Blackwater Gold Project
British Columbia
NI 43-101 Technical Report on Feasibility Study

Prepared for:
New Gold Inc

Prepared by:

Gary Christie, P.Eng., AMEC
Ignacy (Tony) Lipiec, P.Eng., AMEC
Ronald G. Simpson, P.Geo., Geosim
Jay Horton, P.Eng., Norwest
Bruno Borntraeger, P.Eng., Knight Piésold

Effective Date:
14 January 2014

Project Number:
173304

CERTIFICATE OF QUALIFIED PERSON

I, Gary Joseph Christie, P.Eng., am employed as a Project Manager with AMEC Americas Inc (AMEC).

This certificate applies to the technical report prepared for New Gold Inc (New Gold) titled "Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 14 January, 2014 (the "technical report").

I am a member of the Association of Professional Engineers and Geoscientists of British Columbia (APEGBC). I am a graduate mechanical engineer from the University of British Columbia with a Bachelors of Applied Science that was awarded in 1979.

I have practiced my profession continuously since 1979 and have been involved in industrial minerals and base and precious metals in projects and operations in North and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Blackwater Gold Project from 22–23 May, 2013.

I am responsible for Sections 1.1, 1.2, 1.3, 1.14.6, 1.14.15, 1.14.17 to 1.14.21, 1.15, 1.16, Section 2, Section 3, Section 4.9, Sections 18.1 to 18.2, 18.10 to 18.12, 18.14 to 18.17, Section 19, Sections 21.1.1 to 21.1.7, 21.2.1, 21.2.5, 21.3, Section 22, Section 23, Section 24, Sections 25.1, 25.12, 25.21 to 25.25, Section 26 and Section 27 of the technical report.

I am independent of New Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with New Gold during the preparation of the 2013 feasibility study and the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: 22 January 2014

"Signed and sealed"

Gary Christie, P.Eng.

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-4315
Fax (604) 669-9516

CERTIFICATE OF QUALIFIED PERSON

I, Ignacy (Tony) Lipiec, P.Eng., am employed as a Director, Process Engineering with AMEC Americas Inc (AMEC).

This certificate applies to the technical report prepared for New Gold Inc (New Gold) titled "Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 14 January, 2014 (the "technical report").

I am a Professional Engineer in the province of British Columbia. I graduated from the University of British Columbia with a B.A.Sc. degree in Mining & Mineral Process Engineering, in 1985.

I have practiced my profession for 29 years, and have previously been involved with metallurgical design and process engineering for precious metal, base metal and specialty product projects in North America and South America.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have not visited the Blackwater Gold Project.

I am responsible for Sections 1.14.1, 1.14.5, 1.14.11 to 1.14.16, 1.16, Section 2, Section 3, Sections 4.7 to 4.8, Section 10.7, Section 11.4, Section 13, Section 17, Section 20, Sections 21.1.6 to 21.1.7, 21.2.3 to 21.2.5, Section 24.1, Sections 25.7, 25.11, 25.17 to 25.22, 25.24, Section 26 and Section 27 of the technical report.

I am independent of New Gold as independence is described by Section 1.5 of NI 43–101.

I have previously been a co-author on the following technical report on the Blackwater Gold Project:

- Simpson, R.G., Welhener, H.E., Borntraeger, B., Lipiec T., and Mendoza, R., 2012: Blackwater Project British Columbia, Canada NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared for New Gold Inc. by GeoSim Services Inc, Independent Mining Consultants Inc, Knight Piésold Ltd. and AMEC, effective date 28 August, 2012.

I have also been involved with New Gold during the preparation of the 2013 feasibility study and the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: 22 January 2014

"Signed and sealed"

Tony Lipiec, P.Eng.

AMEC Americas Limited
111 Dunsmuir Street, Suite 400
Vancouver, B.C. V6B 5W3
Tel (604) 664-4315
Fax (604) 669-9516

CERTIFICATE OF QUALIFIED PERSON

I, Ronald G. Simpson, P.Geo. am employed as a Principal with Geosim Services Inc. (Geosim).

This certificate applies to the technical report prepared for New Gold Inc (New Gold) titled "Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 14 January 2014 (the "technical report").

I am a Professional Geoscientist (19513) with the Association of Professional Engineers and Geoscientists of British Columbia. I graduated with a Bachelor of Science in Geology from the University of British Columbia in 1975.

