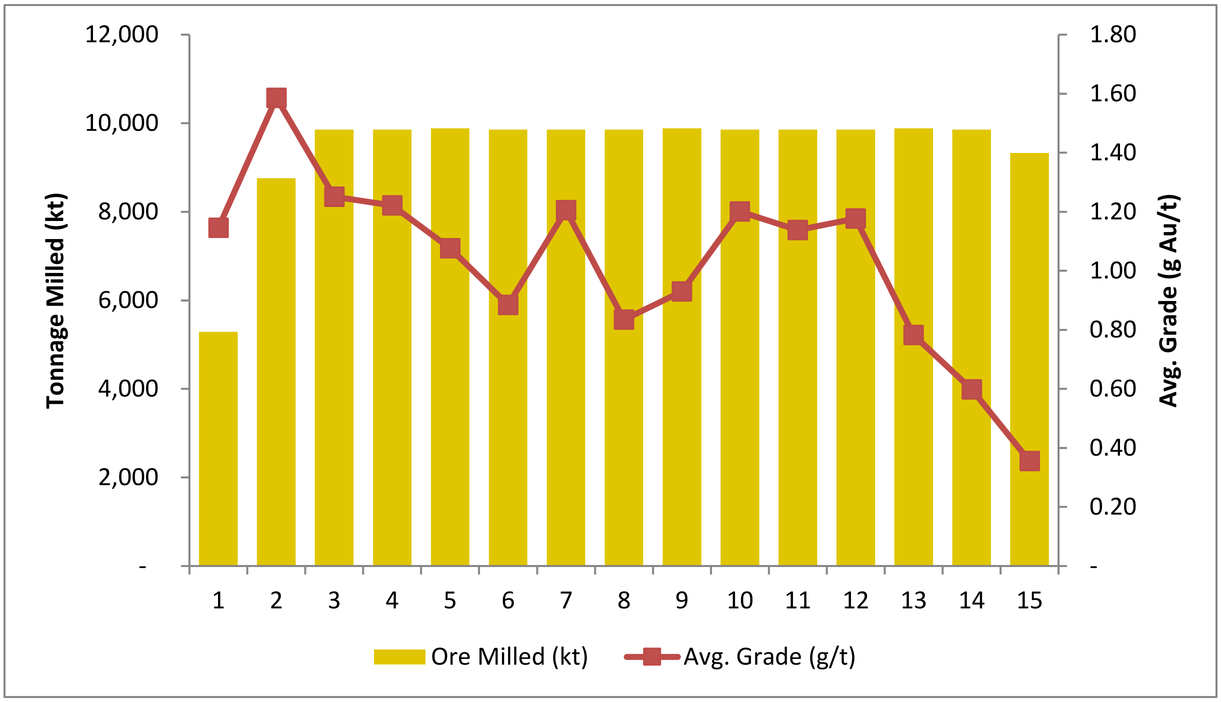


Figure 16.8: Annual Gold Production

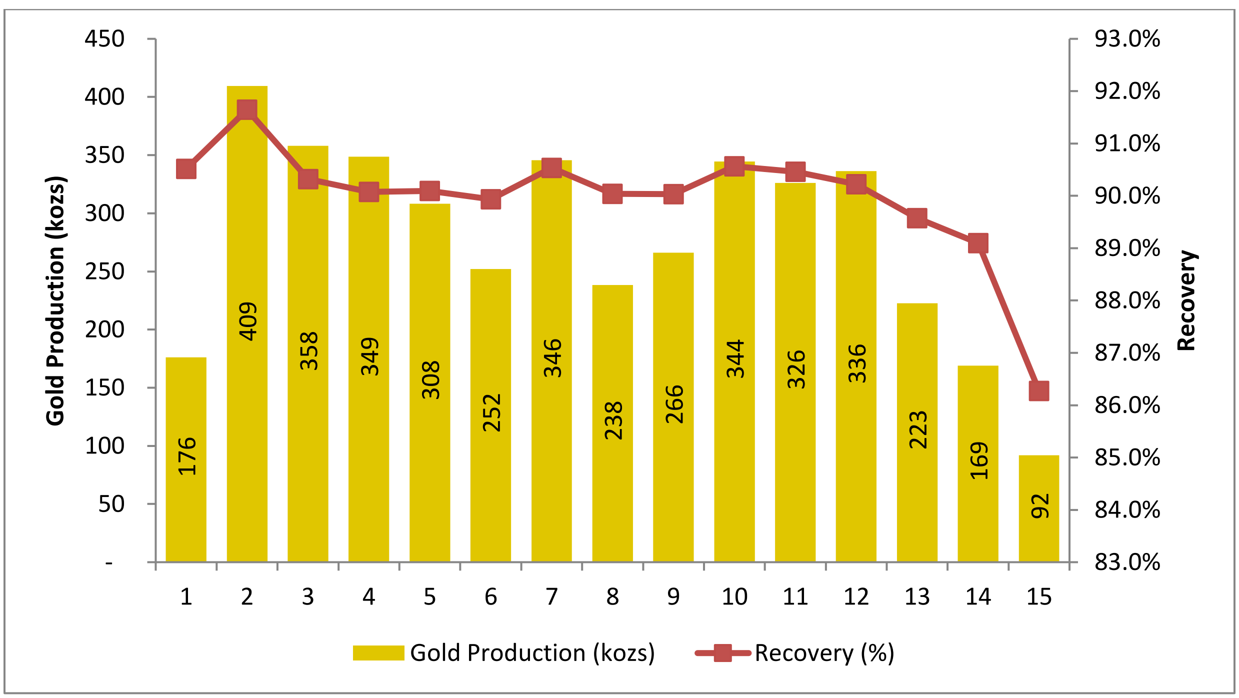
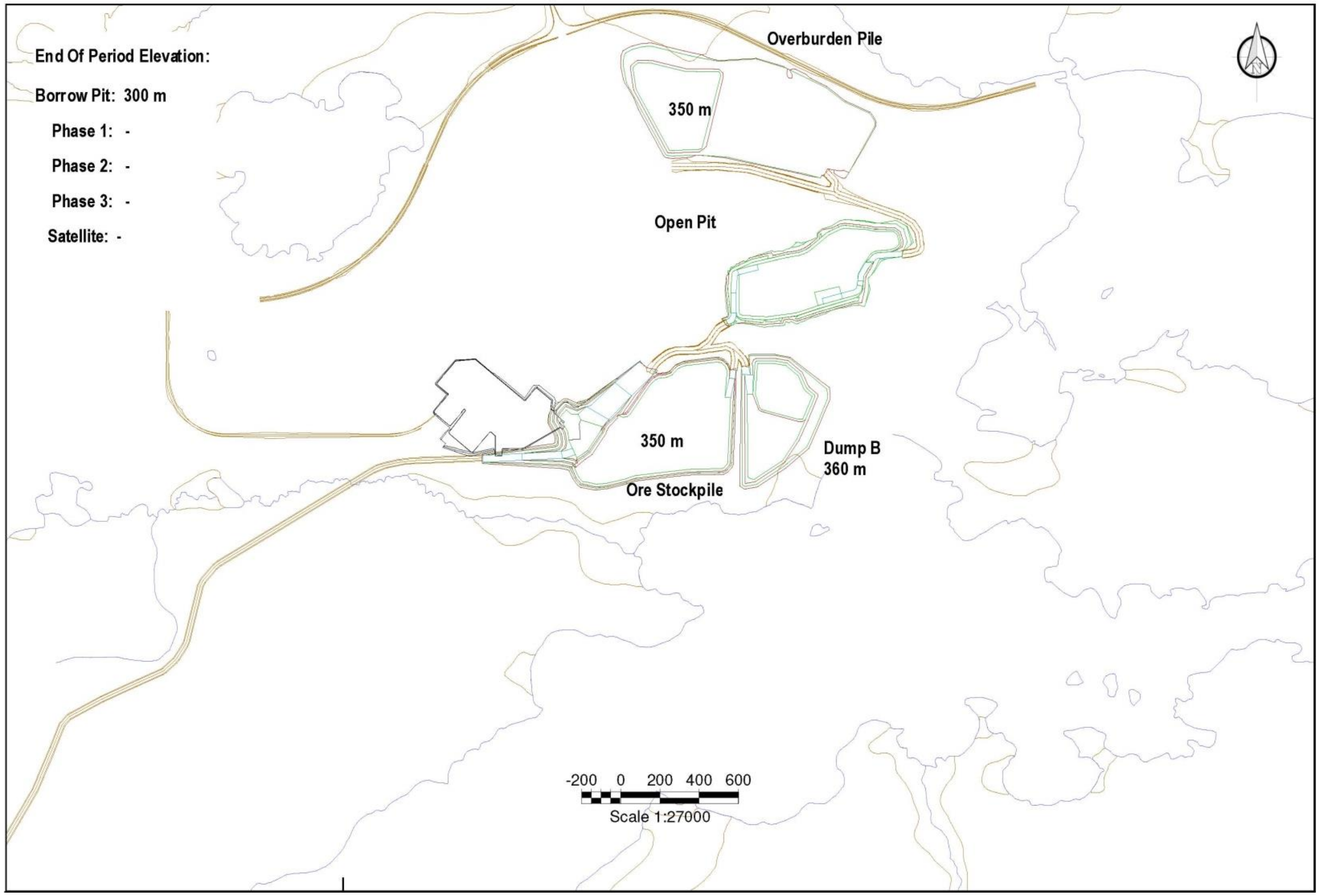
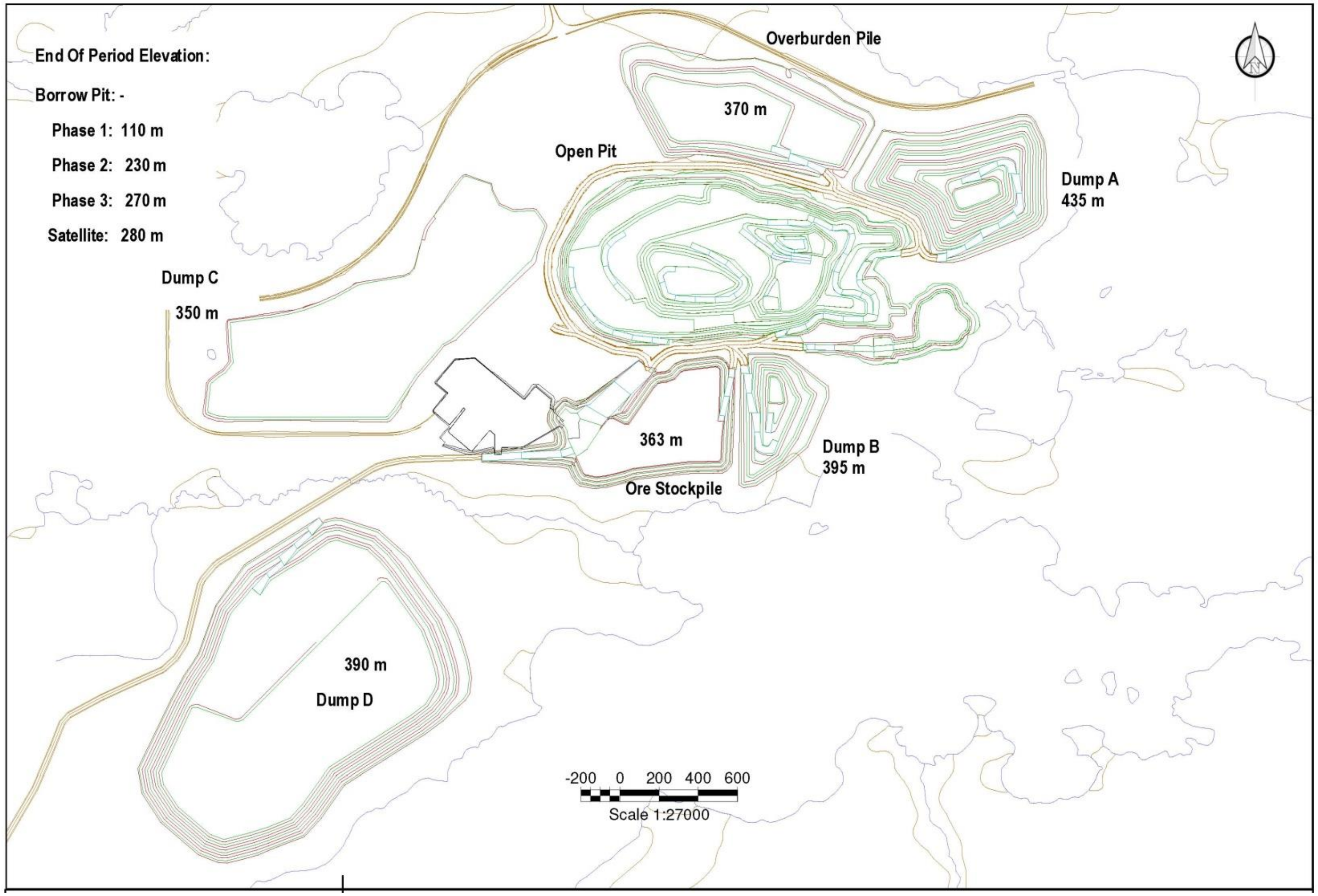


Figure 16.9: Production Schedule - Year 1



MINING SERVICES INC.	Pit Schedule – 2019 Year 1 - M12	Source: LP Gignac Date: Aug. 2016

Figure 16.10: Production Schedule - Year 5

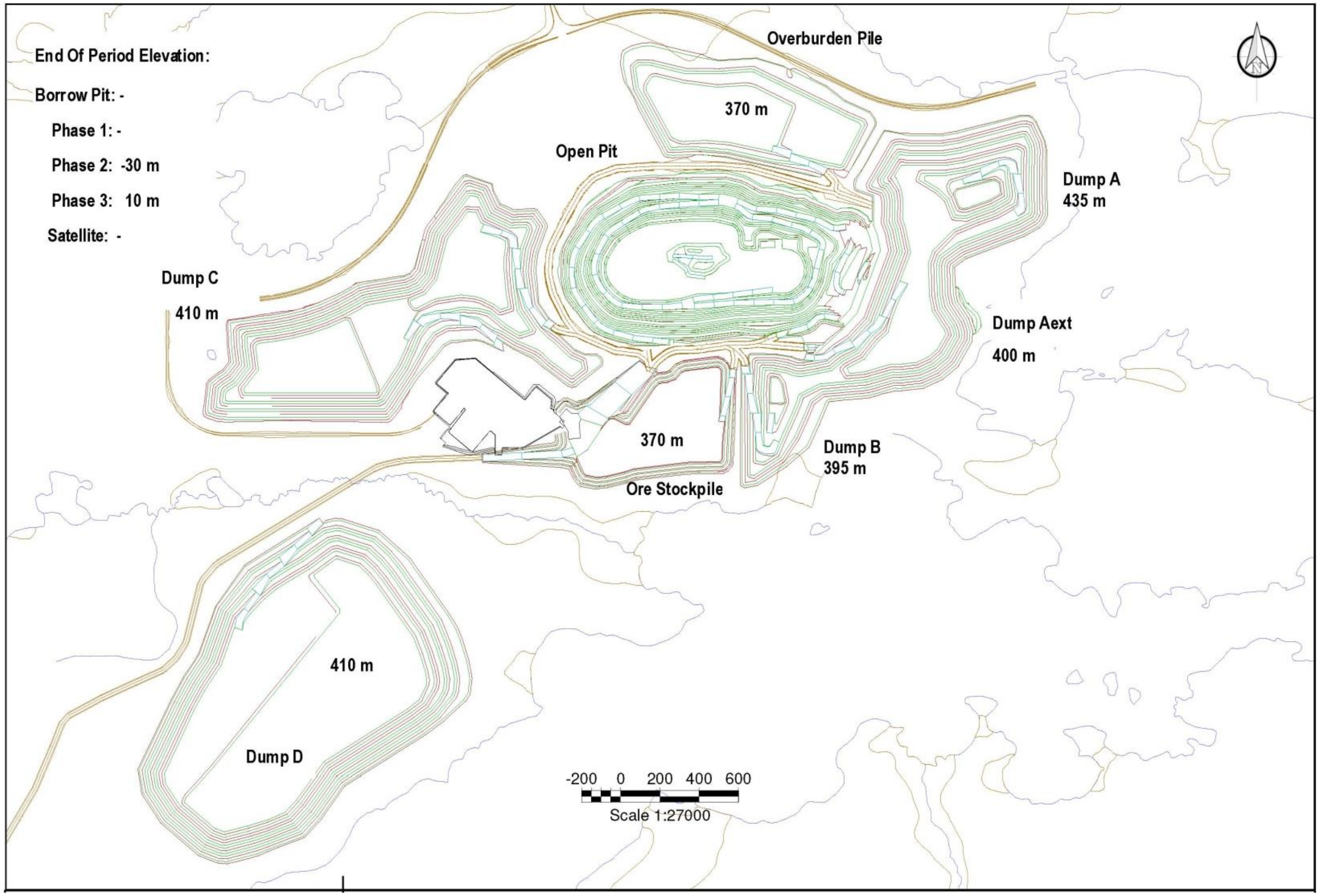


MINING SERVICES INC.

Pit Schedule – Year 5

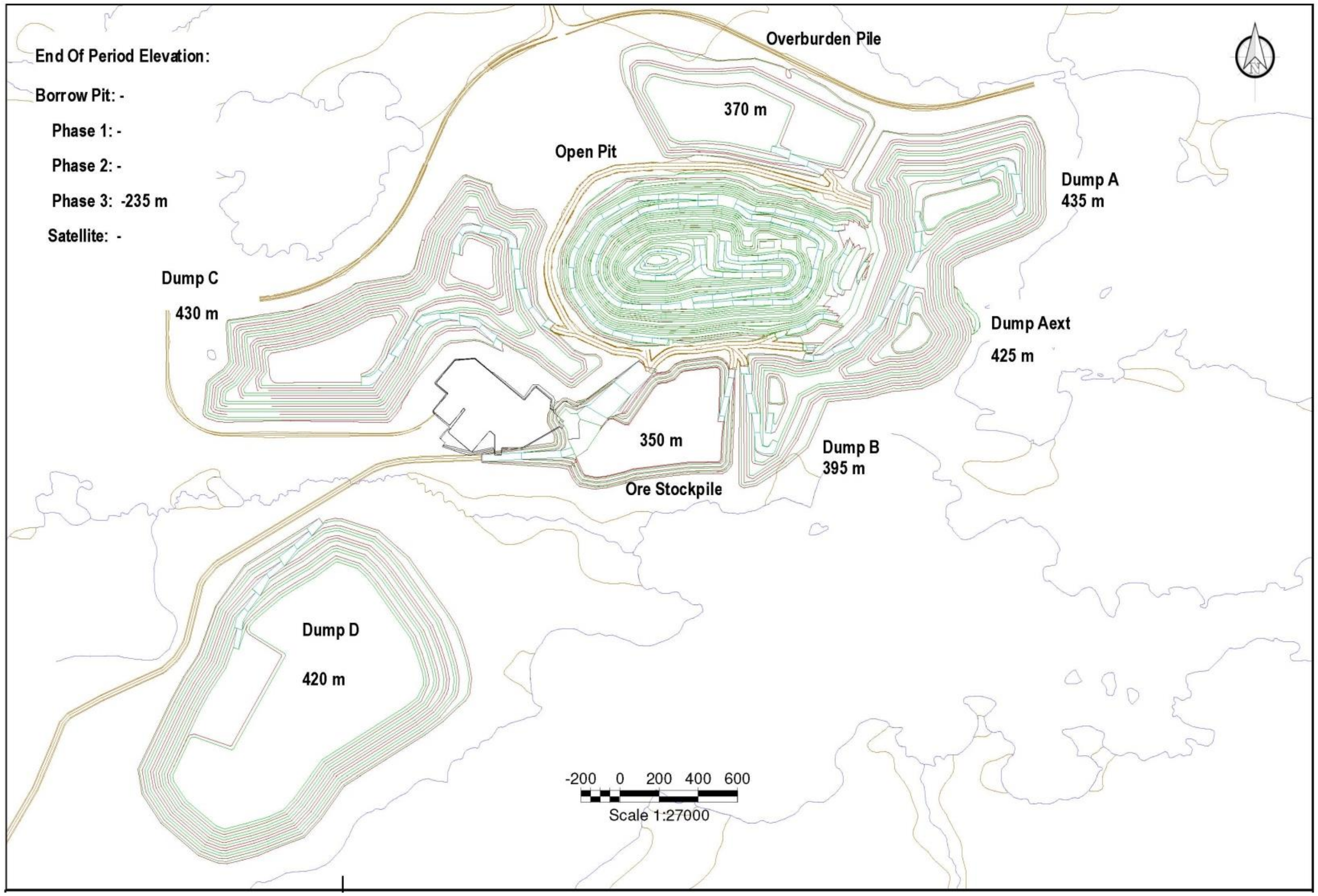
Source: LP Gignac
Date: Aug. 2016

Figure 16.11: Production Schedule - Year 10



MINING SERVICES INC.	**Pit Schedule – Year 10**	**Source: LP Gignac** **Date: Aug. 2016**

Figure 16.12: Production Schedule - Year 14



MINING SERVICES INC.	**Pit Schedule – Year 14**	**Source: LP Gignac** **Date: Aug. 2016**

The process plant production schedule is presented in Table 16.6. A processing rate of 24,000 t/d is planned for Year 1 and Year 2 and increases to 27,000 t/d from Year 3 onwards. The metallurgical recovery during the ramp up and commissioning period has been adjusted downwards from normal steady-state operating performance expectations.

Gold production averages 356 koz for the first four full years of production (Year 2 to Year 5) with an average head grade of 1.27 g Au/t and an average metallurgical recovery of 90.6%. Over the LOM 4.19 Moz of gold are produced.

Table 16.6: Life-of-Mine Production Schedule

Year	Mining					Processing			
	Ore Mined (Mt)	Grade (g Au/t)	Contained Gold (koz)	Waste Mined (Mt)	Total Mined (Mt)	Ore Milled (Mt)	Grade (g Au/t)	Contained Gold (koz)	Recovered. Gold (koz)
-1	4.83	1.07	166	17.47	22.31				
1	10.30	0.97	323	41.48	51.78	5.29	1.15	195	176
2	9.31	1.01	301	59.18	68.48	8.76	1.59	447	409
3	13.38	1.01	436	54.78	68.16	9.86	1.25	396	358
4	11.04	1.09	385	56.43	67.47	9.86	1.22	387	349
5	8.75	1.12	315	59.18	67.93	9.88	1.08	342	308
6	7.96	1.01	258	56.49	64.45	9.86	0.88	280	252
7	13.50	0.98	423	48.89	62.39	9.86	1.20	382	346
8	9.32	0.86	258	43.46	52.77	9.86	0.84	265	238
9	10.38	0.90	301	37.48	47.86	9.88	0.93	296	266
10	12.19	1.04	407	27.61	39.80	9.86	1.20	380	344
11	12.03	1.00	386	21.73	33.76	9.86	1.14	360	326
12	10.49	1.13	380	13.87	24.35	9.86	1.18	373	336
13	5.36	1.14	197	8.46	13.83	9.88	0.78	249	223
14	2.90	1.18	110	2.42	5.32	9.86	0.60	190	169
15						9.33	0.36	107	92
Total	**141.71**	**1.02**	**4,647**	**548.94**	**690.65**	**141.71**	**1.02**	**4,647**	**4,193**

16.4 Mine Operations and Equipment Selection

16.4.1 Mine Operations Approach

Mining is to be carried out using conventional open pit techniques with hydraulic shovels and mining trucks in a bulk mining approach with 10 m benches. An Owner mining open pit operation is planned with the outsourcing of certain support activities such as explosives manufacturing and blasting activities.

16.4.2 Production Drilling and Blasting

Drill and blast specifications are established to effectively single pass drill and blast a 10 m bench. For this bench height, a 203 mm blast hole size is proposed with a 6.0 m x 6.5 m pattern with 1 m of sub-drill. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.30 kg/t. Blast holes are initiated with NONEL® detonators and primed with 450 g boosters. The bulk emulsion product is a gas sensitized pumped emulsion blend specifically designed for use in wet blasting applications.

Several rock formations are present in the pit including greywacke, gabbro, porphyry and BIF. The average rock properties based on testing show a range in hardness between 80 and 175 MPa with a weighted average hardness estimated at about 100 MPa.

A drilling test was conducted on site on various outcrops in the pit. A total of ten 165 mm test holes for a total 100 m was completed in three of the main rock formations (Porphyry, Greywacke and Iron Formation). The penetration rates did not vary significantly between the formations with an instantaneous penetration rate of 40.2 m/h.

The drilling test results were used to calibrate expected instantaneous penetration rates for the larger 203 mm diameter production blast holes. The average drill productivity for the production rigs is estimated at 39.1 m/h instantaneous with an overall penetration rate of 21.5 m/h. The overall drilling factor represents time lost in the cycle when the rig is not drilling such as move time between holes, moves between patterns, drill bit changes, etc. The average drilling productivity is estimated at 2,127 t/h.

Table 16.7: Drill & Blast Parameters

Drill and Blast Parameters		Production Holes
Drill Pattern		
Explosive Type		Emulsion
Explosive Density	g/cm^3	1.2
Hole Diameter	in	8.0
Diameter (D)	m	0.203
Burden (B)	m	6
Spacing (S)	m	6.5
Subdrill (J)	m	1
Stemming (T)	m	2.5
Bench Height (H)	m	10
Blasthole Length (L)	m	11
Pattern Yield		
Rock Density	t/bcm	2.79
BCM/hole	bcm/hole	390
Yield per Hole	t/hole	1,088
Yield per Metre Drilled	t/m drilled	99
Powder Factor	kg/t	0.30
Weight of Explosives per Hole	kg/hole	331
Drill Productivity		
Re-drills	%	5%
Pure Penetration Rate	m/h	39.1
Hole Length	m	11
Overall Drilling Factor (%)	%	55%
Overall Penetration Rate	m/h	21.5
Drilling Productivity	t/h	2,127
Drilling Efficiency	holes/h	1.96

The blast hole rig selected for production drilling will have a hole size range of 152 mm to 270 mm with a single pass drill depth of 12.2 m with a 40 ft tower configuration. This rig will have both rotary and down-the-hole ("DTH") drilling capability. It is expected that DTH drilling mode will be most efficient.

Blasting activities will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The mine engineering department will be responsible for designing blast patterns and relaying hole information to the drills via the wireless network.

16.4.3 Ore Control

The ore control program will consist of establishing dig limits for low grade (COG to 0.50 g Au/t), medium grade (0.50 to 1.10 g Au/t), high grade (>1.10 g/t) and waste in the field to guide loading unit operators. A high precision system combined with an arm geometry system will allow shovels to target small dig blocks and perform selective mining. The system will give operators a real-time view of dig blocks, ore boundaries and other positioning information.

The ore control boundaries will be established by the technical services department based on grade control information obtained through blast hole sampling with post-blast boundaries adjusted for blast movement measurements made using a BMM® system. A blast movement monitoring system has been included in the blasting cost.

The blast hole samples collected will be sent to a nearby off-site laboratory for sample preparation and assaying. Blast hole samples will be collected on the bench and properly tagged by grade control technicians on each shift.

Concurrent to the blast hole sampling a RC drilling campaign is planned every year for an optimal ore-waste boundaries identification. The RC campaign will target 75% of all ore material and also capture around 11% of the total waste in the pit (mainly the contact zones with ore).

16.4.4 Pre-Split

Pre-split drill and blast is planned to maximize stable bench faces and to maximize inter-ramp angles along pit walls as prescribed by the geotechnical pit slope study by Golder. The pre-split consists of a row of closely-spaced holes along the design excavation limit of interim and final walls. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves. As a best-practice, it is recommended that operations restrict production blasts to within 50 m of an unblasted pre-shear line. Once the pre-split is shot, production blasts will be taken to within 10 m of the pre-shear and then a trim shot used to clean the face. Pre-split holes spaced 1.5 m apart will be 20 m in length and drilled with a smaller diameter of 127 mm (5 in.).

As presented in Table 16.8, blasting of the pre-split holes will use a special packaged pre-split explosive internally traced with 5 g/m detonating cord that ensures fast and complete detonation of the decoupled charge. For this specific application, a 41 mm diameter cartridge, 17 m long will be used which corresponds to a complete case of 25 kg. This load factor of 1.47 kg/m allows for a targeted charge weight of 0.83 kg/m^2 of face.

Table 16.8: Pre-Split Parameters

Pre-Split Parameters		Pre-Split Holes
Drill Pattern		
Hole Diameter	in	5
Diameter (D)	m	0.127
Spacing (S)	m	1.5
Bench Height (H)	m	20
Pre-Split Hole Length (L)	m	20
Face Area	m^2	30
Explosives Charge	kg	25
Charge Factor	kg/m^2 face	0.83
Cartridge Charge		
Number of Cartridges	qty	41
Cartridge Length	m	0.41
Cartridge Loading Factor	kg/m	1.47
Decoupled Charge Length	m	17
Decoupled Charge	kg	25
Drill Productivity		
Pure Penetration Rate	m/h	41.2
Overall Drilling Factor (%)	%	58%
Overall Penetration Rate	m/h	23.9
Drilling Efficiency	holes/h	1.2
Metres of Drilling per m Crest	m/m of crest	13.33

The drill selected for this application is more flexible type of rig capable of drilling angled holes for probe drilling and pit wall drain holes. The hole size range of this rig is between 110 mm and 203 mm with a maximum hole depth of 45 m.

16.4.5 Loading

The majority of the loading in the pit will be done by three hydraulic face shovels, two 26 m^3 and one 19 m^3. The shovels will be matched with a fleet of 181 t payload capacity mine trucks. The hydraulic shovels will be complemented by two production front-end wheel loaders ("FEL") with 21 m^3 buckets.

Although interchangeable, the hydraulic shovels will primarily be operating in ore and overburden while the wheel loaders will primarily be operating in waste.

The loading productivity assumptions for both types of loading tools in ore, waste and overburden are presented in Table 16.9.

The 26 m^3 shovel is expected to achieve a productivity of 3,402 t/h based on a three to four pass match with the mine trucks and an average load time of 2.3 minutes. The productivity in overburden will decrease at 2,952 t/h due to lower density of material and extra pass required for an average load time of 2.98 minutes. The 19 m^3 shovel, used for better selectivity, will have a 2,498 t/h production rate in ore and waste rock at a cycle time of 2.98 minutes, whereas in overburden the smaller shovel will reach a productivity of 2,167 t/h with a loading cycle time of 3.95 minutes.

The wheel loader is expected to achieve a productivity of 2,087 t/h based on a four to five pass match and an average load time of 3.7 minutes in ore and waste. The productivity in overburden is estimated at 1,906 t/h.

Table 16.9: Loading Specifications

Loading Unit		Shovel (26 m³)	Shovel (26 m³)	Shovel (19 m³)	Shovel (19 m³)	FEL (21 m³)	FEL (21 m³)
Haulage Unit		**Truck (181 t)**	**Truck (181 t)**	**Truck (181 t)**	**Truck (181 t)**	**Truck (181 t)**	**Truck (181 t)**
Material		**Ore/Wst**	**Ovb**	**Ore/Wst**	**Ovb**	**Ore/Wst**	**Ovb**
Rated Payload	t	181	181	181	181	181	181
Heaped Volume	m^3	108	108	108	108	108	108
Bucket Capacity	m^3	26.0	26.0	19.0	19.0	21.0	21.0
Bucket Fill Factor	%	90%	92%	90%	92%	85%	92%
In-Situ Dry Density	t/bcm	2.79	2.00	2.79	2.00	2.79	2.00
Moisture	%	3%	5%	3%	5%	3%	5%
Swell	%	30%	25%	30%	25%	30%	25%
Wet Loose Density	t/lcm	2.21	1.68	2.21	1.68	2.21	1.68
Actual Load Per Bucket	t	51.73	40.19	37.80	29.37	39.46	32.46
Passes (Decimal)	#	3.50	4.50	4.79	6.16	4.59	5.58
Passes (Whole)	#	3.50	4.50	4.50	6.00	4.50	5.50
Actual Truck Wet Payload	t	181	181	170	176	178	179
Actual Truck Dry Payload	t	176	172	165	168	172	170
Actual Heaped Volume	m^3	82	108	77	105	80	106
Payload Capacity		100%	100%	94%	97%	98%	99%
Heaped Capacity		76%	100%	71%	97%	74%	98%
Cycle Time							
Truck Exchange	min	0.60	0.60	0.60	0.60	0.70	0.70
First Bucket Dump	min	0.10	0.10	0.10	0.10	0.10	0.10
Average Cycle Time	min	0.65	0.65	0.65	0.65	0.83	0.83
Load Time	min	2.33	2.98	2.98	3.95	3.72	4.55
Cycle Efficiency	%	75%	85%	75%	85%	75%	85%
Number of Trucks Loaded per hour	#	19.35	17.14	15.13	12.91	12.11	11.21
Production / Productivity							
Avg. Prod. dry tonnes per hour	t/h	3,402	2,952	2,498	2,167	2,087	1,906
Avg. Prod. dry BCM per hour	BCM/h	1,219	1,476	895	1,083	748	953

16.4.6 Hauling

Haulage will be performed with a 181-tonne class mine trucks. The truck fleet productivity was estimated in Talpac software. Several haulage profiles were digitized in Geovia GEMS with haul routes exported to Talpac to simulate cycle times. Cycle times have been estimated for each period and all possible destinations as there are several waste storage areas.

The assumptions and input factors for the Talpac simulations are presented in Table 16.10, Table 16.11 and Table 16.12.

No speed limits were applied except for the bottom of the pit until the truck reaches the ramp where a speed limitation of 35 km/h was imposed to reflect the lack of proper road and less favourable rolling conditions in addition to having stopes in the pit floor. Otherwise, the maximum truck speed reaches 56 km/h in the simulations.

Table 16.10: Speed Limits

Site Location	Speed Limit (km/h)
Pit on working bench, near dump face	35
Downhill Ramp < -5%	35
Mine Road and Ramps	No Imposed Limit

Table 16.11: Rolling Resistance

Road Type	Rolling Resistance (%)
Main Road	2.50
Ramp	3.00
Pit floor and near dump face	4.00

Table 16.12: Cycle Time Components

Cycle Time Component	Duration (min)
Truck Average Load Queue Time [1]	1.42
Truck Average spot Time at Loader	0.60
Truck Average Loading Time	2.24
Truck Average Dump Queue Time	0.00
Truck Average Spot Time at Dump	0.30
Truck Average Dumping Time	0.20

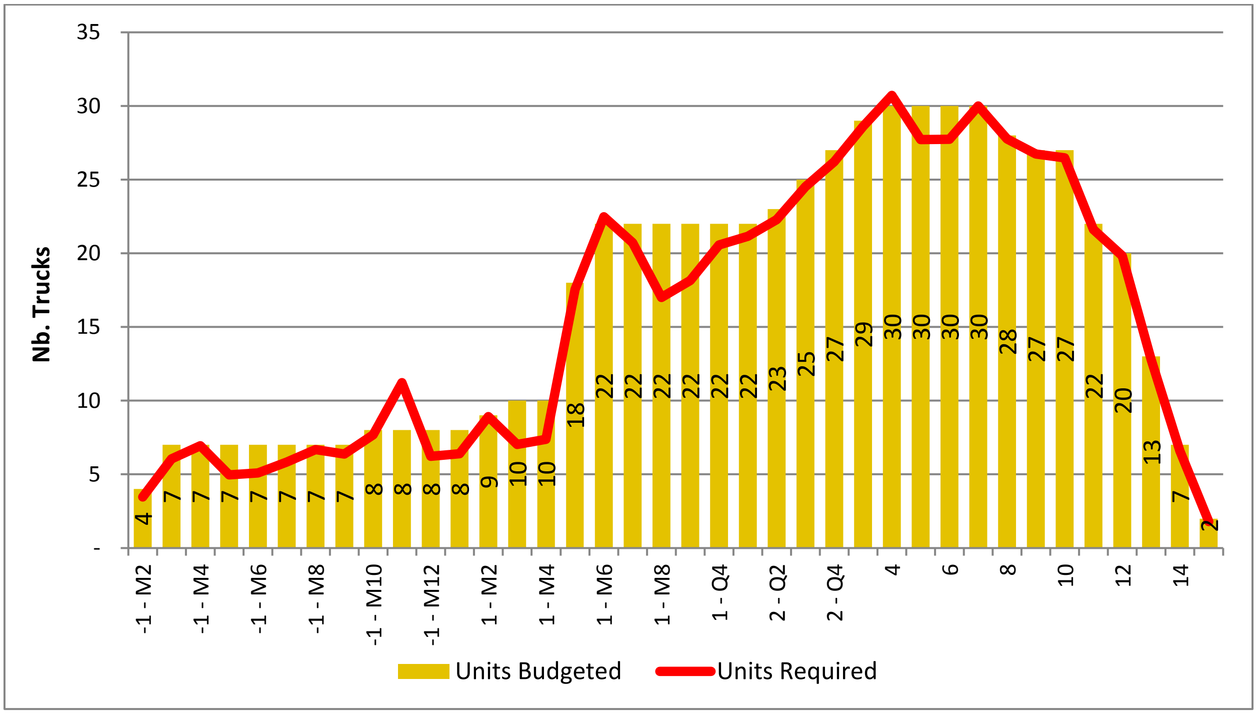
Note 1: The Average Load Queue Time was set to half the Loading Time and Spot Time

The multiple waste dumps were used to help level the truck requirements for the Project. During the critical years of the Project, leveling was achieved by sending waste rock to the closest dumps.

The fuel consumptions were also estimated with Talpac which generates a specific engine load factor depending on the proportion of the travel on ramp grades and on flatter gradients. Generally, the fuel burn rate increases with depth as a longer period of time is spent on grade.

The total haul hours required by period used to determine the number of trucks required throughout the LOM. The truck fleet reaches a maximum of 30 units in Year 4 and remains at this level until Year 7 before it starts decreasing as a result of a decrease in the mining rate.

Figure 16.13: Truck Requirement - Required vs. Budgeted



16.4.7 <u>Dewatering</u>

The open pit dewatering strategy will consist of using the underground opening and the connectivity of the past underground mines (Hard Rock, MacLeod and Mosher) to keep the water level 25 m below the working benches. This groundwater dewatering will be performed using submerged electric pumps.

Surface water will be pumped by mobile diesel pumps placed in sumps on the mining level. With the deepening of the pit additional pumping capacity and HDPE pipes will be added to the dewatering system.

16.4.8 Road and Dump Maintenance

Waste and ore storage areas will be maintained by a fleet of six 630 HP track-type dozers. Also, a 687 HP wheel dozer will be bought and dedicated to mine roads and the loading areas.

Mine roads will be maintained by three 4.9 m (16 ft) blade motor graders. A water/sand truck will be used to spray roads to suppress dust or spread road aggregate during winter months. A small water truck will also be purchased and kept standby in case the larger (76 kL) water truck is unavailable.

16.4.9 Support Equipment

All construction related work, such as berm construction, and water ditch cleaning will be done by three 49 t excavators (one of them will be equipped with a hydraulic hammer) and one 90 t excavator for pit wall scaling.

Two pit buses will transport workers to their assigned workplace and 20 pick-ups trucks will be purchased for the various mine departments.

Several other equipment purchases are planned to support the mining activities: one 60 t crane, three boom trucks (28 t crane), one tire handler, one 425 HP wheel loader for smaller work and a 250 HP utility wheel loader, one backhoe loader as well as a vibratory compactor for road construction.

16.4.10 Mine Maintenance

The Hardrock Project does not intend to enter into a maintenance and repair contract for its mobile equipment fleet. Consequently, the maintenance department has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support for major components is planned such as through component exchange programs.

A maintenance control system will be used to manage maintenance and repair operations. This system will keep up to date status, service history and maintenance needs of each machine.

16.4.11 Mine Management and Technical Services

The mine is headed by a Mine Operations Manager who is responsible for the overall management of the mine. Superintendent positions in engineering, geology, operations and maintenance report directly to the Mine Operations Manager.

The operations department is composed of two General Foremen and two Foremen per crew (eight in total). A mine dispatcher is planned on each shift. To increase operator level performance and organize structured training programs, two mine trainers are planned on day shift only. The operations department includes 14 staff employees at peak level.

The engineering and geology team will provide support to the operations team by providing short-term and long-term planning, grade control, surveying, Mineral Reserves estimation and all other technical functions.

16.4.12 Roster Schedules

A 5 on / 5 off rotating schedule has been planned at this time. Four crews are required to operate on a continuous basis 24 hours per day 365 days per year.

16.4.13 Equipment Usage Model Assumptions

The typical equipment usage model assumptions are established by equipment groupings as presented in Table 16.13. The annual net operating hours varies approximately between 5,000 and 6,000 hours per year.

Table 16.13: Equipment Usage Model Assumptions

Equipment Usage Assumptions		Shovels	Loaders	Trucks	Drills	Ancillary
Days in period	days	365	365	365	365	365
Availability	%	85.0	82.0	85.0	80.0	85.0
Use of Availability	%	90.0	90.0	92.0	90.0	85.0
Utilization	%	76.5	73.8	78.2	72.0	72.25
Effectiveness	%	85.0	85.0	87.0	85.0	80.0
Overall Equipment Effectiveness	%	65.0	62.7	68.0	61.2	57.8

16.5 Fleet Management

A fleet management system will be implemented to manage the operation, monitor machine health, and track key performance indicators. The system will be managed by a dispatcher on each crew who will control the system which will send operators onscreen instructions to work at peak efficiency. A system administrator will be required to assure proper functioning of system hardware and software with ongoing annual vendor support.

A high-precision global positioning system for machine guidance is considered to mitigate the associated risk of working around historic underground workings. This will enable shovel operators to navigate safely in potentially hazardous areas. In addition to protecting people and equipment, the high precision system will improve the productivity and bench grade control. The results and usefulness of such a system have proven to be worthwhile at other mines where past underground mines have been developed. Similarly, high precision drill navigation systems will be installed on the production drills and auxiliary drills to guide rigs into position and assure holes are drilled to the correct depth and location.