I have practiced my profession for 38 years. I have been directly involved in mineral exploration, mine geology and resource estimation with practical experience from feasibility studies.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have visited the Blackwater Gold Project on December 13, 2010; September 8, 2011; November 28; 2011, and September 20, 2012.

I am responsible for Sections 1.4, 1.6 to 1.12, 1.13.1, 1.13.2, 1.15, 1.16, Section 2, Section 3, Section 4.1 to Section 4.6.5, Section 6, Section 7, Section 8, Section 9, Sections 10.1 to 10.5, 10.8 to 10.11, Sections 11.1 to 11.3, 11.5 to 11.9, Section 12, Section 14, Sections 25.2 to 25.6, 25.8, Section 26, and Section 27 of the technical report.

I am independent of New Gold as independence is described by Section 1.5 of NI 43–101.

I have previously authored, or been a co-author on the following technical reports on the Blackwater Gold Project:

- Simpson, R., 2011a: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date March 2, 2011, re-addressed June 6, 2011.
- Simpson, R., 2011b: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date September 19, 2011.
- Simpson, R., 2012: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date March 7, 2012.
- Simpson, R.G., Welhener, H.E., Borntraeger, B., Lipiec T., and Mendoza, R., 2012: Blackwater Project British Columbia, Canada NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared for New Gold Inc. by GeoSim Services Inc, Independent Mining Consultants Inc, Knight Piésold Ltd. and AMEC, effective date 28 August, 2012.

I have also been involved with New Gold during the preparation of the 2013 feasibility study and the preparation of this technical report.

Geosim Services Inc.
1975 Stephens St.
Vancouver, BC Canada V6K 4M7
Phone: (604) 809-7470

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: 22 January 2014

"Signed and sealed"

Ronald G. Simpson, P.Geo.

Geosim Services Inc.
1975 Stephens St.
Vancouver, BC Canada V6K 4M7
Phone: (604) 809-7470

NORWEST
CORPORATION

CERTIFICATE OF QUALIFIED PERSON

I, Jay Horton, P.Eng., am employed as the Manager, Technical Services with Norwest Corporation (Norwest).

This certificate applies to the technical report prepared for New Gold Inc (New Gold) titled "Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 14 January, 2014 (the "technical report").

I am a Professional Engineer of the Association of Professional Engineers and Geoscientists of British Columbia (Member #29093). I graduated with a Bachelor of Applied Science degree from the University of British Columbia in 1999.

I have practiced my profession for 15 years. I have been directly involved in both surface and underground mine design over a range of commodities. I have worked on gold mining projects for 11years.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I have visited the Blackwater Gold Project on 22 May, 2013.

I am responsible for Section 15, Sections 16.3 to 16.7, Section 21.2.2 and those portions of Sections 1, 24.1, 25, 26, and 27 that pertain to those Sections as well as portions of Sections 2, 3, 21.1.6, 21.1.7, and 21.2.5 of the technical report.

I am independent of New Gold as independence is described by Section 1.5 of NI 43–101.

I have been involved with New Gold during the preparation of the 2013 feasibility study and the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

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

Dated: 22 January 2014

"Signed and sealed"

Jay Horton, P.Eng.

Jay Horton, P.Eng.
Suite 1830, 1066 W. Hastings Street
Vancouver, BC V6E 3X2
Telephone (604) 602-8992

Knight Piésold CONSULTING

CERTIFICATE OF QUALIFIED PERSON

I, Bruno Borntraeger, P.Eng. am employed as a Specialist Engineer/Project Manager with Knight Piésold Ltd (Knight Piésold).

This certificate applies to the technical report prepared for New Gold Inc (New Gold) titled "Blackwater Gold Project, British Columbia, NI 43-101 Technical Report on Feasibility Study" that has an effective date of 14 January 2014 (the "technical report").

I am a Professional Engineer registered in British Columbia. I graduated with a Bachelor of Applied Science in Geological Engineering from the University of British Columbia in 1990.

I have practiced my profession for 24 years since graduation. I have been directly involved in geotechnical aspects of the project and the waste and water management engineering studies.

As a result of my experience and qualifications, I am a Qualified Person as defined in National Instrument 43–101 *Standards of Disclosure for Mineral Projects* (NI 43–101).

I visited the Blackwater Gold Project on September 28, 2011, December 20, 2011, February 21, 2012 and April 24, 2012 to view proposed locations of site facilities, potential construction material borrow areas and review geotechnical site investigation progress.

I am responsible for Sections 1.5, 1.14.2, 1.14.3, 1.14.7 to 1.14.10, 1.14.15, 1.16, Section 2, Section 3, Section 5, Section 10.6, Sections 16.1 to 16.2, Sections 18.3 to 18.9.5, 18.13, Sections 21.1.6, 21.1.7, 21.2.5, Sections 24.1.1, 24.1.2, Sections 25.10.1, 25.10.2, 25.13 to 25.16, 25.21, 25.22, 25.24, Section 26, and Section 27 of the technical report.

I am independent of New Gold as independence is described by Section 1.5 of NI 43–101.

I have previously been a co-author on the following technical report on the Blackwater Gold Project:

- Simpson, R.G., Welhener, H.E., Borntraeger, B., Lipiec T., and Mendoza, R., 2012: Blackwater Project British Columbia, Canada NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared for New Gold Inc. by GeoSim Services Inc, Independent Mining Consultants Inc, Knight Piésold Ltd. and AMEC, effective date 28 August, 2012.

I have also been involved with New Gold during the preparation of the 2013 feasibility study and the preparation of this technical report.

I have read NI 43–101 and the sections of the technical report for which I am responsible have been prepared in compliance with that Instrument.

Bruno Borntraeger
Suite 1400, 750 West Pender St.
Vancouver, BC, V6C 2T8
Phone 604.685.0543
Fax 604.685.0147

Knight Piésold CONSULTING

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

Dated: January 22, 2014

"Signed and sealed"

Bruno Borntraeger, P.Eng.

Bruno Borntraeger
Suite 1400, 750 West Pender St.
Vancouver, BC, V6C 2T8
Phone 604.685.0543
Fax 604.685.0147

IMPORTANT NOTICE

This report was prepared as National Instrument 43-101 Technical Report for New Gold Inc (New Gold) by AMEC Americas Ltd (AMEC), Geosim Services Inc. (Geosim), Norwest Corporation (Norwest), and Knight Piésold Ltd (Knight Piésold), collectively the “Report Authors”. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors’ services, based on i) information available at the time of preparation, ii) data supplied by outside sources, and iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by New Gold subject to terms and conditions of its respective contracts with the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party is at that party’s sole risk.

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1.0 SUMMARY

1.1 Introduction

AMEC Americas Ltd. (AMEC), Geosim Services Inc. (Geosim), Norwest Corporation (Norwest), and Knight Piésold Ltd (Knight Piésold) prepared a technical report (the Report) for New Gold Inc (New Gold) on the Blackwater gold project (the Project), located in British Columbia, Canada.

The Report was prepared to support a Feasibility Study (the 2013 Feasibility Study) on the Project as disclosed in New Gold's press release entitled "New Gold Announces Blackwater Feasibility Study Results" dated 12 December 2013.

All currencies are expressed in Canadian dollars (CAD$) unless otherwise stated. Years expressed in this summary are for illustrative purposes only, as the decision to implement production is at the discretion of New Gold, and permits to support operation still have to be obtained. For the purposes of the report, two terms are used for the mine production: life-of-mine (LOM) refers to the life of mine including the pre-production period; the operational period refers to the mine life excluding the pre-production duration.

1.2 Key Outcomes

The key findings of the 2013 Feasibility Study are:

- Measured and Indicated Mineral Resources amenable to direct processing reported at a gold equivalent cut-off grade of 0.4 g/t AuEq total 306.0 Mt grading 0.88 g/t Au and 5.8 g/t Ag
- Measured and Indicated Mineral Resources that are amenable to being stockpiled for future treatment, reported at a gold equivalent cut-off grade of 0.3 g/t AuEq, total 90.9 Mt grading 0.3 g/t Au and 4.3 g/t Ag
- Inferred Mineral Resource amenable to direct processing, reported at a gold equivalent cut-off grade of 0.4 g/t AuEq, total 13.8 Mt grading 0.76 g/t Au and 4.1 g/t Ag
- Inferred Mineral Resources that are amenable to being stockpiled for future treatment, reported at a gold equivalent cut-off grade of 0.3 g/t AuEq, total 3.8 Mt grading 0.31 g/t Au and 3.6 g/t Ag
- Proven and Probable Mineral Reserves total 344.4 Mt grading 0.74 g/t Au and 5.5 g/t Ag