16.6 Pit Slope Monitoring and Voids Management

16.6.1 Pit Slope Monitoring

Rock mass failure is not considered as a risk due to the high overall rock mass strength. However, slope movement monitoring is planned for the open pit to gather measurements and confirm assumptions in order to assure safe working conditions. Initial slope movement monitoring would consist of using prisms read by manual or automated surveys with at least two permanent total stations established in climate controlled enclosures to ensure full coverage of the open pit. The initial prism monitoring will provide movement response data to verify visual observations and if the slope is performing adequately.

Pit wall mapping using routine digital mapping techniques using photogrammetry is recommended. Physical geological mapping is also recommended to supplement and qualify data derived from photogrammetry.

The slope movement monitoring data will be important for the calibration of numerical models required for detailed design updates during the mine life. The pit phasing approach will allow for adjustments to the final design based on observations and knowledge gained with the interim pit phases.

16.6.2 Voids Management

A number of open pit mines in Canada and Western Australia are mining ore bodies that have previously been mined by underground methods. There are hazards with high risk potential when approaching and then progressively mining through underground workings.

The hazards related to underground workings include:

- Sudden and unexpected collapse of the open pit floor and/or walls;
- The loss of people and/or equipment into unfilled or partially filled underground workings;
- Loss of explosives from charged blast holes that have filled cavities connected to the blast hole;
- Overcharging blast holes where explosives have filled cavities connected to the blast hole;
- Risk of flyrock from cavities close to the pit floor and adjacent blast holes.

Previous historical underground workings are well documented and available in 3D electronic format. The workings are not a concern for the overall stability of the final walls. In the pit designs, the ramps were kept away from the known historical underground openings. Larger berms were designed to create access points around the bigger underground openings at different heights. As part of detailed design, each stope that will underlie the pit will require a detailed assessment to determine the best operating practices for safe mining. The assessment should be initiated at least a year before the pit deepens to a critical proximity.

Additional boom trucks with various attachments have been planned in the OPEX and CAPEX to facilitate work around the underground openings.

16.7 Mine Equipment Requirements

The main factors which influenced the selection of the major mine equipment included the annual production requirements and optimization of the fleet size.

An extensive analysis was performed to determine the optimal fleet size, equipment type and preferred suppliers. Table 16.14 presents the equipment purchase schedule.

16.8 Mine Workforce Requirements

Table 16.15 presents the mine workforce requirements over the mine life with a reduction occurring when the tonnage decreases during the Year 9 of operation. The first and last year, Year 2 and Year 14, are fractional years and explain the reduction in number of employees. The total mine workforce is 199 the first year of operation and reaches a peak of 392 individuals by the fifth year.

Table 16.14: Equipment Purchase Schedule

Equipment Purchase Schedule	Total	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Major Equipment																	
Mining Truck (181t)	30	7	1	14	7	1	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (26 m³)	2	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Diesel Hydraulic Shovel (19 m³)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Wheel Loader (21 m³)	2	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Production Drill (6-10")	6	1	2	3	-	-	-	-	-	-	-	-	-	-	-	-	-
Track Dozer (630 HP)	9	2	2	2	-	-	-	-	-	-	1	2	-	-	-	-	-
Motor Grader (16ft)	5	1	2	-	-	-	-	-	2	-	-	-	-	-	-	-	-
Wheel Dozer (687 HP)	2	-	1	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Water/Sand Truck (76kL tank)	1	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Auxiliary Pre-Split Drill (4.5-8")	3	-	1	1	-	-	-	-	-	-	1	-	-	-	-	-	-
Support Equipment																	
Excavator (49t)	3	-	3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Excavator (90t)	2	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Wheel Loader (425HP)	2	-	1	-	-	-	-	-	1	-	-	-	-	-	-	-	-
Small Water Truck (16kL)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Vibratory Compactor - (130HP)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Backhoe Loader - (117HP)	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Utility Wheel Loader - (250HP)	2	-	1	-	-	-	-	-	-	1	-	-	-	-	-	-	-
Boom Truck (28t crane)	3	-	2	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Mechanic Service Truck	7	-	3	-	-	-	-	3	-	-	-	-	-	1	-	-	-
Tire handler Truck	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Fuel/Lube Truck	3	-	1	1	-	-	-	-	-	-	1	-	-	-	-	-	-
Lowboy and Tractor	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Pickups	50	-	15	5	-	-	5	10	5	-	-	-	5	5	-	-	-
Pit Busses	4	-	1	1	-	-	-	-	1	1	-	-	-	-	-	-	-
Compressors	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Welding Machines	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Forklifts	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Lighting Towers	18	-	3	3	-	6	-	-	-	-	-	6	-	-	-	-	-
Spare Box for Haul Trucks	1	-	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-
Spare Bucket for Shovels	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mobile Welding Machine	2	-	2	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Gensets	3	-	3	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Equipment Simulator	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Dewatering Pump - 10in	10	-	2	-	-	-	-	-	2	-	-	-	-	-	2	2	2
Dispatch System	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Slope Monitoring System	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Hydraulic Hammers for Excavator 49t	1	-	1	-	-	-	-	-	-	-	-	-	-	-	-	-	-

Table 16.15: Workforce Requirements

Department	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14
Mine Operations	3	125	224	240	248	252	252	252	252	240	236	216	190	167	117	81
Mine Maintenance	3	49	90	94	99	99	99	99	99	90	90	85	70	65	54	41
Mine Geology	3	12	23	23	23	23	23	23	23	23	23	23	18	17	11	11
Mine Engineering	5	13	17	17	17	18	18	18	18	18	18	18	13	11	8	8
Total Workforce	**14**	**199**	**354**	**374**	**387**	**392**	**392**	**392**	**392**	**371**	**367**	**342**	**291**	**260**	**190**	**141**

17. RECOVERY METHODS

17.1 Processing Plant Design Criteria

The process design criteria have been established based on: testwork results, trade-off studies, GGM client and vendor recommendations and standard industry practices.

The plant is designed to operate at a throughput of 27,000 t/d (grind size of 80% passing 90 µm). For the first 18 months of operation, the plant will operate at a throughput of 24,000 t/d (grind size of 80% passing 72 µm). No plant modifications are planned to achieve the higher throughputs and is a result of coarsening the grind size. The grinding circuit includes two identical ball mills and two identical gravity concentrators. The mill operation schedule is 24 h/d, 365 d/y with an overall availability of 92%. Crushing plant and processing plant equipment design factors allow for a margin of error in the sizing of the equipment. They are used in the calculations of the equipment feed rates and residence times. The key general process design criteria are presented in Table 17.1.

Table 17.1: Key General Process Design Criteria

Parameter	Units	Value
Throughput - Design	t/y	9,855,000
Throughput - Design	t/d	27,000
Throughput - Design	t/h	1223
Design Grind Size (P80)	µm	90
Crusher Utilisation	%	67
Concentrator Availability	%	92
Operating Time	d/y	365
Operating Time - Concentrator	h/d	24
Au Feed Grade - Average	g/t	1.24
Au Feed Grade - Design	g Au/t	1.30
Ore Moisture	%	3.0
Ore Specific Gravity		2.81
Gold Production - Design	oz/y	364,984
Strip Vessel Capacity	t	12
Crushing Plant Equipment Design Factor	%	30
Processing Plant Equipment Design Factor	%	15

17.1.1 Comminution Design Values

The comminution testwork program determined grinding characteristics for the various lithologies. Based on the run-of-mine expected composition (refer to Table 17.2), the weighted averages were calculated to establish the plant feed grindability parameters. The results are compiled in Table 17.3. The 90^{th} percentiles of hardness are used for design purposes to ensure sufficient error margins.

Table 17.2: Run-of-Mine Composition

Lithology	Units	Content
Greywacke (S3E) and Gabbro (I1A)	% w/w	52.5
Iron Formation (C2A)	% w/w	31.2
Porphyry (I3P)	% w/w	15.9
Conglomerate (S4)	% w/w	0.3
Ultramafic (I0)	% w/w	0.1
Total	**% w/w**	**100.0**

Table 17.3: Comminution Parameters (Weighted Averages)

Comminution Parameters	Units	Value
Standard Bond Work Index (BWI) (average)	kWh/t	15.49
Modified Bond Work Index (BWI) (90th percentile)	kWh/t	15.63
Abrasion Index (Ai) (average)	g	0.13
Unconfined Compressive Strength (UCS) (average)	MPa	104
JK Breakage Resistance Number (10th percentile)	(Axb)	25.2
JK Abrasion Resistance (10th percentile)	(ta)	0.22
JK Drop Weight Index (90th percentile)	(DWI)	11.7
JK Mia Parameter (90th percentile)	(Mia)	27.6
JK Mih Parameter (90th percentile)	(Mih)	22.7
JK Mic Parameter (90th percentile)	(Mic)	11.8

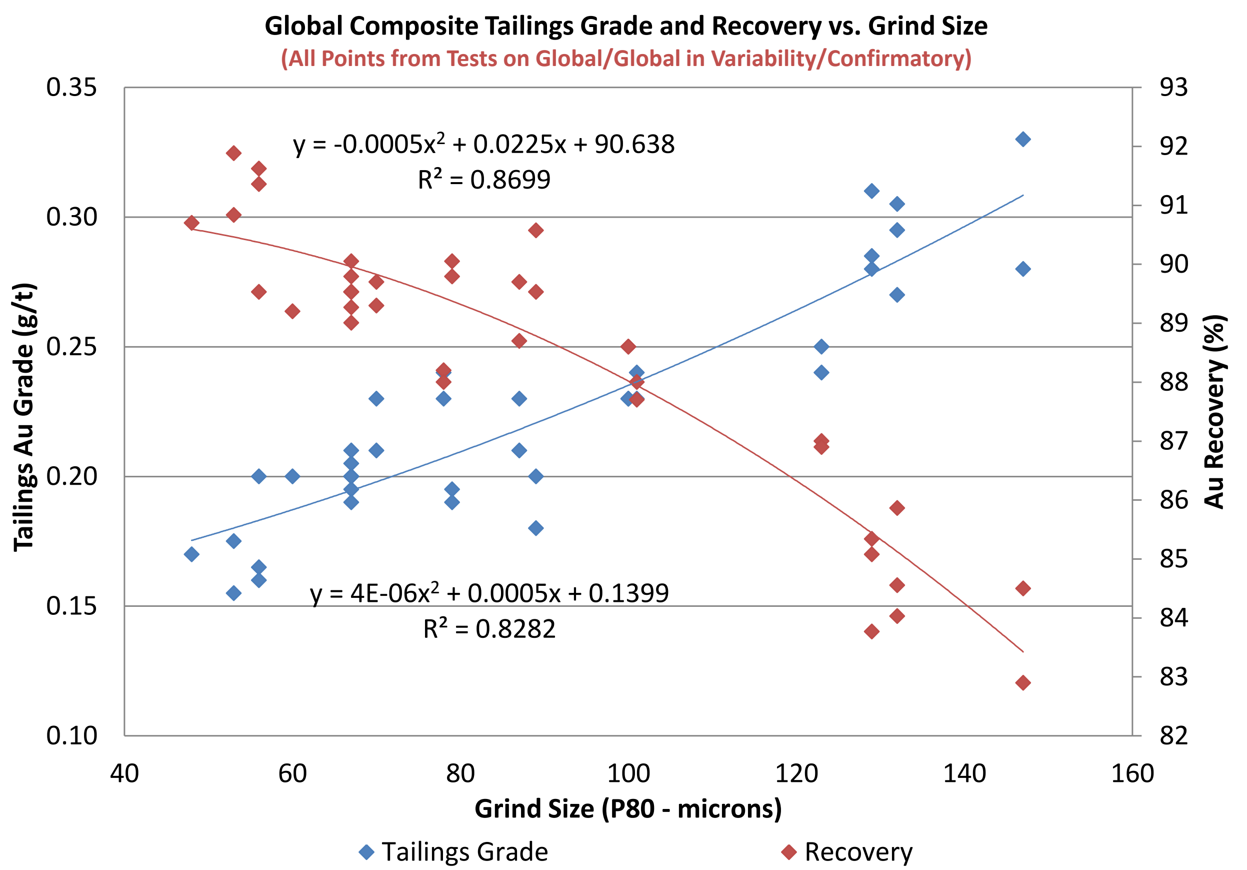
17.1.2 Grind Size Determination

The cyanidation testwork established a strong correlation between grind size and gold recovery (and tailings grade) and for that reason, the test results have been compiled to determine the optimal grind size.

Since the Global Composite is considered to be the most representative of the run-of-mine over the life-of-mine, the results of the leach tests on the Global Composite are used to determine the optimal grinding P80 (refer to Figure 17.1).

The analysis has also been conducted on the results from cyanidation tests on the Variability Composites A to I, the Master Composite representing the first three years of operation and the Low Grade Composites. Results from these other samples are used to evaluate the impact of ore variability.

Figure 17.1: Global Composite Tailings Grade and Recovery vs. Grind Size



Title: Global Composite Tailings Grade and Recovery vs. Grind Size	Author: Soutex/WSP Date: 14/07/21

The difference between the revenue from gold sales (based on recovery) and the expenditure on energy and grinding media was plotted against the corresponding grind size. The P80 value where the highest differential is obtained is around 72 µm and corresponds to the highest profit. At 27,000 t/d, a compromise is made between grind fineness and throughput and a 90 µm grind is considered optimal. The loss in recovery between 72 and 90 µm is counterbalanced by the improvement in the Project's overall economics by optimizing the life-of-mine. At 24,000 t/d, a grind size of 72 µm is selected to optimize the recovery; the mine plan is adjusted accordingly.

17.1.3 Impact of Mineralogical Composition on Leach Performance

It was suspected that leaching performances may be correlated to the amount of arsenopyrite in the deposit since the mineral may contain gold which is difficult to recover via leaching.

A multivariate linear regression analysis was used to determine the correlation between the residual gold grade and the deposit mineralogical composition. Multivariate regression models make it possible to describe how one variable (response) reacts to simultaneous changes in other variables (predictors). The method enables the joint impact of each predictor variable to be quantified on the response variable, which is not possible via simple regression analysis.

The results of the cyanidation tests conducted during the feasibility study stage, described in section 13.2.2, were used as the basis for the analysis. The residual gold grade from the cyanidation testwork was found to be highly correlated to the gold, arsenic and sulfur head sample grades. The strong correlation between the residual gold grade and arsenic and sulfur head grades suggests that arsenopyrite (FeAsS) contains locked gold which is not recovered via leaching. Table 17.4 shows the composites used for the analysis, the number of tests made on each composite and the head grades.

Table 17.4 Composite Used for Multivariate Analysis

Composite	Number of Leach tests	Head Au (g/t)	Head As (%)	Head S (%)
Global	5	1.74	0.010	1.70
A	3	2.56	0.190	1.56
B	3	2.04	0.150	0.85
C	3	1.71	0.070	1.37
D	3	1.68	0.120	3.56
E	3	1.18	0.110	0.99
F	3	1.36	0.029	1.78
G	2	1.59	0.062	0.68
H	2	2.65	0.074	2.92
I	3	2.29	0.280	1.48
Master	10	1.94	0.200	1.88
S3E-0.5-WCE	2	0.55	0.040	0.37

Composite	**Number of Leach tests**	**Head Au (g/t)**	**Head As (%)**	**Head S (%)**
S3E-0.7-WCE	2	0.67	0.027	0.51
I3P-0.5-WCE	2	0.46	0.002	0.27
I3P-0.7-WCE	2	0.75	0.029	0.42
C2A-0.5-WCE	2	0.34	0.027	1.06
C2A-0.7-WCE	2	0.85	0.014	1.55
Total	52			
Maximum value		2.65	0.280	3.56
Minimum value		0.34	0.002	0.27

A less predominant correlation between the residual gold grade and the head sample grind size was also established. This is due to the lack of variability in the tested grind sizes (a large number of tests conducted around the optimal grind size). The Table 17.5 shows the range of values for both P80 and residual gold grade.

Table 17.5 Leach Tests Parameters Range

Parameter	Unit	Maximum Value	Minimum Value
P80	µm	121	30
Residual Gold Grade	g Au/t	0.34	0.03

The impact of grind size along with gold, arsenic and sulfur head grades on the residual gold grade was modeled via the following multivariate linear regression equation:

$$\text{Tails g Au/t} = -0.0436 + 0.0733 \text{ x Head g Au/t} + 0.301 * \text{Head \%As} + 0.0269 \text{ x Head \%S} + 0.000632 \text{ x Head P80}$$

As described in section 16.3, this formula has been used in the block model and open pit optimisation process to calculate the gold recovery of each individual block based on chemical composition and grind size which depends on the throughput of the mill at the moment of the extraction. The overall gold recovery can then be computed for a determined period of time.

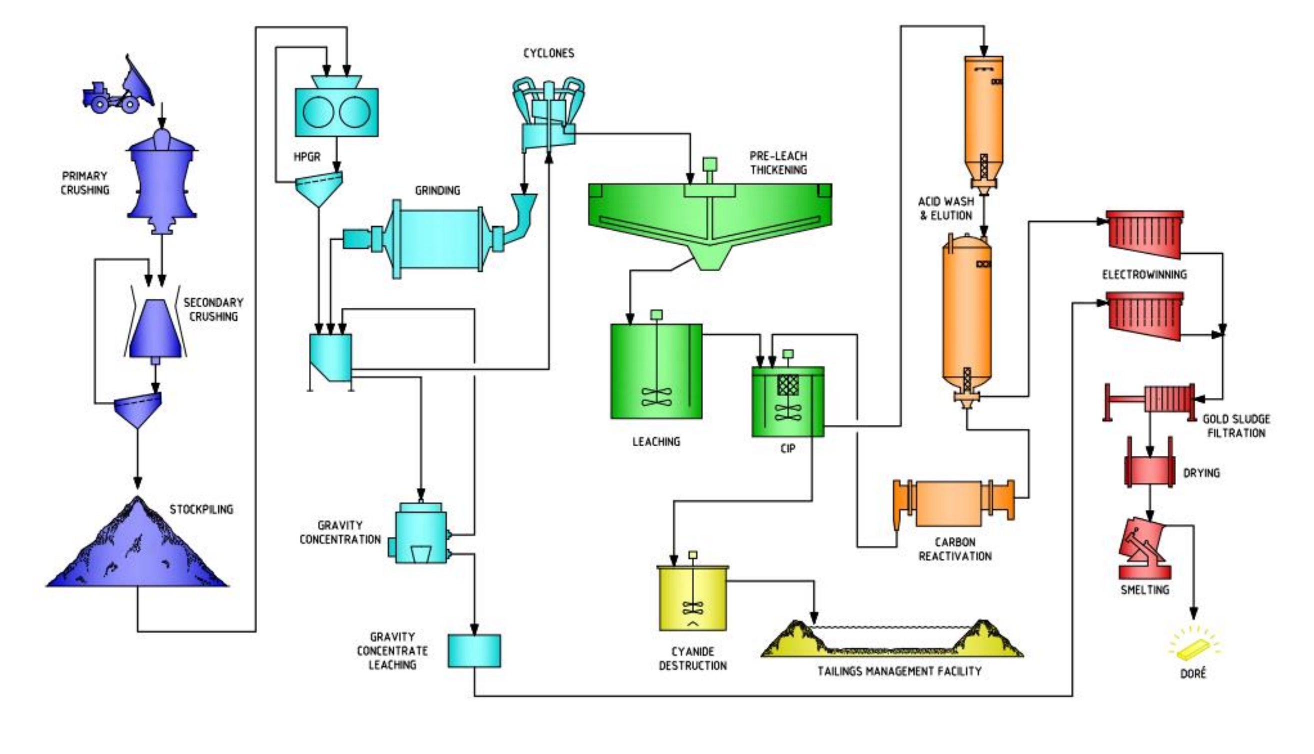
17.2 Flowsheet and Process Description

The gold recovery process for the Hardrock Project consists of a crushing circuit (gyratory and cone), a grinding circuit HPGR and ball mill, pre-leach thickening, a leach and carbon-in-pulp ("CIP") circuit, cyanide destruction and tailings disposal, carbon elution and electrowinning, carbon regeneration and a gold refinery.

The service areas include all the reagents preparation, compressed air, oxygen and the sulfur oxidation plant. The water management system covers all the fresh, reclaim, process, fire and gland water storage and pumping needs.

A simplified flowsheet that summarizes the process is presented below in Figure 17.2.

Figure 17.2: Hardrock Project Simplified Flowsheet



Title: Hardrock Project Simplified Flowsheet	Author: Soutex/WSP Date: 16/01/11

17.2.1 Crushing, Crushed Ore Storage and Reclaim Circuit

The crushing plant is a two-stage circuit consisting of a primary gyratory crusher and a secondary cone crusher. The crushing plant is expected to operate 67% of the time in order to achieve the daily throughput. A 30% design factor has been selected for the crushing circuit equipment so that the conveyors, crushers and other crushing equipment are sized to handle up to 2,194 t/h. The design factor allows for extra production capacity in order to handle the normal processing fluctuations due to disturbances in ore feed rate and ore hardness. Such utilization is standard industry practice. The crushing circuit objective is to reduce the size of the run-of-mine ore to a size that is acceptable for the subsequent HPGR and ball mill circuit.

17.2.1.1 Primary Crushing

The run-of-mine ore is delivered by mine haulage trucks or by loader to the crushing plant. The ore is dumped into a 300 m^3 concrete receiving hopper that feeds the primary crusher. A rock breaker will be used to break oversized rocks. The 652 kW gyratory crusher crushes the ore from a 1,000 mm top size (275 mm P80) to an 80% passing 123 mm product. The crushed ore falls into a concrete hopper and is reclaimed via an apron feeder. A sacrificial conveyor is installed before the secondary crushing feed conveyor which feeds the secondary crusher feed bin.

The primary crushing area is serviced by a dedicated compressed air system, an overhead 45 t capacity crane, a dust collector and a sump pump. A dust suppression foam system is also installed to control dust emissions at the crusher discharge. A magnet and a metal detector are installed on the secondary crushing feed conveyor to prevent any tramp iron from entering the cone crusher.

17.2.1.2 Secondary Crushing

The cone crusher feed is controlled by an apron feeder. The secondary crusher is a 933 kW standard cone with a 55 mm closed side setting (“CSS”). The secondary cone crusher is installed in closed-circuit with a single deck screen. This arrangement is preferred in order to control the top size feeding the high pressure grinding rolls. The screen oversize is returned to the secondary crushing feed conveyor. With a 69 mm closing screen opening, the crushing circuit produces a final crushed product with a 55 mm top size and a 42 mm P80. The final product is conveyed to the crushed ore stockpile dome. This dome has a base diameter of 80 m and an overall height of 40 m. It will be made of a galvanized steel structure and galvanized steel and painted cladding.

An emergency hopper and feeder are installed on the stockpile feed conveyor to reclaim material using a surface loader, if necessary. The secondary crushing area is serviced by a dedicated air compressor, a 40 t capacity overhead crane, a dedicated 2 t capacity crane in the screen area, a dust collector and a sump pump. A dust suppression foam system is also installed to control dust emissions at the crusher discharge. The secondary crushing recirculation conveyor and the stockpile feed conveyor are equipped with belt scales in order to monitor throughputs.

17.2.1.3 Crushed Ore Stockpile and Reclaim

The crushing circuit product is stored in a 27,550 t live capacity stockpile (78,000 t dead capacity). The stockpile is located between the secondary crushing plant and the process plant.

A single stockpile reclaiming line is planned; three apron feeders are located in a tunnel underneath the stockpile. These apron feeders feed a grinding feed conveyor which discharges onto the HPGR feed bin located inside the HPGR building. If required, the material can also be diverted to a bunker/pad. Insertable dust collectors are installed in the transfer chutes between the apron feeders and the HPGR feed bin.

The reclaim area is serviced by a dedicated air compressor and a sump pump. A hopper is installed on the HPGR feed conveyor to allow the tunnel area to be cleaned using a loader. One monorail and hoist is available in the tunnel for maintenance purposes. On the HPGR feed conveyor, a scale is installed to control throughput as well as a metal detector to avoid tramp iron further in the process. A magnet with a cleaning belt conveyor is not considered to be effective at this location considering the particle size and magnetite content.

17.2.2 Grinding and Gravity Circuit

The grinding and gravity circuit consists of one operating line made up of a HPGR, two ball mills, a cycloning stage and two gravity concentrators. The gravity products from both concentrators are combined in a single gravity concentrate leaching unit.

Considering a plant availability of 92%, the nominal circuit throughput is 1,223 t/h. A 15% design factor is used for the sizing of the equipment such as pumps, thickener, tanks, etc. The design factor accounts for the process fluctuations that can vary the flow and ensures that the plant availability is met in upset conditions.

In the case of the ball mill, the 15% design factor was not included; the ball mill power is based on the 90th percentile of power meaning that 90% of the time, the grinding power will be higher than required. Since the power of the mill will not have an effect on the throughput, it is not necessary to have a design factor of the same value as other handling equipment. Also, the P80 of the grinding circuit has been established by finding the point where the energy cost for changing the P80 is equivalent to the economic change in revenue related to the variation in gold recovery. It means that when the operating point is near the target value, a minor change in recovery due to a modification in specific power will not have any effect on the net revenue. The ore hardness data available when the ball mill was designed was measured on the composite samples made up of a blend of 53 different core intervals originating from different lithologies representing the entire deposit. The weighted average of the composite samples BWI obtained was 15.5 kWh/t which is the result used for the design of the ball mills. The resulting ball mill power was 10,000 kW. Modified BWIs, considered as more accurate for the prediction of the grinding circuit behavior, were also measured on samples from various lithologies. The overall 90th percentile result obtained for the samples from various lithologies was 15.6 kWh/t. That confirms that the design value of 15.5 kWh/t is satisfactory and conservative. Furthermore, the variability between the results obtained for each lithology is very small justifying the use of only one value of BWI to represent the entire deposit for the design of the ball mills. The ball mill power was increased to 10,500 kW to provide operational flexibility in case of harder ore / feed variations. The HPGR generates microcracks on the ore particles which typically reduce the power required at the ball mill. The microcrack effect was not taken into account when designing the ball mills which provides additional reserve for contingency.

The grinding circuit objective is to reduce the particle size to the pre-determined optimal liberation size to maximize gold recovery in the leach and carbon-in-pulp circuit. The gravity circuit objective is to recover gravity recoverable gold in the grinding circuit in order to decrease the load on the leach and carbon-in-pulp (“CIP”) circuit

17.2.2.1 High Pressure Grinding Rolls (HPGR)

A detailed comminution trade-off study recommended a two stage crushing followed by HPGR and ball milling circuit over other typical comminution flowsheets such as crushing followed by semi-autogenous (“SAG”) milling and ball milling, for the reasons of reduced risk in meeting the design throughput, and increased energy efficiency.

The grinding feed conveyor discharges into the HPGR feed bin located above the HPGR. The HPGR is equipped with two motors of 2,650 kW for a total of 5,300 kW. The HPGR roll dimensions are 2.4 m in diameter by 1.65 m in length. The HPGR discharge falls onto the HPGR discharge conveyor and then into the screens feed bin that divides the ore between two single deck screens. Water is added in the HPGR screen chute, sprayed on the HPGR screens and added in the HPGR screen undersize chute in order to reduce dust emissions and help flake deagglomeration. Screening is done at a cut size of 6 mm. The screen undersize is about 80% passing 3.5 mm and falls into the ball mill pump box.

The screens oversize is recirculated on the HPGR recirculation conveyor to be processed again in the HPGR. A 100% circulating load is expected.

The HPGR is installed in a dedicated building while the grinding mills and HPGR screens are installed in the process plant building. A 110/45 t capacity overhead crane is installed to service the HPGR in the HPGR building. A portable sump pump and a dust collector are also installed for the HPGR. A foam dust suppression system is installed for the HPGR. In the processing plant building, a scrubber and a 15 t overhead crane are installed to service the HPGR screens area. A heated corridor is also planned for between the HPGR and process plant operating floors to allow staff traffic and services transfer.

17.2.2.2 Ball Mill

The HPGR screened product is discharged to the ball mill pump box. The pump box is equipped with two sets of two slurry pumps (one operating and one on stand-by): one set feeding the ball mill cyclones and one set feeding the gravity circuit. Approximately 60% of the fresh feed is diverted to the gravity circuit.

The cyclone overflow cut size is 72 to 90 µm. The grinding circuit recirculating load is estimated at 250%.

The cyclone overflow feeds the pre-leach thickener trash screen while the underflow (approximately 70% solids) is separated into two parts each directed to a ball mill feed chute. Lime is added to the ball mill feed to raise the slurry pH to between 10 and 11. The grinding mills are twin pinion ball mills equipped with motors totalling 10,500 kW per mill. Both mills are 6.7 m in diameter (inside liners) by 12.3 m in length (EGL). The ball mill discharge falls into the ball mill pump box where it is combined with the HPGR discharge slurry and pumped to the cyclones for classification. The plant can be operated at a lower throughput, by operating only one ball mill and one HPGR screen.

A 70/10 t capacity overhead crane is installed to service both grinding mills. A single liner handler can be used for the liner changes in either ball mill. The ball magnet jib crane and ball bucket, the hydraulic jacking unit and the inching drive are also shared between the two mills. A dedicated sump pump is installed in each ball mill area.

17.2.2.3 Gravity

The gravity feed pump transfers a portion of the ball mill pump box content to the gravity circuit to recover gravity recoverable gold. One gravity screen and two gravity concentrators are installed to process the material.

The vibrating gravity scalping screen prevents particles coarser than 3.36 mm from entering the gravity concentrator. The screen oversize falls into the gravity recirculation launder. The screen undersize falls into the gravity distribution box and then onto the two gravity concentrators. A line is installed on the gravity distribution box to by-pass the gravity concentrators when required. After each cycle, the gold concentrate is removed from the concentrators and is transferred to the gravity concentrate leaching circuit. Reclaim water is used to flush the concentrate and antiscalant is added to the water stream to avoid scale build-ups.

The gravity concentrator tailings fall into the gravity recirculation launder where they are combined with the gravity screen oversize and returned to the ball mill pump box.

17.2.2.4 Gravity Concentrate Leaching

The gravity concentrate from both gravity concentrators is transferred to the single gravity concentrate leaching circuit. This system is a packaged unit consisting of a feed tank, a drum leach reactor, a solution storage tank and a transfer pump.

Both the gravity and gravity concentrate leaching equipment are secured in a fenced area with limited and controlled access. Also, cameras dedicated for security are installed in this area (not linked to the process camera network).

The gravity recoverable gold is leached in the reactor. A 98% dissolution efficiency is expected. The gravity leach tailings are returned to the ball mill pump box while pregnant solution is fed to a dedicated electrowinning cell.

17.2.3 Pre-Leach, Leach and Carbon-In-Pulp

The circuit is made up of a pre-leach thickener, a series of leach tanks (one pre-leach and five leach tanks) followed by seven CIP tanks. The objective of the pre-leach, leach and CIP circuit is to dissolve gold from the ground ore, adsorb it onto activated carbon and transfer the loaded carbon to the elution circuit.

17.2.3.1 Pre-Leach Thickening

The ball mill cyclones overflow from each line feeds a dedicated trash screen located above the pre-leach thickener. The screens undersize are sampled and fall into the pre-leach thickener feed box.

The 42.5 m diameter thickener is located outdoors; the tank does not require insulation because the slurry is always in movement and its temperature remains above 10°C. The thickener increases the slurry density from 35 to 55% solids. A very high molecular weight and slightly anionic polyacrylamide flocculant is added at a dosage of 15 g/t (grams of flocculant per tonne of dry solids) to promote flocculation and sedimentation.

17.2.3.2 Leach

The leach circuit consists of one agitated pre-leach tank and five agitated leach tanks. The leach tanks are located outdoors.

The slurry transfers from one tank to the next by overflow through an upcomer. Any tank can be bypassed if maintenance is necessary. The leach tanks are equipped with 220 kW agitators. The total residence time in the leach circuit is 36 hours and an additional seven hours is available in the pre-leach tank.

Lime is added in the pre-leach tank, the second and the fourth leach tank to readjust the pH level between 10 and 11 when required. A 25% sodium cyanide solution is added to leach tanks #1, #3 and #5 to ensure a 0.5 g/L NaCN concentration is maintained throughout the leach circuits. A cyanide analyser is installed in each circuit to measure cyanide levels and control cyanide addition. Finally, oxygen is injected to reach the targeted 15 mg/L concentration of dissolved oxygen. The leach discharge from each line falls by gravity to the CIP circuit.

17.2.3.3 Carbon-In-Pulp

The leach circuit discharge falls into the CIP launder located above the CIP tanks. The circuit is composed of seven CIP tanks but the required 1.5 hours of residence time is achieved in six tanks. The seventh tank is included to ensure the residence time is maintained when one tank is not in operation. The CIP tanks are located indoors.

A carrousel type of operation is selected, which allows the CIP tanks to be installed on the same level and to be of the same dimensions (7.0 m in diameter by 11.5 m in height). In the carrousel mode, the slurry feed and discharge positions are rotated to ensure a counter-current movement between slurry and carbon, without transferring carbon from one tank to another. The slurry passes through all the CIP tanks using the agitator mechanism for pumping between tanks. After it reaches the last tank, the slurry falls by gravity to the carbon safety screen to remove carbon particles from the tailings.

The carbon safety screen undersize falls into the CIP tailings pump box and is pumped to the cyanide destruction circuit. The pump box is equipped with two slurry pumps (one in operation and one on stand-by). The slurry is pumped to the cyanide destruction tanks.

Once a day, the lead tank is taken off-line and the entire tank contents are emptied and transferred to the loaded carbon screen via a recessed impeller centrifugal pump. One carbon transfer is planned daily and is done in two hours. The carbon batch is 12 t.

The loaded carbon screen is located in the acid wash and elution area. New carbon is added into the circuit after being screened on the carbon sizing screen located above the CIP tanks and directed to the correct tank by the carbon distribution box. A carbon concentration of 29 g/L is obtained.

17.2.4 Cyanide Destruction and Final Tailings

The cyanide destruction and tailings area comprises the equipment required for tailings detoxification, final tailings collection and pumping to the TMF.

17.2.4.1 Cyanide Destruction

The cyanide destruction circuit consists of two agitated tanks. The tailings from the CIP circuit are pumped to the cyanide destruction distribution box and then to the first of the two cyanide destruction tanks. The cyanide destruction tanks are normally used in series but they can also be used in parallel if required.

The SO_2/Air process is used to lower weak acid dissociable cyanide levels (CN_{WAD}) in the tailings solution down to 1 mg/L. The SO_2 is added in a gaseous form at the bottom of the tanks, oxygen is injected through the tank walls using gas spargers to maintain a 2.0 mg/L O_2 concentration in the tanks. Copper sulfate is used as a catalyst for the reaction and is added to the slurry in a liquid form at a 25% concentration. About 20 mg/L of Cu ions are required. Finally, lime is added to control the pH of the reaction to approximately 8.5. 120 minutes of residence time is required in each tank to achieve the desired extent for the reaction. Cyanide destruction is carried out at a slurry density of 55% solids in order to minimize the volume of water pumped to the TMF.

17.2.4.2 Final Tailings

The cyanide destruction tank overflows to the tailings collection box. The box also collects various tailings streams from the process, especially from sump pumps. Reclaim water is added to the box when the plant throughput is low in order to decrease the slurry density and maintain adequate velocity in the tailings pipeline.

The tailings collection box discharge is sampled and falls into the tailings pump box. Two parallel pump trains (one in operation and one on stand-by) of three pumps installed in series are installed to pump the tailings to the TMF.

The tailings pipeline is separated into four sections to fulfill the pressure requirements. A wireless leak detection system will be installed on the double wall section and a spigotting system at the TMF end.

17.2.5 Acid Wash, Elution and Carbon Regeneration

The 12 t capacity acid wash, elution and carbon regeneration circuit covers the process steps that handle the gold-loaded carbon, strip gold to a pregnant solution and regenerate carbon.

The objective of the circuit is to maximize gold recovery to the pregnant solution feeding the electrowinning circuit. It also supplies activated carbon (new and regenerated) to the CIP circuit.

17.2.5.1 Acid Wash and Elution

The loaded carbon from the CIP circuit is pumped to the loaded carbon screen located ahead of the acid wash column. When a carbon batch is transferred, the screen oversize containing the loaded carbon falls into the acid wash column. The slurry that reports to the screen undersize is returned to the CIP launder.

The acid wash step removes carbonate scale and some adsorbed metals that build onto the activated carbon during the adsorption process. A dilute hydrochloric acid solution (2% HCl) is circulated through the column to dissolve this scale. The acid wash waste solution is pumped to the tailings pump box.

The acid washed loaded carbon is transferred to the elution column where the adsorbed metals are stripped using the pressure Zadra process. A heated diluted caustic (1.0% $NaOH$) and cyanide (0.1% $NaCN$) solution is prepared in the elution solution tank and is circulated through the column to strip the carbon. The heat source for the elution solution is a natural gas heater. A carbon elution cycle is completed within eight to twelve hours.

17.2.5.2 Carbon Regeneration

The eluted carbon is transferred to the eluted carbon dewatering screen. The dewatered reusable carbon is recovered at the screen oversize and feeds the regeneration kiln. The water and the fine carbon that pass through the screen are recovered in the carbon water tank. The water is reused as transfer water while the carbon is filtered using a filter press and is collected into fine carbon bags.

The natural gas fired kiln heats the carbon to a temperature ranging from between 550 and 650°C. At this temperature and under a slightly oxidizing atmosphere, the fouling organics are removed from the carbon. The regenerated carbon exits the kiln and falls into the carbon quench tank. Newly attrited carbon is used to make-up for the fine carbon loss and is combined with the regenerated carbon in the quench tank. When required, a carbon batch is transferred to the carbon sizing screen located ahead of the CIP circuit.

17.2.6 Electrowinning and Smelting

The gold from the pregnant solutions (gravity concentrate leaching and elution) is recovered onto the cathodes in the electrowinning circuit. The electrowinning gold sludge is recovered and smelted into doré bars in the refinery.

17.2.6.1 Electrowinning

The pregnant solution from the gravity concentrate leaching is pumped to its dedicated electrolyte tank located in the electrowinning area. The electrolyte solution is circulated between the gravity electrowinning cell and the electrolyte tank.

The pregnant solution from the elution circuit is transferred to a flash tank prior to being split between four electrowinning cells.

The gold sludge is washed from the cell cathodes in the cathode wash pump box and is pumped to the plate and frame sludge filter. The barren solution from the cells falls by gravity to the elution circulation tank. The sludge filter filtrate is recirculated to the cathode wash pump box.

17.2.6.2 Smelting

Following filtration, the gold sludge is dried in the oven in preparation for smelting. The dried sludge is transferred to the mixer where refining fluxes are added. The mixture of sludge and fluxes is fed to the induction furnace where the slag material is separated from the gold as the doré bars are poured.

17.2.7 Gas and Reagents

The process plant includes a compressed air system and oxygen supply system as well as various reagents reception, preparation and storage equipment.

17.2.7.1 Compressed Air

The compressed air system is composed of three air compressors (two operating, one on stand-by). The compressed air is stored in two air receivers with an air dryer between them. Compressed air is produced at 690 kPa. A first distribution loop handles the instrument air while a second delivers compressed air to the various equipment requiring compressed air (filters, vents, dust suppression systems, etc.).

17.2.7.2 Oxygen (O_2)

The oxygen requirements for leaching and cyanide destruction are met by a vacuum swing adsorption (“VSA”) plant. The VSA plant is a packaged unit installed outside the plant. Two liquid oxygen tanks are also installed on site as a back-up source of oxygen.

17.2.7.3 Cyanide (NaCN)

Cyanide is delivered in isotainers (or equivalent) containing 22 t of solid NaCN briquettes. Up to four isotainers can be stored in the plant, providing about twelve days of storage at 27,000 t/d. Water is added to the preparation tank and the solution is circulated between the tank and the isotainer until complete dissolution of the briquettes. The water addition is controlled in order to produce a 25% NaCN concentration solution.

The storage tank contains two batches of solution and is equipped with two distribution pumps (one in operation and one on stand-by). Cyanide is distributed to the gravity concentrate leaching circuit, the leach circuit and the elution circuit.

17.2.7.4 Caustic (NaOH)

Caustic is delivered in bulk, in a liquid form at a 50% concentration. It is transferred to the storage tank, which can hold the content of approximately one and a half tanker trucks. The onsite storage capacity is sufficient to last about one week. Since the caustic users are all intermittent (cyanide preparation, elution circuit and gravity concentrate leaching), only one distribution pump is required.

17.2.7.5 Quicklime (CaO)

Quicklime is received in bulk in a powder form. It is transferred to a 300 t capacity silo which provides five days of storage at 27,000 t/d. Dry lime is fed to the slaker via a screw feeder. Water is added to the slaker to produce hydrated lime slurry. The hydrated lime production rate closely matches the consumption rate in order to ensure the slaker operates as continuously as possible.

The slaker discharges onto the vibrating grits screen that removes oversize particles from the lime slurry. The slurry falls into the agitated tank where it is further diluted to produce 20% lime slurry. The lime storage tank is equipped with two distribution pumps (one in operation and one on stand-by). The distribution line is in closed-loop with the storage tank. Hydrated lime addition points include the ball mill circuit, the leach circuit and the cyanide destruction circuit.

17.2.7.6 Flocculant

Flocculant is received in a powder form in bulk bags. The bags are unloaded in a hopper and a screw feeder transfers the flocculant to an eductor, mixed and metered to the circuit Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in bulk in a 32% concentrated solution. It is unloaded in a storage tank that has the capacity to hold a one and a half truck delivery. The HCl is distributed to the acid wash and elution circuit using a single pump.

17.2.7.7 Copper Sulfate ($CuSO_4·5H_2O$)

Copper sulfate pentahydrate is received in bulk bags. It is unloaded in the mixing tank where water is added to produce a 25% concentration solution. The copper sulfate storage tank is located right under the mixing tank and the transfer is done via a valve once a batch is complete. Two distribution pumps are installed (one operating and one on stand-by) to transfer copper sulfate to its addition point in the tailings pump box prior to cyanide destruction.

17.2.7.8 Sulfur Dioxide (SO2)

Elemental sulfur is delivered in a molten state in a tanker truck. It is transferred to a 160 t capacity insulated storage tank equipped with steam coils to maintain the sulfur in its molten state. The tank provides almost twenty days of storage at 27,000 t/d. Sulfur is metered to the burner in ratio with combustion air. The burner is a natural gas fired furnace that reaches operating temperatures of 1 450°C and higher. The SO_2 gas produced is approximately at a concentration of 18% by volume.

17.2.7.9 Antiscalant

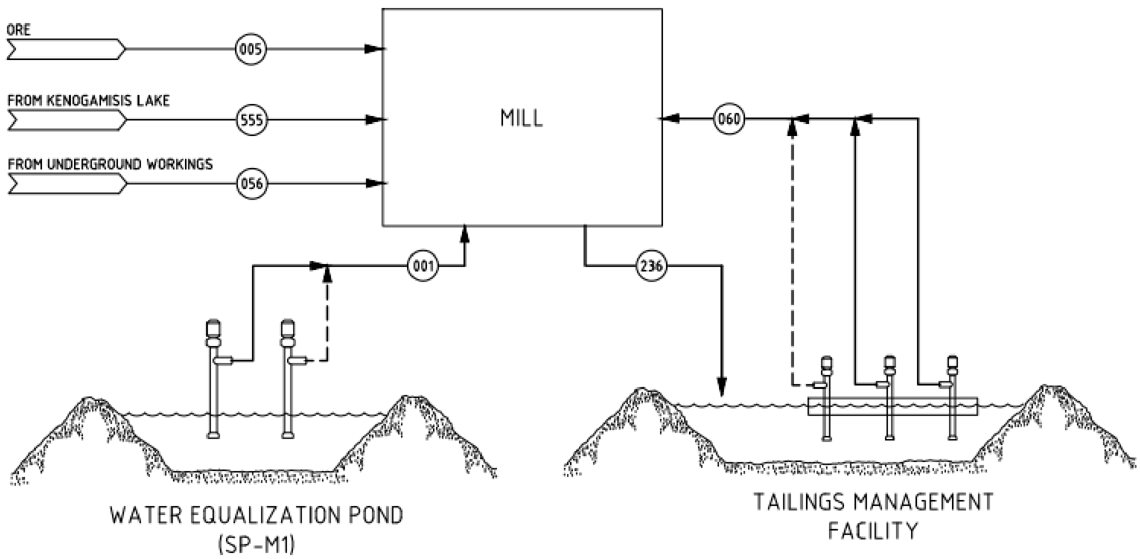
Antiscalant is received in a liquid form in containers. It is transferred to a storage tank from where it is pumped to the process. Antiscalant is added in the gravity concentrators' water lines, the elution solution tank, the process water tank and the reclaim water tank.

17.3 Mass and Water Balance

A detailed mass balance was developed for the concentrator to track all flows in and out of the equipment. The detail level of this mass balance was advanced in order to size all equipment and all piping of the process plant.

A comprehensive water balance was developed to track all fresh and waste water flows to ensure that appropriate management is planned for each type of water. The process plant requirements in fresh water will come from the underground workings collected in the water equalization pond M1. No water is planned to be withdrawn from the Kenogamisis Lake. The majority of the water required for the process plant operations comes from the TMF. The TMF is fed by the solids discharged in the process plant that contains significant amounts of water. Figure 17.3 outlines the water balance results calculated for the process plant.

Figure 17.3: Hardrock Project Process Plant Water Balance



STREAM No.	001	005	056	060	236	555
SOLIDS (t/h)	0.0	1222.8	0.0	0.0	1222.7	0.0
SOLUTION (t/h)	200.0	37.8	8.6	797.5	1038.0	0.0
SLURRY (t/h)	200.0	1260.6	8.6	797.5	2260.7	0.0
% SOLIDS	0.0%	97.0%	0.0%	0.0%	54.1%	0.0%
S.G. SLURRY	1.00	2.66	1.00	1.00	1.53	1.00
SLURRY (m^3/h)	200.0	473.4	8.6	797.5	1473.5	0.0

Title: Hardrock Project Process Plant Water Balance

Author: Soutex/WSP
Date: 16/01/29

17.4 Process Equipment

The equipment included in the process plant was designed using the information obtained from the mass balance and design criteria. For most pieces of equipment, at least three budget quotations were requested from suppliers. A bid evaluation was then conducted including a technical and a commercial evaluation of the proposed equipment. During the technical evaluation, the process, mechanical, electrical and other relevant aspects were analyzed to determine if the equipment complied with the specifications. The commercial evaluation, in turn, was made considering the budget price of the equipment, the transportation cost and the delivery lead time. The selected bidder was determined for each piece of equipment by combining the technical and commercial recommendations.

17.5 Cyanide Management

The International Cyanide Management Institute (“ICMI”) was established for the purpose of administering the "International Cyanide Management Code for the Manufacture, Transport and Use of Cyanide in the Production of Gold" (ICMI Code), and to develop and provide information on responsible cyanide management practices and other factors related to cyanide use in the gold mining industry. Throughout the design of the process plant, ICMI Code requirements were considered, to allow a third party to confirm compliance if necessary.

17.6 Power Requirements

The power requirement was determined for the process plant using the power demand indicated by the selected equipment supplier. The total load demand for the process plant is 35.8 MW at 27,000 t/d.

17.7 Plant Layout

17.7.1 Process Plant Location

The identified area for the process plant site is located southwest of the open pit, surrounded by the ore pile and the waste rock piles B, C and D. The site main entrance is on the west end to allow for a connection to Highway 11. Also, the natural gas power plant location is kept on the eastern side of the process plant and other infrastructures in function of the prevailing winds.

17.7.2 Building Architecture

All buildings designed for the process plant, including crushing and transfer towers, are planned to be covered with pre-assembled insulated wall and roof panels to reduce construction time and workforce on site.

A National Building Code study was conducted to ensure appropriate emergency and safety requirements are met in terms of design and construction methods (fireproof staircases, etc.). The National Building Code study also confirmed that the requirements for the buildings which house workers were met (toilets, wall composition, travel spaces, office arrangements, etc.).

17.7.3 Heating, Ventilation and Air Conditioning (HVAC)

The mill building and the ore storage dome tunnel are heated with a water/glycol mix system having 45% / 55% proportions to prevent freezing. The glycol mix temperature will be 140°F for the heating distribution circuit with a 110°F return temperature. The system is designed to heat the functional spaces and the heating load of the ventilation created by four make-up air units (two for the process plant, one for the HPGR building and one for the ore reclaim tunnel).

A primary/secondary system type is selected. This type of system allows a separation of the heat generation equipment installed in the power plant from the heat distribution equipment in the process plant (circulation pumps, manifolds expansion tank, pressurized glycol tank, etc.). The estimated capacity of the water/glycol heating system for the process plant is 19 MBTU/h.

The ventilation system was designed considering the different functions of the areas in the mill building; four functional areas were created to meet the specific needs. These areas are the following:

- Ventilation of the mill building work areas;
- Ventilation of the offices;
- Ventilation and cooling of compressor rooms;
- Cooling of electrical rooms.

17.7.4 Fire Protection

The fire protection water reserve will be in the fresh water tank located at the process plant. A dedicated portion of this tank is exclusively used for fire protection water, sized according to the largest consumer building. As the tank will be installed outside, a heating system will be installed to prevent freezing.

Water sprinkler systems will be installed for the main offices, the electrical and mechanical shops, as well as the process plant areas that have been identified as a fire hazard (conveyors, hydraulic units, etc.). Fire protection cabinets containing water hoses and portable extinguishers will also be installed throughout the plant. The electrical room will be protected by an inert gas system. In addition, for unheated conveyor galleries, fire protection will be provided by dry-pipe sprinkler system.

17.7.5 Electrical Distribution

The process plant main electrical room will be fed directly from two power plant 13.8 kV feeders. The process plant main 13.8 kV switchgear will have a 3,000 A capacity and will include a tie breaker to feed the plant from only one feeder in case of a problem. 3,000 A breakers are also the standard in the power plant. The three-storey electrical room will contain transformers on the first floor (all dry type), 13.8 kV, 4.16 kV and 600 V switchgears on the second floor and motor control centers and programmable logic controllers on the third floor.

17.7.6 Control System

The process plant control system is designed with a main PLC cabinet wired with fibre optics to remote I/O Ethernet PLCs. A server and communication room is planned near the process plant control room.

A camera system will be implemented to help operations and control the process. The network will use the fibre optics backbone to transfer images to the operator in the control room via a network video recorder with recording, viewing and remote monitoring capacities. Cameras will be powered over ethernet whenever possible. The fire alarm communication system will also be the fibre optics backbone and will be linked to a main system located in the administrative building gate. Each building is equipped with an annunciator installed at its main entrance.

18. PROJECT INFRASTRUCTURE

18.1 General

This section describes the infrastructure and service facilities required to support the Project's mining and processing facilities such as information technology ("IT") systems, roads, water and tailings management, camp, maintenance shops, warehouses, laboratories and offices, power generation equipment, diesel fuel and the natural gas distribution pipeline.

18.2 Water and Tailings Management

18.2.1 Administrative Water Services

Potable Water: Most of the infrastructures on site will require potable water to support the change room, the lunch room, the various washrooms and showers and the emergency showers. A tie-in to the municipal network will be required to the Project site.

Raw Water: The wash-bay system will require an input of raw water to wash the mine fleet. Water will be provided from the water equalization pond, fed primarily from open pit dewatering water provided by the Mosher Shaft dewatering pump station. Wash water effluent will be clarified, then treated if required prior to release to the environment.

18.2.2 Effluents

Two types of effluents will be generated during Project activities: mine effluent and sanitary effluent. The water quality standards applicable to mine effluent are the Provincial Water Quality Objectives ("PWQOs") (MOE, 1994), Ontario Regulation ("O.Reg.") 560/94-MISA Metal Mine Sector Effluent Criteria, and Federal Metal Mining Effluent Regulations ("MMER") Effluent Criteria. The water quality standards applicable to sanitary effluent are the PWQOs. The Assimilative Capacity Study (Stantec, 2016) conducted for the Project identified effluent discharge locations and proposed effluent criteria for both mine and sanitary effluents discharging to the Southwest Arm of Kenogamisis Lake which are protective of the receiving environment. The proposed effluent criteria meet and exceed MMER and O.Reg. 560/94 criteria at end of pipe and the PWQOs for all parameters are met within a reasonably small mixing zone in the receiving waterbody.

Mine Effluent: All collected mine water is directed through distributed runoff and seepage collection ponds to the centralized mine water Collection Pond M1, which is designed to provide on-site treatment/storage for all mine water with controlled release to the Southwest Arm of Kenogamisis Lake following treatment through the effluent treatment plant. Runoff and seepage collection from the TMF will be collected in a series of ponds and pumped back to the TMF for reuse to meet process plant reclaim demands.

Sanitary Effluent: The Project has two sources of sanitary sewage: the temporary construction camp located near Barton Bay and the main mine site which includes the offices, process plant and mine buildings. The temporary construction camp will be connected to the municipal sewage system by incorporation of a new sewage lift station

For the main mine site buildings, a permanent sewage treatment plant will be constructed to accommodate up to 100 persons at any given time.

18.2.3 Site Runoff and Spillage Control

Runoff refers to runoff over ground surfaces as well as seepage to the surface from groundwater/subsurface sources. Four main sources of site runoff produced by mining operation have been identified:

- **Runoff from Open Pit:** Precipitation on the open pit will be directed to the historical underground workings associated with the MacLeod-Cockshutt and Hard Rock mines. The use of the historical underground workings provides a storage reservoir for open pit seepage and runoff during the mining operation and allows for flexibility in the water management approach;
- **Runoff from WRSA and Ore Stockpiles:** Runoff from the WRSAs and ore stockpiles will be collected in a series of perimeter ditches which will drain by gravity or be pumped to one of seven local collection ponds and directed to one centralized Mine Water Collection Pond M1. The seepage collection ditches and ponds will be designed to the 1:100 year design event. After mine closure and WRSA rehabilitation, water will continue to be collected and directed to the open pit to accelerate the filling of the pit to form a pit lake. Once water quality is acceptable for discharge, the ponds will be decommissioned and water directed to the natural environment.
- **Runoff from Mill Yard:** Surface runoff and contact water from the processing plant, truck shop yard and ore stockpiles will be collected in perimeter ditches draining to the Collection Pond M1.

  Runoff from TMF: The TMF will included a perimeter seepage collection system to collect runoff and seepage from the perimeter dams. Water from the collection ponds will be pumped back to the TMF for reuse to meet process plant reclaim demand. The TMF has been designed to contain up to the environmental design flood with emergency spillways designed for the probable maximum flood.

18.2.4 Tailings Management

Amec performed specialized geotechnical and hydrologic engineering services for the design of the TMF for the Project. Services include geotechnical site investigations, tailings deposition planning and design of the tailings dams and ancillary hydraulic structures.

The TMF is formed mostly using perimeter embankment dams raised in stages using mine rock with relatively low-permeability till forming an upstream core keyed to relatively low permeability foundation soils or grouted shallow or outcropping dam foundation bedrock.

The TMF site was selected based on a balance of environmental, social, economic and operational risk parameters. Prior to construction of the TMF, Goldfield Creek will be diverted around the north side of the ultimate TMF into a permanent channel designed to provide fisheries compensation. The TMF is set back from other waterbodies by an environmental buffer width of approximately 125 m.

Figure 18.1 shows the general arrangement of the TMF.

The TMF dams have been designed to meet the requirements of the Lakes and River Improvement Act (MNR, 2011) and the Canadian Dam Association guidelines (CDA, 2014). Filter and transition zones are provided downstream of the till core and between the foundation soils and rockfill embankment to protect against piping of fines into the rockfill due to seepage forces. The rockfill embankment is constructed of geochemically benign mine rock.

18.2.4.1 Design Criteria

The Project involves the production of approximately 140Mt of process plant tailings following with gold extraction using cyanide leaching followed by in-plant cyanide destruction to treat process water. An allowance for 5Mt of historical tailings is considered in the design of the TMF for a total tonnage of 145M t.

The TMF dam have been classified as ‘Very High’ hazard potential based on the potential environmental impacts in the event of a catastrophic failure. The maximum starter dam section is about 11 m high with the ultimate dams raised to a maximum section height of about 35 m.

Dam design criteria include maintaining storage to contain the Environmental Design Flood (“EDF”), a 100-year return hydrologic event (24-hour storm or freshet), with no discharge through the spillway. An emergency spillway will be maintained to safely pass the Inflow Design Flood (“IDF”) consisting of a routed Probable Maximum Flood.

Conventional tailings slurry has been adopted for the Project. It provides flexibility for flooding of the tailings for closure should it be required for geochemical stability. Selection of conventional slurry is premised on the inclusion of spigotting from the dams to produce a wide exposed tailings beach and displace the pond away from the perimeter dams during operations to enhance dam stability.

18.2.4.2 Tailings Characteristics

Particle size distribution tests for two tailings samples were provided by GGM. The tailings are non-plastic hard rock particles and are predominantly silt and sand sized with approximately 75% to 82% of the particles by mass finer than 0.075 mm diameter (silt and clay sized), out of which 11% to 16% by mass are clay sized. Settling and consolidation tests have not been performed to date, but are recommended for the detailed design, to better understand the tailings settled density in the TMF.

Tailings geochemistry has been evaluated by Stantec which indicate that less than 10% of the ore is considered potentially acid generating (“PAG”) and will be further oxidized by up to 26% during processing, reducing the overall Acid Rock Drainage (“ARD”) potential for the tailings. As the ARD on-set time for the tailings is estimated at 13 years, ongoing deposition of tailings will reduce the ARD potential. Prior to rehabilitation of the TMF, or a portion thereof, testing of tailings beaches is recommended to determine if localized areas or pockets of PAG tailings exist due to preferential settling of sulphide minerals near final deposition locations. If required, remedial measures will be undertaken to mitigate ARD conditions during closure.

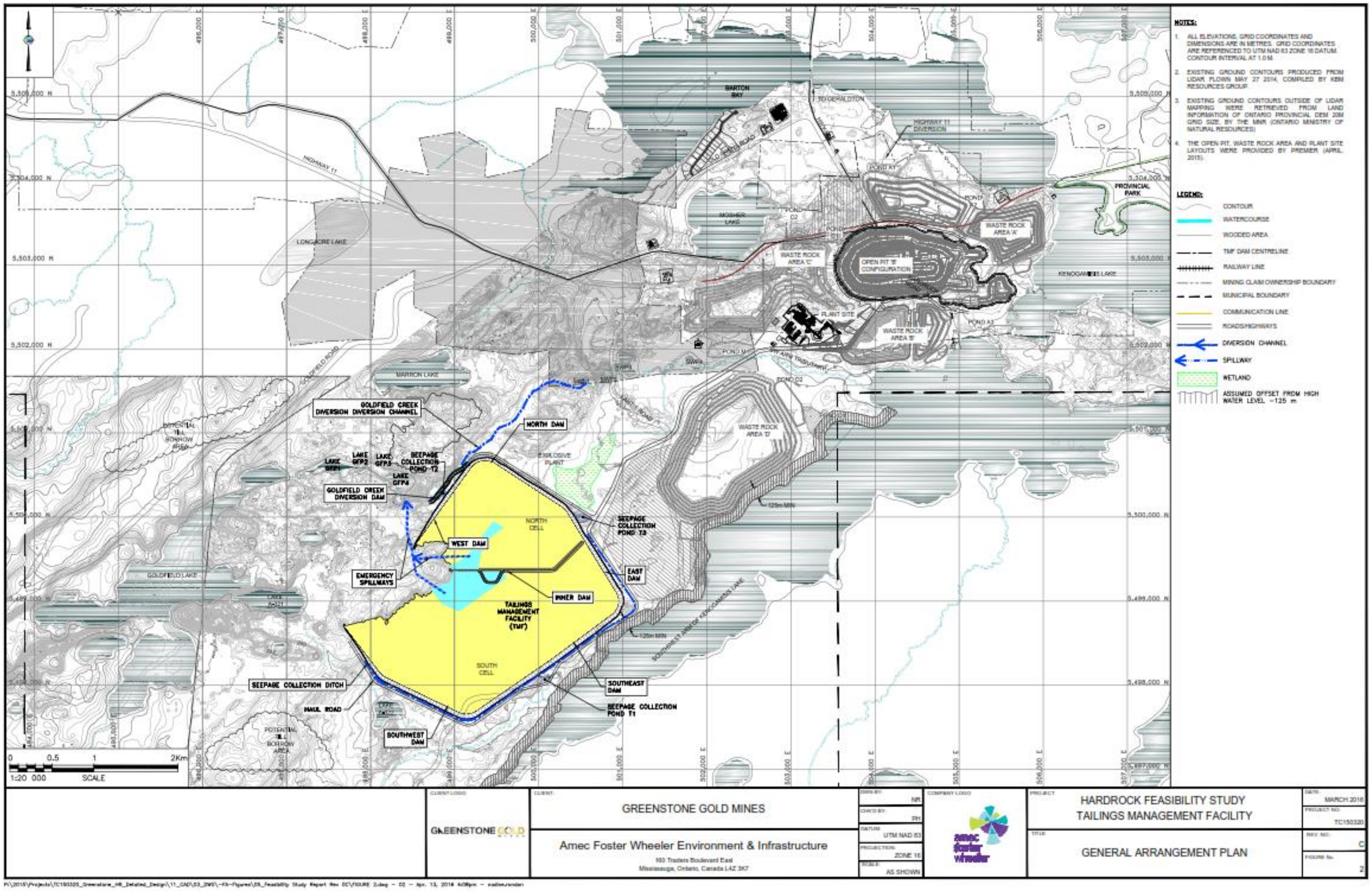
18.2.4.3 Tailings Deposition Plan

Key tailings operating data include:

- Total tailings production: 145 Mt
- Dry density of deposited tailings: 1.34 t/m3 (for short-term planning)
- Total deposited tailings volume: 108.24 Mm3

The tailings deposition plan involves spigotting tailings from the crest of the embankment dams to produce a wide beach to enhance dam safety and minimize seepage under or through the dams. The TMF South Cell has been designed with capacity to hold the tailings for the initial one year of operation. The South Cell dams will be raised during the first operating year to accommodate tailings for the second year of operation. During this initial period of two years a temporary diversion channel will be maintained to divert freshwater from the North Cell around the TMF, to reduce the quantity of water to be managed and pumped to the process plant. Thereafter tailings deposition will alternate between the North and South Cells with deposition into the North Cell targeted for early completion to facilitate early reclamation and closure. A key objective of the deposition plan is to maintain the TMF pond against natural ground on the northwest side of the TMF. This design will help to mitigate dam safety risks during operations, reduce dam seepage rates, and facilitate the ultimate closure design for the TMF. Refer to Table 18.1 for Dam Construction Sequence and Staging.

Figure 18.1: General Arrangement Plan



The key drivers for the deposition plan is pushing the pond away from the perimeter dams, establishing the permanent pond against natural ground, where the emergency and closure spillways will be located.

18.2.4.4 Water Management

The site-wide water balance by Stantec has determined a positive water balance for the site. As such, the TMF will be developed in stages, initially with only one of two cells to minimize the surplus runoff reporting to the tailings system and therefore requiring treatment. Similarly, efforts will be taken to complete tailings deposition onto one cell early for progressive rehabilitation, and direct discharge of runoff the environment, thus reducing runoff collection and treatment requirements.

A system of seepage collection ditches and ponds is provided downstream of the dams to facilitate capture and pump-back of seepage and runoff from the dams into the TMF.

18.2.4.5 Dam Design

The following alternative sections were considered.

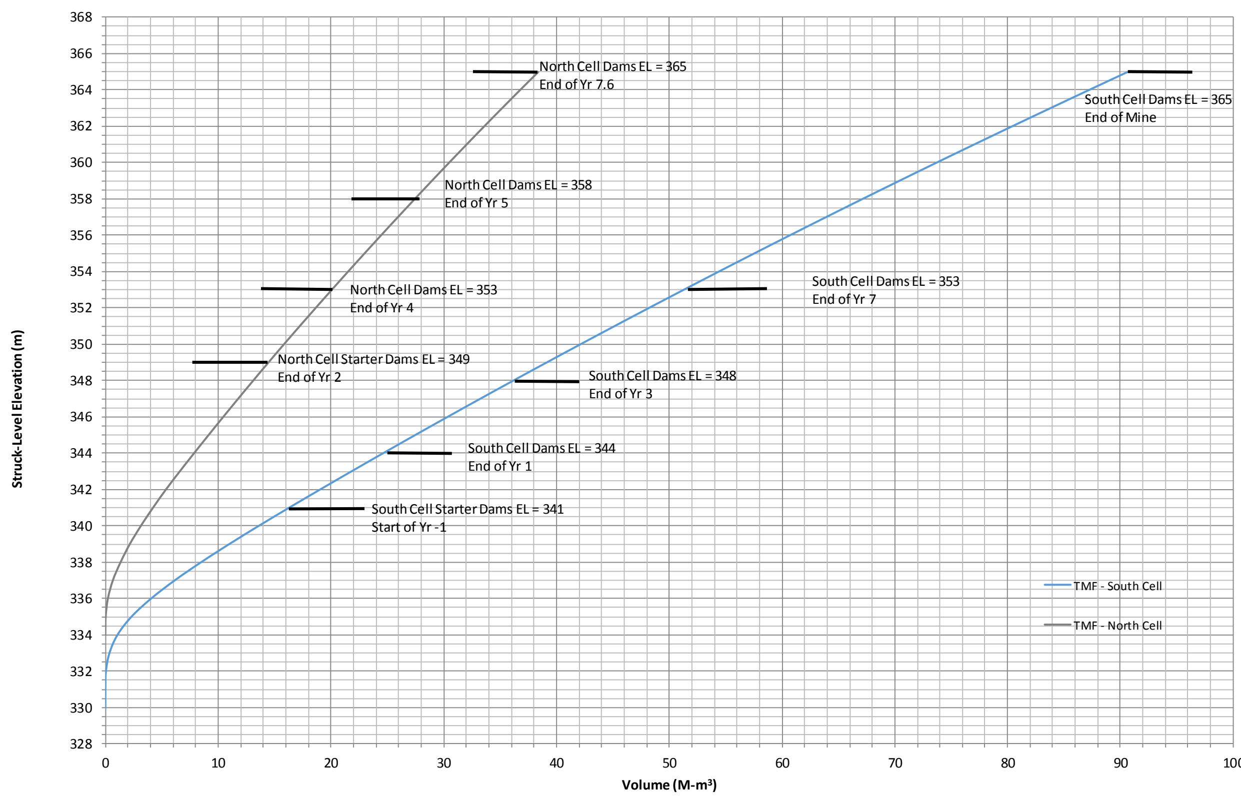
- Central low permeability till core abutting compacted waste rock buttresses;
- Compacted waste rock with upstream low permeability liner;
- Compacted waste rock with upstream inclined low permeability till core.

The compacted waste rock with upstream inclined low permeability till core was judged to be the preferred section.

The perimeter dams and inner dam for TMF cell division will be raised in stages depending upon the capacity requirements. Dam foundation preparation, till core, filter construction and approximately 5 m width of mine rock abutting the till core will be executed by contractors, with the mine rock embankment raised using the mine fleet.

Figure 18.2 shows the struck-level capacity curve for the TMF.

Figure 18.2: TMF Struck Level Capacity



CLIENT : GREENSTONE GOLD MINES

Amec Foster Wheeler Environment & Infrastructure
160 Traders Blvd East
Mississauga, Ontario, L4Z 3K7

PROJECT: HARDROCK FEASIBILITY STUDY

TITLE: TMF STRUCK-LEVEL CAPACITY CURVE & DAM STAGING

DATE:	JOB NO.:	PREP'D BY:	FIGURE	REV.
MAR, 2016	TC150320	MAS/PH	6	C

The crest level of the dam is determined from required tailings capacity. The tailings discharge elevations at each stage of TMF operation will be 0.3 to 0.5 m below the dam crest at each stage. The emergency spillway invert level for each stage dam is set 1.5 m below the dam crest for passage of the IDF.

The objectives of dam foundation preparation are to ensure stability of dams, control and mitigate foundation seepage. For dam safety, foundation filters are provided under the full width of the downstream dam shells to prevent piping of fine soil particles into the rockfill.

Foundation treatment in addition to removal of organics and unsuitable materials involves provision of seepage cut-off trenches in overburden soils and bedrock grouting.

18.2.4.6 Closure Considerations

Closure of the TMF involves lowering of the spillways and re-vegetation of the exposed beaches. Runoff will be directed through overflow spillways constructed in natural ground when deemed suitable for discharge to the environment.

The use of thickened non-potentially acid generating ("non-PAG") tailings to cap the non-active tailings cells will be considered to promote runoff and ultimately produce more stable land masses for closure. To do this, a better understanding of the distribution of PAG and non-PAG ores in the deposit is required and mining sequence developed to suit, such that non-PAG tailings be processed toward the end of mine life.

18.2.4.7 Geotechnical Subsurface Investigations

Geotechnical investigations carried out in the area of the TMF have been carried out by Stantec, for environmental baseline data - hydrogeological and geotechnical components and by Amec in 2014 and 2015.

Generally, the boreholes penetrated 15 m into bedrock before termination. Piezometers/monitoring wells were installed in the boreholes at various depths to facilitate groundwater monitoring and testing. Standard Penetration Tests (“SPT”) were carried out in the overburden at regular depth intervals in the boreholes. Soil samples were collected from the boreholes and test pits and tested in a geotechnical laboratory for various index properties such as particle size distribution, Atterberg limits and moisture content. Consolidated drained direct shear testing was completed on select samples of the overburden.

Overburden hydraulic conductivity values were estimated using rising head slug testing from screened intervals of select piezometer/monitoring wells in the TMF area. Bedrock hydraulic conductivity values were estimated using rising and falling head slug and constant head test methods from continuous single packer testing in the TMF area.

A standalone geotechnical investigation report outlining the factual findings in detail was prepared and submitted (2014 Investigations) to GGM (Amec, 2015b). Organics/peat up to a maximum thickness of 2.5 m were encountered at surface at the majority of investigation locations. The groundwater table was found to vary from near surface to about 3.5 m below ground surface in the TMF area. Bedrock encountered was generally good to excellent quality based on RQD measurements.

The expected subsurface conditions inferred from test pits along the proposed Goldfield Creek diversion channel alignment (up to the watershed divide near the corner of the North and West dams) are up to 2 m of peat/organics underlain by outwash deposits of sands to silts in excess of 4 m thick.

18.2.4.8 <u>Construction Borrow Materials</u>

Two potential till borrow source areas have been identified. One area is located in the high ground south of Goldfield Lake and the other in the high ground north of Goldfield Lake off of Goldfield Road. Both glacial till deposits underlay a thin organics layer (0.1 to 0.5 m thick) and are predominantly fine grained till varying in composition from silty sand to sandy clayey silt. The till contains trace gravel with cobbles and boulders. The indicated maximum thicknesses of the areas south and north of Goldfield Lake are 21 m and at least 6 m, respectively.

The maximum operating pond volume has been assumed to be between 4.0M m^3 in the Phase 1 (Years 1 to 2) and 5.5M m^3 thereafter, corresponding to scenario of 70% recirculation from the TMF for a 1 in 100 wet year. The Maximum Operating Water Level varies accordingly. The emergency spillway invert levels will be maintained at least 1.5 m below the dam crest levels at all stages of operation to ensure capacity to contain the EDF. Maximum pond capacity available in TMF below spillway invert level is discussed in Section 11.

Table 18.1: Dam Construction Sequence and Staging

Operational Years	Construction Stages	Active Cell Deposition	TMF Dam Crest Elevations			South Cell Perimeter Dam Quantities		North Cell Perimeter Dam Quantities		Comments
			South Cell (m)	North Cell (m)	Inner Dam (m)	Till Core (Mm^3)	Mine Rock (Mm^3)	Till Core (Mm^3)	Mine Rock (Mm^3)	
1	1	South	341	-	341	0.7	1.6	-	-	South Cells dams raised to EL 344 m by end of year 1 (3 m raise)
2	2	South	344	-	344	-	-	0.4	0.7	North Cells dams raised to EL 349 m by end of year 2 (13 m height)
3	3	North	344	349	344	0.4	0.8	-	-	South Cell dams raised to EL 348 m by end of year 3 (4 m raise)
4	4	South	348	349	348	-	-	0.2	1.5	North Cell dams raised to EL 355 m by end of year 4 (6 m raise)
5 to 7.57 End of North Cell	5	North	348	365	365	-	-	0.2	1.1	North Cell dam raising Continues while deposition of tailings in North Cell
Years 7.57 to 15 End of North Cell	6	South	365	365	365	0.7	6.2	-	-	South Cell dam raising Continues while deposition of tailings in South Cell
Total						**1.8**	**8.6**	**0.8**	**3.3**	

18.2.4.9 Recommendations

Future work recommendations include:

- Supplemental geotechnical investigations and laboratory testing for better definition of strength and consolidation properties of the interbedded silt layers encountered in the subsurface soils near the southwest and southeast dams;
- Deformation modelling of critical dam sections to confirm sufficiently robust protection against core cracking;
- Settling and consolidation testing to better understand tailings behavior and density progression to optimize the TMF design as the currently assumed properties are believed to be conservative;
- Further study of the geochemistry of the ore and tailings to allow optimization of the TMF design, operation, and closure planning;
- Detailed tailings deposition planning to optimize the dam raising schedule and inner dam construction requirements;
- Detailed water balance modelling to confirm design assumptions and set operating guidelines for the TMF pond. Adequate process plant make-up water supply storage will be required before winter;
- Site-specific seismic ground motion hazard assessment;
- Site-specific seismic hazard analysis to determine appropriate earthquake design parameters for the dam design;
- Detailed geotechnical investigations to support construction documents.

18.2.5 Effluent Treatment Plant

The effluent treatment plant is made of a series of stages that aim to increase water clarity and adjust pH. In the first step, acid and caustic are added to the water in the first of three metal precipitation reactors to adjust the pH to the desired value. The presence of OH^- ions in contact with residual metallic ions allows the formation of a precipitate of insoluble hydroxides.

The next step is pre-coagulation and coagulation where coagulant is added to the second metal precipitation reactor to assist in the decantation and in the precipitation of arsenic. Proper mixing in the last two metal precipitation reactors ensures a homogenous diffusion of the coagulant in the water. Water is then transferred to the maturation tank where flocculant and microsand are added. The baffles and efficient mixing accelerate the contact between the flocs, the flocculant and microsand, thereby ensuring the formation of ballasted flocs. Water is then transferred to a counter-current lamellar settling tank where the ballasted flocs sink to the bottom forming a sludge and the clarified water is collected at the surface. The sludge is drawn out of the clarifier and directed to hydrocyclones to separate the microsand from the sludge. The microsand exits the hydrocyclones at the underflow to be recirculated in the system while the sludge evacuated at the overflow is directed to the sludge pumpbox before being transferred to the tailing pumpbox.

In order to achieve the final water clarity and meet the suspended solids criterion, the clarified water is further polished in a disc filter. The clarified water is sent to an intercoagulation tank where coagulant and flocculant are added to help in the formation of flocs. A final pH adjustment is made by the addition of caustic or acid. After proper mixing and retention time, the water is sent to a disc filter to remove the precipitated flocs and obtain the clarified water that can be sent to the environment.

18.3 Fuel Supply

18.3.1 Diesel and Gasoline Storage and Distribution

Diesel, gasoline and urea will be stored on site at the fuel storage area, with each product having a designed storage volume of 150,000, 9,000 and 9,000 L, respectively. Liquid urea will be used with diesel fuel to reduce the nitrogen oxide ("NOx") emissions of the mine fleet.

The area is designed so that it drains to an underground oil/water separator.

18.4 Power Supply and Distribution

18.4.1 Power Demand Estimates

The average power demand for the Project was calculated using the Project's process plant load list compiled by WSP and complemented by GMS for the average load required for the supporting infrastructure, such as the truck shop/warehouse, the administration office, the sewage treatment plant, etc. The average power demand was then increased by 10% to provide an assessment of peak demand the power supply facility would have to sustain during operation. Table 18.2 illustrates the estimated average and peak power:

Table 18.2: Average and Peak Power Demand

Description	Demand (MW)
Average Power Demand	40.08
Peak Power Demand	44.08

18.4.2 Power Supply Options

Two power supply options were thoroughly examined during the FS. The first was for a connection on the local electrical transmission grid, and the second was self-generation on site using natural gas reciprocating engines. A hybrid solution, consisting of a grid connection and limited power generating capacity on site was also considered.

18.4.2.1 Power Supply Selection

The main drawbacks arising from a tie-in on the local transmission grid are the following:

- A limited short-circuit capacity at the Project substation's bus-bar, which could be short of the ratings required to properly operate the large variable frequency drives for the large process equipment such as ball mills;
- The grid connection did not provide the full power requirement of the Project, and would require the implementation of generating capacity on site using natural gas;
- Electrical energy cost from the grid was conservatively assessed at CAD 0.09/kWh, CAD 0.031/kWh more expensive than the estimated electrical energy generated with an on-site power generating plant;
- TransCanada Energy East project potential future electricity requirements were not factored in the availability model of the grid power availability analysis performed by the Independent Electricity System Operator ("IESO").

Following these conclusions and for the purposes of the FS, it was decided by GGM that the best technical and economic solution for the Project was to implement an on-site power generating facility operating on natural gas for all the Project's electricity requirements.

18.4.3 Power Plant Design

The power plant design was prepared to optimize the fuel efficiency and minimize the capital expenditures while also providing enough spinning power capacity for peak loads. The optimized operating point of a reciprocating engine usually hovers close to the 85% engine load figure.

The power plant will operate on an N+2 operating philosophy, whereby five units would generate power to the Project network to meet average and peak power demand, one unit would be a hot standby off grid, and one unit would be in maintenance. The availability of the power plant would be close to or considered to be at 100%. Using the results illustrated in Table 18.2, the total online generating capacity would be approximately 48.5 MW.

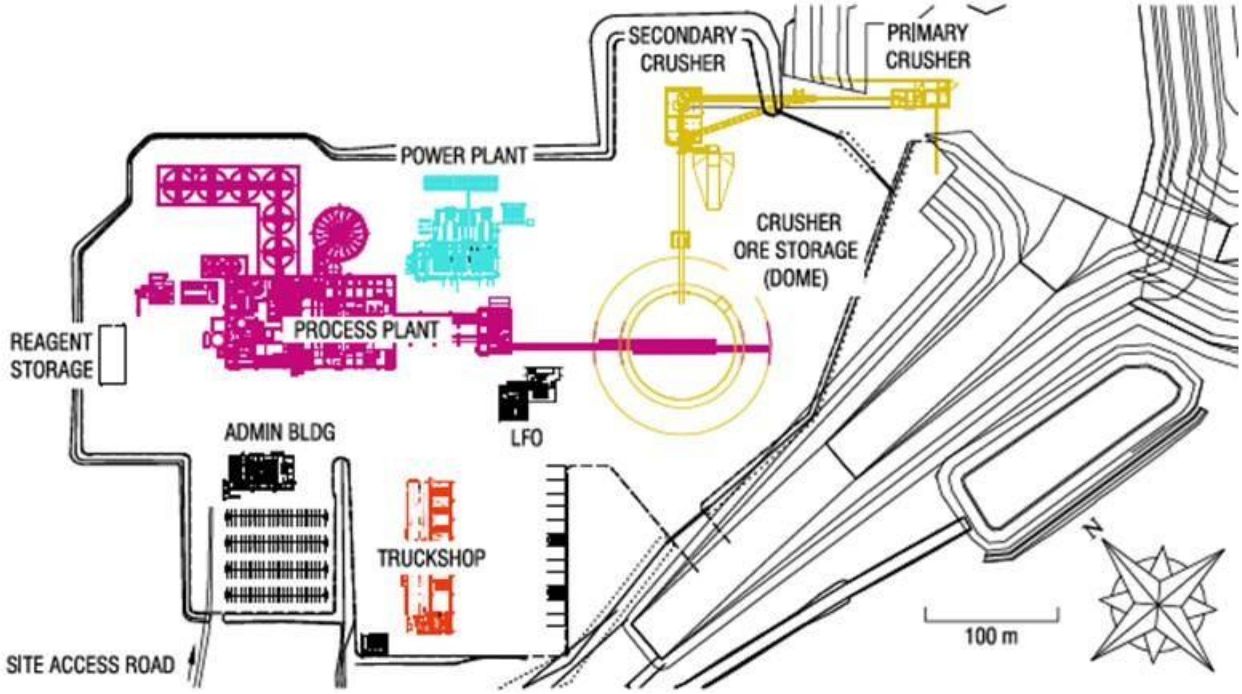
Considering the design requirements, it was decided to design the power plant with seven 9.3 MW generating sets having an output voltage of 13.8 kV at 60 Hz, along with all necessary auxiliary equipment.

18.4.4 Natural Gas Distribution Pipeline

In order to supply the quantity of natural gas required for power generation for the Project, a new tap off the TCPL Mainline and a new distribution pipeline need to be constructed from the Geraldton metering point on the TCPL Mainline to the Project site.

Union Gas, TransCanada’s delivery agent for the Greenstone, Ontario region, was contracted to perform a scoping study and preliminary economic analysis of the construction of the tap and pipeline.

Figure 18.3: Location – Power, Fuel, and Mine Administration



Source: G Mining Services Inc., 2016

18.4.5 Power Distribution

The process plant main electrical room will be fed from the two 3,000 A breakers at the power plant. All power reticulation for the system inside the main building will be done through the process plant main electrical room. Two 1,200 A breakers will be used to feed 13.8 kV overhead power lines, which will service remote areas such as the TMF, the primary and secondary crushers, the office buildings and the truck shop, and the various pumping stations of the Project.

18.5 Mine Support Infrastructures

18.5.1 Truck Shop, Warehouse, Offices

The truck shop and warehouse complex will be located on the southwest side of the concentrator building.

18.5.1.1 Heavy Duty Maintenance

Seven heavy duty maintenance bays will be provided. Each bay will provide 210 m^2 of working area and will be equipped with a roll-up door. The height of the roof will be designed to allow the movement of a 35/5 t overhead crane and to service the different pieces of mining mobile equipment.

18.5.1.2 Mine Fleet Wash System

The wash bay will be located inside the truck shop complex. Water cannons and steam washers will be located on both sides of the trucks. Water, rocks and sediments will be collected to allow re-circulation of water, but also the removal of hydrocarbons and the collection and disposals of solids.

18.5.1.3 Warehouse

The warehouse will be 10.6 m wide x 56 m long with a total storage space of 595 m^2. Heavy and light duty racking will be installed. The warehouse will be used to store consumables and maintenance parts such as hoses, fittings, tools and filters.

18.5.1.4 Truck Shop Offices / Staff Services

A total of 300 m^2 of office space will be provided in the western end of the truck shop spread out on two floors. Washrooms for both men and women and a 75 m^2 lunch room will be located on the bottom floor. On the second floor, 40 m^2 will be used to set up a training meeting room and the remaining 110 m^2 will be used for office space for the mine maintenance staff.

18.5.1.5 Services

Compressed air will be provided throughout the maintenance facility. Lubricants will be stored in the truck shop.

18.5.1.6 Truck Shop HVAC System

The HVAC system will have a total of three roof top units. Natural gas operated heaters will be installed beside roll-up doors and will be used during winter. The system installed in the wash bay will be designed to melt frozen chunks of mud on the mine fleet equipment being washed and serviced.

18.6 Other Project Infrastructure

18.6.1 Information Technology and Communications Systems

The site will be provided with a fibre optic communications link to the global internet backbone, connecting at the intersection of Michael Power Boulevard and the new realignment of Highway 11 on a combination of existing and new electrical poles that will run to the site administration building along the new site access road.