- The process plant will have a throughput of 60,000 t/a. The plant will use a conventional whole ore leach flowsheet. Over the LOM (including pre-production), 86.6% of the gold and 49.1% of the silver will be recovered into a gold–silver doré product
- The estimated operating mine life of 16 years is based on a conventional truck-and-shovel operation. Open pit mining will be undertaken for the first 14 years, and stockpiles will provide feed thereafter. The strip ratio over the life-of-mine, including pre-production, averages 2.00:1 (waste tonnes to ore tonnes)
- Over the first nine years of operation, the average gold feed grade will be 0.85 g/t Au and average annual gold production will be 485,000 oz per year
- Total life-of-mine (LOM) metal production including lower-grade stockpiles will be 7.0 million oz of gold and 29.6 million oz. of silver
- The capital cost estimate is $1,963 million (US$1,865 million), including an overall Project contingency of $200.4 million (US$190 million)
- The LOM operating costs are estimated at $14.49/t of ore milled, and $12.48/t of ore milled after accounting for silver credits. Total LOM all-in sustaining cash costs are estimated at $14.47/t of ore milled
- The net present value (NPV) base case at 5% discount rate, from Yr -2, (2015) is estimated at $1,044 million (US$991 million). The after-tax net present value at 5% discount rate is estimated at $616 million (US$585 million) at an exchange rate of 0.95 (US$/CAD$)
- The internal rate of return is 11.3% before tax and 9.3% after tax
- Payback is estimated at 6.2 years before tax and 6.4 years after tax
- The average LOM cash costs, net of royalties and silver credits, is US$670/oz
- Gold will account for 93% of the revenue and silver will account for 7%.

1.3 Cautionary Notes

1.3.1 Caution Regarding Forward-Looking Information

Section 1.3 of this Report applies to forward-looking statements in this section and throughout the Report.

Certain information and statements contained in this report are “forward looking” in nature. Forward-looking statements include, but are not limited to, statements with respect to the economic and feasibility parameters of the Blackwater Project: the cost and timing of the development of the Project; the proposed mine plan and mining method, stripping ratio, processing method and rates and production rates; grades; projected metallurgical recovery rates; infrastructure, capital, operating and sustaining costs; the projected life of mine and other expected attributes of the Blackwater Project; the net present value (NPV) and internal rate of return (IRR) and payback period of capital; cash costs and all-in sustaining costs; the success and continuation of exploration activities; estimates of Mineral Reserves and Mineral Resources; the future price of gold; the timing of the environmental assessment process; government regulations and permitting timelines; estimates of reclamation obligations that may be assumed; requirements for additional capital; environmental risks; and general business and economic conditions.

All forward-looking statements in this report are necessarily based on opinions and estimates made as of the date such statements are made and are subject to important risk factors and uncertainties, many of which cannot be controlled or predicted. Material assumptions regarding forward-looking statements are discussed in this report, where applicable. In addition to, and subject to, such specific assumptions discussed in more detail elsewhere in this Report, the forward-looking statements in this Report are subject to the following assumptions:

1. There being no signification disruptions affecting the development and operation of the Project

2. The exchange rate between the Canadian dollar and U.S. dollar being approximately consistent with the assumptions in the 2013 Feasibility Study

3. The availability of certain consumables and services and the prices for diesel, natural gas, fuel oil, electricity and other key supplies being approximately consistent with assumptions in the 2013 Feasibility Study

4. Labour and materials costs being approximately consistent with assumptions in the 2013 Feasibility Study

5. Permitting and arrangements with First Nations and other Aboriginal groups being consistent with current expectations

6. That all environmental approvals, required permits, licenses and authorizations will be obtained from the relevant governments and other relevant stakeholders within the expected timelines

7. Certain tax rates, including the allocation of certain tax attributes to the Project

8. The availability of financing for New Gold's development activities

9. The timelines for exploration and development activities on the Project

10. Assumptions made in Mineral Resource and Mineral Reserve estimates, including geological interpretation, grade, recovery rates, gold price assumption, and operational costs; and general business and economic conditions.