There is also cellular phone coverage in the area.

18.6.2 Roads

The Project benefits from direct access to Trans-Canada Highway 11, approximately 275 km northeast of Thunder Bay and 600 km west of Timmins/Matheson. A section of Trans-Canada Highway 11 will have to be deviated and reconstructed to avoid the Hardrock open pit mine.

A new site access road will be constructed off Trans-Canada Highway 11 to the administration, mineral processing, power generation, and shop facilities.

18.6.3 Assay Laboratory

Assaying services for the Project will be outsourced.

18.6.4 Buildings

The administration offices complex will be located on the southwest side of the process plant building. The building will house the change rooms for both men and women workers, the dry, offices, conference rooms, and a lunch room for the office staff. The first aid station will be located on the first floor.

18.6.5 Temporary Camp

A temporary camp, located on GGM land approximately 2.5 km north of the process plant site and adjacent to Michael Power Boulevard, will be required for the construction and pre-production period to lodge the construction workers since the municipality cannot accommodate that many workers with the existing facilities available in the area. The plan is to hire a camp catering contractor to operate and maintain the camp with an average occupancy of between 400 and 500 persons, with a peak capacity of 650 persons for several months.

18.6.6 Fire Protection

Automated fire detection and protection systems will be installed for all mission critical process areas, such as the crushing, grinding, and process plant buildings and interconnecting conveyor galleries and tunnels, and certain critical infrastructure facilities such as the power plant, warehouses, and fuel storage areas. A fire hydrant network will be installed around the perimeter of the Project infrastructure and process plant site, with fire hose cabinets installed in administrative buildings and the truck maintenance facility.

18.6.7 Security

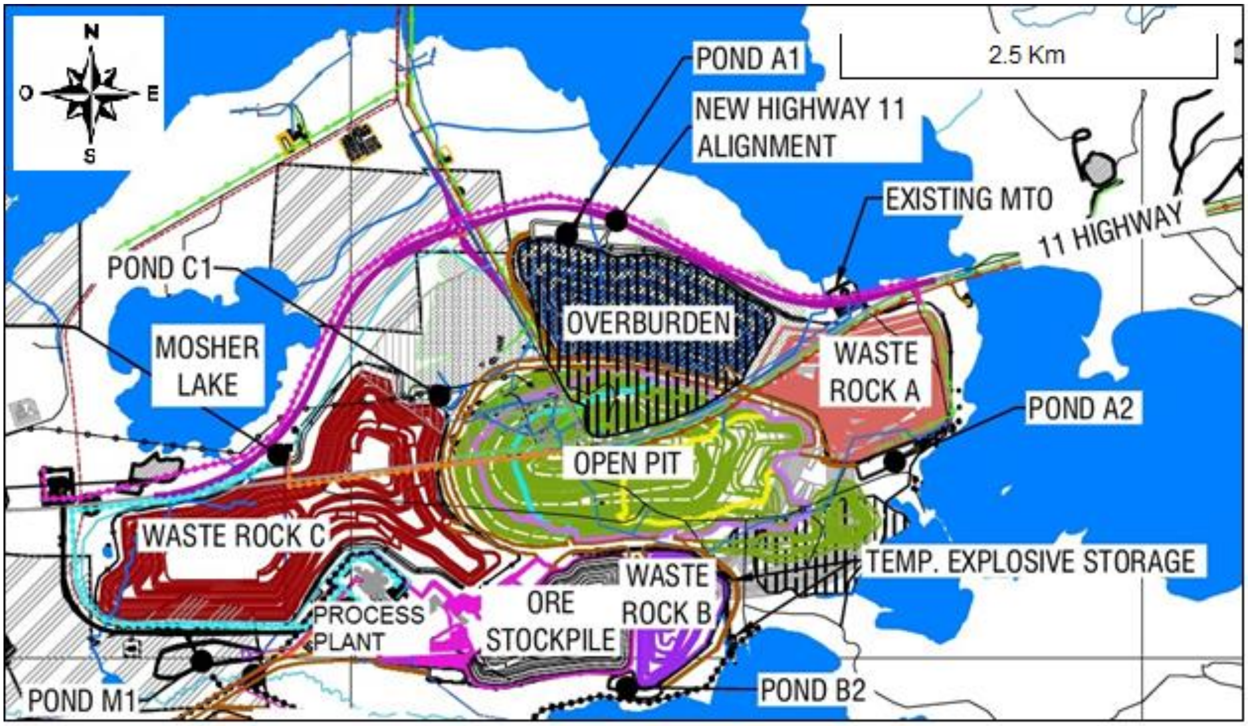
Access to the processing, power, and administration area will be secured by a remotely operated vehicle gate, controlled by security guards on 24-hour duty. The processing facilities and truck shop will be monitored by closed circuit video ("CCTV") surveillance. There will be additional CCTV surveillance of the yard area around the processing, power and administration areas, including the employee parking area and main gate. The refinery and gravity will employ an additional level of security.

18.7 Infrastructure Relocation

18.7.1 Private and Public Infrastructure Relocation

The Hardrock Project is located over or close to existing infrastructure. The plan is to relocate (rebuild or dismantle) the affected infrastructures.

Figure 18.4: Hardrock Project and Existing Infrastructure



Source: G Mining Services Inc., 2016

Currently, Highway 11 passes through the future Hardrock open pit mine and will need to be relocated. Consequently, the configuration of the Michael Power Boulevard will be impacted. Newer segments of Highway 11 and Michael Power Boulevard will be constructed, respectively 4.9km and 0.6km in length, in order to deviate traffic away from the mining operation. The various stakeholders including the MTO and the local community were involved during the preliminary design report. The relocation of the 4.9km segment for Highway 11 involves the construction over historical MacLeod tailings.

The municipality's golf course is located just north of the future open pit mine on land owned by GGM. As a result of the Hardrock Project, part of the golf section will need to be removed to make room for a waste rock dump. The original golf course west of Michael Power Boulevard will remain until the end of the preproduction phase.

An ("OPP") station is located in the vicinity of the future open pit mine operation and needs to be relocated. The new proposed location is at the junction of the new Highway 11 and existing Michael Power Boulevard.

The future Hardrock open pit mine is located very close to the MacLeod and Hardrock townsites. GGM's intention is to acquire all the private properties in the vicinity of the Project.

The Geraldton Heritage Interpretive Centre is located over the MacLeod High Tailings area next to the golf course. The plan is to demobilize this infrastructure at the end of the mine pre-production period or during the first year of operations.

A historical mine headframe was esthetically refurbished to be displayed as a monument at the junction of the existing Highway 11 and Michael Power Boulevard. The plan is to demobilize this headframe at the end of the mine pre-production period or during the first year of operations. The mine does not plan to build a new headframe.

There is an existing gas station located at the junction of Highway 11 and Michael Power Boulevard. This gas station will need to be removed after the realignment of Highway 11.

The MTO owns a highway patrol station located east of the Hardrock Project. The future section of Highway 11 passes through this infrastructure.

GGM proposed a new location to the MTO to relocate this infrastructure. The MTO has agreed on the new location and will provide specifications for the future patrol station. GGM will manage the construction and once the infrastructure is completed, it will be turned over to the MTO.

18.7.2 Historical Tailings Relocation

The historical underground mine operation has an old tailings storage facility, which has been restored. Some parts of this old facility will need to be removed and relocated in the new tailing facility in the first few years of operation, and part of the waste rock/overburden from the mine will be stored over this tailings facility.

18.7.3 Relocation of Electrical Infrastructure

The electrical infrastructure near the Project will need to be moved out of the footprint of the pit to avoid conflicts with the mining operations. GGM and Hydro One have developed a plan by which the existing 115 kV Longlac Transmission Station ("TS") would be relocated approximately 2.2 km west of its current location, along with the incoming 115 kV power line and the outgoing 44 kV feeders servicing the municipalities of Geraldton and Longlac, amongst others.

The dismantling of the existing Longlac TS will be carried out only when the new Longlac TS and all feeders are ready to be energized. GGM will perform the dismantling and all equipment and material will become the property of GGM.

The exiting Hydro One Networks Inc. ("HONI") operating centre that is annexed to the existing substation will be relocated separately to its own location in the Geraldton's designated industrial area.

19. MARKET STUDIES AND CONTRACTS

Neither GMS nor GGM has conducted a market study in relation to the gold metal that will be produced from the Project. Gold is freely traded, at prices that are widely known, so that prospects for sale of any production are virtually assured. Prices are usually quoted in USD dollars per troy ounce.

The gold doré refining agreement will be negotiated once the Project is approved for construction.

20. ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Introduction

This Section provides an overview of the environmental studies and consultation efforts that have been completed to support the federal and provincial environmental approval requirements for the Project. Information on environmental studies and preliminary environmental effects is summarized from the draft environmental effects assessment submitted in January 2016 (Stantec, 2016). Any known environmental issues that could materially impact the Project design through operations and closure are also discussed.

Environmental baseline studies were initiated for the Project in 2013 and were used to identify environmental constraints during the development of preliminary layouts and designs for the Project. This includes consideration of siting and layout of Project infrastructure as well as consideration of design alternatives from an environmental management and approvals perspective. This environmental baseline was the basis for determining incremental changes and predicting environmental effects associated with the Project.

The Project is subject to both federal and provincial environmental assessment ("EA") (refer to Subsection 20.4.1). A draft environmental impact statement / environmental assessment ("EIS/EA") has been completed and submitted to regulatory agencies, Aboriginal groups and the public for review and comment. Thirteen valued components ("VCs") were identified by GGM in accordance with the approved EIS Guidelines issued by the Canadian Environmental Assessment ("CEA") Agency as relevant to the effects assessment process in the EA. They included the atmospheric environment, acoustic environment, groundwater, surface water, fish and fish habitat, upland vegetation and wetlands, wildlife and wildlife habitat, labour and economy, community services and infrastructure, land and resource use, heritage resources, traditional land and resource use ("TRLU") and human and ecological health. Project interactions with the VCs were analyzed to determine potential environmental effects associated with the Project for construction, operation, and closure phases (refer to Subsection 20.4.1.3). In addition to the VCs, the effects assessment also considered effects of the environment on the Project, accidents and malfunction scenarios and cumulative effects.

A preliminary series of follow-up monitoring and environmental management plans were recommended in the draft EIS/EA, including measures related to both compliance and EIS/EA monitoring for all phases of the Project. The preliminary program (refer to Subsection 20.6) is intended to demonstrate the commitment of GGM, as the proponent, to an appropriate and thorough process of verifying predicted effects from the

Project and the effectiveness of mitigation measures. The collective monitoring activities associated with the Project will also be used to inform adaptive management for the Project, if required.

The draft EIS/EA also included a Conceptual Closure Plan, described in Subsection 20.7. The Conceptual Closure Plan includes preliminary details on closure and rehabilitation that may be refined following EIS/EA approval as part of standard process for Closure Plan approval by the MNDM.

The results of the draft EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects. There are no issues identified to date that would materially affect the ability of GGM to extract minerals from the Project; however, agency comments on the draft EIS/EA received to date and potential future conditions of approval could require refinements to Project components or additional mitigation measures to be implemented. Since completing the draft EIS/EA, GGM has completed slight modifications of Project components in response to agency comments, which form the basis for the final mine plan used for this Report. Additional refinements may be made for the final EIS/EA and during detailed engineering. As discussions with agencies are still ongoing, the extent of changes to the Project cannot fully be captured in this Report or FS.

Consultation with stakeholders (community members, agencies, interested parties) and Aboriginal communities is an integral part of the Project. Active participation through consultation during Project planning helps to achieve an open and transparent process, build trust, enhance awareness of the Project and strengthen the quality and credibility of results. Active consultation has been undertaken throughout Project planning including the preparation of the draft EIS/EA, and will continue as the Project progresses. Consultation and engagement activities are described in Subsection 20.5.

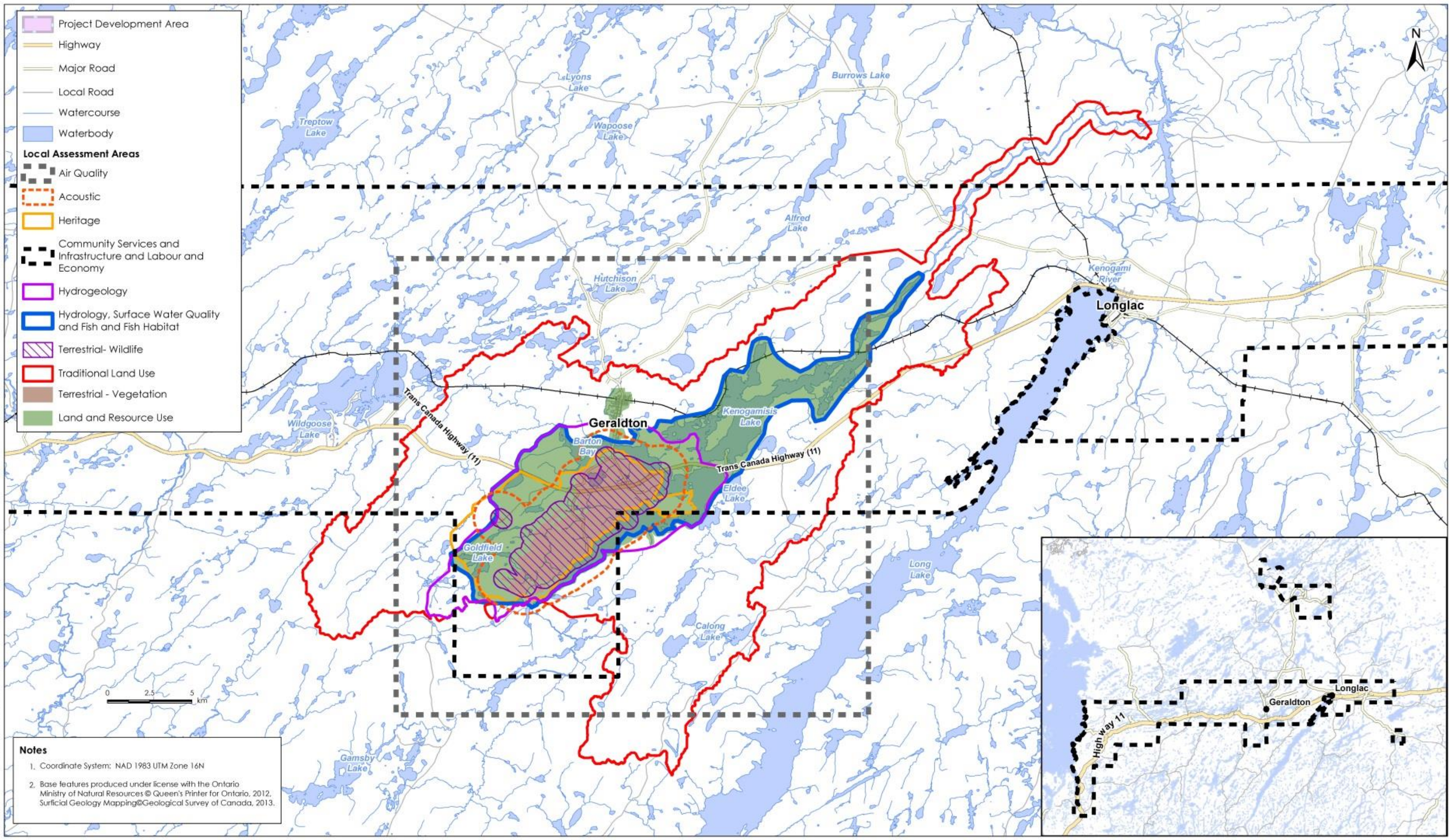
20.2 Environmental Studies

20.2.1 Overview

Baseline environmental studies were completed to characterize the natural, social, economic, cultural and built aspects of the environment that may be potentially impacted by the Project or affect Project design.

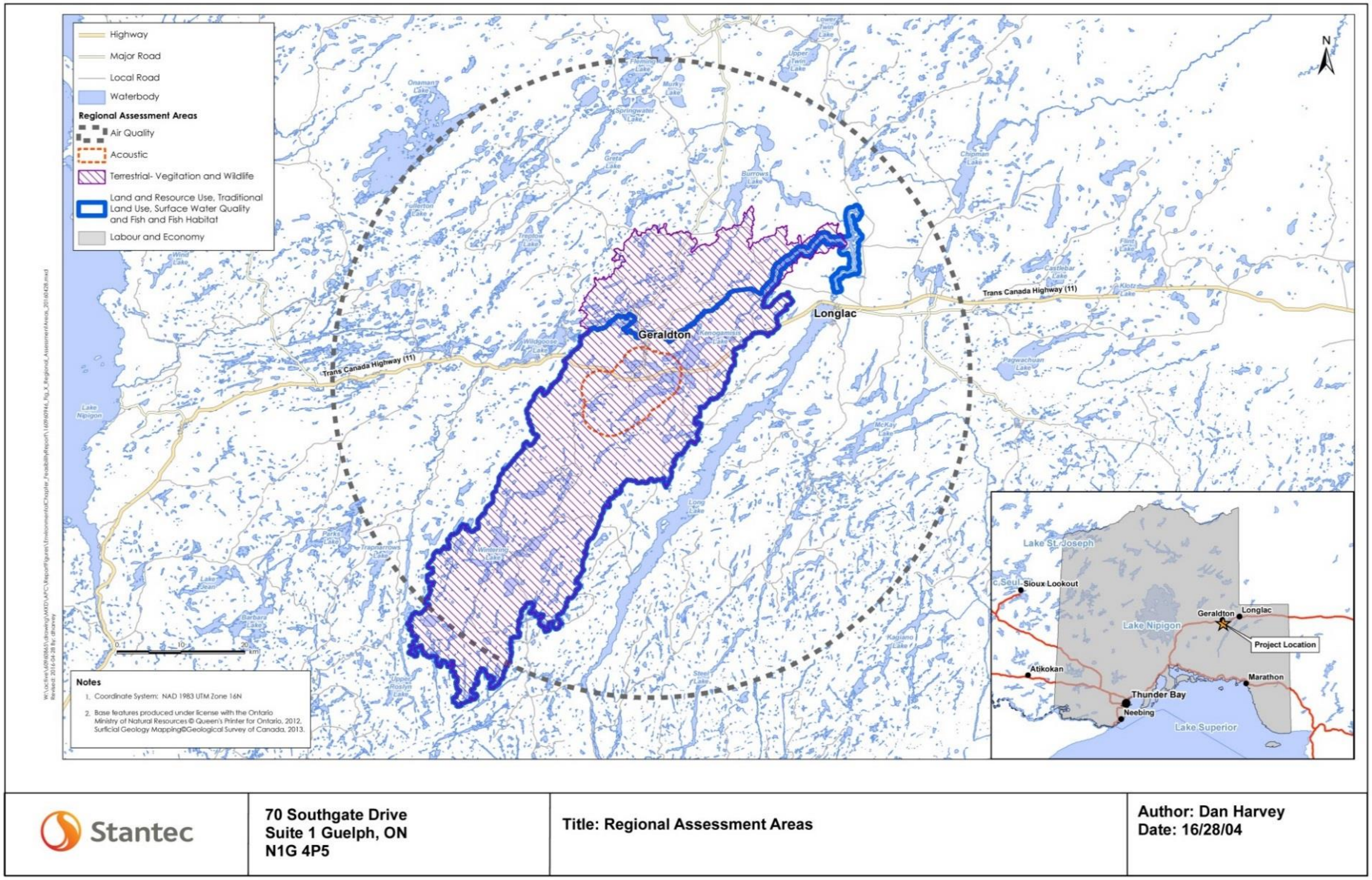
A Project development area ("PDA") was identified (Figure 20.1) and encompasses the Project footprint and is the anticipated area of physical disturbance associated with the construction and operation of the Project. In addition, local assessment areas ("LAA") and regional assessment areas ("RAA") were identified to encompass the areas where there is potential for effects on the environment from the Project (refer to Figure 20.1 and Figure 20.2). Spatial boundaries are defined in each baseline study component, but for the purpose of this Report, the LAAs and RAAs are referred to broadly as the “study area”.

Figure 20.1: Local Assessment Areas



Stantec	70 Southgate Drive Suite 1 Guelph, ON N1G 4P5	Title: Local Assessment Areas	Author: Dan Harvey Date: 16/28/04

Figure 20.2: Regional Assessment Areas



20.2.2 Geology and Geomorphology

20.2.2.1 Physiography

The Project lies within the Boreal Shield, a Canadian ecozone where the Canadian Shield and the boreal forest overlap. Precambrian bedrock at or near the surface plays an important role in shaping the biophysical landscape. Lakes, ponds and wetlands are abundant in this landscape and drainage patterns are typically dendritic, with sporadic angular drainage as influenced by bedrock outcrops.

Topography in the study area is relatively flat to gently rolling with ground surface elevations ranging from 375 masl in the western portion of the PDA to 335 masl along the shoreline of Kenogamisis Lake. Lower lying areas within the PDA are characterized by swamps and ponds with poor drainage throughout the area. The PDA is bounded to the south, east and north by Kenogamisis Lake, which forms the main watershed within the study area. Local water features and topography where an important consideration in the siting and design of key Project components, including the TMF and associated watercourse diversions and the WRSAs.

20.2.2.2 Surficial Soils and Geology

The surficial soils and geology in the study area are typical of the Boreal Forest region overlying the pre-Cambrian shield in northern Ontario. Soils are relatively young, exhibiting less than 10,000 years of development and consist of organic muck (comprising about 36% of the total area) and well-drained brunisols over thin bedrock (comprising about 35% of the area), with poorly drained gleysols accounting for 13% of the area. The remaining 16% of the PDA is either developed land or water.

Surficial geology consists of large areas of glacial till, outwash and glaciolacustrine sediments, and glaciofluvial deposits. Unique to this area is the presence of a high percentage of calcareous (carbonate rich) substrates. Carbonates are commonly found throughout all modes of soil deposition within the study area. Till and other discontinuous drift (gravelly silty sand to sandy silt) is mapped in the northern and western portions of the PDA, generally near the proposed open pit and northern portion of the TMF. Subaqueous outwash and associated glaciolacustrine sediment (rippled silty fine to very fine sand, silts, and minor clay as thin interbeds) occur along the eastern portion of the PDA, primarily to the south of the open pit in the areas of WRSA D and the southern portion of the TMF. Organic deposits such as peat or muck are present in wetlands and river valleys and are typically between one metre and three metres thick. Ice contact glaciofluvial sediments (sand and gravel) or thick till (gravelly clayey silt to gravelly sandy silt) are located along the western boundary of the PDA and correspond to an esker that originates just north of Goldfield Creek and extends southwest through Goldfield Lake.

20.2.2.3 Bedrock Geology

A detailed description of the bedrock geology and controls on mineralization is presented in Section 7.0.

20.2.3 Acid Rock Drainage/Metal Leaching Potential

A comprehensive geochemical testing program was initiated in 2013 to characterize waste rock, ore, overburden and tailings associated with the Project. Testing included Acid-Base Accounting ("ABA"), Shake Flask Extraction, total metals and laboratory and field kinetic tests with the field kinetic testing program continuing through 2015/2016. Subsequent testing has been initiated to further refine the geochemical characterization of waste rock and, when complete, will be used for mine planning and development of a detailed waste rock management plan. The following section presents a summary of the results of the geochemical testing program up to the end of 2015.

Overall, the ore, waste rock and tailings materials contain relatively low Acid Rock Drainage ("ARD") potential but will still require consideration of how to best manage effects from existing Potentially Acid Generating ("PAG") material in the design of these Project components. Overburden will not require any management for ARD potential. Measures to mitigate potential effects to water quality due to metal leaching have been documented in the preliminary Water Management Plan for the Project included in the draft EIS/EA.

20.2.3.1 Overburden

Overburden is classified as non-Potentially Acid Generating ("non-PAG") material and is unlikely to generate acidic leachate. The potential for leaching of arsenic, and potentially cobalt and copper, was identified for soils in the area of the historical MacLeod-Mosher and Hard Rock plant sites. Average field leaching rates for these elements declined between 2014 and 2015, indicating a declining trend over time.

20.2.3.2 Waste Rock

Up to 4.0 weight percent of PAG material was estimated based on the C_{total}/S_{total} threshold and mainly corresponds with waste rock associated with the sulphide replacement zones. The onset time for ARD conditions is estimated to be 70 years after exposure to atmosphere. The relatively low percentage of PAG rock and the long ARD onset time provide management flexibility for this material. Co-deposition of PAG and non-PAG waste rock has been identified as the preferred option as outlined in the preliminary Waste Rock Management Plan developed for the Project and included in the draft EIS/EA.

The data from the field kinetic tests were demonstrated to be more reflective of actual site conditions than leaching rates obtained in the laboratory humidity cell data, providing a longer testing period under field conditions for evaluation of long-term leaching behaviour. Results from field kinetic testing indicate that average concentrations were below the Schedule 4 criteria under the MMER for all lithologies with the following parameters above the Ontario Provincial Water Quality Objectives ("PWQO"):

- Clastic sediments – WR-S (72% of waste rock): the PWQO for arsenic and cobalt and the Interim PWQO for arsenic, antimony, aluminum and uranium;
- Intrusive rocks – WR-I (11% of waste rock): the PWQO for arsenic and the Interim PWQO for arsenic, antimony and aluminum;
- Chemical sediments – WR-C (17% of waste rock): the Interim PWQO for arsenic and antimony.

Field leaching rates of these elements generally declined between 2014 and 2015 in samples representing major rock types (WR-S, WR-I and WR-C), indicating a significant reduction in leaching rates over time. Additional laboratory and field testing is being conducted to investigate relationships of arsenic leaching with other parameters to support the development of potential waste rock management strategies. Metal leaching under neutral conditions was a key issue evaluated in the draft EIS/EA and in the development of the mine plan.

20.2.3.3 Future Tailings

Ore samples and tailings have similar ABA characteristics before and after metallurgical tests. Ore and tailings also have similar neutralization potential ratio thresholds for ARD classification with PAG tailings estimated at 9.7% with a minimum ARD onset time for PAG tailings is 12 years based on laboratory neutralizing potential depletion rates. These rates are expected to be slower under field conditions and addressed through progressive rehabilitation and closure of the TMF.

In the TMF pond, concentrations of metals and total cyanide are predicted to meet MMER criteria based on results of ageing tests. Unionized ammonia, cobalt, copper, arsenic, antimony, silver and free cyanide were identified as parameters of potential concern during operation based on comparison with the PWQO. Water from the TMF will not discharge directly to the environment and toe seepage will be collected and pumped back to the TMF pond during operation.

At closure, water and seepage collected in the TMF will be sent to the open pit to help expedite filling of the open pit. Once water quality meets acceptable criteria for discharge, the emergency spillway will be lowered and water will be discharged to the Goldfield Creek diversion.

20.2.4 Atmospheric Environment

The Project is located in a rural location of northern Ontario where air quality is primarily influenced by the Ward of Geraldton and traffic on Highway 11. Measured levels of nitrogen dioxide, sulphur dioxide and inhalable particulate matter were below their applicable provincial criteria. The maximum measured concentrations of total suspended particles and all metals with the Ministry of the Environment and Climate Change (“MOECC”) air quality criteria were well below their applicable criteria. The maximum measured concentrations of all volatile organic compounds with MOECC air quality criteria were well below their applicable criteria.

20.2.5 Acoustic Environment

The major contributors to baseline acoustical environment were found to be the traffic noise from Highway 11, Michael Power Boulevard and the natural environment. Baseline sound levels were found to be dominated by traffic noise during the daytime and natural environment during the nighttime. No “non-traffic anthropogenic sources” were found to be major contributors to the acoustic environment, and no tonal or excessive low frequency noise was encountered during field studies. The field observations and measurements of traffic noise indicate that the receptor area is characteristic of a Class 2 acoustical environment under baseline conditions as defined by the MOECC.

20.2.6 Groundwater

Field activities to confirm hydrogeological conditions were completed from 2013 to 2015 and included borehole drilling and groundwater monitoring well installation, well development, hydraulic response testing, test pits, drive point piezometer and pressure transducer installation, water level monitoring and groundwater quality sampling.

The overburden and shallow bedrock are considered to be hydraulically connected. Groundwater levels are generally found at 1 to 2 m below ground surface with groundwater flow strongly influenced by topography. Overall, the regional groundwater flow within overburden is toward the east, southeast toward Kenogamisis Lake in the southern portion of the PDA, with radial flow towards Barton Bay and the Central Basin of Kenogamisis Lake in the area of the open pit. Significant water producing fractures or faults were not encountered during the drilling and testing completed, suggesting that significant water inflow issues are not expected during open pit development. This is supported by the historical underground mining that did not identify significant water inflow issues.

Elevated concentrations of hardness, iron, manganese and colour were consistently observed at the majority of background monitoring wells in the overburden and bedrock and are typical of groundwater in Ontario and are reflective of the natural mineralization and geochemical processes in the area. Overburden and bedrock water quality away from historical mining areas was generally of good quality with parameters occasionally above the Ontario Drinking Water Standards reflective of location conditions.

There are several historical or existing land uses identified that have contributed to the degradation of water quality in the Project area. The historical MacLeod and Hard Rock tailings contain elevated concentrations of cyanide, arsenic, cobalt and nickel with the historical Hard Rock tailings generally similar to the MacLeod tailings with the exception of a small area that is considered acid generating (referred to as the reactive tailings) and elevated cadmium and zinc above the MOECC Aquatic Protection Values. Seepage from the historical tailings has been identified to affect water quality within Barton Bay and the Central Basin of Kenogamisis Lake, particularly arsenic concentrations which are well above the PWQO within the historical tailings.

The historical underground workings associated with the MacLeod-Mosher and Hard Rock mines are currently flooded and will need to be dewatered prior to mining. Water quality associated with Hard Rock Shaft No. 1 had concentrations of cobalt that consistently exceeded the PWQO and concentrations of chloride, arsenic, copper, iron and zinc that at times exceeded the PWQO or Interim PWQO. Water quality associated with the MacLeod-Mosher underground workings was characterized from samples collected at Mosher Shaft No. 1. They showed concentrations of iron and zinc that consistently exceeded the PWQO and concentrations of arsenic that consistently exceeded the Interim PWQO. No discharge from the MacLeod-Mosher underground workings currently occurs.

20.2.7 Soil Quality

Soil investigations in the area of the former MacLeod-Mosher and Hard Rock plant sites identified elevated arsenic concentrations at approximately 65% of the test pits/test holes and antimony and cobalt at approximately 25% of the test pits/test holes. Various other metals were found to exceed the applicable standards in approximately 10% of the test pits/test holes. Hydrocarbon impacts were identified associated with active and former fuelling stations that will be decommissioned as part of the Project as they are located within the open pit footprint. Hydrocarbon and metal affected soil exceeding applicable criteria may be managed through a combination of remediation by land farming and management within the waste rock storage areas as applicable under the *Environmental Protection Act* and *Ontario Regulation 347*. A soil management plan will be prepared to provide guidance on the management of excess soil generated during the development and operation of the Project, including soil contaminated by historical mining activities and management of historical tailings.

20.2.8 Surface Water

20.2.8.1 Hydrology

The PDA is located in the Kenogamisis River watershed, adjacent to Kenogamisis Lake. The lake is long, narrow and shallow and consists of four main basins referred to as the Southwest Arm, Barton Bay Basin, the Central Basin (sometimes called MacLeod Basin) and Outlet Basin (sometimes called the Northeast Arm). Water levels within Kenogamisis Lake are controlled by the Kenogamisis Lake Dam, which is operated under the guidance of the Aguasabon River System Water Management Plan. The normal operating water level range for Kenogamisis Lake is between 329.32 and 329.70 masl with two Cautionary Compliance Zones to provide flexibility during winter and spring freshet conditions.

The Kenogamisis River is the major river in the study area. Its watershed area upstream of the Southwest Arm of Kenogamisis Lake is 760 km^2 and provides approximately 92% of total inflow into the Southwest Arm of Kenogamisis Lake and 65% of total inflow into the Outlet Basin of Kenogamisis Lake. The flow regime of the Kenogamisis River is similar to other rivers in the area with high spring flows in April–May (sometimes early June, as in 2014) and low flows in summer (July–August) and winter (November–March).

The two primary permanent watercourses located in the PDA are the Southwest Arm Tributary and Goldfield Creek. The Southwest Arm Tributary is a second order tributary draining directly to Kenogamisis Lake. The main branch of this watercourse originates in a wetland that drains eastward for a distance of approximately 3.3 km before discharging into Kenogamisis Lake. Goldfield Creek is a larger watercourse with a watershed area of 32 km^2. The creek originates at Goldfield Lake and drains in an easterly direction towards Kenogamisis Lake. Goldfield Creek will be diverted to allow construction of the new TMF and connected to the Southwest Arm Tributary. The realignment will form part of the overall fish habitat compensation plan for the Project. Other areas of the PDA drain towards Mosher Lake and Barton Bay and the Central Basin of Kenogamisis Lake.

20.2.8.2 Surface Water Quality

Baseline water chemistry data were collected monthly or bimonthly since 2013 and compared with historical results spanning almost 40 years of data. Surface water quality was generally moderately hard (moderately high mineral content), circumneutral in pH (mean values of 6.1 to 8.1), with mean total dissolved solids concentrations in the range of 107 to 131 mg/L and are typical of northern Ontario lakes. Nutrient levels tended to be low, except for Barton Bay which is affected by discharge from the municipal sewage treatment plant. With the exception of arsenic, copper, iron and lead metal concentrations in Kenogamisis Lake were present at levels below the Canadian Water Quality Guidelines for Freshwater Aquatic Life ("CWQG") and PWQO. Seasonal and spatial trends were evident in the data with the lowest concentrations during the spring freshet, and increased gradually through the summer and fall. Barton Bay and the Central Basin of Kenogamisis Lake have the highest metal concentrations and are attributed to the effects of historical mine operations and sewage treatment plant discharges in Barton Bay. In lakes and creeks sampled as unaffected background or reference lakes, the majority of metals were below CWQGs and PWQOs with the exception of arsenic and iron.

Historical mining activities have contributed to the degradation of groundwater and surface water quality within the area of the PDA. An assessment of arsenic loading to Kenogamisis Lake was completed using a mass balance approach, which provides an accounting of the total arsenic loading on both an individual basin and overall lake perspective. The mass balance calculations indicate that while a small component of flow, the discharge of groundwater from historical tailings represents approximately 60% of the total arsenic load leaving the Outlet Basin, and about 55% of the total load leaving Kenogamisis Lake at the control dam. By the time water from Barton Basin mixes with water from the Central Basin and Southwest Arm, mean arsenic concentrations are at 9 µg/L, just above the Interim PWQO of 5 µg/L, with concentrations remaining similar through the Outlet Basin.

20.2.9 Fish and Fish Habitat

Characterizing fish and fish habitat in the study area included a review of background information and the completion of fish habitat assessments during six separate periods in 2013, 2014 and 2015. Lakes within the study area provide cool water habitat with larger lakes such as Kenogamisis and Goldfield Lake providing a diversity of aquatic vegetation, cover and substrate types. Larger lakes also provided greater bathymetric structure (i.e., humps, shoals, flats, etc.).

There was an abundance of potential spawning habitat for Northern Pike and Yellow Perch throughout most lakes within the study area. Important spawning and feeding habitat for species like Walleye and Lake Whitefish was documented where the Kenogamisis River and Magnet Creek flow into Kenogamisis Lake.

Important spawning habitat for these species may also be provided by rocky mid-lake shoals in Kenogamisis Lake and Goldfield Lake.

Moderate sized streams such as Goldfield Creek and its main tributary provided a variety of cover types and habitats, although riffle habitat was limited throughout the study area. These streams provided an abundance of potential Northern Pike spawning habitat in adjacent wetlands when they become inundated in the spring. Despite good cover, fish abundance and species diversity were low in the study area streams. The exception to this was large numbers of small bodied fish that may use lower stream mouths to spawn.

Shallow, isolated ponds and first order watercourses in the study area likely freeze to the bottom in winter, limiting fish use of these types of habitat. Highly organic substrates and ice cover may also create anoxic conditions in these areas, further limiting fish distribution.

More than 6,080 individual fish, consisting of 24 species were captured by Stantec in the study area between September 2013 and October 2015. No species identified were listed as federal or provincial species at risk (“SAR”), nor are SAR expected to occur in the study area. Game and sustenance fish species, including Walleye, Lake Whitefish, Northern Pike, Yellow Perch and Burbot, were present in Kenogamisis and Goldfield Lakes.

Extensive data on metals in fish tissue from Kenogamisis Lake have been collected by the MOECC for more than 30 years. These data were collected for large bodied fish, primarily sport fish. Mean total arsenic concentrations in forage fish were higher than in game fish. There is no standard provincial or federal consumption guideline for arsenic; however, sport fish from the study area did not exceed consumption guidelines published for other countries. Background concentrations of total mercury in Walleye were above the partial restriction guideline for human consumption (0.26 mg/kg), but no spatial trend was evident for the species sampled. A bioavailability study was completed and concluded that, while the current elevated levels of arsenic and other metals in water and sediments of Barton Bay and the Central Basin may lead to bioaccumulation, they do not lead to any recognizable adverse effects on phytoplankton, benthic invertebrates, or fish populations studied.

20.2.9.1 Sediment Quality

Sediment samples were collected throughout the study area in 2013 and 2014 to supplement sediment data collected from Kenogamisis Lake in 2011. Copper and arsenic commonly occur in sulphide-based minerals and the Geraldton area is rich in such minerals, so some naturally elevated levels of copper, arsenic and other metals are expected. Arsenic exceeded the MOECC Lowest Effect Level (“LEL”) and

Severe Effect Level ("SEL") at most sampling stations including Lake A-322, Goldfield Lake and Mosher Lake. Within Kenogamisis Lake, the SEL for arsenic was only exceeded in Barton Bay and in the Central Basin. Common parameters that exceeded the LEL in the study area were cadmium, chromium, copper, lead and nickel. Exceedances of the LEL for zinc occurred in individual replicates from the Central Basin and Barton Bay.

20.2.10 Upland Vegetation and Wetlands

The Project is located along the southern boundary of the Boreal Forest Region, in northern Ontario. Typical forest cover is a mix between deciduous and coniferous forest cover as well as coniferous swamp; vegetation communities are predominantly coniferous with deciduous associates. White and black spruce, tamarack, balsam fir and jack pine are common throughout the area with frequent occurrences of deciduous vegetation communities and species, including white birch, trembling aspen and balsam poplar. Anthropogenic disturbances in the Project area have resulted in a variety of vegetation communities, ranging from open disturbed sites showing early successional growth to mature naturalized deciduous and coniferous forest communities. In the PDA, ecosites were approximately 40.9% conifer dominated upland forest, 9.8% hardwood dominated forest, 12.4% early successional/disturbed forest, 35.9% conifer dominated swamp and <1% open wetland (marsh, bog, and fen) communities. The remaining <1% cover was shallow open water.

A total of 245 species of vascular plants were recorded in the study area, of which 91% (223 species) were native and 9% (22 species) were exotic. No plant SAR or Species of Conservation Concern ("SOCC") were recorded in the study area during botanical inventories, and are assumed to not be present in the study area. No known Provincially Significant Wetlands or provincially rare wetland communities were identified in the study area. One sensitive, but not provincially designated as rare, wetland community was identified adjacent to the northeast limits of the proposed TMF. Although this ecosite community type (B136) is not listed as a provincially rare vegetation type, it could be considered a sensitive feature due to its dependence on nutrient rich springs and groundwater, and its ecological characteristics.

20.2.11 Wildlife and Wildlife Habitat

Field investigations identified a variety of wildlife species in the study area. Fifteen SAR and SOCC were recorded during baseline surveys for the Project. Of these, six are confirmed to be either resident or breeding within the vicinity of the Project: Canada warbler, eastern wood-pewee, common nighthawk, northern myotis, little brown myotis and taiga alpine butterfly.

The analysis of results identified a number of terrestrial features and associated wildlife habitat for federal and provincial SAR and SOCC within the study area. The following habitat for federal and provincial SAR, and SOCC were identified during baseline field investigations:

- Barn Swallow nesting habitat;
- Canada Warbler breeding habitat;
- Common Nighthawk breeding habitat;
- Eastern Wood-pewee breeding habitat;
- Olive-sided Flycatcher breeding habitat;
- Little Brown Myotis and Northern Myotis habitat in deciduous and mixed forests with stand size >25 cm diameter at breast height and tree height >10 m.

To provide a comprehensive approach to identifying and evaluating wildlife habitat, significance at the provincial level has been assessed based on guidance provided in the Ministry of Natural Resources and Forestry ("MNRF") Natural Heritage Reference Manual. The following significant wildlife habitats were identified during baseline field investigations:

- Habitats of seasonal concentrations of animals for: moose late winter cover; aquatic waterfowl stopover and staging areas (in the combined area of Kenogamisis Lake and 100 m radius including wetland/shoreline ecosites); turtle wintering areas in Kenogamisis Lake; and snake hibernacula.
- Rare vegetation communities and specialized habitats for: sparse treed fen; waterfowl nesting areas; moose aquatic feeding areas; woodland raptor nesting habitat; seeps and springs.
- Habitats of SOCC for Taiga Alpine.

20.2.12 Labour and Economy

Between 2006 and 2011, the population of Ontario increased by approximately 5% while the populations of the Thunder Bay District and the Municipality of Greenstone decreased by approximately 2% and 4%, respectively. Available population projections indicate that the municipality will continue to see population decline, with an estimated population of 4,618 residents in 2018 and 4,480 residents in 2023.

The Northwestern Ontario economic region includes the Districts of Thunder Bay, Rainy River and Kenora. Spatially, this is the largest economic region in the province, while also having the smallest population of all Ontario economic regions. Mining is a key component of the economy in Northwestern Ontario with over 80 active exploration projects during 2012, as well as six operational mines.

Key industries providing employment locally in the Municipality of Greenstone include: trades, transport and equipment operations; processing, manufacturing and utilities; agriculture and resource-based industries, including mining and forestry. Baseline economic conditions indicate that the Greenstone economy has been in decline, with the number of people in the labour force decreasing by 11% between 2006 and 2011, and the unemployment rate increasing by two percentage points. In comparison, the size of the labour force in the District of Thunder Bay decreased by 4% over the same period, while the Ontario labour force increased by 4%. Within the Municipality of Greenstone, there are higher rates of unemployment in Aboriginal communities than in non-Aboriginal communities.

The Thunder Bay District is expected to experience a shortage of skilled workers for mining projects, primarily because there is a lack of younger people with appropriate skills coming into the regional labour market. Increased recruitment and retention challenges are also anticipated as competition for workers increases.

20.2.13 Community Services and Infrastructure

The Town of Geraldton, centrally located in the Municipality of Greenstone, is the service support centre for the surrounding region including government services (MNRF/Regional Fire Management), Medical Services (District Hospital), financial services and retail. Overall, the Project is located relatively close to existing municipal and provincial services, including water and wastewater, waste, transportation, power, recreational and emergency services. Key local community services and infrastructure in the study area include:

- Municipal features, including a park, public boat launches and public beaches, among other urban land uses;
- Kenogamisis Golf Club;
- MacLeod and Hard Rock townsites;
- Hydro One infrastructure, including a substation and power lines;
- Discover Geraldton Interpretive Centre;
- Highway 11 and Michael Power Boulevard;
- Gas station;

- MacLeod-Cockshutt Mining Headframe;
- Ontario Provincial Police station.

Some municipal services and infrastructure has been reported to be at or near capacity, including waste water systems and solid waste facilities. Health care availability in the Municipality of Greenstone may become an issue since the area is considered to be underserviced by health care professionals. Meanwhile, due to population decline, there has been a surplus of housing in some communities in the Municipality of Greenstone and there are some underdeveloped, designated residential areas to accommodate larger-scale future growth in the Project vicinity, including in Beardmore, Longlac, Nakina and Geraldton.

20.2.14 Land and Resource Use

Land and resource use has been shaped by mining and forestry activity. In the early 1930s, the region became known for gold mining; however, extraction ceased during the 1970s leaving forestry as the main industry and land use in the region. Today, the most extensive land uses in the area are forestry, hunting, trapping and fishing, and local features include:

- MacLeod Provincial Park, which includes a campground, walking trail, cross-country skiing trails and public beach;
- Ward of Geraldton, which includes a municipal park, public boat launches and public beaches, among other urban land uses;
- Crown land campsite;
- Kenogamisis Lake Resort (guide outfitter);
- Snowmobile trails;
- Canoe routes along the Kenogamisis River;
- Planned forest harvest areas and forest access roads;
- Trapline areas (GE021, GE022, GE065);
- Bear Management Areas (GE-21A-032, GE-21A-027);
- Bait Harvesting Areas (NI5035, NI5036, NI5027, NI5028).

20.2.15 Heritage Resources

20.2.15.1 Archaeology Resources

A Stage 1 Archaeological Assessment was completed for the Project to compile all available information about the known and potential archaeological heritage resources within the study area and to provide specific direction for the protection, management or recovery of these resources. A Stage 2 assessment was subsequently completed for areas of high archaeological potential, including areas near water sources, transportation routes and townsites. The Stage 2 assessment concluded that no archaeological resources were found in the PDA with no further archaeological assessments recommended.

20.2.15.2 Architectural/Historical Resources

A Cultural Heritage Evaluation Report has been completed to screen for resources of potential cultural heritage value or interest ("CHVI"), as defined by Ontario Regulation 9/06. Twenty-eight heritage resources were identified on properties which may be affected by the Project, the vast majority in residential developments constructed by mining companies; six resources are situated within Rosedale Point, 14 are situated within the MacLeod townsite, and one is situated in the Hard Rock townsite.

20.2.16 Traditional Land and Resource Use

TLRU includes traditional activities, sites, and resources identified by Aboriginal communities. Project engagement activities and the review of Project-specific traditional knowledge ("TK") studies, land use survey results and existing literature have confirmed the potential for Project effects on TLRU. One gathering site (a former family settlement) was identified immediately north of the PDA. Campsites, cabins, and sacred sites were also identified in the regional area but not within the PDA. Traditional activities (e.g. hunting, fishing and trapping) also occur in the Project area.

In 2012, the predecessor to GGM supported the efforts of MNDM and local Aboriginal communities in initiating comprehensive cultural impact assessment ("CIA") studies to be carried out by Ginoogaming First Nation, LL #58 First Nation, and Aroland First Nation. The intent of these studies was to provide communities with the information required to make an informed decision regarding the advanced exploration phase, which was never carried out by the predecessor to GGM. The CIAs have been completed by Ginoogaming First Nation, LL #58 First Nation and Aroland First Nation, and provide information on the community values associated with the PDA and continue to set the stage for additional dialogue and assessment by communities of effects going forward. As these studies are confidential in nature, they are not provided in the current draft EIS/EA, although they have helped to inform decision-making on the Project.

The opportunity to provide TK/traditional land use input into Project planning has been discussed with communities. The approach to TK sharing (for both the draft and final versions of the EIS/EA) is dependent on the specific preference of individual communities involved in this aspect. GGM has made early and ongoing efforts to provide opportunities for TK sharing. Consultation on TK will be ongoing throughout the Project and will evolve over time with applicable Aboriginal communities.

20.3 Environmental Constraints

The Project is located within an area bounded by Kenogamisis Lake to the north, south and east, with wetland and low-lying areas and associated surface water features to the west. These constraints have been incorporated into the design of the Project, which has focused on minimizing the environmental footprint of the Project while respecting environmental features and required setbacks.

Acquisition of a number of properties is required as they will need to be removed to allow development of the Project. These include provincial infrastructure related to the MTO patrol yard, Hydro One transmission and distribution power lines and associated substation, OPP station, the Discover Geraldton Interpretive Centre, properties within the MacLeod and Hard Rock townsites and Dan’s General Store (Husky Gas Station). An environmental screening report has been completed by TBTE (TBTE, 2015) for the proposed Highway 11 realignment, which included the MTO patrol yard, Mosher portal area, historical MacLeod tailings and the MacLeod Mine landfill. Soil and groundwater impacts were identified at the Mosher portal area and at the historical tailings and MacLeod Mine landfill with a soil management plan being developed for implementation during construction. The potential for soil and groundwater impacts were identified for the MTO patrol yard based on typical land use; however, actual site specific investigations have not been completed at this time. Modified Phase 1/2 Environmental Site Assessments (TBTE, 2015) were completed for a former gas station property (former Larry’s Esso) and the current gas station property located at the intersections of Highway 11 and Michael Power Boulevard. Soil impacts associated with petroleum hydrocarbons and arsenic and groundwater impacts associated with petroleum hydrocarbons were identified. A Soil Management Plan will be prepared to provide guidance on the management of excess soil generated during the development and operation of the Project.

Historical mining activities have contributed to the degradation of groundwater and surface water quality within the area of the PDA, particularly with respect to the historical tailings. As discussed in Subsection 20.4.1.3.4, it is anticipated that the Project will result in an improvement in water quality within Kenogamisis Lake, having a positive effect on arsenic and iron concentrations due to the reduction in groundwater discharge associated with the historical MacLeod and Hard Rock tailings. This will be achieved through removal of a portion of the historical tailings for storage within the newly constructed TMF, installation of seepage collection around the historical tailings as part of the berm and buttress construction to address long term physical stability, improved cover design for the remaining historical tailings and changes in groundwater flow during operations that will allow impacted groundwater to be captured within the open pit and treated prior to discharge. The Conceptual Closure Plan, and ultimately the Closure Plan, will account for any required rehabilitation activities of historical tailings and will be completed in accordance with O. Reg. 240/00.

Historical mine openings exist within the PDA and are currently capped or secure. The condition of the caps and security of the existing mine openings will be evaluated with respect to the requirements of O. Reg. 240/00 during preparation of the draft Closure Plan and upgrades will be completed as required. For the majority of the mine openings, they will be removed during development of the Project and, as a result, a limited number of openings will remain at closure.

Seven provincial SAR or their habitats have the potential to occur on site: American White Pelican (*Pelecanus erythrorhynchos*), Bank Swallow (*Riparia riparia*), Barn Swallow (Hirundo rustica), Eastern Whip-poor-will (*Caprimulgus vociferous*), Little Brown Myotis (*Myotis lucifugus*) and Northern Myotis (*Myotis septentrionalis*). All species and their habitats are protected by the *Endangered Species Act*, 2007 with authorizations being provided by MNRF during permitting to develop these lands. GGM will submit applications for the appropriate authorizations to the MNRF prior to Project development (refer to Subsection 20.4.2).

Development of the Project will alter existing activities and facilities within the PDA, including the MacLeod-Cockshutt Mining Headframe, the Discover Geraldton Interpretive Centre and the Kenogamisis Golf Club. Discussions between GGM, MNRF, the Municipality of Greenstone and other affected stakeholders are ongoing.

MacLeod Provincial Park is located 350 m east of the PDA. There are no other provincially or federally protected areas such as national parks, protected areas, ecological reserves, or conservation reserves near the Project. There are no Areas of Natural and Scientific Interest, or evaluated Provincially Significant Wetlands within or near to the Project site. One sensitive, but not provincially designated, rare, fen community was identified immediately adjacent to the PDA. There are no areas of archaeological resources identified through baseline studies at the Project site.

20.4 Environmental Approval Requirements

20.4.1 Environmental Assessment

20.4.1.1 Overview

Federal EA is regulated under the *Canadian Environmental Assessment Act 2012* ("CEAA 2012"), and is administered by the CEA Agency. Under CEAA 2012, "designated" projects included in the *Regulations Designating Physical Activities* require a federal EA. The Hardrock Project has been confirmed as a Designated Project and a federal EA is being implemented in accordance with the approved EIS Guidelines issued to GGM by the CEA Agency on August 5, 2014, with subsequent amendments on February 11, 2016 to include consideration of greenhouse gas emissions and February 12, 2016 related to changes in the list of Aboriginal communities with which GGM is expected to engage.

Under Ontario's *Environmental Assessment Act* ("EAA"), mining development projects are not subject to provincial individual EA requirements because they are carried out by private sector proponents. GGM entered into a Voluntary Agreement with the MOECC to make the entire Project subject to a single individual EA process in accordance with the approved Terms of Reference ("ToR") received from the province. A final ToR was submitted to the MOECC on January 2, 2015, and an editorial amendment submitted on March 31, 2015 for completion of the provincial individual EA under the EAA. The final ToR was approved with amendments on June 24, 2015; it provides the framework for the individual EA and outlines key steps and requirements to undertake an EA process and prepare an EA report compliant with EAA.

GGM is proceeding with a coordinated EA to address both federal and provincial EA requirements through a single process, which will result in the filling of a single body of information that addresses both provincial and federal EA processes, culminating in one single EIS/EA document. The draft EIS/EA was submitted to the CEA Agency, MOECC, Aboriginal communities and public on February 1, 2016. GGM will be completing consultation events with the regulatory agencies as well as the Aboriginal communities and local community to present the draft EIS/EA and solicit input and comments. Following receipt of all comments on the draft EIS/EA, updates will be completed and the final EIS/EA will be submitted for government review and approval.

After GGM submits the final EIS/EA report, the provincial and federal EA processes will continue in a parallel manner to the extent possible, according to regulatory requirements. GGM will review provincial and federal comments received on the final EIS/EA and will provide a response with regard to how/why comments were incorporated or why it may not be feasible to do so. Construction of the Project cannot proceed until the

EIS/EA has been approved, and the appropriate regulatory approvals (as described below) have been attained. Environmental approvals to initiate construction are dependent on the EIS/EA approval.

20.4.1.2 Consultation

Consultation is a key component of both federal and provincial EA processes as a means to engage interested parties to identify and address concerns with Project planning and implementation. Consultation with government, Aboriginal communities and the public has been ongoing since before the formal start of the EA processes, and has included opportunities to review Project information and provide input at key stages in EA development. GGM's consultation program reflects the requirements of the federal EIS Guidelines and approved provincial ToR. Refer to Subsection 20.5 for further details regarding consultation and engagement activities undertaken in support of the Project.

20.4.1.3 Preliminary Effects Assessment

The methods that are used to conduct the environmental effects assessment have been designed to meet the combined requirements of CEAA 2012 and the EAA. These methods are based on a structured approach that, particularly:

- Considers the federal and provincial regulatory requirements for the assessment of environmental effects as defined by CEAA 2012 and the EAA, with specific consideration of the requirements of the ToR and EIS Guidelines;
- Considers the issues raised by the public, Aboriginal communities and other stakeholders during consultation and engagement activities conducted to date;
- Focuses on issues of greatest concern that arise from the above considerations;
- Considers existing environmental conditions of the area, particularly historical activities and resulting environmental effects that might have affected baseline conditions;
- Integrates engineering design and programs for mitigation and monitoring into a comprehensive environmental planning and management process that will be applied during the design and implementation of the Project;
- Considers the Project in a careful and precautionary manner, to avoid significant adverse environmental effects.

The environmental effects assessment methods address both Project-related and cumulative environmental effects based on the Project description at the completion of the draft EIS/EA. Project-related environmental effects and cumulative environmental effects are assessed using a standardized methodological framework for each VC. Two conditions must be met to initiate an assessment of cumulative effects on a VC: 1) the Project is assessed as having adverse residual environmental effect on a VC; and 2) the adverse residual effects from the Project overlap spatially or temporally with residual effects of other physical activities on a VC. Cumulative environmental effects are assessed to determine whether they could be significant, and to consider the contribution of the Project to them.

The following sections present a summary of the environmental effects assessment, proposed mitigations and determination of significance for each VC from the draft EIS/EA. As the EIS/EA is still being finalized, the following sections are considered preliminary and may be subject to change as assessments and refinements are currently being completed.

20.4.1.3.1 <u>Atmospheric Environment</u>

The potential environmental effects of the Project on the atmospheric environment include change in air quality, climate change and change in lighting. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

For construction and closure, mitigation measures include: using dust suppressants, maintaining vehicles and implementing a ‘no idling’ policy to reduce emissions, applying speed limits to reduce dust from vehicles travelling on gravel roads and minimizing of haul routes to reduce vehicles use.

During operation, mitigation measures will include the use of dust suppressants and other dust controls, reducing diesel fuel consumption where practical through the use of energy efficient equipment, limiting off-site light effects through the use of downlighting and implementing a greenhouse gas (“GHG”) management plan to minimize and track GHG emissions.

With mitigation in place, air quality emissions resulting from construction and closure are expected to be temporary and within applicable regulatory objectives, standards and guidelines. Overall, the Project’s contribution to total Canadian annual GHG emissions would be up to 0.039% (based on 2012 GHG emission levels). Short term GHG emissions from equipment are expected during construction and closure. During operation, the Project is expected to emit no more than 270 kt of CO_2e per year. In regard to lighting, the change in ambient lighting during operation is expected to be comparable in extent to baseline conditions (i.e., similar to other light sources in a rural area characterized by low district brightness).

Residual adverse environmental effects on the atmospheric environment were determined to be not significant.

20.4.1.3.2 Acoustic Environment

The potential environmental effects of the Project on the acoustic environment include change in noise and change in vibration levels. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

During construction and closure, major construction activities will be scheduled during daytime hours (e.g., 07:00 to 19:00), where possible, to avoid sensitive nighttime periods. Other mitigation measures include applying noise mitigation measures (e.g., muffler systems, ‘no idling’ policy) on construction equipment and properly maintaining equipment, and responding to any noise complaints received (also applies to operation).

During the operation phase, mitigation measures include maintaining stationary equipment in good working order, applying appropriate sound transmission standards for buildings that are used to enclose noise generation equipment, keeping doors of buildings with noise generating equipment closed and equipping generator inlets and exhaust stacks in the power house with silencers.

With mitigation measures in place, predicted sound levels are expected to meet regulatory requirements at all Points of Reception. The magnitude of vibration effects from Project-related activities is not predicted to exceed MOECC criteria.

Residual environmental effects on the acoustic environment were determined to be negligible and not significant.

20.4.1.3.3 Groundwater

The potential environmental effects of the Project on the groundwater include change in groundwater quantity and flow, and change in groundwater quality. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures for groundwater quantity and flow include: limiting the construction footprint (i.e., PDA) to the extent possible to reduce the potential for reductions in groundwater recharge; and considering the potential to accelerate open pit filling to re-establish groundwater levels to pre-mining conditions as quickly as possible.

Mitigation measures for groundwater quality include: implementing progressive rehabilitation (placement of soils and vegetation) to reduce infiltration into the WRSAs and TMF; designing the WRSAs to increase the amount of runoff and reduce the amount of infiltration through the WRSAs; removing approximately 25% of the historical MacLeod tailings and 82% of the historical Hard Rock tailings; enhancing the cover over the remaining MacLeod tailings beneath the overburden stockpile; and constructing runoff and seepage collection ditches and ponds around WRSAs and the TMF.

With regard to groundwater quantity and flow, the water table levels will be lowered by one metre in the local area due to the open pit, however, there are no groundwater supply users within the area affected and the lands are owned or under lease by GGM. No effect is predicted to water quantity and flow as changes in groundwater discharge to surface water bodies represent a very small component of overall flow. Groundwater quality is predicted to meet regulatory criteria at the point of discharge. In addition, there will be an overall reduction in loading to surface water features as a result of the removal of a portion of the historical tailings and changes in groundwater flow resulting in a positive change. Arsenic loading from groundwater discharge to surface water bodies is predicted to decrease by 95% during operations and 57% during closure.

Residual adverse environmental effects on groundwater were determined to be not significant.

20.4.1.3.4 Surface Water

The potential environmental effects of the Project on the surface water include change in surface water quantity and change in surface water quality. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures related to surface water quantity and quality include limiting the Project footprint to reduce contact water volumes and management requirements, implementing progressive rehabilitation to reduce infiltration into the WRSAs and TMF, improving water quality within the TMF through cyanide detoxification to reduce cyanide concentrations, designing water management and storage infrastructure to control peak discharges to surface water, re-using water to reduce freshwater intake needs, effluent treatment and discharge requirement and treating effluent prior to discharge.

With regard to water quantity, changes in drainage patterns, including from the realignment of Goldfield Creek, will be contained within the LAA with flow continuing to the Southwest Arm, there will be no changes to flows in Kenogamisis Lake and the flow will be within the range of background variability. With the design and mitigation for the Goldfield Creek diversion, a significant effect on water quantity is not predicted from the Project.

With regard to water quality, mine effluent discharge is predicted to meet baseline levels or PWQO values within a relatively small mixing zone that does not extend beyond the Southwest Arm of Kenogamisis Lake. The Project is predicted to reduce baseline arsenic concentrations in Barton Bay by 50% during operation and by 33% in closure. Arsenic concentrations in the Outlet Basin are predicted to decrease by 11% during operation, and stabilize at a concentration that is under existing baseline conditions during closure. Overall, the Project is anticipated to result in an improvement in water quality within Barton Bay, having a positive effect on arsenic, sulphate and iron concentrations due to the reduction in groundwater discharge associated with the historical MacLeod tailings.

Residual adverse environmental effects on surface water were determined to be not significant.

20.4.1.3.5 Fish and Fish Habitat

The potential environmental effects of the Project on fish and fish habitat include change to fish mortality and change in fish habitat (permanent alteration and loss). As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures for fish and fish habitat include managing construction effects on fish by working “in the dry” by isolating work areas, performing fish salvages to transfer fish from work areas, and complying with in-water timing restrictions, implementing an offsetting plan for impacts to fish that cannot be fully mitigated, developing and implementing effluent discharge criteria and a Spill Response Plan and designing water intake and effluent outfalls to prevent entrainment or impingement of fish.

Fish mortality can be avoided during all phases of the Project such that there is no substantive residual effect on fish mortality. The Project has been designed to reduce the potential for causing fish mortality through avoidance and mitigation measures. Alteration of fish habitat will result from changes to flow and drainage. Project designs have minimized effects on local waterbodies. Effects on sustainability and productivity of fish within the local area are not anticipated. Approximately 9 ha of low value fish habitat, including artificial ponds and drainages, will be lost or permanently altered, but the creation of new fish habitat in conjunction with the diversion of Goldfield Creek will offset these areas such that there is no residual effect.

Residual adverse environmental effects on fish and fish habitat were determined to be not significant.

20.4.1.3.6 Upland Vegetation and Wetlands

The potential environmental effects of the Project on upland vegetation and wetlands include change in abundance of vegetation species of interest, change in abundance or condition of upland vegetation communities and change in wetland function and connectivity. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

The primary mitigation for upland vegetation and wetlands is progressive rehabilitation of the PDA which will commence at the end of construction. In addition, GGM will implement a Vegetation Management Plan designed to mitigate adverse effects on vegetation and wetlands during construction and operation, including timely restoration of affected vegetation communities, control of invasive species (e.g., truck washing stations) and protection of sensitive species. Other mitigation measures include those implemented to reduce effects from dust and sedimentation such as the use of dust suppressants, enclosure of dust sources and implementation of erosion protection measures until vegetative cover is established.

With regard to change in the abundance of vegetation species of interest, clearing of vegetation during construction will result in the removal of plant species. However, plant species in the PDA are common species throughout the region. Effects will be reversible in most areas through revegetation and there are no anticipated effects on a species listed on Schedule 1 of the S*pecies at Risk Act* or listed as threatened or endangered under the *Endangered Species Act.*

With regard to change in abundance or condition of upland vegetation communities, during construction, the removal of approximately 147.7 ha of upland vegetation in the PDA is unlikely to return to a vegetated naturalized habitat, while the removal of approximately 1,081.1 ha of upland vegetation in the PDA is likely to return to a vegetated naturalized habitat. Overall, residual effects resulting from the Project will not result in the loss of long-term viability of a vegetation community type in the RAA.

With regard to change in wetland function and connectivity, during construction, loss of wetland area is anticipated to include the short-term removal of 14.5 ha within the Southwest Arm Tributary and is expected to be reversible, the loss of 54.3 ha associated with permanent linear facilities (Highway 11) or habitat conversion (creek realignment) not likely to return to a naturalized habitat, and the short term loss of 640.0 ha likely to return to a naturalized habitat. Dewatering of the open pit will affect 142.0 ha of wetlands as a result of groundwater drawdown in excess of 0.5 m; however, the overall effects will not result in the loss of the long-term viability of wetland communities given their common occurrence throughout the LAA.

Residual adverse environmental effects on upland vegetation and wetlands were determined to be not significant.

20.4.1.3.7 Wildlife and Wildlife Habitat

The potential environmental effects of the Project on wildlife and wildlife habitat include change in wildlife habitat, change in health and mortality risk and change in movement. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation of potential Project effects on wildlife and wildlife habitat will be accomplished through the implementation of a Wildlife Management Plan and the progressive restoration of vegetation communities and wildlife habitat. Additionally, mitigation measures proposed for other VCs (e.g., upland vegetation and wetlands) or as part of other construction and environmental management plans (e.g., noise abatement) will either directly or indirectly reduce effects on wildlife.

With regard to change in wildlife habitat, effects on SAR and significant wildlife habitat are not predicted to affect the sustainability of wildlife within the region and will be partially reversible following closure. In addition, indirect effects from habitat avoidance due to sensory disturbance will be reversed following the completion of active closure activities. Project effects will not result in the irreversible loss of critical habitat for a species listed on Schedule 1 of the *Species at Risk Act.* While the Project will affect existing wildlife movement in the local area, the effects will be limited spatially and temporally and new wildlife movement patterns are predicted to be established in response to rehabilitation within the PDA.

With regard to change in wildlife health and mortality, no residual adverse effect to wildlife health was predicted by the ecological risk assessment. The effects on mortality risk will be similar to baseline conditions and Project effects will not result in the permanent, irreversible loss of a species listed on Schedule 1 of the *Species at Risk Act* or listed as threatened or endangered under *Endangered Species Act.*

Residual adverse environmental effects on wildlife and wildlife habitat were determined to be not significant.

20.4.1.3.8 Labour and Economy

The potential environmental effects of the Project on labour and economy include change in labour and change in economy. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

In order to mitigate any potential adverse environmental effects, GGM will post job qualifications in advance and identify available training programs and providers so that local and Aboriginal residents can acquire the necessary skills and qualify for potential employment. Project purchasing requirements will also be posted in advance so that local and regional businesses can position themselves to effectively compete to supply goods and services needed for construction and operation. GGM will work with the affected local communities and the municipal government to develop a strategy for addressing economic implications of final mine closure that will provide advance notice of the potential effects of closure and identify actions by which the resulting job losses and impacts on businesses can be reduced. Standard mitigation measures related to the loss of timber by salvaging merchantable timber in accordance with provincial requirements will be implemented. GGM will continue to communicate with MNRF and Enhanced Forest Resource Licence holder regarding its effects on the Kenogami Forestry Management Unit. GGM has consulted with the municipality and developed an agreement to mitigate potential adverse effects on tourism resulting from the removal of existing structures, in particular the Kenogamisis Golf Club, MacLeod-Cockshutt Mining Headframe and the Discover Geraldton Interpretive Centre.

The overall Project effect on labour and economy is positive given the direct, indirect and induced benefits of Project expenditures. The Project will result in increases in the size of the labour force and reductions in the unemployment rate. The Project is also anticipated to result in increases in household incomes, increased opportunities for local and Aboriginal businesses and contributions to municipal taxes.

Residual adverse environmental effects on labour and economy were determined to be not significant.

20.4.1.3.9 Community Services and Infrastructure

The potential environmental effects of the Project on community services and infrastructure include change in capacity of housing and accommodation, change in capacity of health and emergency services and infrastructure, change in capacity of recreation and entertainment facilities and change in capacity of provincial and municipal services and infrastructure. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures for housing and accommodation include a temporary mine camp to accommodate the peak number of construction workers. Mitigation measures for health and emergency services and infrastructure include developing protocols with responsible agencies to deal with worker access to emergency and other medical services. Mine rescue vehicles and trained first responders will be available at the Project site and new employees will be required to take mandatory safety orientations. Employees will be trained in fuel handling, equipment maintenance, fire prevention and response measures. The Project site will be controlled through security measures. Mitigation measures for recreation and entertainment services and infrastructure include providing a temporary camp with dining services and a basic recreational area to accommodate the peak number of construction workers. GGM will maintain the Kenogamisis Golf Club clubhouse and the front nine holes for the longest duration possible and act in accordance with the agreement developed with the municipality regarding future plans for the MacLeod-Cockshutt Mining Headframe, the Discover Geraldton Interpretive Centre and the golf course. Further, mitigation measures for provincial and municipal services and infrastructure include providing notice to the local school board regarding construction and operation scheduling in order for the school board to prepare for the enrollment of additional students. In order to mitigate effects on local infrastructure and utilities, GGM will bus construction workers to and from the temporary camp to limit Project-related traffic, use an on-site natural gas-fueled power plant and electrical/recovered heat distribution system to supply heat and power for Project operation and have Project-dedicated sewage treatment facilities.

With the implementation of mitigation measures residual adverse environmental effects on community services and infrastructure were determined to be not significant.

20.4.1.3.10 Land and Resource Use

The potential environmental effects of the Project on land and resource use include change in recreational land and resource use, change in commercially-based land and resource use and change in navigation. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures include engaging the MNRF, affected tenure holders (trappers, bait harvesters and guide outfitters) and local recreational harvesters to discuss changes to resource use as a result of the Project. GGM will communicate Project activities, locations and timing to stakeholders throughout all phases of the Project. GGM will also continue working with the Municipality of Greenstone on the potential relocation of land and resource use features and alternate route planning for trails. The Project will be designed to avoid obstructions to navigation, and signs will be posted to alert boaters of the treated effluent discharge location.

The area where residual effects will occur has been disturbed by previous mining and forestry activities; however, there will be access restrictions to the PDA. Navigation between Kenogamisis Lake and Goldfield Lake will be maintained and land and resource use is expected to continue at current levels in the regional area where there is an abundance of trails, and wildlife resources for hunting, trapping, fishing, guide outfitting and bait harvesting.

Residual adverse environmental effects on land and resource use were determined to be not significant.

20.4.1.3.11 <u>Heritage Resources</u>

The potential environmental effects of the Project on heritage resources include loss or displacement of archaeological resources determined to have CHVI, and loss, displacement, or disruption of architectural or historical resources determined to have CHVI. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Mitigation measures for archaeological resources include: ceasing construction or operation within a 20 m radius, contacting relevant authorities prior to the implementation of procedures and mitigation if an archaeological resource is discovered. GGM will work collaboratively with Aboriginal communities to develop a protocol for communications should previously undocumented archaeological resources be discovered. Key construction and operations staff will be trained in the recognition of basic archaeological artifacts such as Aboriginal material culture and Euro-Canadian material culture.

The mitigation strategies to be used for architectural/historical resources include: commemoration to create a record of past mining activity and the associated architectural/historical resources; detailed documentation (i.e., creating a public record of the structure or structures, which provides researchers and the general public with a land use history, construction details and photographic record of the resource) and salvage (i.e., recovering architectural or historical resources) where retention or relocation are not feasible; and continuing discussions with the municipality regarding an appropriate approach to commemorate architectural or historical resources.

No archaeological resources have been identified, and therefore no residual effects are anticipated. Further, protocols to protect archaeological resources will be implemented in the event of a chance find during the construction or operation phases.

The remaining cultural heritage resource within the PDA will be isolated from Project activities with the implementation of a 60 m buffer during all Project phases, and is not expected to experience any residual effects. For other affected architectural and historical resources, some of the CHVI of each property will be retained through the selective salvage of identified heritage attributes and other materials, or commemoration.

No residual environmental effects on heritage resources are anticipated.

20.4.1.3.12 Traditional Land and Resource Use

The potential environmental effects of the Project on TLRU include change to distribution of plant species and plant harvesting sites and activities; change to distribution of fish species and fishing areas and activities; change to distribution of hunted and trapped species and hunting and trapping areas and activities; and change in cultural or spiritual practices, sites or areas. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

Potential environmental effects on TLRU were determined based on the Project specific TK studies, Project engagement activities, past project experience and literature review. Other valued component assessments provided additional relevant information regarding effects on resources, and aspects of the biophysical and socio-economic environment that may affect TLRU.

To mitigate potential adverse effects, the mitigation measures identified under wildlife and wildlife habitat, land and resource use, fish and fish habitat, and upland vegetation and wetlands will be applied to avoid or limit effects on components of the environment related to TLRU. GGM will work with community representatives to address potential sites of importance that may be identified through TK sharing.

It is predicted that residual effects on TLRU is limited to reduced access to the PDA for the pursuit of traditional activities. However, with the historical impacts through much of the PDA, the reduced access is not anticipated to be an issue and while access to the PDA will be limited for the lifetime of the Project, TLRU sites and areas within the local assessment area will continue to be accessible.

Based on the findings of the biophysical and socio-economic assessments related to TLRU (i.e., wetlands, fish and fish habitat, wildlife and wildlife habitat, heritage resources, land and resource use, and human and ecological health) and the characterization of effects to known and assumed TLRU sites and areas, it is predicted that the ability of Aboriginal communities to maintain current use of lands and resources for traditional purposes outside of the PDA will be retained.

Residual adverse environmental effects on TLRU are determined to be not significant.

20.4.1.3.13 Human and Ecological Health

The potential environmental effects of the Project on human and ecological health include change in human health and change in ecological health. Project emissions include releases into the terrestrial, aquatic and atmospheric environment. As part of this assessment, mitigation measures were identified that could be applied to the Project to avoid or reduce effects.

A number of mitigation measures have already been incorporated in the Project to eliminate or reduce environmental effects of the Project which also serve to address human and ecological health effects. These mitigation measures include, but are not limited to, the use of dust suppressants, dust collectors and protective covers, a water management plan and progressive rehabilitation that address pathways related to water.

The human health and ecological risk assessments identified negligible risks from exposure (i.e., inhalation and ingestion) of Project-related emissions. With the implementation of the planned mitigation measures for air and surface water, the potential increase in health risk as a result of the Project is negligible. As such, adverse health effects are not expected and, correspondingly, a change to human or ecological health is not expected.

20.4.1.4 Cumulative Effects Assessment

Based on the characterization of the residual cumulative effects (i.e., after mitigation has been applied) of the Project, in combination with the effects associated with other future projects in the regional assessment area, no significant residual adverse cumulative effects are predicted as a result of the Project.

20.4.1.5 Status of the EIS/EA

The preceding subsections presented an overview of the potential effects, mitigation measures and residual effects as described in the draft EIS/EA. As previously noted, comments on the draft and final EIS/EA lead to the refinement of Project components. Discussions with stakeholders are still ongoing in regard to addressing comments on the draft and the extent of changes to the Project cannot fully be captured in this Report.

A list of key comments received on the draft EIS/EA to date, and the anticipated resolution in the final EIS/EA, are provided in Table 20.1.

Table 20.1: Key Comments and Proposed Resolution for the Final EIS/EA

Key Topic	Outstanding Comments/Concerns	Proposed Approach to Resolve/Reduce Outstanding Comments/Concerns
Current Land Users	• Provision of additional Project specific information on current land users.	• To be addressed through ongoing meetings with current land users and provision of additional information where appropriate within the final EIS/EA.
Wetland Evaluation	• Wetland evaluation suggested to determine provincial significance.	• Additional discussions will be undertaken with government agencies. GGM anticipates that the comment can be addressed by adding focussed information on wetland functionality to the final EIS/EA report.
TLRU	• Clarification on the role of TK/TLRU in the EIS/EA and incorporation of community specific assessments.	• Community driven TK/TLRU that were available at the time of the draft EIS/EA have been incorporated unless confidential in nature. GGM will continue to work with Aboriginal communities to provide the opportunity to provide TK/TLRU information that will be incorporated in the final EIS/EA where appropriate. The completion of additional TK studies is anticipated.
Tailing Management Facility	• TMF location, methods and design details presented in draft EIS/EA and permitting requirements under the *Lakes and Rivers Improvement Act.*	• GGM anticipates that concerns can be addressed through meetings with government agencies and provision of additional design and operational clarification in the final EIS/EA report.
Goldfield Creek Realignment	• Specific information on downstream flooding and erosion for the re-aligned channel should be considered in EIS/EA.	• The conceptual offsetting plan, included in the draft EIS/EA will be updated for the final EIS/EA to consider a fluvial geomorphology concept to address downstream flooding and erosion concepts and protection. • Downstream flood lines associated with the channel realignment are being considered in the location of Project infrastructure and some refinements may be carried out for the final EIS/EA.
Setbacks	• Confirmation of Project infrastructure setbacks required from high water mark.	• Water levels within Kenogamisis Lake are controlled in accordance with the Aguasabon River System Water Management Plan. The normal operating water level range for Kenogamisis Lake between 329.32 and 329.70 masl. The upper Cautionary Compliance Zones that is allowed during spring freshet (April 15 and June 30) is 329.85 masl. • The highest water level recorded within the lake was 330 masl. This has been determined as the high water mark and is being confirmed through a legal survey to establish the 120 m reserve of release around Kenogamisis Lake in the area of the lands proceeding from claims to leases. • Project setbacks have been established at 120 m for the claim to lease lands and 30 m for the patent lands.

Key Topic	Outstanding Comments/Concerns	Proposed Approach to Resolve/Reduce Outstanding Comments/Concerns
Effluent Discharge into South West Arm of Kenogamisis Lake	• Discharge location and final effluent requirements based on policy of receiver.	• To be addressed through ongoing meetings with government agencies. The Project has incorporated conservative estimates in predicting water quality effects on the receiving environment. Even with these conservative estimates, the Project still results in an overall improvement for key water quality parameters. Final effluent modelling to confirm discharge criteria and diffuser design will be addressed during the permitting phase of the Project.
Closure Design Requirements	• Space availability between the lake and WRSAs and TMF to monitor groundwater seepage and potentially implement contingency measures if criteria are exceeded.	• The water quality modelling and closure requirements have been completed using conservative assumptions providing confidence that criteria will be met. GGM has increased setbacks in a number of areas in the final mine plan submitted for the feasibility study. • GGM will continue to consider the WRSA design to shed water to reduce infiltration and will include additional information in the closure plan as appropriate.
Seepage Quality	• Seepage quality from historical tailings, the WRSAs, and TMF.	• GGM has incorporated design of seepage and mitigation measures for the historical tailings and included an enhanced cover design over the tailings to reduce infiltration and seepage. • For the WRSAs and TMF, the effects assessment was conservative and did not include seepage collection measures in the assessment of loadings to the natural environment. However, these seepage collection measures have been included in the feasibility design as well as other mitigation measures to reduce infiltration to the WRSAs. The effects of the mitigation measures will be included in the final EIS/EA to address the comments.

Consultation will be ongoing as the EA process proceeds, and following EIS/EA approval, as other permit approvals are pursued. Some issues may require further resolution, or a level of detail not typically available in the EIS/EA stage, and these issues will be tracked and further discussed as Project planning continues into detailed design. Any commitments made in the EIS/EA will be brought forward for resolution at the appropriate stage in Project planning. Resolution of the comments identified in Table 20.1 are anticipated without incurring risk to the Project as it is currently defined; the design of some components may have to be refined, however, modifications are not expected to pose a significant liability to the feasibility of the Project.

20.4.2 Permits or Approvals to Obtain

A range of other permits and approvals may also be required for mining activities and operations through numerous federal, provincial and municipal authorities. The specific requirements for many of these approvals will be confirmed in consultation with regulatory agencies. A preliminary list of federal, provincial and municipal permits, licences and/or authorizations is provided below in Table 20.2.

Table 20.2: Potential Permits / Approvals

Permits / Approvals	Associated Activities
Federal Permits / Approval	
Authorization for Works Affecting Fish Habitat Legislation: *Fisheries Act* Responsible Agency: Department of Fisheries and Oceans ("DFO") (with some provisions administered by Environment and Climate Change Canada)	• Work that may result in serious harm to fish that are part of a commercial, recreational or Aboriginal fishery, or to fish that support such a fishery.
MMER Legislation: *Fisheries Act* Responsible Agency: Environment and Climate Change Canada	• Use of fish bearing waters to deposit mine waste. Environmental effects monitoring program.
Approval of Works in Navigable Waters Legislation: *Navigation Protection Act* Responsible Agency: Transport Canada	• Construction of any works in or over navigable waters. • Deposition of material that is liable to interfere with navigation into a water body where there is not at least approximately 36.6 m of water depth at all times.
License for an Explosives Factory Legislation: *Explosives Act* Responsible Agency: Natural Resources Canada	• Manufacturing, use/storage of blasting explosives.
Transportation of Dangerous Goods Legislation: *Transportation of Dangerous Goods Act* Responsible Agency: Transport Canada	• Transportation of hazardous materials.
Provincial Permits / Approvals	
Mine Closure Plan Legislation: *Mining Act* Responsible Agency: MNDM	• Closure Plan for the Project.
Permit to Take Water Legislation: *Ontario Water Resources Act*, Ontario Regulation 387/04 Responsible Agency: MOECC	• Surface water and groundwater taking and dewatering activities.
Well Drilling Legislation: *Ontario Water Resources Act*, Ontario Regulation 903 Responsible Agency: MOECC	• Monitoring well installation.
Environmental Compliance Approval –Air/Noise Legislation: *Environmental Protection Act*, Ontario Regulation 419/05, and Guideline A-7 Responsible Agency: MOECC	• Air and noise emissions from Project components and activities.

Permits / Approvals	Associated Activities
Environmental Compliance Approval – Industrial Sewage Works Legislation: *Ontario Water Resources Act* Responsible Agency: MOECC	• Mine process water • Sewage treatment plants and discharge
Spill Prevention and Contingency Plans Legislation: *Environmental Protection Act* and Ontario Regulation 224/07 Responsible Agency: MOECC	• Discharge of industrial sewage.
Effluent Monitoring and Effluent Limits Legislation: *Environmental Protection Act*, Ontario Regulation 560/94 Responsible Agency: MOECC	• Metal mining operation discharges.
Environmental Compliance Approval – Waste Disposal Site Legislation: *Environmental Protection Act*, Ontario Regulation 232/98 Responsible Agency: MOECC	• Disposal of construction and/or operation waste materials at an on-site location.
Environmental Compliance Approval – Waste Disposal Site Legislation: *Environmental Protection Act* Responsible Agency: MOECC	• Transportation of waste to a MOECC-approved facility in Ontario or outside of the province.
Record of Site Condition Legislation: *Environmental Protection Act*, Ontario Regulation 153/04 Responsible agency: MOECC	• Remediation of contaminated land (i.e., gas station, historical tailings).
Waste Audit and Waste Reduction Plan Legislation: *Environmental Protection Act*, Ontario Regulation 102/95 Responsible Agency: MOECC	• Construction of a large project.
Waste Generator Registration Legislation: *Environmental Protection Act*, Ontario Regulation 347 Responsible Agency: MOECC	• On-site storage of materials such as oils, greases (or any other types of waste defined as hazardous or liquid industrial under Ontario Regulation 347).
Work Permit Legislation: *Public Lands Act* Responsible Agency: MNRF	• Water crossings and road construction/upgrading on Crown land. • Permits for any additional activities or tenure on Crown land, if required.
Aggregate Pit License/Permit Legislation: *Aggregate Resources Act* Responsible Agency: MNRF	• Extraction of aggregate for construction activities.
Permits and Licences (various) Legislation: *Fish and Wildlife Conservation Act* Responsible Agency: MNRF	• Pre-development fish/wildlife studies. • Initial fish and wildlife relocation. • Destruction of beaver dams, furbearer/bear dens, and nests/eggs of birds wild by nature.