Forward-looking statements involve known and unknown risks, uncertainties and other factors which may cause the actual results, performance or achievements to be materially different from any of the future results, performance or achievements expressed or implied by forward-looking statements. These risks, uncertainties and other factors include, but are not limited to, the assumptions underlying the 2013 Feasibility Study and economic parameters discussed herein not being realized; decrease of future gold prices; cost of labour, supplies, fuel and equipment rising; actual results of current exploration; adverse changes in Project parameters; discrepancies between actual and estimated production, Mineral Reserves, Mineral Resources and recoveries; exchange rate fluctuations; delays and costs inherent in consulting and accommodating rights of First Nations and other Aboriginal groups; title risks; regulatory risks and political or economic developments in Canada; changes to tax rates; risks and uncertainties with respect to obtaining necessary surface rights and permits or delays in obtaining same; risks associated with maintaining and renewing permits and complying with permitting requirements; and other risks involved in the gold exploration and development industry; as well as those risk factors discussed elsewhere in this Report.

New Gold and the Qualified Persons who authored of this report undertake no obligation to update publicly or otherwise revise any forward-looking statements whether as a result of new information or future events or otherwise, except as may be required by law.

1.3.2 Cautionary Note to U.S. Readers Concerning Estimates of Mineral Reserves and Mineral Resources

Information concerning the Blackwater Property has been prepared in accordance with Canadian standards under applicable Canadian securities laws, and may not be comparable to similar information for United States companies. The terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” used in this Report are Canadian mining terms as defined in the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Definition Standards for Mineral Resources and Mineral Reserves adopted by CIM Council on November 27, 2010 and incorporated by reference in National Instrument 43-101 (“NI 43-101”). While the terms “Mineral Resource”, “Measured Mineral Resource”, “Indicated Mineral Resource” and “Inferred Mineral Resource” are recognized and required by Canadian securities regulations, they are not defined terms under standards of the United States Securities and Exchange Commission. As such, certain information contained in this Report concerning descriptions of mineralization and resources under Canadian standards is not comparable to similar information made public by United States companies subject to the reporting and disclosure requirements of the United States Securities and Exchange Commission.

An “Inferred Mineral Resource” has a greater amount of uncertainty as to its existence and as to its economic and legal feasibility. It cannot be assumed that all or any part of an “Inferred Mineral Resource” will ever be upgraded to a higher confidence category. Readers are cautioned not to assume that all or any part of an “Inferred Mineral Resource” exists or is economically or legally mineable.

Under United States standards, mineralization may not be classified as a “Reserve” unless the determination has been made that the mineralization could be economically and legally produced or extracted at the time the Reserve estimation is made. Readers are cautioned not to assume that all or any part of the Measured or Indicated Mineral Resources that are not Mineral Reserves will ever be converted into Mineral Reserves. In addition, the definitions of “Proven Mineral Reserves” and “Probable Mineral Reserves” under CIM standards differ in certain respects from the standards of the United States Securities and Exchange Commission.

1.4 Project Description and Location

1.4.1 Location

The Blackwater Project is located in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver. The Project site is readily accessible by forest service and mine roads. Driving time from Vanderhoof to the property is about 2.5 hours. Helicopter access is available from bases in Vanderhoof, Quesnel, or Prince George.

1.4.2 Mineral Tenure

New Gold holds a 100% recorded interest in 227 mineral claims covering an area of 104,678 ha distributed among the Blackwater, Capoose, Auro, and Key claim blocks. The Blackwater claim block comprises 75 mineral cell claims totalling 30,578 ha. All Blackwater claims are 100% held in the name of New Gold. Sixty-nine claims expire in 2022. Two claims expire in October 2014 and four claims expire in January 2015. There are no other parties with beneficial interests in these mineral rights. None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves. The Blackwater deposit spans the Davidson claim (509273), the Dave claim (515809) and the Jarrit claim (515810).

1.4.3 Surface Rights

A review of surface rights in the vicinity of the Blackwater claim block was undertaken in December, 2013 and January 2014. The majority of the Blackwater mineral claims are located on Crown lands. The review identified an overlapping private parcel, land reserves/notations, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitters, and an ungulate winter range.

A review of surface rights in the vicinity of proposed electrical transmission lines, water pipeline, airstrip and access roads (Linear Infrastructure) was undertaken in December, 2013. This review identified private parcels, a Land Act licence rights of way, land reserves/notations, a transfer of administration/control area, grazing tenures, forest tenures, forest recreation sites, traplines, guide outfitter areas, a wildlife management area, an agriculture land reserve, and third-party mineral tenures overlapping or in close proximity to proposed linear infrastructure routes.