Permits / Approvals	Associated Activities
Various Approvals Legislation: *Lakes and Rivers Improvement Act* Responsible Agency: MNRF	• Location Approval, and Plans and Specifications Approval for the polishing pond dam and tailings dams. • Approvals for diversions and channelizations.
Overall Benefit Permit Legislation: *Endangered Species Act* Responsible Agency: MNRF	• Activities with potential to contravene Sections 9 (Species Protection) or 10 (Habitat Protection) of the *ESA*.
License to Harvest Forest Resources and/or Release of Reservation Legislation: *Crown Forest Sustainability Act* Responsible Agency: MNRF	• Release of Reservation required for Crown timber on private or patented land. • Forestry Resource Licence for Crown timber on Crown land.
Encroachment Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	• Any work upon, over or under provincial highway right-of-way (except entrances).
Entrance Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	• Change in use of an existing entrance, construction of a new entrance or temporary entrance (for construction).
Sign Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	• New signs for highway right-of-way.
Building and Land Use Permits Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	• Construction of buildings or facilities close to or adjacent to a provincial highway,
Order-in-Council - Legal Highway Transfer Process Legislation: *Public Transportation and Highway Improvement Act* Responsible Agency: Ministry of Transportation	• Transfer of ownership of new highway by-pass to the province and transfer of the existing section to private from province.
Letter of Compliance for Archaeology Legislation*: Ontario Heritage Act* Responsible Agency: MTCS	• Disturbance of any potential archaeological sites.
Official Plan Amendment Legislation: *Planning Act* Responsible Agency: MMAH	• Change to existing land use designation(s) in the Municipality of Greenstone and within the Thunder Bay North District Unorganized Territory.

Permits / Approvals	Associated Activities
Municipal Permits / Approvals	
Official Plan and Zoning By-Law Amendment Legislation: *Planning Act* Responsible Agency: Municipality of Greenstone	• Change to existing zoning provision(s).
Building Permit Legislation: *Building Code Act* and Building By-law 01-58 Responsible Agency: Municipality of Greenstone	• Construction of buildings.
Demolition Permit Legislation: *Building Code Act* and Building By-law 01-58 Responsible Agency: Municipality of Greenstone	• Demolition of buildings.

20.5 Consultation Activities

GGM has undertaken active participation through consultation during the planning and preparation of the draft EIS/EA. Consultation will be ongoing as the Project progresses through preparation of the final EIS/EA and permitting phases. GGM's consultation program reflects the requirements of the consultation guidelines set out in the Code of Practice for Consultation in Ontario's Environmental Assessment Process (MOECC, 2014). In addition, the consultation program was designed to follow the federal EIS Guidelines and approved provincial ToR for the Project.

During the preparation of the draft EIS/EA, GGM has consulted with a wide range of stakeholders, Aboriginal communities and agency reviewers through various stage of the Project approval process and is currently completing final consultation on the draft EIS/EA to support completion and submission of the final EIS/EA.

20.5.1 Aboriginal Engagement

Through the federal EIS Guidelines and subsequent correspondence with the CEA Agency, GGM was provided direction to consult and engage with: Aroland First Nation, Ginoogaming First Nation, LL #58 First Nation, the Métis Nation of Ontario (MNO) and Animbiigoo Zaagi'igan Anishinaabek (AZA First Nation) as part of the EA.

Provincially, the MOECC identified that three communities hold or claim Aboriginal or treaty rights that may be adversely impacted by the Project (Aroland First Nation, Ginoogaming First Nation and LL #58 First Nation), and that it was delegating aspects of consultation to GGM. MOECC also indicated that in addition to GGM's consultation obligations and delegation of procedural aspects with the Aboriginal communities identified above, MOECC also requires engagement with people or groups who may have an interest in the Project. These communities included the:

- AZA First Nation;
- Biigtigong Nishnaabeg (formerly known as Ojibways of the Pic River First Nation);
- Biinjitiwaabik Zaaging Anishinaabek (BZA First Nation) (formerly known as Rocky Bay First Nation);
- Bingwi Neyaashi Anishinaabek (BNA First Nation) (formerly known as Sand Point First Nation);
- Constance Lake First Nation;
- Eabametoong First Nation;
- Greenstone Métis Council;
- Marten Falls First Nation;
- Pays Plat First Nation;
- RSMIN.

Aboriginal Environmental Review Teams were formed during the EA process; numerous meetings have taken place with review teams as well as individual community meetings. Comments from communities have been, and will continue to be, received during the EA process on environmental baseline, alternative methods, comparative analysis results and effects/mitigation. Concerns/issues identified by each community will continue to be discussed and addressed as the Project progresses. GGM is working with Aboriginal communities, and their technical review consultants, to address comments on the draft EIS/EA in preparation for the completion of the final EIS/EA.

20.5.2 Summary of Influence of Consultation and Engagement on the Project

Since the initiation of the EA process, consultation has been carried out related to key aspects of the Project, including baseline studies, the identification and evaluation of alternatives, assessment of environmental effects and the overall design of the Project. A summary of the key changes/refinements made to the draft EIS/EA based on the results of consultation are provided in Table 20.3. GGM will continue its ongoing engagement with interested parties throughout the EA and permitting process and into construction, operation and closure of the Project.

Table 20.3: Influence of Consultation of the Draft EIS/EA

Draft EIS/EA Item	Influence of Consultation
Baseline Studies	• The scope of baseline studies included key data sources and comments identified for consideration in the draft EIS/EA. This information was used throughout the EA process to inform the evaluation of alternatives and the assessment of Project effects. • 2015 baseline results were incorporated into the draft EIS/EA. Fieldwork will continue to be completed as the Project progresses and will be used to inform permitting/long-term monitoring phases. • Baseline studies and the analysis in the draft EIS/EA were refined to include additional information or clarification to respond to questions or concerns.
Identification of Alternatives	• The "long list" of alternatives for the initial screening was expanded to account for additional alternatives identified for consideration. • The results of the initial screening were refined to include consideration of new alternatives in the comparative analysis. • The rationale for the screening of alternatives was refined to include additional environmental rationale.
Alternatives Assessment Methodology and Results	• The list of criteria and indicators was revised to include additional consideration of key aspects of the environment. • The location of some Project components, such as the TMF location, Goldfield Creek realignment, Highway 11 realignment and WRSA "A", were modified based on consultation input. • Climate change and source protection planning were specifically considered as part of the alternatives assessment in the draft EIS/EA. • The results of the comparative analysis were updated to consider the range of environmental effects identified as key areas of concern. • Further detail was included in the description of the comparative analysis results to address environmental concerns in the decision-making process, and provide additional rationale for the selection of preferred alternatives.
Environmental Effects Assessment	• Key areas of interest were identified and considered in the detailed assessment of effects for each VC. Further information provided through the completion of supplemental baseline reports has been incorporated into the assessment for each VC. • The effects of historical mining activities were characterized through baseline studies and considered as part of the environmental effects assessment and cumulative effects assessment to clearly delineate between existing contributions to water quality and Project effects. • New data sources were identified and considered through the effects assessment process for VCs. • The rationale for the selection of measurable parameters used in the effects assessment was updated for VCs.
Project Design	• The locations of provincial facilities were refined in ongoing consultation with key agencies to meet operational needs. • WRSA layout was adjusted to attempt to avoid or delay effects on the Kenogamisis Golf Club to the extent possible and effects to the Southwest Arm Tributary. • The size and configuration of the TMF were optimized to avoid unnecessary disturbance of watercourses and fish habitat, including avoiding the infilling of Lake A-322. The TMF construction and operation sequence was also refined to facilitate progressive rehabilitation. • The need for a temporary mine camp was confirmed. Sewage disposal options have been confirmed and will be directed to the municipal sewage treatment plant. • Collection ponds were sited to collect discharge and runoff from Project components. • Refinements were made to watercourse diversions to reduce the overall environmental effects to flow regimes, water transfer between sub-watersheds, fish and fish habitat, and to enhance Project efficiencies. • The route of Highway 11 was refined to optimize highway geometry and improve safety.

20.6 Follow-up Monitoring and Environmental Management Plans

As part of the EA process, a monitoring framework will be advanced for all subsequent phases of the Project. The framework in the draft EIS/EA includes monitoring related to both compliance monitoring and effects monitoring during construction, operation, and closure and to fulfill anticipated compliance monitoring requirements. Environmental management plans ("EMP") will outline the proposed environmental protection measures and commitments to be carried out by GGM and their contractor and subcontractors, during construction and operation, respectively to avoid or reduce potential effects. These EMPs will be tied to the follow-up and monitoring plans, and will outline contingency measures to respond to any exceedances of regulatory standards related to environmental discharges or other adverse effects. Contingency measures specific to each EMP will be implemented in the event that regular environmental and compliance monitoring programs detect deviations from standard operating conditions that result in, or may lead to, adverse effects on worker safety or the environment.

Upon approval of the final EIS/EA, and completion of permitting, refinements to the follow-up and monitoring programs will incorporate outcomes of the approval processes, and refinements will be considered throughout the Project. Program plans are iterative by nature and the monitoring activities associated with the Project will be used to inform adaptive management, which is a process identified in the draft EIS/EA for continuously improving environmental management practices.

20.7 Closure, Decommissioning and Reclamation

Before mining operations can begin, MNDM requires that a Closure Plan with Financial Assurance be submitted and approved under the *Mining Act R.S.O. 1990*, Chapter M.14 (amended by S.O. 2010, 18. 23); Part VII under the Act, O. Reg. 240/00 as amended, and Schedule 1 and 2, Mine Rehabilitation Code of Ontario.

A Conceptual Closure Plan has been developed as part of the draft EIS/EA to provide an early opportunity to discuss the closure approach and inform initial costing. The Conceptual Closure Plan includes preliminary details on closure that may be refined following EIS/EA approval through further discussion with regulatory agencies, including the MNDM, MNRF and MOECC. At the end of mining operations, the main features requiring closure will include the main open pit, water management and drainage systems, WRSAs, TMF, site access roads and buildings and associated infrastructure. After the closure activities have been carried out, a post-closure monitoring program will be carried out to verify that the closure objectives and criteria have been met and confirm that the Project can proceed to final close out under the *Mining Act*.

The main elements of the Conceptual Closure Plan include progressive rehabilitation during Project operation for certain components, and final closure measures following the end of mine operations. When practical, areas that are no longer required may be rehabilitated during mining operations. These activities, known as progressive rehabilitation, contribute to the overall rehabilitation efforts that would otherwise be carried out at closure, or efforts carried out in support of the closure activities (e.g., field trials). Once the mine advances from the development stage to the operational stage, progressive rehabilitation activities can commence, as applicable. Progressive rehabilitation opportunities may include:

- Removal of construction-related buildings, laydown areas, and access roads;
- Stabilization and re-vegetation of WRSAs, where practical;
- Rehabilitation of the north cell of the TMF, upon completion of deposition anticipated after Year 7, consisting of a vegetated store and release cover with runoff from the cell directed to the south cell, or to the environment once water quality meets acceptable regulatory requirements;
- Backfilling of the eastern portion of the main open pit;
- Removal of hazardous and non-hazardous waste materials from the site on a regular basis, where possible.

While progressive rehabilitation activities will be carried out throughout the mine life, the majority of rehabilitation work will take place once mining has been completed. The following list summarizes the main activities associated with closure:

- All infrastructure, equipment and mining materials (including buildings, pipelines, site lighting and security, service water supply, water management facilities and petroleum products) will be removed.
- Some facilities (e.g., access roads and the effluent treatment plant) may be required for the proper care and maintenance of the site during closure and will be removed/rehabilitated once they are no longer required during closure.
- The pit will be partially backfilled with waste rock during operations and the remainder filled with water, creating a pit lake.
- The open pit, WRSAs and TMF will be stabilized (chemically and physically) and the top surfaces and benches covered with a store and release cover to facilitate vegetation growth. The TMF will be fully revegetated.

- In preparation for revegetation efforts, the ground surface will be prepared by scarification or ripping of compact surfaces, amending soil to support vegetative growth, and implementing erosion protection measures to protect the soil cover until vegetation is established.
- The majority of the closure measures will be implemented over a five year period after the cessation of mining and ore processing activities; however, rehabilitation of the main open pit will take significantly longer due to the time required to fill the open pit with water. To reduce the filling time, water will be pumped into the open pit from the TMF, mine contact water management ponds and from the Southwest Arm of Kenogamisis Lake. Pond water may also be put through passive wetland treatment or released directly to the environment, as appropriate. The site can be considered to be in a state of post-closure when the site has been shown to be stable and able to meet the closure criteria.

The overall objective of closure is to return the site to a chemically and physically stable state which is self-sustaining and supports the desired future land uses. The landscape will be revegetated using locally available, non-invasive plant species to encourage the return of wildlife and fish species to the area. Most access restrictions will be lifted after closure; however, a boulder fence will be erected around the open pit to restrict vehicular access for safety purposes. It is anticipated that recreational activities such as hunting, hiking, snowmobiling and other passive activities, as well as economic uses such as forestry, would be permitted.

Monitoring will be completed to assess the physical, chemical and biological stability of the Project and confirm that closure objectives have been met and when the Project has reached a condition suitable for moving to closed out status as defined under the *Mining Act*. The site will be monitored by GGM according to a set schedule to verify the site is performing as expected and that effluent criteria are being met. This includes the monitoring of effluent water quality (surface and groundwater) and physical stability (embankments and the open pit slopes).

The final Closure Plan and all technical details will be confirmed with regulatory authorities as the EIS/EA and related technical and engineering studies for the Project progress. The final Closure Plan will be completed and submitted to the MNDM upon EIS/EA approval and the EIS/EA conceptual plan facilitates this process. The Conceptual Closure Plan contains sufficient detail on the closure approach to allow for the development of a cost estimate for the FS and this Report. Following standard practice, a contingency amount has been carried forward with the preliminary cost estimate to account for changes from the Conceptual Closure Plan to the Closure Plan.

21. CAPITAL AND OPERATING COSTS

21.1 Capital Expenditures

21.1.1 Responsibility Matrix

Responsibility for the CAPEX estimate has been divided among the FS contributors as follows:

- GMS – mine fleet, mining infrastructure, and general infrastructure including power. GMS provided a check estimate for the relocation of the Hydro One Geraldton TS and 115 kV power lines. GMS also maintained the overall CAPEX database and libraries;
- WSP and Soutex – Work breakdown structure ("WBS") Area 300 for surface water management infrastructure and WBS Area 600 for process plant. WSP also developed the unit labour and material costs to be applied to all industrial activities throughout the estimate;
- Amec – WBS Area 300 related to the TMF material takeoffs (quantities), including fish habitat compensation, and the Goldfields Creek realignment/Southwest Arm extension;
- TBT – re-alignment of Trans-Canada Highway 11 through the Project area of influence, and the relocation of the MTO patrol station;
- Union Gas – natural gas pipeline from the TCPL Canadian Mainline Pipeline to the Project natural gas distribution header and power plant;
- GGM – infrastructure relocation budgets other than the Hydro One infrastructure, MTO Patrol Station, and Trans-Canada Highway 11 relocations. GGM was also responsible for the construction indirect costs which were reviewed by GMS, and for the Owner's cost estimates.

21.1.2 Basis of Estimate

The base date of the CAPEX estimate is Q2 2016.

The pre-production CAPEX period is planned for 42 months with a 23 month construction period.

The accuracy of the estimate is ±15%, based on a global engineering completion of greater than 25% (Class 3 according to AACE International Recommended Practice No. 47R-11).

The CAPEX estimate is aligned with an Owner-managed project delivery model.

21.1.3 CAPEX Summary

The capital cost estimate is summarized in Table 21.1.

Table 21.1: Capital Expenditures Summary

Work Breakdown Structure	Total (M CAD)
100 - Infrastructure	62.6
200 - Power & Electrical	72.4
300 - Water & Tailings Management	79.9
400 - Mobile Equipment	178.1
500 - Infrastructure Repositioning	45.6
600 - Process Plant General	343.1
700 - Construction Indirect Cost	175.4
800 - General Services - Owner's Cost	59.8
900 - 980 - Preproduction, Startup, Commissioning	94.1
990 - Contingency	131.3
Grand Total	**1,242.4**

21.1.3.1 Direct Costs

Estimating responsibilities were assigned at the WBS level to the FS contributors, who were responsible for the budgetary tenders, and calculating the material take off estimates for their respective areas. WSP was responsible for developing the unit costs for labour and materials for all industrial disciplines, while GMS developed the unit costs for labour and materials for mine operations and general services disciplines. These unit costs were normalized and applied to the CAPEX database consistently across the material take off estimates of all contributors. WBS Areas 100 to 600 are direct costs, and WBS Areas 700 to 900 are indirect costs.

The CAPEX estimate for infrastructure is summarized in Table 21.2. The CAPEX for the temporary camp WBS Area 130 represents the cost of site preparation only, as the rental for the camp buildings is captured in WBS Area 740. Support facilities include the site administration building, the explosives reagents storage, and the temporary explosives storage. The predominant cost in WBS Area 170 is the natural gas tap and pipeline to the power plant, and also includes the diesel fuel storage and distribution systems for the mine fleet refueling. WBS Area 180 includes the recycling and waste management facilities.

Table 21.2: Infrastructure Capital Expenditures

Work Breakdown Structure	Total (M CAD)
100 - Infrastructure	
110 - General Site Preparation	14.2
120 - Workshops/Storage	22.1
130 - Support Facilities	11.5
140 – Camp	0.8
170 - Fuel Systems	13.7
180 - Other Facilities	0.3
Grand Total	**62.6**

The CAPEX estimate for power supply and electrical is summarized in Table 21.3. The power plant WBS Area 210 includes material, equipment and construction costs. The detailed engineering and commissioning costs for the power plant are in WBS Areas 710 and 950 respectively. The IT estimate reflects the systems required for G&A (finance, accounting, purchasing, and inventory), mining operations (slope stability monitoring fleet and maintenance management) and the geology and mining engineering departments.

Table 21.3: Power Supply and Communications Capital Expenditures

Work Breakdown Structure	Total (M CAD)
200 - Power & Electrical	
210 - High Voltage	59.5
240 - Site Power Distribution	4.8
260 - IT and Site Communications	8.1
Grand Total	**72.4**

The CAPEX estimate for water and tailings management is presented in Table 21.4. The TMF estimate includes the scope that will be completed during the pre-production phase. The topography derived from the 2014 Lidar survey map was used for developing the TMF dam drawings and quantities. All foundation preparation works are based on findings of the feasibility geotechnical investigations. The initial phases of the South Cell and North Cell dams are planned to be constructed by the earthworks contractors, while further dam raises are by the Owner’s mining fleet.

Effluent and surface water management consists primarily of the mine water effluent treatment plant, collection ditches and ponds. Potable water and domestic sewage systems consist primarily of connections to municipal systems from the Project site. Reclaim water consists primarily of the pump station at the TMF and reclaim return water pipeline to the process plant.

Table 21.4: Water and Tailings Management Capital Expenditures

Work Breakdown Structure	Total (M CAD)
300 - Water & Tailings Management	
310 - Potable Water	2.2
320 - Reclaim Water	3.8
340 - Tailings Management Facility (TMF)	58.9
350 - Surface Water Management	7.2
360 - Effluent Water Management	6.1
370 - Fire water	0.7
380 - Domestic Sewage	1.0
Grand Total	**79.9**

The CAPEX estimate for mobile equipment is summarized in Table 21.5. The mine equipment fleet requirements are outlined in Subsection 16.7. A tender process was undertaken for the mine equipment fleet. The equipment pricing includes tires, fire suppression, transport to the Project site, assembly and commissioning.

Table 21.5: Mobile Equipment Capital Expenditures

Work Breakdown Structure	Total (M CAD)
400 - Mobile Equipment	
410 - Mine Equipment	176.7
430 - Plant and Surface Mobile Equipment	1.3
Grand Total	**178.1**

The CAPEX estimate for WBS Area 500 infrastructure repositioning totals CAD 45.6M. The capital cost estimate for the Trans-Canada Highway 11 deviation was developed by TBT. The preliminary design report includes all foundation requirements, based on detailed geotechnical evaluations, as well as typical unit costs from northern Ontario by experience. The OPP station and MTO patrol station relocation costs were also developed by TBT. The CAPEX estimate for the relocation of the existing substation and power lines (44 kV) were developed in collaboration with GMS based on available information from the existing substation, drawings, and the results of a limited geotechnical study. GGM negotiated with Hydro One for GGM to rebuild and reposition power lines, as specified by Hydro One.

The CAPEX estimate for the process plant is summarized in Table 21.6. The estimate includes all direct costs for the processing facilities. The processing facilities are further described in Section 17.

Table 21.6: Process Plant Capital Costs

Work Breakdown Structure	Total (M CAD)
600 - Process Plant General	
600 - Process Plant	62.5
610 - Crushing and Ore Handling	78.0
620 - Grinding & Gravity	79.0
630 - Pre-Leach / Leach / CIP	61.2
640 - CN Detox & Final Tails	15.3
650 - Acid Wash, Elution, Carbon Regeneration	7.6
660 – Refinery	1.8
670 - Electrical Process Plant	20.7
680 - Plant Reagent & Services	16.3
690 - Plant Supply	0.6
Grand Total	**343.1**

21.1.4 Indirect Costs

Indirect costs have been developed primarily from detailed estimates, including some internal estimates by GGM, where noted.

- Construction indirect costs were developed primarily by GGM based on the execution strategy, and include infrastructure such as temporary site facilities;
- Owner's costs were developed by GGM;
- Pre-production mining costs were developed by GMS and are consistent with the basis of the OPEX costs;
- Commissioning costs were based on a ramp-up schedule and are consistent with the basis of the OPEX costs;
- Freight and duty were estimated based on quantities and previous project experience.

The CAPEX estimate for indirect costs is summarized in Table 21.7.

Table 21.7: Indirect Costs

Work Breakdown Structure	Total (M CAD)
700 - Construction Indirect Costs	175.4
710 - Engineering, CM, PM	83.7
720 - Construction Facilities & Services	29.2
730 - Contractor Mobilization/Demobilization and Indirects	35.9
740 - Construction Camp Facilities & Operation	26.7
800 - General Services - Owner's Cost	59.8
810 - Departments	27.8
820 - Logistics / Taxes / Insurance	30.8
830 - Operations Accommodations	1.2
900 - Preproduction, Startup, Commissioning	225.4
910 - Mining Preproduction / Commissioning	84.3
920 - Mining Haul Roads	1.6
940 - Spares & First Fills	11.1
950 - Process Plant Preproduction / Commissioning	13.4
960 - Operational Readiness Support	1.1
970 - Pre-production Revenue	17.5
990 - Contingency	131.3
Grand Total	**460.6**

21.1.4.1 Allowances, Contingency, and Escalation

The total contingency provision is CAD 131M, which represents 11.8% of the total CAPEX. The contingency amount was established through a facilitated quantitative risk assessment ("QRA") process. The QRA process considered the underlying level of scope definition, estimate inputs and assumptions, with the objective of providing for an appropriate contingency provision on direct and indirect costs including design growth, mining preproduction, mining equipment and schedule delays.

21.1.5 Sustaining Capital

Sustaining capital is presented in Table 21.8

Sustaining capital for the mine includes additional equipment purchases for a total of CAD 49M. Major equipment repairs are capitalized which represents CAD 108M over the LOM. The sustaining capital estimate also includes the remaining mining construction civil works for ditches, ponds and dump shear keys totalling CAD 4.3M.

Major dam raises for the TMF are accounted for in sustaining capital, a total of CAD 86.4M over two major work periods: Year 2-3 and Year 7-9 of mine operations. This work would be done by the mine operations team.

Table 21.8: Sustaining Capital Costs

Sustaining Capital (Year)	Mine Equipment Capital Repairs (M CAD)	Mine Equipment Purchases (M CAD)	TMF Dam Construction (M CAD)	Other (M CAD)	Total (M CAD)
1	0.5	-	-	0.4	0.8
2	3.4	26.7	17.9	2.1	50.0
3	11.6	4.1	16.7	-	32.4
4	16.9	0.4	0.9	-	18.2
5	13.0	2.1	0.9	10.1	26.2
6	10.1	5.3	-	0.6	16.0
7	13.8	2.1	17.3	0.1	33.3
8	10.3	3.8	16.4	-	30.5
9	10.4	2.8	16.4	-	29.6
10	14.2	0.4	-	-	14.6
11	1.8	1.1	-	-	3.0
12	0.1	0.2	-	-	0.3
13	1.8	-	-	-	1.8
14	-	-	-	-	-
Total	**107.9**	**49.0**	**86.4**	**13.3**	**256.6**

21.2 Operating Costs

21.2.1 Operating Costs Summary

Operating costs are summarized in Table 21.9. The operating costs include mining, processing, G&A, transportation and refining, royalties and other costs. The average operating cost is CAD 705/oz of gold or CAD 20.95/t milled over the LOM.

Table 21.9: Operating Costs Summary

Category	Total Costs (M CAD)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz Au)
Mining	1,412	10.03	338
Processing	1,061	7.54	254
G&A	205	1.45	49
Transportation & Refining	13	0.09	3
Other Costs	56	0.40	13
Royalties	203	1.45	49
Total Operating Cost	**2,950**	**20.95**	**705**

The operating organization consist of three departments: mine, including mine operations, geology, engineering and maintenance; process and power plant; and G&A including human resources, environment, health and safety, site services and accounting. The peak total operating workforce is 544 employees (reached in Year 4).

Table 21.10: Peak Operations Workforce

Operations Department	Peak Workforce
Mine	392
Process Plant	100
G&A	52
Number of Employees	**544**

A summary of the total operating costs, by year, is presented in Table 21.11 and Figure 21.1.

Figure 21.1: Operating Cost by Year

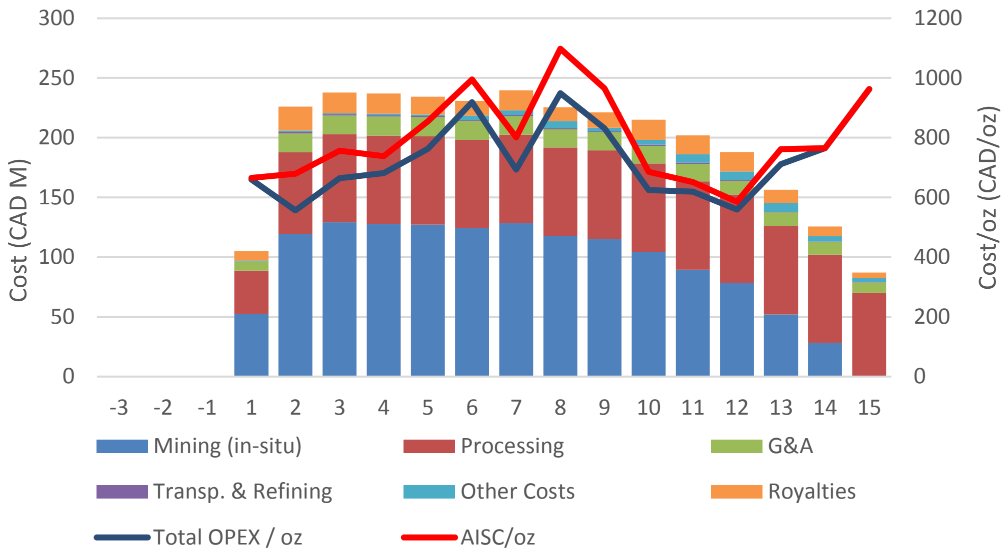


Table 21.11: Total Operating Costs Summary

Commercial Production	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Tonnage Milled (Mt)	140.81	4.38	8.76	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.33	
Tonnage Mined (Mt)	648.47	31.90	68.48	68.16	67.47	67.93	64.45	62.39	52.77	47.86	39.80	33.76	24.35	13.83	5.32	0.00	
Gold Sales Ops (koz)	4,181	161	409	358	349	308	252	346	238	266	344	326	336	223	169	96	
Operating Costs (M CAD)																	
Mining (in-situ)	1,395	53	119	129	128	127	124	128	118	115	105	90	79	52	28	0	
Mining (rehandling)	17	1	2	0	1	1	1	0	1	0	0	0	0	2	3	5	
Processing	1,061	36	68	74	74	74	74	74	74	74	74	74	74	74	74	70	
G&A	205	8	16	16	16	16	16	16	15	15	15	15	11	11	11	8	
Transportation & Refining	13	0	1	1	1	1	1	1	1	1	1	1	1	1	1	0	
Other Costs	56	0	1	1	1	1	4	4	6	3	4	7	7	7	4	3	1
Royalties	203	8	20	17	17	15	12	17	12	13	17	16	16	11	8	5	0
Total Operating Cost	**2,950**	**106**	**228**	**238**	**238**	**235**	**232**	**240**	**226**	**221**	**215**	**202**	**188**	**158**	**129**	**92**	**1**
Total OPEX CAD / oz	705	660	557	664	682	763	919	693	949	832	624	619	559	712	765	963	
Total OPEX CAD / t milled	20.95	24.30	26.01	24.13	24.12	23.79	23.49	24.32	22.95	22.40	21.82	20.49	19.09	16.03	13.11	9.90	
Mining Cost CAD / t mined	2.18	1.69	1.77	1.89	1.90	1.88	1.94	2.06	2.24	2.41	2.63	2.65	3.23	3.92	5.94		

21.2.2 Mining Costs

Table 21.12 presents the breakdown of mining costs, by department, while Table 21.13 presents the major cost drivers for the mine department.

The mine operating costs are estimated from first principles for all mine activities. Equipment hours required to meet the production requirements of the LOM plan are based on productivity factors or equipment simulations. Each piece of equipment has an hourly operating cost which includes operating and maintenance labour, fuel and lube, maintenance parts, tires (if required) and ground engaging tools (if required). Quotations have been received for various consumables such as tires, drilling tools, explosives and accessories.

The average mining cost during operations is estimated at CAD 2.18/t mined including re-handling costs. The mining costs are lower than average during the early years and increase with increased haulage distances and pit deepening, in the later years. This operating cost estimate excludes capital repairs which treated as sustaining capital.

Haulage is the major mining cost activity representing 36% of total costs followed by blasting (13%), loading (9%) and drilling (9%). Some haulage costs have been back-charged to the TMF dam construction as this represents incremental haulage. Loading and haulage for stockpile re-handling is also captured as a separate activity cost.

Fuel is the dominant cost, by element, representing 28% of total costs, followed by salaries (27%), maintenance parts (15%) and bulk explosives (11%).

Table 21.12: Mining Cost Summary Total

Mining Costs (M CAD)	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mine Operations	38.6	1.6	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	3.1	2.9	2.6	1.8	1.5	-
Mine Maintenance Admin.	65.3	3.1	6.2	6.3	5.1	5.1	5.1	5.1	5.0	5.0	4.9	4.0	3.8	3.4	2.9	0.4
Mine Geology	12.8	0.5	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.9	0.8	0.6	0.6	-
Mine Engineering	31.3	1.3	2.5	2.5	2.6	2.6	2.6	2.6	2.6	2.6	2.6	2.0	1.8	1.4	1.4	-
Grade Control	52.3	2.1	4.6	4.9	4.5	4.3	4.4	4.7	4.2	4.2	4.2	3.6	3.1	2.0	1.5	-
Voids Management	23.8	0.7	1.4	1.4	1.4	1.4	1.4	1.4	4.2	5.1	2.2	1.2	1.2	0.7	0.3	-
Topo Drilling Contract	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Drilling	122.4	4.8	12.9	12.8	12.0	12.3	12.3	11.9	10.5	9.7	7.5	6.6	4.7	3.1	1.3	-
Blasting	184.6	7.3	19.6	19.5	18.3	18.7	17.9	17.4	15.1	13.9	11.9	10.5	7.5	4.6	2.5	-
Pre-Split D&B	41.9	1.6	2.7	3.3	3.5	5.3	4.3	4.9	3.6	3.9	2.6	2.0	1.9	1.4	1.0	-
Loading	128.2	6.4	13.6	13.6	13.5	13.5	13.0	12.4	9.4	9.0	7.8	7.0	5.1	2.7	1.3	(0.0)
Hauling	520.8	16.0	37.4	46.1	48.1	45.2	44.5	49.1	45.1	43.5	44.6	37.7	34.8	21.9	6.9	-
Dump Maintenance	60.9	2.9	5.8	5.8	5.8	5.8	5.8	5.8	5.1	5.1	3.1	3.1	3.1	2.0	2.0	-
Road Maintenance	56.5	2.3	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	4.6	3.7	3.7	2.9	2.9	-
Dewatering	4.5	0.1	0.1	0.1	0.2	0.2	0.3	0.3	0.3	0.3	0.3	0.3	0.5	0.5	0.7	-
Overburden Mining Contract	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Support Equipment	51.4	2.0	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	4.1	3.2	1.4	-
Sub-Total In-Situ Mining	1,395.2	52.6	119.5	129.1	127.8	127.2	124.2	128.4	117.8	115.2	104.5	89.6	78.6	52.0	28.1	0.4
Rehandling	16.7	1.3	1.9	0.0	0.7	0.6	0.7	-	0.6	-	-	-	-	2.1	3.5	5.2
Total Mining	**1,411.9**	**53.9**	**121.4**	**129.1**	**128.5**	**127.8**	**125.0**	**128.4**	**118.4**	**115.2**	**104.5**	**89.6**	**78.6**	**54.2**	**31.6**	**5.6**
Total Mining / t mined	**2.18**	**1.69**	**1.77**	**1.89**	**1.90**	**1.88**	**1.94**	**2.06**	**2.24**	**2.41**	**2.63**	**2.65**	**3.23**	**3.92**	**5.94**	**-**

Table 21.13: Top Three Mining Costs by Cost Type

Top Mining Costs (M CAD)	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Diesel/Fuel	398.3	13.4	30.8	36.3	37.7	34.8	33.0	37.0	34.5	33.4	32.1	26.8	24.6	16.2	6.1	1.6
Salaries	373.1	14.8	30.4	32.0	32.4	32.4	32.4	32.4	31.1	30.8	28.6	24.2	21.8	16.1	12.1	1.6
Maintenance Supply	214.6	8.6	19.7	20.5	20.6	20.9	20.3	20.3	17.5	16.6	14.5	12.2	10.5	7.1	4.2	1.1
Sub-Total Top Three	**986.0**	**36.8**	**81.0**	**88.8**	**90.8**	**88.1**	**85.7**	**89.8**	**83.1**	**80.7**	**75.1**	**63.3**	**56.9**	**39.3**	**22.4**	**4.3**
% of Total	**70%**	**68%**	**67%**	**69%**	**71%**	**69%**	**69%**	**70%**	**70%**	**70%**	**72%**	**71%**	**72%**	**73%**	**71%**	**77%**

21.2.3 Processing Costs

The process plant operating costs were evaluated based on metallurgical testwork, recent supplier quotations, a recent salary survey and standard industry practice. The process costs are divided into eight categories: workforce, electrical power, wear parts, maintenance parts, grinding media, reagents, material handling, research and development ("R&D") and laboratory. Surface water pumping, as well as effluent treatment plant, are included in the costs.

The total process plant operating costs were estimated at CAD 8.21/t milled for a production rate of 24,000 t/d in Year 1 and Year 2 and CAD 7.50/t milled for a production rate of 27,000 t/d for Year 3 and beyond. The process plant operating cost is summarized in Table 21.14.

Table 21.14: Process Operating Costs Summary

OPEX Cost Category	24,000 t/d			27,000 t/d		
	Total OPEX (M CAD/y)	% of Total	Unit Cost (CAD/t milled)	Total OPEX (M CAD/y)	% of Total	Unit Cost (CAD/t milled)
Labour	8.6	12.0%	0.99	8.6	11.7%	0.88
Electrical Power	17.4	24.2%	1.99	17.4	23.6%	1.77
Wear Parts	14.4	20.0%	1.64	14.4	19.5%	1.46
Maintenance Parts	5.9	8.3%	0.68	5.9	8.0%	0.60
Grinding Media	8.7	12.1%	0.99	8.7	11.8%	0.88
Reagents	15.9	22.0%	1.81	17.8	24.1%	1.81
Material Handling	0.3	0.3%	0.03	0.3	0.3%	0.02
R&D and Laboratory	0.7	1.0%	0.08	0.7	1.0%	0.07
Total	**71.9**	**100%**	**8.21**	**73.9**	**100%**	**7.50**

The processing plant electrical power requirements are based on the electrical demands specified in the load list which take into account the installed power, the utilization factor, the mechanical load factor and the process availability. The power requirements for 24,000 t/d are estimated to be equal to those at 27,000 t/d (higher throughput but coarser grinding product size results in approximately the same power consumption in the comminution circuit). The installed power for most of the equipment (and all the major equipment) has been validated with suppliers during the budgetary quotation process. Some minor equipment power has been estimated in-house based on similar applications.

The wear parts cost category includes all the major equipment replacement parts (crusher liners, ball mill liners, HPGR rolls, etc.) and an allowance for the contractual workforce required to execute these replacements. The life cycle estimation and replacement parts costs are based on data provided by the selected manufacturer for each major type of equipment.

The maintenance parts cost category includes all the minor normal operation replacement parts such as pump casings, screen decks, chute liners, conveyor belts, etc. These costs are calculated to represent 5% of the total mechanical equipment costs.

Grinding media consumption is based on the ore abrasion index and is calculated to be CAD 0.04 kg/kWh. The ball mill power consumption and grinding media costs are used to evaluate an annual grinding media cost.

Most reagents consumption data is derived from testwork. For some low consumption reagents, such as antiscalant and refining flux, the requirements have been estimated based on similar projects. Pricing has been requested from suppliers and a selection has been made based on their products technical acceptability and cost.

Oxygen is produced on site by a vacuum swing absorption (“VSA”) plant. The plant is built, owned, and operated by a third party. A fixed monthly fee is associated with this service. Sulfur dioxide is also produced on site from elemental sulfur. The power required for sulfur melting is assumed to come from the heat recovery system of the power plant.

An allowance has been made for material handling costs (crushed ore handling, reagents transport, etc.) that includes mobile equipment fuel and maintenance costs. The associated mobile equipment labour is included in the workforce cost category.

R&D costs are fixed annual costs and laboratory fees have been estimated as a fraction of the sample load during each phase.

The power cost of site generated power was derived from three major components: i) forecasted energy price (natural gas); ii) workforce required to operate and maintain the power plant; and iii) maintenance costs over the LOM. The total power cost is estimated at CAD 0.059/kWh.

The natural gas price used for power requirements was evaluated at CAD 5.05/GJ. With annualized power demands, this represents CAD 0.043/kWh.

21.2.4 General and Administration Costs

The G&A costs, by year, are summarized in Table 21.15 and peak at approximately CAD 16M per year. The labour costs for G&A represent 26% of the total G&A budget. The G&A costs reflect the operating model which assumes a locally sourced workforce with no camp related costs.

Table 21.15: General and Administration Operating Costs Summary

G&A Costs (M CAD	Total	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
General Management	15.9	0.6	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	1.2	0.9	0.9	0.9	0.8
Accounting / Finance	8.8	0.3	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.5	0.5	0.5	0.5
Supply Chain	17.9	0.7	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	1.4	0.8	0.8	0.8	0.5
Information Technology	9.3	0.4	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.4	0.4	0.4	0.3
Human Resources	12.4	0.5	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	1.0	0.6	0.6	0.6	0.5
Health and Safety	11.5	0.4	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.9	0.6	0.6	0.5	0.4
Surface Support	38.6	1.4	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.8	2.3	2.3	2.3	2.0
Environment	27.2	1.0	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.9	1.8	1.8	1.8	1.7
Security	9.0	0.4	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.4	0.4	0.4	0.3
Corporate	8.1	0.3	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.7	0.3	0.3	0.3	0.2
Customs, Taxes and Duties	13.7	-	0.8	1.0	1.6	1.6	1.5	1.5	1.1	1.1	0.7	0.7	0.7	0.7	0.3	0.3
Insurance and Banking Fees	32.5	1.7	3.1	2.7	2.6	2.5	2.5	2.4	2.3	2.2	2.1	2.0	1.9	1.9	1.8	0.9
Total G&A Costs	**204.8**	**7.7**	**15.8**	**15.6**	**16.1**	**16.0**	**15.9**	**15.8**	**15.4**	**15.3**	**14.8**	**14.7**	**11.3**	**11.2**	**10.6**	**8.4**
Total G&A Costs CAD/t milled	**1.45**	**1.76**	**1.81**	**1.59**	**1.64**	**1.62**	**1.62**	**1.60**	**1.56**	**1.55**	**1.50**	**1.49**	**1.15**	**1.14**	**1.08**	**0.90**

22. ECONOMIC ANALYSES

This section presents all elements of the economic model which principally consist of metal production and revenues, royalty agreements, operating costs, capital costs, sustaining capital, salvage value, closure and reclamation costs, taxation and net Project cash flow.

The economic analysis is carried out in real terms (i.e. without inflation factors) in Q3 2016 Canadian dollars without any project or equipment financing assumptions. The economic results are calculated as of the start of the 42 months pre-production CAPEX phase which includes engineering and procurement with all prior costs treated as sunk costs but considered for the purposes of taxation calculations. The economic results such as the net present value ("NPV") and internal rate of return ("IRR") are calculated on an annual basis.

22.1 Assumptions

The key assumptions influencing the economics of the Project include:

- Gold price in USD/oz;
- Exchange rates, mainly the CAD/USD exchange rate;
- Diesel price in CAD/L;
- Natural gas price for power generation.

22.1.1 Gold Price

The base case gold price selected for the economic evaluation is USD 1,250/oz. This price assumption is supported by independent forecasts and consensus pricing. The average long-term gold price of 16 independent forecasts is USD 1,299/oz with a high of USD 1,500/oz and a low of USD 1,013/oz.

22.1.2 Exchange Rates

The base case Canadian dollar exchange rate for economic evaluation is CAD/USD 1.30. Most operating costs are estimated in Canadian dollars with the US dollar denominated gold revenue converted to Canadian dollars. The average Canadian dollar exchange rate used by gold producers for 2015 was CAD/USD 1.27, with a range from CAD/USD 1.10 to 1.37.

The Euro exchange rate assumption is CAD/EUR 1.40, which is relevant in purchasing some mining and power plant equipment sourced from Europe. An exchange rate range of CAD/EUR 1.35 to 1.45 is considered appropriate based on an average of forecasts from various financial institutions.

22.1.3 Fuel

The reference diesel fuel price used for estimating operating costs is CAD 0.75/L, which is an estimated delivered price to site for coloured diesel destined for off-road vehicles. It is exclusive of provincial road taxes and sales taxes which are reimbursable but includes a federal excise tax of CAD 0.04/L. The reference price is benchmarked off the Thunder Bay, Ontario rack price for ultra-low sulfur diesel no. 1. The price assumption for the FS and the Report is in line with the 18-month average price but is above the 6-month and 12-month average.

22.1.4 Natural Gas

The long-term natural gas price used in the power costs is the 2016-2020 average of (1) the current forward prices of the Alberta Energy Company ("AECO") and (2) the Delta Energy price assumptions for that same period. The assumption for the Report and FS is CAD 5.05/GJ including transportation and charges to site. The power generation plant is fueled by natural gas and is therefore an important consumable for the processing cost.

22.2 Metal Production and Revenues

Gold production over the Project life is 4,193 koz based on an average recovery of 90.2%. Gold production during pre-production is 11.1 koz, generating estimated revenue of CAD 17.5M (net of transportation, refining and royalty costs) which offsets pre-production CAPEX. Gold production during operations is 4,181 koz and gross revenue is CAD 6,795M.

During the commissioning and start-up phase there is a build-up of 4 koz of inventory in the process circuit which is recovered at the end of operations. The commissioning and ramp-up schedule is presented in Table 22.1. Beginning in Year 1, small tonnages are fed into the crushing circuit for cold commissioning, with processing starting at an average of 6 kt/d and ramping up to 18 kt/d over a 4-month period. At this point, commercial production is achieved with the plant processing 75% of nameplate throughput for 30 days, which meets the commercial production definition of at least 60% of nameplate throughput over 30 days. The metallurgical recoveries during the pre-production period have been lowered below the expected recoveries during normal operations.

Table 22.1: Mill Commissioning and Ramp-up

Mill Commissioning and Ramp-up	Tonnage (kt/month)	t/d (avg.)	% Nameplate	% Gold Recovery
Pre-Prod month 1	93	3,000	12.5%	75.0%
Pre-Prod month 2	90	3,000	12.5%	75.0%
Pre-Prod month 3	186	6,000	25%	75.0%
Pre-Prod month 4	540	18,000	75%	85.0%
Commercial Production Year 1	707	22,800	95%	91.5%
Operations	720	24,000	100%[1]	91.6%

Note: Represents 100% of capacity which is 24,000 t/d vs. full nameplate capacity at 27,000 t/d

The annual mine and mill production is summarized in Table 22.2 and Figure 22.2. The gold production profile is presented in Figure 22.1. Commercial production is reached in Year 1 and ends in Year 15 for a 14.5-year mine life. Additional details on the production schedule are presented in Subsection 16.3.

Figure 22.1: Annual Gold Production Profile

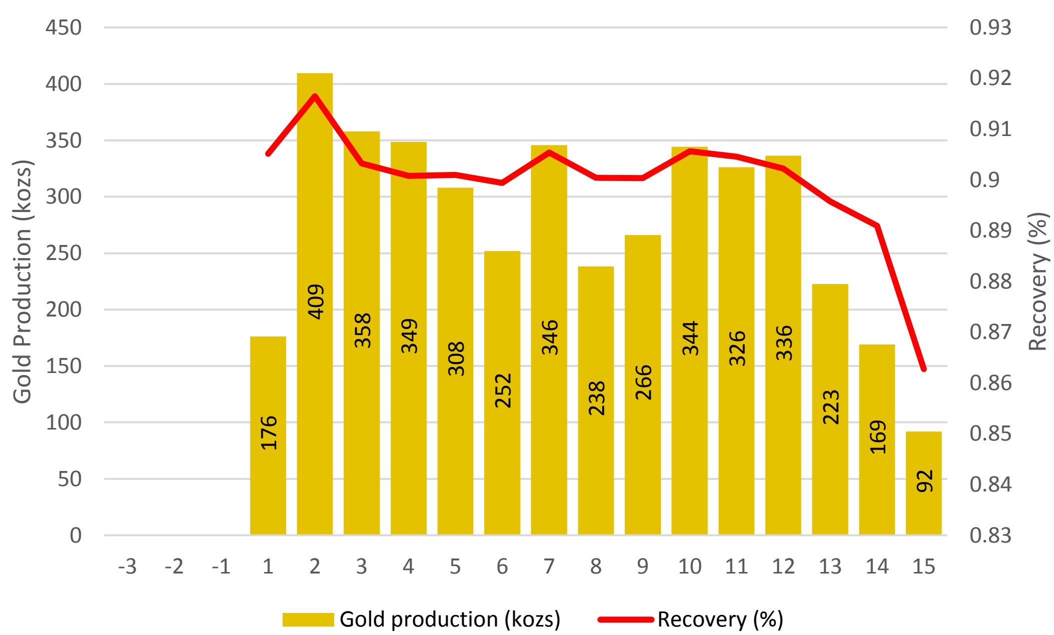
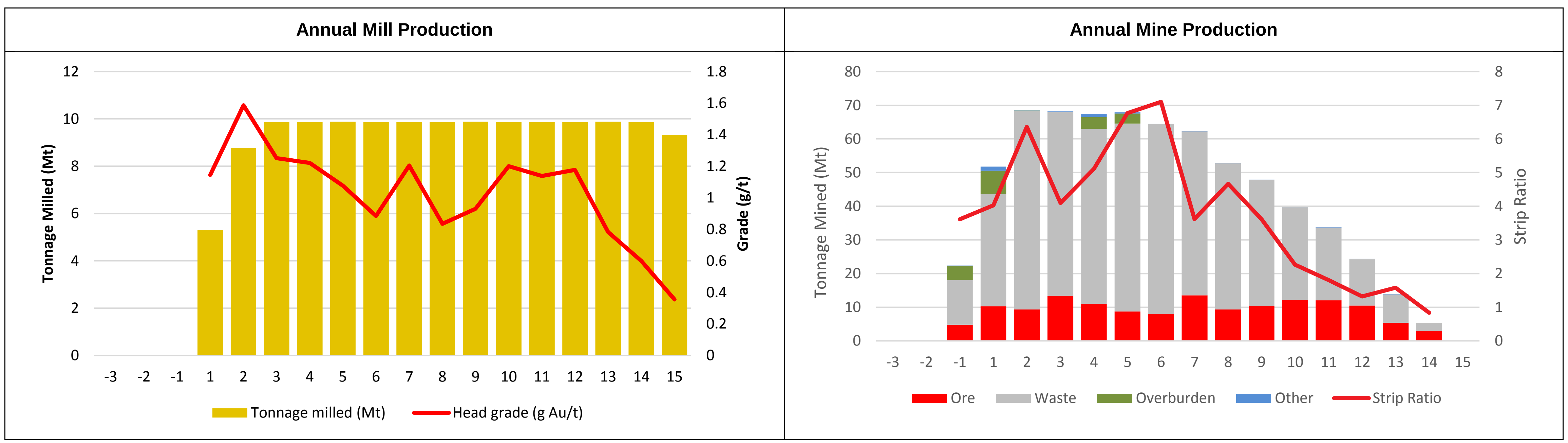


Table 22.2: Annual Mine and Mill Production Summary

Production Summary	Total	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15
Mill Production																			
Tonnage Milled (Mt)	**141.71**			-	5.29	8.76	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.86	9.86	9.88	9.86	9.33
Gold Processed (koz)	**4,647**			-	195	447	396	387	342	280	382	265	296	380	360	373	249	190	107
Head Grade (g Au/t)	**1.02**			-	1.15	1.59	1.25	1.22	1.08	0.88	1.20	0.84	0.93	1.20	1.14	1.18	0.78	0.60	0.36
Gold Production (koz)	**4,193**			-	176	409	358	349	308	252	346	238	266	344	326	336	223	169	92
Recovery	**0.902**				0.905	0.916	0.903	0.901	0.901	0.899	0.905	0.900	0.900	0.906	0.905	0.902	0.896	0.891	0.863
Mine Production																			
Waste (Mt)	**527.67**	-	-	13.19	33.36	58.95	54.61	51.92	55.80	56.47	48.75	43.42	37.43	27.51	21.66	13.76	8.43	2.42	-
Overburden (Mt)	**17.80**	-	-	4.25	6.86	0.17	-	3.55	2.97	-	-	-	-	-	-	-	-	-	-
Other (Mt)	**3.46**	-	-	0.03	1.26	0.06	0.17	0.97	0.41	0.02	0.14	0.04	0.05	0.11	0.07	0.11	0.03	0.00	-
Ore (Mt)	**141.71**	-	-	4.83	10.30	9.31	13.38	11.04	8.75	7.96	13.50	9.32	10.38	12.19	12.03	10.49	5.36	2.90	-
Total Mined (Mt)	**690.65**	-	-	22.31	51.78	68.48	68.16	67.47	67.93	64.45	62.39	52.77	47.86	39.80	33.76	24.35	13.83	5.32	-
Strip Ratio (W:O)	**3.87**		-	3.62	4.03	6.36	4.09	5.11	6.76	7.10	3.62	4.66	3.61	2.27	1.81	1.32	1.58	0.83	-

Figure 22.2: Mine and Mill Production Profile



22.3 Royalties

Certain mining claims in the Hardrock deposit are subject to a 3% net smelter royalty ("NSR") payable to Franco Nevada Corporation. Over the course of the LOM, payments under this royalty are expected to total CAD 203M.

22.4 Operating Cost Summary

Operating costs include mining, processing, G&A services, transportation and refining of gold. The operating cost summary is presented in Table 22.3.

Detailed operating cost budgets have been estimated from first principles based on detailed wage scales, consumable prices, fuel prices and productivities. The transportation and refining cost used in the economic model is CAD 3.00/oz and is based on indicative pricing from a Canadian refiner.

Table 22.3: Operating Cost Summary

Category	Total Costs (M CAD)	Unit Cost (CAD/t milled)	Cost per oz (CAD/oz)
Mining	1,412	10.03	338
Processing	1,061	7.54	254
G&A	205	1.45	49
Transportation & Refining	13	0.09	3
Other Costs	56	0.40	13
Royalties	203	1.45	49
Total Operating Cost	**2,950**	**20.95**	**705**
Closure & Reclamation	54	0.38	13
Sustaining Capital	257	1.82	61
All-in Sustaining Cost (AISC)	**3,261**	**23.16**	**780**

The average operating cost for the LOM is CAD 705/oz and is lower at CAD 659/oz for the first four full years of operations. The costs increase in Year 6 to Year 8 when feed grade is lower and increases for the last few years when low grade stockpile ore is processed. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs averages CAD 780/oz over the mine life.

22.5 Capital Expenditures

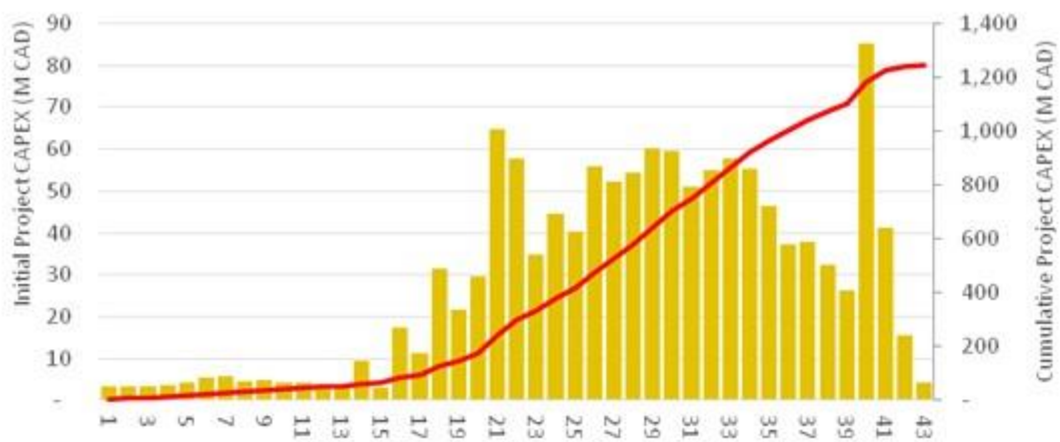
The capital expenditures include initial capital ("CAPEX") as well as sustaining capital to be spent after commencement of commercial operations.

22.5.1 Initial Capital

The CAPEX for Project construction, including processing, mine equipment purchases, pre-production activities, infrastructures and other direct and indirect costs is estimated to be CAD 1,242M. The total initial Project capital includes a contingency of CAD 131M which is 11.8% of the total CAPEX. Other non-project related expenditures during the construction period bring the total initial capital to CAD 1,247M.

The monthly CAPEX is presented in Figure 22.3. The higher expenditures at the end of the CAPEX phase (Month 40) correspond to the last deliveries of mining equipment prior to commercial production.

Figure 22.3: Initial CAPEX by Month



The native currency assumptions of the CAPEX is 78% in Canadian dollars, 17% in US dollars and 5% in Euros.

22.5.2 Sustaining Capital Expenditures

Sustaining capital is required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works and additional infrastructure relocation. The sustaining capital is estimated at CAD 257M (Table 22.4).

Table 22.4: Sustaining Capital Summary

Sustaining Capital Costs	(M CAD)
Mine Equipment Capital Repairs	107.9
Mine Equipment Purchases	49.0
TMF Dam Construction	86.4
Mine Civil Works (ponds, roads, ditches, etc.)	4.3
Infrastructure Relocation/Compensation	9.0
Total Sustaining Capital	**256.6**

22.5.3 Salvage Value

A salvage value is estimated for some mining equipment purchased during operations that will not have been utilized to its useful life. A residual value is estimated for some of the major process plant equipment such as grinding mills, crushers and tank agitators. The power plant will have a residual value as the units will have a remaining useful life of 10 to 15 years at the end of operations. The salvage value is summarized in Table 22.5.

Table 22.5: Salvage Value

Salvage Value	(M CAD)
Process Plant Equipment	15.8
Power Plant Equipment	20.0
Mine Major Equipment	2.3
Total Salvage Value	**38.1**

22.6 Working Capital

Working capital is required to finance supplies in inventory. Given the accessibility of the site, the working capital requirements are considered low compared to remote operations.

22.7 Reclamation and Closure Costs

Reclamation and closure costs include infrastructure decommissioning, site preparation and revegetation, maintenance and post closure monitoring. The reclamation cost is funded with cash outflows provisioned in the economic model from Year 3 to Year 13 and spent over three years at the end of operations. The total reclamation and closure cost is estimated at CAD 54M, as summarized in Table 22.6.

Table 22.6: Reclamation and Closure Cost

Reclamation and Closure	(M CAD)
Progressive Rehabilitation	7.9
Linear Infrastructure	2.0
Plant Site Area	6.4
Tailings Management Facility (TMF)	7.7
Waste Rock, Overburden and Stockpiles	4.5
Water Management	3.8
Mine Hazards	0.4
Monitoring and Studies	8.9
Other	12.5
Total	**54.1**

22.8 Project Financing

The economic model excludes any Project debt or equipment financing and is therefore 100% financed through equity for the purposes of the Report and FS. The uses and sources of funds is summarized in Table 22.7. The funding requirement is CAD 1,246M.

Table 22.7: Funding Summary

Funding Summary	(M CAD)
Uses of Funds	
Construction Costs	1,242.4
Working Capital Adjustments	(0.5)
Other Costs During Construction	4.5
Total	**1,246.4**
Sources of Funds	
Equity	1,246.4
Total	**1,246.4**

22.9 Taxation

Partnerships are not legal tax paying entities under the Canadian *Income Tax Act* as the income or loss is calculated at the partnership level and allocated to the partners. Centerra and Premier will bear the responsibility for paying tax on profits generated by the Partnership. The after-tax results are based on the assumption that the Partnership is a taxable Canadian entity and tax is calculated based on the tax rules in Ontario. The calculations do not reflect the benefit of any historical tax positions held by either Centerra or Premier. Losses have been included from the date of the Partnership agreement. The Ontario mining tax, federal income tax and provincial income tax during the LOM totals CAD 690M.

22.9.1 Ontario Mining Tax

Ontario mining tax is levied at a rate of 10% on taxable profit in excess of CAD 0.5M derived from a mining operation in Ontario. There are specific guidelines for the calculation of profit and depreciation for the purpose of the Ontario mining tax. A mining tax exemption on up to CAD 10M of profit during a 3-year period is available to each new non-remote mine, of which Hardrock does not qualify. The total Ontario mining taxes are CAD 161M over the Project life.

22.9.2 Income Taxes

The federal and provincial income taxes have both been estimated from an identical taxable income which is arrived at by deducting the Ontario mining tax and various tax depreciations allowances. The federal income tax rate is 15% while the Ontario income tax rate is 10%. The total federal income tax is estimated at CAD 317M and the provincial income tax at CAD 211M.

22.9.3 Carbon Taxes

The Ontario provincial government is implementing a program called Cap and Trade that will take effect from January 1, 2017 in an effort to limit carbon and similar emissions by business. The program requires that facilities with emissions of 25,000 t or more of GHG emissions per year are defined as “Mandatory Participants” and required by law to participate in the program. For the mining industry, the regulations mandate that entities engaged in the smelter or refining of certain metals and emit 10,000 CO_2e or more annually are required to begin quantifying and reporting their emissions. The power plant would be considered a combined heat and power ("CHP") plant and the current regulations for CHP’s are ambiguous. GGM is in the process of clarifying the degree of impact. Any potential impacts are not considered in the economic analysis.

22.10 Economic Results

The main economic metrics used to evaluate the Project consist of net undiscounted after-tax cash flow, net discounted after-tax cash flow or NPV, IRR and payback period. The discount rate used to evaluate the present value of the Project corresponds to the weighted average cost of capital. The discount rate represents the required rate of return that an investor would expect based on the risks inherent in achieving the expected future cash flows.

A 5% discount rate is commonly used for gold projects located in a developed and stable mining jurisdiction. The relative country and project risk is assessed as low for the Hardrock Project. Sensitivities have been presented at various discount rates ranging from 5 to 8% (Table 22.9).

A summary of the Project economic results is presented Table 22.8 and the annual Project cash flows are presented in Table 22.11. The total after-tax cash flow over the Project life is CAD 1,636M and after-tax NPV 5% is CAD 709M. The after-tax Project cash flow results in a 4.5-year payback period from the commencement of commercial operations with an after-tax IRR of 14.4%.

Table 22.8: Project Economic Results Summary

Project Economics		Base Case Results
Production Summary		
Tonnage Mined	Mt	691
Ore Processed	Mt	142
Average Head Grade	g Au/t	1.02
Gold Processed / Contained Gold	koz	4,647
Recovery	%	90.2%
Gold Production	koz	4,193
Cash Flow Summary		
Gross Revenue	M CAD	6,795
Mining Costs (including rehandle)	M CAD	(1,412)
Processing Costs	M CAD	(1,061)
G&A Costs	M CAD	(205)
Royalty, Transportation, Refining and Other Costs	M CAD	(272)
Total Operating Costs	M CAD	(2,950)
Operating Cash Flow Before Taxes	M CAD	3,845
Initial CAPEX	M CAD	(1,247)
Sustaining Capital	M CAD	(257)
Total Capital	M CAD	(1,504)
Salvage Value	M CAD	38
Closure Costs	M CAD	(54)
Taxes (mining, provincial and federal)	M CAD	(690)
Before-Tax Results		
Before-Tax Undiscounted Cash Flow	M CAD	2,325
NPV 5% Before-Tax	M CAD	1,095
Project Before-Tax Payback Period	years	3.9
Project Before-Tax IRR	%	17.9%
After-Tax Results		
After-Tax Undiscounted Cash Flow	M CAD	1,636
NPV 5% After-Tax	M CAD	709
Project After-Tax Payback Period	years	4.5
Project After-Tax IRR	%	14.4%

Table 22.9: Project Net Present Values at Various Discount Rates

Discount Rate	Before-Tax Project NPV (M CAD)	After-Tax Project NPV (M CAD)
5%	1,095	709
6%	933	587
7%	791	481
8%	667	387

22.11 Sensitivity Analysis

A sensitivity analysis was performed for ±10% and ±15% variations for gold price, exchange rate, OPEX and CAPEX. Each parameter was calculated independent of any correlations that may exist between variables such as for gold price and exchange rate, which tend to be negatively correlated.

The Project is most sensitive to gold price followed by exchange rate, initial capital costs and operating costs. The Project is somewhat less sensitive to the CAD/USD exchange rate than the gold price in USD/oz as some of the CAPEX are in US dollars. The sensitivity on gold grade is identical to that of the gold price and is therefore not presented in the following figures.

The results of the sensitivity analysis on an after-tax undiscounted cash flow, NPV 5% and IRR is presented in Figure 22.4, Figure 22.4: Project After-Tax Undiscounted Cash Flow

Figure 22.5 and Figure 22.6 respectively.

Table 22.10: Project After-Tax Sensitivities

	NPV 5%			IRR		
Feasibility Study (FS) Variable	**-15 % (CAD M)**	**FS (CAD M)**	**+15 % (CAD M)**	**-15 % (% IRR)**	**FS (% IRR)**	**+15 % (% IRR)**
Operating Costs	873	709	543	16.3	14.4	12.4
Capital Costs	824	709	590	17.4	14.4	12.1
Exchange Rate (CAD/USD)	314	709	1,093	9.6	14.4	18.5
Gold Price	293	709	1,113	9.2	14.4	19.0

Table 22.11: Project Cash Flow Summary

Life-of-Mine Cash Flow	Total (3)	-3	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16
Sales and Revenue																				
Gold Price (US$/oz)	1,250	-	-	-	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	1,250	-
Exchange Rate (C$/US$)	1.30	-	-	-	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	1.30	-
Gold Sold (koz)[2]	4,181				161	409	358	349	308	252	346	238	266	344	326	336	223	169	96	
Gold Revenue (M $)	6,795				262	665	582	566	501	409	562	387	432	560	530	547	362	275	156	-
Operating Costs (M CAD)																				
Mining	1,412				53.9	121.4	129.1	128.5	127.8	125.0	128.4	118.4	115.2	104.5	89.6	78.6	54.2	31.6	5.6	-
Processing	1,061				36.0	68.3	73.9	73.9	74.1	73.9	73.9	73.9	74.1	73.9	73.9	73.9	74.1	73.9	69.9	-
Administration	205				7.7	15.8	15.6	16.1	16.0	15.9	15.8	15.4	15.3	14.8	14.7	11.3	11.2	10.6	8.4	-
Royalties	203				7.8	19.9	17.4	17.0	15.0	12.3	16.8	11.6	12.9	16.8	15.9	16.4	10.8	8.2	4.7	-
Refining & Other	68				0.9	2.4	1.8	2.2	2.1	4.4	4.7	6.9	3.9	5.1	7.9	7.9	8.2	4.9	3.7	1.3
Total Direct Costs	**2,950**				**106.4**	**227.8**	**237.8**	**237.7**	**235.0**	**231.5**	**239.7**	**226.2**	**221.3**	**215.0**	**202.0**	**188.1**	**158.5**	**129.2**	**92.3**	**1.3**
Capital and Other Costs (M CAD)																				
Construction Capital	1,111	42.8	312.5	573.7	182.0	-														
Contingency	131	4.6	15.2	49.6	61.9	-														
Other Capital	5	2.6	1.2	0.5	0.3	-														
Sustaining Capital[1]	257	-	-	-	0.8	50.0	32.4	18.2	26.2	16.0	33.3	30.5	29.6	14.6	3.0	0.3	1.8	0.0	-	-
Working Capital	-	-	-	1.4	6.8	0.2	0.1	(0.4)	(1.1)	(1.5)	2.6	(2.6)	1.1	2.9	0.3	0.9	(2.2)	(0.3)	(1.2)	(7.1)
Reclamation Fund	54	-	-	-	-	-	0.4	1.4	2.2	3.1	4.0	4.9	5.8	6.7	7.6	8.5	9.4	-	-	-
Salvage Value	(38)	-	-	-	-	-	-	-	-	-	-	-	(1.4)	-	(0.8)	(0.2)	-	-	-	(35.8)
Total Capital & Other	**1,519**	**49.9**	**329.0**	**625.2**	**251.8**	**50.3**	**32.8**	**19.2**	**27.3**	**17.7**	**39.9**	**32.9**	**35.1**	**24.2**	**10.1**	**9.5**	**8.9**	**(0.3)**	**(1.2)**	**(42.8)**
Cash Flow (M CAD)																				
Pre-Tax Cash Flow	2,325	(49.9)	(329.0)	(625.2)	(96.4)	387.1	311.1	309.6	238.3	160.2	282.1	128.1	175.9	320.5	317.8	349.0	194.4	145.6	64.7	41.5
Cash Taxes	690	-	-	-	-	28.9	31.1	40.5	35.9	18.1	68.4	26.0	45.7	91.6	89.6	102.4	54.0	38.8	14.3	4.1
After-tax cash flow	**1,636**	**(49.9)**	**(329.0)**	**(625.2)**	**(96.4)**	**358.2**	**280.0**	**269.1**	**202.4**	**142.2**	**213.7**	**102.1**	**130.2**	**228.8**	**228.2**	**246.6**	**140.4**	**106.8**	**50.4**	**37.4**

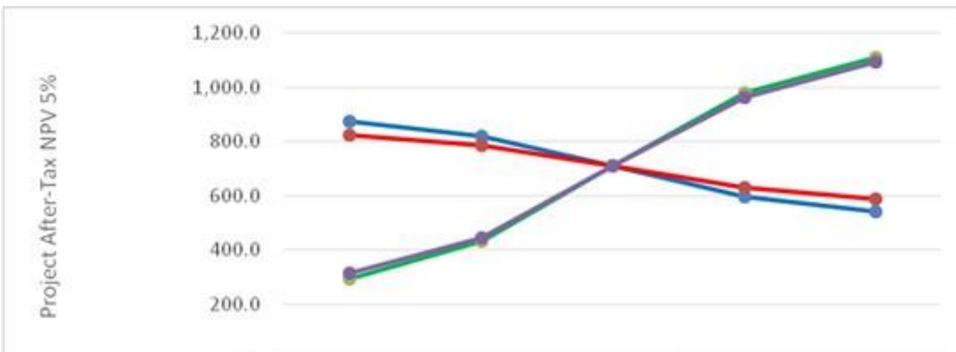
Notes:

1. Non-GAAP measure.
2. Pre-production gold sales treated as credit against pre-production costs in construction capital.
3. Numbers may not add due to rounding.

Figure 22.4: Project After-Tax Undiscounted Cash Flow

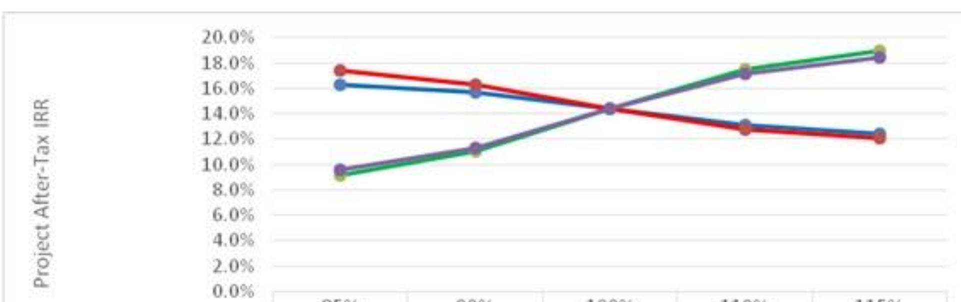
	85%	90%	100%	110%	115%
OPEX	1,904	1,815	1,636	1,456	1,365
Initial CAPEX	1,755	1,715	1,636	1,554	1,512
Gold Price	963	1,189	1,636	2,075	2,293
CAD/USD Exch. Rate	985	1,204	1,636	2,061	2,273

Figure 22.5: Project After-Tax NPV 5% Sensitivity



	85%	90%	100%	110%	115%
OPEX	873.2	818.5	709.0	598.8	542.7
Initial CAPEX	824.4	785.9	709.0	630.2	589.9
Gold Price	293.1	433.3	709.0	978.5	1,112.6
CAD/USD Exch. Rate	314.0	447.2	709.0	965.5	1,093.2

Figure 22.6: Project After-Tax IRR Sensitivity



	85%	90%	100%	110%	115%
OPEX	16.3%	15.7%	14.4%	13.1%	12.4%
Initial CAPEX	17.4%	16.3%	14.4%	12.8%	12.1%
Gold Price	9.2%	11.0%	14.4%	17.5%	19.0%
CAD/USD Exch. Rate	9.6%	11.3%	14.4%	17.2%	18.5%

23. ADJACENT PROPERTIES

23.1 Hardrock

There are no adjacent properties that have any significant information relating to the Project. GGM maintains a significant land position in the Geraldton mining camp, and most of the camp's historical mineral deposits (Figure 23.1 and Table 23.1) are located within the boundaries of the GGM projects.

Table 23.1: Gold Production Statistics for the Bankfield, Little Long Lac, Magnet, Talmora Long Lac and Tombill Mines (from Ferguson et al., 1971; Mason and White 1986)

	Bankfield Mine	Little Long Lac Mine	Magnet Mine	Talmora Long Lac Mine	Tombill Mine	Total
Years of Production	1937-1942, 1944-1947	1934-1954, 1956	1936-1943, 1946-1952	1942,1947-1948	1938-1942,1955	
Ore Milled (short tons)	229,009	1,782,516	359,912	9,570	190,623	2,571,630
Ore Milled (metric tonnes)	207,757	1,617,099	326,512	8,682	172,933	2,332,983
Au Grade (oz/t)	0.290	0.340	0.423	0.147	0.361	0.348
Au Grade (g/t)	9.94	11.65	14.49	5.04	12.36	11.92
Gold Ounces	66,416	605,449	152,089	1,406	68,737	894,097
Silver Ounces	7,590	52,750	16,879	67	8,595	85,881

23.1.1 Talmora Long Lac Mine (Past-Producer)

This description was mostly taken from Ferguson et al. (1971) except where noted.

The past-producing Talmora Long Lac Mine is located in Errington Township, on the south side of Barton Bay, Kenogamisis Lake, and 4 km southwest of the Town of Geraldton (Figure 23.1).

Between 1934 and 1936, an extensive surface trenching and diamond drilling program was performed by Longlac Lagoon Gold Mines, revealing three mineralized zones.

Between 1938 and 1940, a shaft was sunk to a depth of 544 ft (165.8 m) with levels at 195 ft (59.4 m), 315 ft (96.0 m) and 515 ft (157.0 m) on which 4,796 ft (1,461.8 m) of drifting and 1,038 ft (316.4 m) of crosscutting were done. Diamond drilling included 400 ft (121.9 m) from surface and 2,449 ft (746.5 m) in four underground holes. All work was performed by Elmos Gold Mines Ltd.

Between 1940 and 1942, trenching, stripping and two underground diamond drill holes totalling 234 ft were carried out by Tombill Gold Mines Ltd. A small 50 t mill was constructed on the mine site during winter

1941-1942. Underground work was resumed in March, 1942, and during the summer, 1,017 ounces of gold and 36.5 ounces of silver were produced from 3,947 t of sorted material. Due to the unfavourable wartime conditions, operations were suspended in November of the same year.

Between 1947 and 1948, Talmora Longlac Gold Mines Ltd completed 1,663 ft (506.9 m) of drifting and 670 ft (204.2 m) of crosscutting. Diamond drilling comprised four surface holes totalling 139 ft (42.4 m) and 91 underground holes totalling 10,776 ft (3,284.5 m). From the start of milling on September 15, 1947 until the cessation of operations on March 31, 1948, a total of 398.5 ounces of gold and 30 ounces of silver were produced from 5,623 t of hoisted material, for an average grade of 0.07 oz Au/t. At the time operations were suspended, it was estimated that about 12,000 t with an average grade of 0.37 oz Au/t remained in the mine (Pye, 1951).

These "reserves" are historical in nature and should not be relied upon. It is unlikely they conform to current NI 43-101 criteria or to CIM Definitions Standards and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context.

In 1968, some geophysical work was carried out by Tombill Mines Ltd.

The geology of the mine consists of greywackes with interbeds of iron formation intruded by a diorite mass, folded into a westerly-plunging anticline (Pye, 1951). A felsic intrusive occurs as a sill-like mass on the south limb. Two steeply dipping diabase dykes up to 30 m wide cross the anticline in a northerly direction. Shear zones striking N060° to N080° and dipping 45° near the diorite-greywacke contact contain quartz lenses averaging less than 30 cm thicknesses. The main sulfides are pyrite and arsenopyrite.

23.1.2 Little Long Lac Mine (Past-Producer)

This description was mostly taken from Ferguson et al. (1971) except where noted.

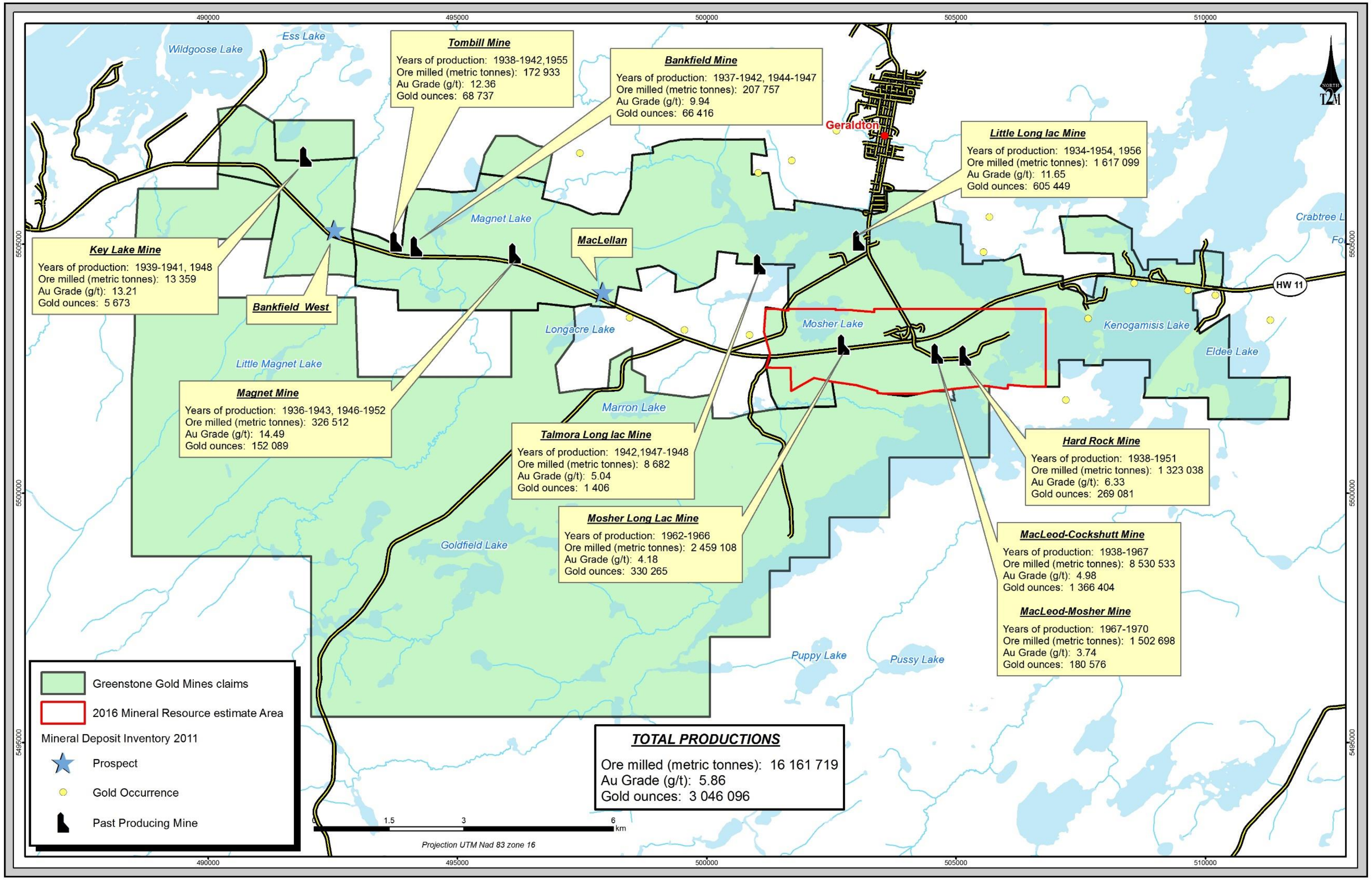
The past-producing Little Long Lac Mine is located in the southeastern part of Errington Township, extends eastwards into Ashmore Township and is bounded to the north by Kenogamisis Lake. The Little Long Lac Mine is located about 2 km south of Geraldton (Figure 23.1).

Between 1933 and 1953, a shaft was sunk to a depth of 2,318 ft (706.5 m) with levels at 200 ft (61.0 m), 300 ft (91.4 m), 445 ft (135.6 m), 570 ft (173.7 m), 695 ft (211.8 m), 850 ft (259.1 m), 1,000 ft (304.8 m), 1,152 ft (351.1 m), 1,300 ft (396.2 m), 1,450 ft (442.0 m), 1,600 ft (487.7 m), 1,750 ft (533.4 m), 1,900 ft

(579.1 m), 2,050 ft (624.8 m) and 2,200 ft (670.6 m). From level 2,200, a winze was sunk to a depth of 3,952 ft (1,204.6 m), with levels at 2,405 ft (733.0 m), 2,558 ft (779.7 m), 2,711 ft (826.3 m), 2,864 ft (872.9 m), 3,013 ft (918.4 m), 3,159 ft (962.9 m), 3,309 ft (1,008.6 m), 3,459 ft (1,054.3 m), 3,609 ft (1,100.0 m), 3,759 ft (1,145.7 m) and 3,920 ft (1,194.8 m). Drifting totalled 37,370 ft (11,390.4 m) and crosscutting 10,596 ft (3,229.7 m). Diamond drilling from surface totalled 105,626 ft (32,194.8 m) and underground drilling totalled 101,558 ft (30,954.9 m). A 150-ton mill was installed, and a small mill for scheelite production was added later. The work was performed by Little Long Lac Gold Mines Ltd.

From 1934 to 1954 and in 1956, a total of 605,409 ounces of gold and 52,750 ounces of silver were produced from 1,780,516 t of hoisted material. Average gold recovery was 0.34 oz/t.

Figure 23.1: Past Gold Producers on the Hardrock Project



Source: Innovexplo, 2015

Between 1967 and 1968, Little Long Lac Gold Mines Ltd drilled a total of 5,000 ft (1,524 m) to test the iron formation.

The geology of the mine consists of arenaceous metasediments with interbeds of iron formation and some mafic intrusive rocks that have been folded into a synclinal structure striking N272° (Pye, 1951). The deposits occur in fracture zones in massive quartz greywacke on the drag-folded north limb of the syncline. The Main vein zone is 3 to 4 ft wide (0.9 to 1.2 m), strikes approximately N075°, dips 80°, and consists of two parallel veins 2 to 6 in wide (5 to 15 cm). Some mineralization was also extracted from the lower grade 09 vein zone located about 600 ft (183 m) to the south of the Main zone; this zone is about 2 ft wide (60 cm), strikes N065°, dips 85°, and contains scheelite. The metallic constituents of quartz veins, which rarely make up more than 2 or 3% of the mineralization, include arsenopyrite, pyrite, pyrrhotite, sphalerite, chalcopyrite, galena and gold.

23.1.3 Magnet Consolidated Mine (Past-Producer)

This description was mostly taken from Ferguson et al. (1971) except where noted.

The past producing Magnet Consolidated Mine is located in the southwest part of Errington Township, about 8 km southwest of the Town of Geraldton (Figure 23.1).

The discovery of native gold on a small island in the southern part of Magnet Lake in 1931 initiated an intensive search for gold in the area. Between 1934 and 1936, trenching was performed by Magnet Lake Gold Mines and 24,641 ft of diamond drilling were carried out by Wells Mines Ltd. Drilling uncovered three mineralized zones, two of which, now known as the Magnet and Wells vein zones, showed considerable promise. In order to explore these zones jointly underground, the two companies amalgamated in 1936 to form the present Magnet Consolidated Mines Limited.

Between 1936 and 1940, a shaft was sunk to a depth of 1,115 ft (339.9 m) with levels at 203 ft (61.9 m), 328 ft (100.0 m), 480 ft (146.3 m), 630 ft (192.0 m), 780 ft (237.7m), 930 ft (283.5 m) and 1,080 ft (329.2 m) on which 11,181 ft (3,408.0 m) of drifting and 1,943 ft (592.2 m) of crosscutting was done. A total of 13 underground diamond drill holes totalling 1,665 ft (507.5 m) was completed. A 100-ton amalgamation-floatation mill was built.

Between 1940 and 1952, the shaft was continued to a depth of 1,772 ft (540.1 m), with additional levels at 1,230 ft (374.9 m), 1,380 ft (420.6 m), 1,555 ft (474.0 m) and 1,730 ft (527.3 m). An inclined winze 228 ft long (69.5 m) was constructed between levels 9, 10 and 11. A winze was sunk 931 ft (283.8 m) from the

1,730 ft level to a total depth of 2,640 ft (804.7 m) with levels at 1,884 ft (574.2 m), 2,037 ft (620.9 m), 2,160 ft (658.4 m), 2,312 ft (704.7 m), 2,460 ft (749.8 m) and 2,610 ft (795.5 m). Drifting totalled 19,585 ft (5,969.5 m) and crosscutting 2,944 ft (897.3 m). The company drilled seven surface diamond drill holes for a total of 4,029 ft (1,228.0 m) and 265 underground holes for a total of 43,054 ft (113,122.9 m).

From 1938 to 1943 and from 1946 to 1952, 152,089 ounces of gold and 16,879 ounces of silver were produced from 359,912 t of hoisted material. Average gold recovery was 0.42 oz/ton.

The geology of the mine consists of metasediments, mostly greywacke with interbeds of iron formation and conglomerate, striking N290° and dipping 75 to 80°. Intrusive rocks consist of dykes and sill-like masses of diorite and porphyry and younger diabase dykes cutting across the formations (Pye, 1951). The two deposits, raking N300 to N315°, consist of lenticular quartz veins and accompanying veinlets predominantly in sheared greywacke. The Magnet vein zone, with an average strike of N285° and a near-vertical dip, was developed over a maximum length of about 1,300 ft (396.2m). The leaner North zone, 50 to 100 ft (15.2 to 30.5 m) to the north, strikes N280° and dips vertically. The deposits at the Magnet mine consist chiefly of quartz with small amounts of carbonate and subordinate sulfides. The metallic constituents, which seldom constitute more than 5% of the mineralization, are arsenopyrite, pyrite, pyrrhotite, chalcopyrite, sphalerite, galena and gold.

23.1.4 Bankfield Mine (Past-Producer)

This description was mostly taken from Ferguson et al. (1971) except where noted.

The past producing Bankfield Mine is located near the southwest part of Magnet Lake in the west-central part of the Errington Township and extend into Lindsley Township. This historical mine is situated about 10 km west-southwest of the Town of Geraldton (Figure 23.1).

The property was originally staked in October 1931 by T. A. Johnson and Robert Wells when they discovered gold-bearing quartz occupying a shear zone cutting a small reef in the southern part of Magnet Lake. Subsequent to this discovery, a mineralized zone was found by surface exploration about 1,000 ft (304.8 m) southwest of the lake. Surface-trenching and diamond drilling indicated sufficient material to merit development by underground methods.

Between 1934 and 1936, a shaft was sunk to a depth of 552 ft (168.2 m) with levels at 150 ft (45.7 m), 250 ft (83.8 m) and 525 ft (160.0 m). Drifting totalled 2,468 ft (752.2 m) and crosscutting 781 ft (240.6 m).

Underground diamond drilling totalled 1,416 ft (431.6 m) and drilling from surface totalled 2,237 ft (431.6 m) during this period. Work was performed by Bankfield Gold Mines Ltd.

Between 1935 and 1942, a winze (located in Lindsley Township) was sunk from the 525-ft level to a depth of 1,297 ft (395.3 m) from the surface with levels at 779 ft (237.4 m), 900 ft (274.3 m), 1,025 ft (312.4 m), 1,150 ft (350.5m) and 1,275 ft (388.6m). Sub-levels were established at 275, 400, 1,025 and 1,150 ft. Drifting totalled 14,516 ft (4,424.5 m) and crosscutting 7,832 ft (2,387.2 m). Diamond drilling included 132 underground holes totalling 21,628 ft (6,592.2 m), six surface holes totalling 2,328 ft (709.6 m) and 10,145 ft (3,092.2 m) of unspecified drilling. A 100-ton cyanide mill was constructed. The work was performed by Bankfield Consolidated Mines Ltd.

From 1937 to 1942 and from 1944 to 1947, a total of 66,417 ounces of gold and 7,590 ounces of silver were produced from 231,009 t of hoisted material. Average gold recovery was 0.29 oz/t.

The geology of the mine consists of greywacke with bands of conglomerate, slate and iron formation striking N290 to 300° and dipping 75 to 80° (Pye, 1951). The rocks have been intruded by diorite and quartz porphyry, and ultimately by a 200 ft (61.0 m) wide diabase dyke which runs parallel to a strike fault near the mine workings. The main mineralized horizon, consisting of a sheared, brecciated and highly silicified zone, occurs near a contact between the sediments and a porphyry-diorite mass. It strikes N275 to N288°, dips 70 to 78°, has an average width of 7 ft (2.1 m) and is, including its extension into the adjacent Tombill property, 2,000 ft long (609.6 m). The deposits at the Bankfield Mine consists mainly of sheared and silicified greywacke and porphyry, mineralized with sulfides and small amounts of gold, and are cut by numerous "opalescent" grey quartz veins. The reported metallic minerals are arsenopyrite, pyrite, pyrrhotite, sphalerite, chalcopyrite, galena and ilmenite.

23.1.5 Tombill Mine (Past-Producer)

This description was mostly taken from Ferguson et al. (1971) except where noted.

The past producing Tombill Mine is located in the east-central part of Lindsley Township, about 10 km west-southwest of the Town of Geraldton (Figure 23.1).

Between 1935 and 1942, a shaft was sunk to a depth of 630 ft (192.0 m), with levels at 215 ft (65.5 m), 400 ft (121.9 m) and 600 ft (182.9 m) on which 3,762 ft (1,146.7 m) of drifting and 4,442 ft (135.9 m) of crosscutting were done. Diamond drilling comprised more than 12 surface holes totalling 15,570 ft (4,745.7 m) and 63 underground holes totalling 4,406 ft (1,342.9 m). A mill with a 100-ton capacity was erected, and was later increased to 150 t. All work was carried out by Tombill Gold Mines Ltd. In 1940, an agreement was reached allowing Bankfield Consolidated Mines Ltd to explore and develop a block below the 500-ft level.

From 1938 to 1942 and in 1955 (mill clean-up), a total of 69,120 ounces of gold and 8,595 ounces of silver were produced from 190,622 tons of hoisted material. Average gold recovery was 0.36 oz/t.

The geology of the mine consists of metasediments and felsic intrusive rocks along a sheared and fractured contact where mineralized zones developed. Associated minerals are pyrite, arsenopyrite and pyrrhotite.

23.1.6 Gold Potential of the Other Historical Mines

GMS has been unable to verify the above information on historical gold mines near the Hardrock Project. The presence of significant mineralization on these adjacent historical mines is not necessary indicative of similar mineralization at the Hardrock Project.

24. OTHER RELEVANT DATA AND INFORMATION

24.1 Project Execution & Organization

The Project execution plan is aligned with the FS and standard industry practices. The Project will be executed using an “Owner-managed” project delivery model. The execution strategy incorporates the key elements required for successful project development, including health and safety, environment, community relations, planning, project controls, document control, risk management, construction management and quality management.

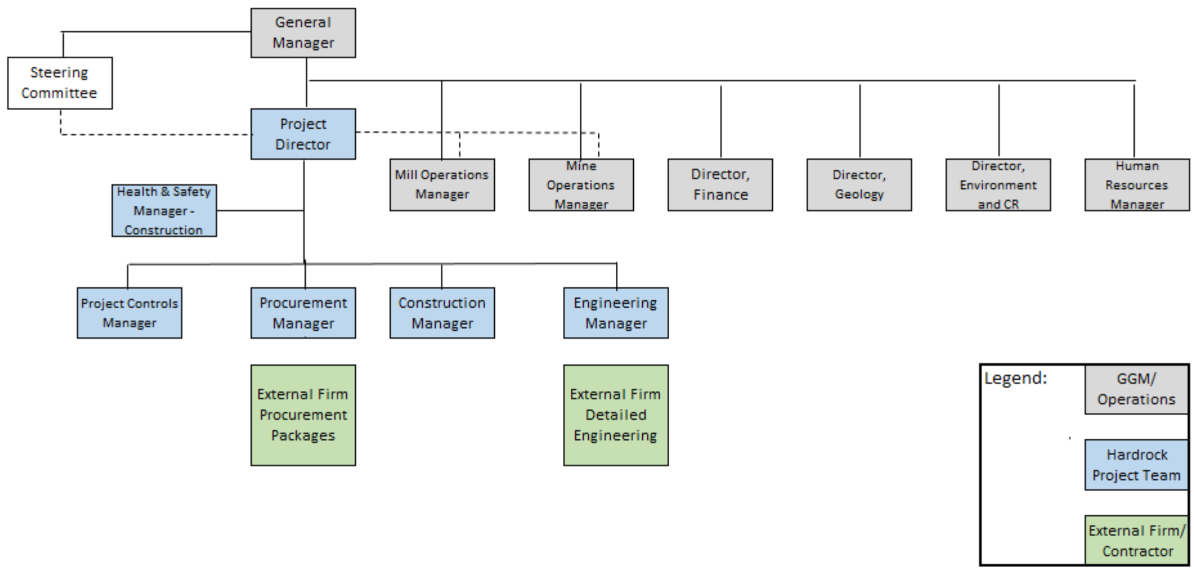
The Hardrock Project team is responsible to deliver all temporary and permanent infrastructure required to provide a fully operational mine capable of processing at a sustainable throughput of 27,000 t/d.

The Hardrock Project team will manage all of health and safety, engineering, procurement and contracts, construction, and pre-commissioning, as well as overall Project management services during the Project. GGM operations will provide environmental management and community relations support to the Project team. Area Managers will be in place for the infrastructure relocation scope (including Highway 11, Substation and MTO station) and the TMF.

An independent TMF review board (“ITRB”) will be established at GGM. The purpose of the ITRB will be to review and advise on the design, construction, operation, performance and closure planning for the TMF, and it will be in place prior to construction through to closure.

The Project team will peak at 64 people during the execution phase. The Project organization chart is shown in Figure 24.1

Figure 24.1: Hardrock Project Organization Chart



24.1.1 Health, Safety and Environment

GGM is committed to protecting the health and safety of workers and the public, as well as safeguarding the environment influenced by GGM's activities. Health, safety and environmental ("HSE") management plans will be developed and implemented for the Project pre-production and operations phases.

A HSE program will be developed for the construction period and associated activities. This program will follow all established regulations and standards for a mine site in construction. This program will also be compliant to the mine operation HSE program. GGM HSE requirements will be clearly communicated to and reinforced with all contractors on site.

An emergency response plan will be established. Ambulance services and a hospital are located in Geraldton, a few kilometres from the site. A nurse will be on-site during day shift to provide first aid.

24.1.2 Engineering and Procurement Management

Detailed engineering will primarily be outsourced to third party engineering consultants, except for the mine design which will be done by internal GGM resources. Detailed engineering for the substation and 44 kV power lines relocation will be done by Hydro One, while Union Gas will complete the engineering for the natural gas pipeline. Certain scopes including the administration building and power plant will be engineered under a design/supply arrangement.

Procurement and contracting activities will include pre-qualification and selection of vendors, sourcing of equipment and bulk materials, expediting of goods and documentation deliverables, inspection surveillance of equipment and materials, transportation, logistics, and warehousing, field procurement, materials management during construction, as well as the contracting of all necessary engineering, consulting, construction or installation services.

24.1.3 Construction Management

The Project will be construction driven, and the construction management team will work with planning and engineering to influence the structure and timing of engineering deliverables, and to confirm the match of equipment and fabrication dates with required “on-site” dates. The master Project schedule will drive the work planning on site, and priority will generally be given to the critical path activities.

Labour relations strategies and plans will be implemented for construction. Construction will be on a 7-day / 10-hour schedule, except for start-up which will be on a 7-day / 24-hour schedule. Permanent facilities will be installed as early as practical to support the construction requirements and minimize the needs for temporary setups. A temporary modular camp is planned to be installed a few kilometres outside of the mine site and close to Geraldton.

The dismantling, relocation and/or reconstruction of existing facilities will be managed by the construction management team in close collaboration with the local community and facilities owner. Certain other properties will be handled through compensation rather than being rebuilt.

Pre-commissioning activities will include final inspections and adjustment of equipment before it is tested and the testing of equipment and components of the plant in an energized state to ensure that the equipment can be handed over to the commissioning team in an acceptable operating condition for process commissioning. Operations representatives will be actively involved in the pre-commissioning process.

24.1.4 Operational Readiness, Commissioning and Ramp-up Strategy

The GGM Operations team will be accountable for all hot commissioning activities with the support and involvement of the Project construction management team. A commissioning, handover and transition plan will be developed jointly with construction and operations teams for each facility.

The ramp-up period to commercial production is expected to take approximately four months. The pre-production phase of the Project is considered complete once commercial production is achieved which is defined as achieving 60% of the design throughput over a period of 30 days.

Third party operational readiness planning will be implemented in the detailed engineering phase to maintain a rigorous and disciplined process around planning, staffing, training, budgeting, and execution of the Project operation and start-up.

24.1.5 Risk Management

GGM's risk identification, assessment and mitigation process will continue to be applied throughout the detailed engineering, construction, operation, and closure phases.

24.1.6 Quality Assurance and Quality Control

Quality Assurance and Quality Control ("QA/QC") will be applied to all levels of the Project. All designs will conform to the codes and standards applicable for the province, and applicable acts and regulations.

Engineering consultants and construction contractors will be responsible for their own quality control as specified in the terms and conditions of their contracts. Certain quality control work will be performed by third parties during execution.

The Project team will be responsible for ensuring that contractors follow their quality control programs, through the implementation of a comprehensive quality assurance program. Third parties will be used as required for on-site testing and laboratory analysis

24.1.7 Project Controls

Project controls will be implemented to provide Project management and other stakeholders with transparent and timely information on project progress, performance and variances. This information will be used to assist management in setting priorities and making decisions that will support the delivery of the Project on time and on budget. Project controls includes planning and scheduling, progressing, cost control, change management, estimating, forecasting, earned value analysis and reporting. Robust change management processes will be in place to identify, mitigate and manage change. The schedule is structured to identify the critical path and link all activities. Measurement of physical progress, quantities and hours against the baseline will be a key component of the schedule updates.

24.2 Project Schedule

The Project executive summary schedule is shown in Figure 24.2. The pre-production period, which includes detailed engineering, procurement, construction and commissioning is planned for 42 months, with a 23-month construction period. Timing of detailed engineering has been aligned with construction requirements. Procurement is aligned with the required-on site dates and construction activities to meet schedule requirements. Activities such as earthworks, concrete and structural steel have been scheduled for the non-winter months where possible.

Figure 24.2: Hardrock Project Level 1 Schedule

GREENSTONE GOLD MINES

Hardrock Project
Engineering Procurement Construction Level 1 Schedule

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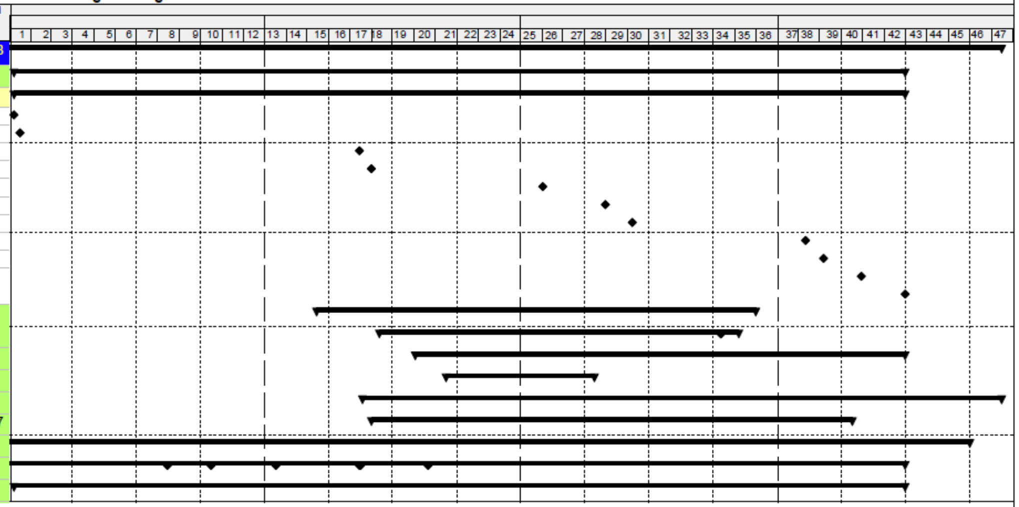
Activity ID	Activity Name	Duration	Budgeted Hours
	Greenstone Gold Mine - Hardrock Project MS 43101	1723	3404848
	000 - General	1228	0
	010 - Project Milestones	1228	0
M04	Start Detailed Engineering (Process Plant)	0	0
M05	Start Long Lead Equipment Procurement	0	0
M11	Approval of Basic Permits & Start of Construction	0	0
M12	Temporary Camp Available for Use	0	0
M14	Start of Mining Pre-production	0	0
M13	Complete Truckshop	0	0
M15	Approval of Schedule 2 Listing Authorizations	0	0
M16	Complete Power Plant Commissioning	0	0
M17	Start of Crushing Plant Hot Commissioning	0	0
M18	Start of Ore Commissioning	0	0
M19	Declare Commercial Production	0	0
	100 - Infrastructure	612	246081
	200 - Power & Electrical	498	148397
	300 - Water & Tailings Management	670	252883
	400 - Mobile Equipment	199	27118
	500 - Repositioning	883	84509
	600 - Process Plant	658	1057217
	700 - Construction Indirects	1677	678030
	800 - General Services - Owner's Cost	1556	145080
	900 - Preproduction, Startup, Commissioning	1228	765533



◆ Milestone
Summary

Page 1 of 1

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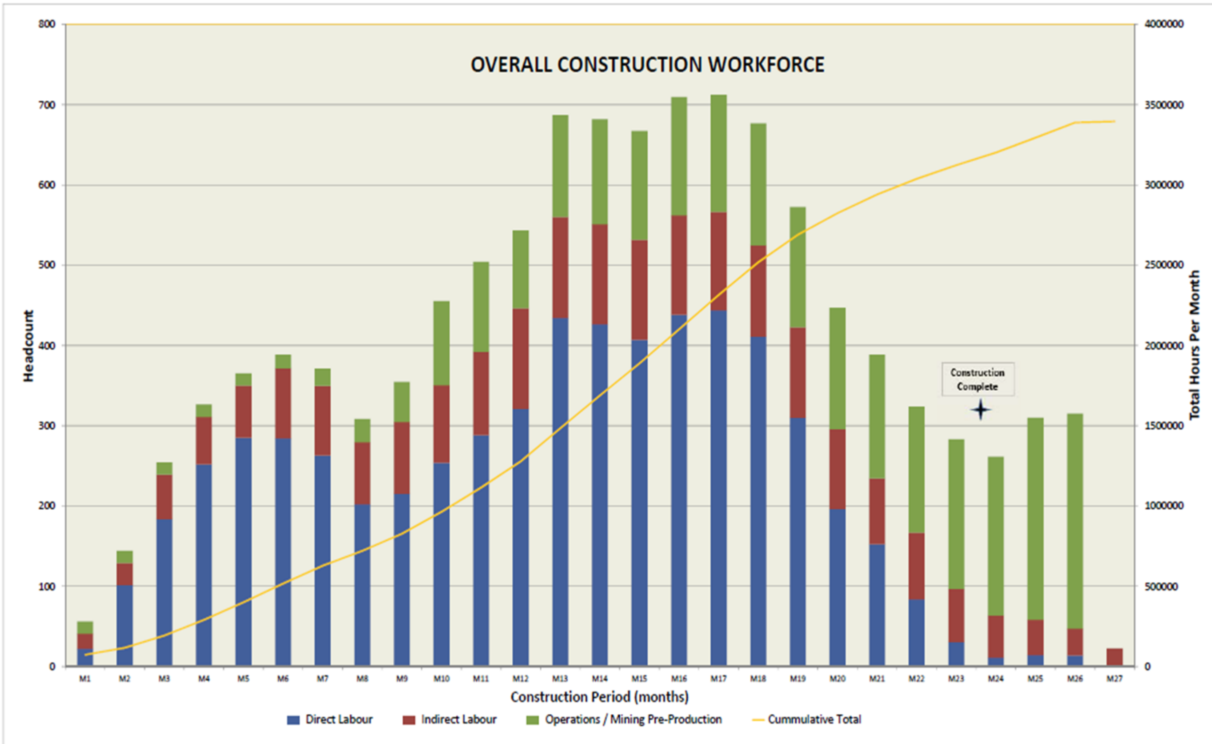
The Project workforce requirements are summarized in Figure 24.3. The total direct and indirect construction hours during the pre-production period is estimated at 3.4 million hours and the on-site workforce peaks at approximately 700 people.

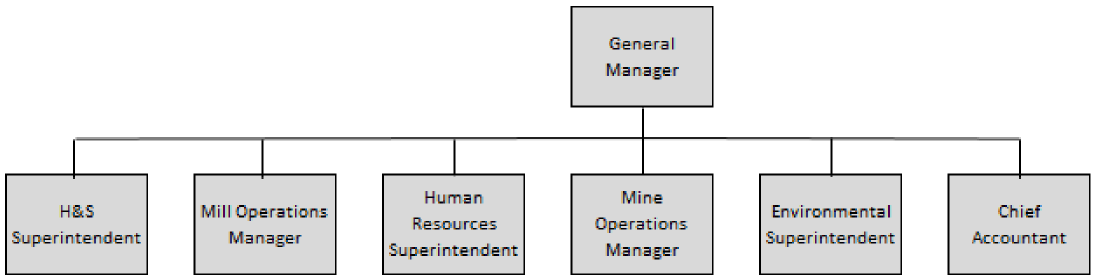
Figure 24.3: Hardrock Project Construction Workforce



24.3 <u>Operating Plan</u>

The overall organization has three main areas: mine, process plant and administration. The mine department will include operations, maintenance, and mine engineering. The process plant department, which includes the power plant involves operations and maintenance. The operation organizational chart is presented in Figure 24.4.

Figure 24.4: Operation General Organizational Chart



The Hardrock General Manager has single point accountability for all aspects of the Hardrock site and business. The General Manager will be assisted by a core team of functional managers who will have accountability for front line and functional management of the various business areas. The operations workforce peaks at 544 in Year 4, based on a 24 hour x 7 day operation.

The management team members are accountable to deliver on the HSE, production, business objectives and to assure management systems are established and effective.

The Environmental Superintendent and Health and Safety Superintendent will ensure that GGM meets or exceed the requirements of the environmental and occupational health and safety legislation respectively. This will be achieved by implementing and maintaining effective HSE management systems that drive continuous improvement.

The mine is headed by a Mine Operations Manager who is responsible for the overall management of the mine. Superintendent positions in engineering, geology, operations and maintenance report directly to the Mine Operations Manager.

The Technical Services group will consist of mine engineers, geologists, planners, plant engineers and technicians. The Technical Services group supports and services operations by ensuring that mine plans, systems, designs, records, budgets and schedules are in place and support the safe and efficient operation of the mine. The group will also provide engineering services, as required, for the process plant and related infrastructure.

The process plant, power plant, tailings area, water management and treatment will be operated and maintained by the process plant department. The process plant is planned to operate 24 hours per day, 365 days per year.

A predictive and planned maintenance strategy will be implemented at the process plant and power plant, and dedicated planners for each discipline will manage and coordinate all preventive maintenance procedures and work plans.

Primary and secondary crushing are ahead of the process plant stockpile area and will require dedicated operation and maintenance teams as the manipulations and overall maintenance aspects of the large machines are time intensive. A lower planned availability, as well as appropriate maintenance workforce, will cover these systems.

To achieve industry standard plant on-line time, plant planned shutdowns will be a focus area as the high pressure grind rolls, wet screens, grinding mills and miscellaneous conveying and feeding systems are interconnected. Plant shutdowns will be managed internally with external contracted assistance as required (workforce and specialized technical people).

The site accounting team reports to the chief accountant. The accounting department is responsible for budgeting, processing, measuring and reporting business results, in an efficient, accurate and timely manner.

A procurement system will be in place and will consist of market research, operation requirements planning, suppliers' management, purchasing and order controlling.

A warehouse management system and logistics system will be set-up that will ensure that each operating unit will be supplied with the required products and consumables.

The information technology ("IT") department is primarily responsible for standardizing, operating and managing the IT systems, applications and infrastructure required by the business. This includes management of business information systems, communication infrastructure, data security and integrity.

Gold balance, bullion preparation, sale to clients as well as sales contract management will be under the supervision of the Mill Operations Manager. The Chief Accountant will be accountable for sales and revenues reporting, as well as gold security and general auditing practices.

25. INTERPRETATION AND CONCLUSIONS

25.1 Conclusions

The completion of this Report and the FS has confirmed the technical feasibility and economic viability of the Project, based on an open pit mining operation with average gold production at 288,000 ounces per year over a 14.5 year LOM.

The principal conclusions by area are detailed below.

25.1.1 Geology and Mineral Resources

- Understanding of the Project geology and mineralization, together with the deposit type, is sufficiently well established to support Mineral Resource and Mineral Reserve estimation.
- Cut-off grades of 0.30 g Au/t for the in-pit resource and 2.00 g Au/t for the underground resource are appropriate for reporting Mineral Resources for the Project.
- At a cut-off grade of 0.30 g Au/t, the in-pit Indicated Mineral Resources are estimated to be 131.9 Mt grading 1.10 g Au/t for 4.7 Moz of gold. In-pit Inferred Mineral Resources are estimated to be 170 kt grading 0.87 g Au/t for 4.8 koz of gold.
- At a cut-off grade of 2.00 g Au/t, the underground Indicated Mineral Resources are estimated to be 13.7 Mt grading 3.91 g Au/t for 1.7 Moz of gold. Underground Inferred Mineral Resources are estimated to be 21.5 Mt grading 3.57 g Au/t for 2.5 Moz of gold.
- Definitions for Mineral Resource categories used in this report are consistent with the CIM definitions and adopted by NI 43-101.

25.1.2 Mining and Mineral Reserves

- The mine design and Mineral Reserve estimate have been completed to a level appropriate for feasibility studies.
- At a cut-off grade of 0.33 g Au/t, Probable Mineral Reserves are estimated to be 141.7 Mt with an average grade of 1.02 g Au/t for 4.65 Moz of gold.
- The LOM plan details 14.5 years of production, with a four month ramp up and commissioning period followed by eighteen (18) months at a throughput rate of 24,000 t/d, increasing to 27,000 t/d for the remainder of the mine life.

- The open pit generates 548.9 Mt of overburden and waste rock (inclusive of historic tailings and underground backfill) for a strip ratio of 3.87:1.
- The Mineral Reserve estimate stated herein is consistent with CIM definitions. The Mineral Reserves are based on Measured and Indicated Mineral Resources, and do not include any Inferred Mineral Resources.

25.1.3 Metallurgical Testing and Mineral Processing

- The process design criteria have been established based on testwork results, Owner and vendor recommendations or requirements and on standard industry practices.
- The processing options for the Project were selected based on the results of this testwork and are well known technologies that are currently used in the mining industry.
- The gold recovery process for the Project consists of a crushing circuit, a grinding circuit (HPGR and ball mill), pre-leach thickening, a leach and CIP circuit, cyanide destruction and tailings disposal, carbon elution and electrowinning, carbon regeneration, and a gold refinery. The process plant is designed to operate at a throughput of 27,000 t/d.
- Overall metallurgical recovery is 90.2%.

25.1.4 Infrastructure

- Existing infrastructure within the footprint of the property limits will need to be relocated or purchased and dismantled. The most significant relocation is that of the TransCanada Highway 11.
- Power availability from the existing grid is deemed insufficient. Construction of a natural gas-fired power plant is planned.

25.1.5 Environmental Considerations

- A draft EIS/EA, which also included a conceptual closure plan, has been completed and submitted to regulatory agencies, Aboriginal groups and the public for review and comment.
- The results of the draft EIS/EA, including implementing the identified mitigation measures, supports the conclusion that the Project will not cause significant adverse environmental effects, including effects from accidents and malfunctions, effects of the environment on the Project and cumulative effects.

- There are no issues identified to date that would materially affect the ability of GGM to extract minerals from the Project; however, agency comments on the draft EIS/EA received to date and potential future conditions of approval could require refinements to Project components or additional mitigation measures to be implemented.
- GGM continues to work with Aboriginal communities to understand potential effects of the Project on traditional land uses and activities and is committed to working towards LTRAs.

25.1.6 Capital and Operating Costs

- The estimate was developed according to AACE International Standards for a Level 3 estimate with a target accuracy of ± 15%.
- The initial CAPEX for Project construction, including processing, mine equipment purchases and pre-production activities, infrastructures and other direct and indirect costs is estimated to be CAD 1,242M. The total initial capital includes a contingency of CAD 131M, which is 11.8% of the total CAPEX. Other costs during the construction period of CAD 5M bring the total initial capital to CAD 1,247M.
- Sustaining capital required during operations for additional equipment purchases, mine equipment capital repairs, mine civil works, TMF dam raises and additional infrastructure relocation is estimated at CAD 257M.
- A salvage value of CAD 38M is estimated for some mining equipment, processing equipment and power plant that will not have been utilized to their useful life.
- The total reclamation and closure cost is estimated to be CAD 54M.
- The average operating cost is CAD 705/oz Au or CAD 20.95 per tonne milled over the life of the mine. The all-in sustaining cost ("AISC") which includes closure, reclamation and sustaining capital costs average CAD 780/oz Au over the mine life.

25.2 Risks and Opportunities

25.2.1 Risks

GGM's risk identification and assessment process is iterative and has been applied throughout the FS phase. Risks are identified in relation to Project objectives and the internal and external context at the time of each assessment, and are summarized into the Hardrock Project risk register. All aspects of the Project (technical, environmental, community, financial, health and safety, etc.) are assessed in order to provide a business or enterprise level perspective.

Risk treatment plans are developed for each risk in order to reduce the risk's probability and/ or impact to an acceptable or practical level. Certain risk mitigation activities were completed as planned during the FS Phase, while other actions are planned for detailed engineering, construction, operations or closure as appropriate. These mitigation plans are incorporated in the project execution plans and where required in the CAPEX and OPEX budgets. A discussion on the key risks as of the Report issue date follows.

25.2.1.1 <u>Tailings Management Facility</u>

Risks identified in relation to the failure of the Tailings Management Facility (TMF) are related to all phases of work including design, permitting, construction and operations. Amec, a recognized engineering firm with extensive tailings design experience, has been engaged to design the TMF and to perform significant geotechnical drilling and hydrogeological field work, which have been completed to support the design basis. The design has been peer reviewed. Various dam construction assessments were made with a more conservative core upstream design chosen. A detailed risk assessment was undertaken by Amec on the preferred alternative to understand the impact of extreme weather resulting in too much or too little water in the TMF.

Extensive consultation is being undertaken to manage the risk of delayed permits received due to regulator concerns with the TMF's proximity to Kenogamisis Lake. A detailed Tailings Facility Construction Management Plan, including a quality assurance/quality control program, will be developed and implemented for construction. A dam raising schedule has been developed to ensure capacity for the mill tailings during operations.

An Independent Tailings Review Board (ITRB) will be established at GGM to provide oversight during the lifecycle of the TMF. The purpose of the ITRB is to review and advise on the design, construction, operation, performance, and closure planning for the TMF. Consultation with stakeholders will be undertaken throughout the Environmental Assessment and permitting process.

25.2.1.2 <u>Relocation of Highway 11</u>

The Project requires a variety of existing infrastructure to be relocated in order to accommodate the proposed pit and plant site. Accordingly, there are risks associated with such proposed relocation. For example, there is a risk associated with delays in obtaining permits or a refusal in granting permits for the highway realignment work as a new section of Highway 11 must be constructed over existing historical tailings. Detailed design and geotechnical investigations for this work have been completed. A mobile water treatment system will be in place during construction to treat any contact water. A consultant familiar with the highway-related technical and regulatory processes in northwestern Ontario has been engaged. As timing of the relocation is critical to the construction schedule, a detailed plan will be put in place to monitor the permitting timelines, construction seasons and tailings stability.

25.2.1.3 Stability of Historical Tailings

Geotechnical investigations and a stability analysis of the MacLeod High Tailings have been completed to assess the risk of overloading the historical tailings. Infrastructure already exists on these historical tailing with no incidents reported. Additional work recommend by the two peer reviewers will be undertaken in subsequent phases of work.

25.2.1.4 Environmental Assessment (EA)

There are several external factors that could contribute to the risk of a delayed EA approval process. Once GGM has submitted the EA, the government can ‘stop the clock’ should information requests (IRs) of a more serious nature be raised. Regulators may also decide that consultation efforts have been insufficient. Capacity constraints within the regulatory agencies may lead to delays, an element which could be exacerbated by the provincial elections scheduled for early 2018. As a mitigation step, GGM, in conjunction with Stantec, submitted a draft EA in February 2016 and comments that were received have been addressed. Significant engagement and consultation is ongoing with the First Nations (14 groups in total) with both the communities and their technical representatives. Work on completing key Traditional Knowledge (TK) studies is ongoing.

25.2.1.5 Water

The Project is surrounded on three sides by lakes and is cross-cut by small streams. There are several risks associated with construction activities and the use, treatment and discharge of water during operations and closure. These risks and associated treatment plans are as follows:

- Groundwater modelling and laboratory testing have been undertaken to understand the risk of unacceptable contaminants such as arsenic seeping from TMF and waste rock storage areas. Design elements include seepage collection ditches, temporary water treatment during construction and collections ponds that allow for water to be recycled to the plant during operations to ensure the required water quality objectives are met. Arsenic loading has been modelled.

- There is a risk of exceeding the forecasted permitting timelines if the Project is subject to Schedule 2 of the MMER, a federal legislative action which is administered by Environment Canada. Extensive discussions with the government are underway to define the applicability of this regulation to the Project.
- Extensive engagement with Department of Fisheries and Oceans Canada and Environment Canada has been undertaken to communicate and received feedback to manage the risk of the Goldfield Creek (GFC) realignment strategy not being accepted. The GFC realignment diversion and offsetting plans are designed to meet both the Fisheries Act and MMER compensation requirements.
- The risks related to water ingress into the open pit is deemed to be manageable as the historical dewatering rate was reasonable and the permeability of the rock is low as determined by geotechnical work. Dewatering is planned for 25m below the mining surface.

25.2.1.6 People and Systems

Managing the risk of the right people not being available at the right time is a critical mitigation measure for a ensuring a smooth execution of the project. An overall project delivery strategy and staffing plan has been established. Engineering and procurement activities will be outsourced. During construction, work will rely on the use of experienced contractors. Efforts are currently underway to define the availability of skills within the local communities as developing a local employee base will be an important mitigation measure for staff attraction and retention risks during operations. Preliminary relocation and compensation policies have been developed and are reflected in the Basis of Estimate. Training has been included in the detailed project and operations plans.

Project management tools and systems and well integrated information technology systems will need to be established to ensure the successful execution of the project. An implementation plan with recommendations for actions and budget for enterprise resource planning (“ERP”) system, cost and procurement controls has been developed.

25.2.1.7 Mineral Resource

Extensive reinterpretation of the block model and careful selection of estimation parameters has decreased the risk of an incorrect estimation methodology selected to low as practical prior to mining. The block model has been peer reviewed.

25.2.1.8 Metallurgy

The timing of hiring critical process plant staff and the overall ramp-up schedule and plan are mitigations to manage the risk of deficiencies in the control system. Work to analyze all in-pit pulps is currently underway to manage the risk of low gold recoveries. Confirmatory leach testing is planned once detailed design engineering has been completed.

25.2.1.9 Mining of Voids

The risk of voids not being properly factored into the mining work is considered to be well managed at this stage. A relatively accurate model of the underground openings is available. Extra ground engineers and equipment are budgeted and the presence of voids has been factored into productivity. A strategy for managing voids during production is in place and includes backfilling, tagging and flagging procedure, CMS surveys, and probe drilling.

25.2.2 Opportunities

There are several opportunities to improve overall Project economics and sustainability.

- Revenue-related potential opportunities:
    - The use of the Hardrock process plant and TMF for the future processing of gold from other GGM Property deposits such as Brookbank, or a potential future Hardrock underground resource to improve the LOM average grade.
    - Extend the LOM by the addition of potential newly defined resources / reserves from the Property and marginal grade Hardrock material stockpiled during the LOM.
    - The use of the Hardrock process plant and TMF to process some portion of the existing surface historic tailings in order to recover gold, generate revenue, and also potentially mitigate environmental liabilities related to sulphides, arsenic and other contaminants.

- Connecting the natural gas power plant to the grid, and selling spare power generation to the grid during times of shutdowns or excess capacity.

- OPEX related potential opportunities:
    - A potential blend of LNG and diesel as a fuel source is possible for the mine haul trucks. Currently, the mine fleet uses 100% diesel.
    - The use of new, commercially available technologies such as automated mine haulage equipment to increase operational efficiencies and reduce OPEX.
    - Use of RC drilling and other studies early in the project to provide a better understanding of reserve continuity resulting in better control of dilution, reducing the amount of waste processed and therefore improving OPEX.
- CAPEX related potential opportunities:
    - Obtain unused or high quality / refurbished used equipment for the process plant.
    - Consideration of site construction labour efficiencies through the use of pre-fabricated or modular structures, equipment packages, and concrete foundations.
    - Consider the use of alternative lower cost sources for materials and equipment for the mine, processing and infrastructures development.
    - Consider the possibility of major equipment vendor / manufacturer financing or leasing arrangements that serve to improve Project economics.

26. RECOMMENDATIONS

26.1 Hardrock Project Recommendations

Given the technical feasibility and positive economic results of the FS, GMS recommends that GGM continue the work necessary to support a decision to fund and develop the Project.

GGM plans the following principal tasks in the next phase of development:

- Completing the financing plan to fund the construction period;
- Continuing stakeholder engagement activities to establish LTRAs;
- Submission and approval of a EIS/EA;
- Securing all required environmental and construction permits; and
- Managing and mitigating key risks and pursuing opportunities to improve project economics.

The cost for this phase of the work are estimated at approximately CAD 12M.

The list of specific recommendations that follow applies to this and successive phases of work. The cost of addressing each of these recommendations have not been individually estimated however are generally considered to be within the scope of Project CAPEX, sustaining capital, closure and OPEX outlined in this Report.

26.1.1 Exploration and Geology:

- Use a second laboratory as an independent review on 5 to 10% of its pulps in future sampling programs;
- Refine the contaminant models of arsenic and sulfur which are used to modulate the expected metallurgical gold recovery.

26.1.2 Open-Pit Mining

- Conduct additional pit slope geotechnical work such as detailed review of variation in structural fabric orientation to identify possible localized sub-domains with stronger controls on achievable bench face angles; and conduct sensitivity analyses on slope saturation and lower effective shear strength. Additional laboratory testing such triaxial testing and intact shear strength of foliation is recommended;

- A runout assessment study using specialized software is recommended to further validate the waste dump set-back criteria.

26.1.3 <u>Water and Environment</u>

- Complete the evaluation of flood protection berms where Project infrastructure is located in proximity to floodlines as a risk mitigation measure;
- Complete additional investigations around the eastern extension of the open pit to evaluate soil and rock permeability and need for mitigation measures to reduce inflows and potential for flooding due to high water levels within Kenogamisis Lake.;
- Consider the options to manage historical tailings that need to be relocated to allow possible future processing as a source of low grade mill feed;
- Manage the potential geotechnical and environmental issues associated with the construction of the Highway 11 deviation over top of historical tailings. Clearly define the divisions of responsibility for highway related engineering, construction, geotechnical engineering, and environmental engineering;
- Continue the implementation of the environmental follow up / monitoring programs described in Section 20 related to air, noise, water, fish, fauna, wildlife and social and the implementation of environmental management plans;
- Advance the design of the drainage and seepage collection systems and ponds to maximize seepage collection, conveyance, and storage potential;
- Refine the water balance to optimize storage requirements within the underground workings, open pit and TMF to equalize flows and discharges to the mine effluent treatment plant;
- Advance geochemical testing and characterization studies and incorporate production and operational management into the Conceptual Waste Rock Management Plan;
- Complete additional geochemical testing of historical tailings to allow better prediction of potential effects to water quality as a result of the relocation and storage of the tailings within the TMF.

26.1.4 Tailings Management Facility

- Conduct supplemental geotechnical investigations and laboratory testing for better definition of strength and consolidation properties of the interbedded silt layers encountered in the subsurface soils near the southwest and southeast dams;
- Conduct deformation modelling of critical dam sections to confirm sufficiently robust protection against core cracking;
- Perform settling and consolidation testing to better understand tailings behavior and density progression to optimize the TMF design as the currently assumed properties are believed to be conservative;
- Conduct further studies of the geochemistry of the ore and tailings to allow optimization of the TMF design, operation, and closure planning.;
- Conduct detailed tailings deposition planning to optimize the dam raising schedule and inner dam construction requirements;
- Conduct detailed water balance modelling to confirm design assumptions and set operating guidelines for the TMF pond. Adequate mill make-up water supply storage will be required before winter;
- Conduct site-specific seismic hazard analysis to determine appropriate earthquake design parameters for the dam design;
- Finalize geotechnical investigations to support construction documents.

26.1.5 Metallurgy and Processing

- Conduct additional metallurgical tests including:
  - Cyanide destruction optimization testwork to confirm reagents and operating conditions. Investigate the possibility of realizing the cyanide destruction and the precipitation of arsenic in two stages;
  - Tests to validate oxygen consumption in the leaching tanks;
  - Abrasion tests to confirm liner and steel ball consumptions in the grinding mills;
  - Consider additional pilot plant tests with a potential HPGR vendor;
  - Additional tests for equipment sizing, as required;

- Testwork to investigate the possibility of thickening the tailings prior to cyanide destruction to increase cyanide recovery.

26.1.6 Power and Other Infrastructure

- Continue to integrate the planning and execution of the infrastructure relocation program and other external infrastructure interfaces, to ensure alignment with the project development schedule and budget, including the Trans-Canada Highway 11 realignment, the relocation of the Hydro One Geraldton Transmission Station and the natural gas distribution pipeline.
- Continue to monitor evolving climate change regulations and evaluate the impact of climate change regulations on processing OPEX related to the consumption of natural gas for power generation, and re-evaluate the heat recovery tradeoffs to consider the cost impact of carbon taxes and/or credit trading and whether any further potential increases in overall project thermal efficiency through the use of heat recovery in the power plant could mitigate the impact of the additional potential costs of carbon emissions regulation.

26.1.7 Project Execution

- Put in place the Project delivery organization as proposed and described in Section 24 of this Report, and implement the associated project controls and management systems for effective project delivery;
- Develop and implement the operations organization required to execute the Project general and administrative and mining preproduction functions;
- Refine and detail hiring plan for project execution team to assure all positions are staffed in a timely manner.

26.2 Brookbank, Key Lake and Viper Recommendations

Consider additional exploration on the surrounding deposits, such as Brookbank underground, as an eventual source of high grade mill feed material when the average grade dips in Year 6 and Years 8 and 9. These potential mines would need to be mined concurrently with the Hardrock Project open pit given the high milling rates.

- Brookbank – 9,000 m drill program, surface stripping and detail mapping, followed by a resource update to include all new information. The cost for this program would be approximately CAD 1.5M.

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