1.4.4 Royalties and Encumbrances

New Gold's 100% interest in the Blackwater claim block is subject to four net smelter return (NSR) agreements:

- Mineral claim 515809 (Dave Option): The optionors retained a 2.5% NSR. New Gold has purchased 40% of the Dave NSR royalty for $1,000,000, and a 1.5% NSR royalty remains. The claim covers a portion of the Blackwater deposit.
- Mineral claim 515810 (Jarrit Option): The optionors retained a 2% NSR royalty. New Gold has purchased half of the Jarrit NSR royalty for $1,200,000, and a 1% NSR royalty remains. The claim covers a portion of the Blackwater deposit.
- Mineral claims 637203, 637205, and 637206 (JR Option): The optionors retained a 3% NSR royalty. New Gold may purchase two-thirds of the JR Claims NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain.
- Mineral claim 835014 (PS Option): The optionor retains a 2% NSR royalty, of which New Gold may purchase half for $1,000,000.

No other material encumbrances that are recorded against the Blackwater claims and are still active have been identified.

1.5 Accessibility, Climate, Local Resources, Infrastructure and Physiography

1.5.1 Accessibility

The Blackwater site will be accessed via the Kluskus Forest Service Road (FSR). New Gold will undertake road improvements over a small section of the FSR. New Gold will likely become the primary operator and user of the FSR by the time the Project is complete, considering that reduced logging operations are anticipated in the area at that time, and will be responsible for primary maintenance. New Gold will upgrade part of the FSR to meet the future year-round operational needs of the Blackwater Project.

A new 16 km long mine access road will replace the existing exploration access road to the site. Some sections of the water supply pipeline, the fibre-optic cable and the power transmission line will parallel this road. The road will be used for heavy traffic during mine construction and has been designed for year-round all-weather access.

An airstrip will be built for use during the construction phase of the Project to increase accessibility and reduce travel time to the Project site. The airstrip will be sized for Dash 8 class aircraft.

1.5.2 Climate

The climate in the Project area is sub-continental. The climate is also influenced by moisture-laden weather systems moving east by way of the low Kitimat Ranges. Temperatures range from a minimum of -40°C in winter to a maximum of 32°C in summer. The mean annual precipitation for the site is estimated to be 636 mm with 49% falling as rain and 51% as snow. Mining activities are expected to be possible on a year-round basis.

1.5.3 Local Resources

The Project area is very sparsely inhabited; three ranches are found within a 20 km radius of the Project site. Some services are available in Vanderhoof, but Prince George is the regional hub with air service from major centres.

There is no grid-connected power in the direct vicinity of the Project. The main BC Hydro 500 kV transmission lines supplying western B.C. are approximately 100 km to the north. Several interconnection points from the 500 kV lines to existing 230 kV substations and transmission lines are possible in an area between Fraser Lake and Vanderhoof. Power for the current Blackwater exploration camp is provided by generators. The deposit is located on the north slope of Mt. Davidson, and the proposed Project infrastructure including the mill, waste and tailings storage will be sited predominantly in the Davidson Creek watershed. Precipitation run-off, ground water from pit dewatering, and supplemental fresh water from a nearby lake are water sources for mineral processing. A ground water well field will supply fresh water for the camp.

1.5.4 Physiography

The elevation of the Blackwater property ranges from just over 1,000 m in low-lying areas northeast of the proposed mine site to 1,800 m at the summit of Mt. Davidson on the southwest side of the property. Outcrop on the property is limited, and most of the area is covered with thick glacial deposits of 2 m or more.

The Nazko Upland subregion is the primary biogeoclimatic region. Low-elevation valley bottoms are dominated by stands of lodgepole pine. Hybrid white spruce tends to dominate on moist to wet sites below 1,500 m, while subalpine fir and Englemann spruce are dominant above 1,500 m. The recent pine beetle epidemic has infested almost all of the lodgepole pine forests within this subregion. The Nazko Upland subregion also contains an extensive network of lakes, rivers, and wetland complexes. Atmospheric heating of these water bodies can result in convective activity and sporadic summer showers.

1.5.5 Seismicity

A design earthquake magnitude 8.5 was selected for earthquake return periods of 500, 5,000, and 10,000 years, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation of the probabilistic seismic hazard. This represents large magnitude earthquakes along the Queen Charlotte fault system and Cascadia subduction zone. The potential for shallow crustal earthquakes closer to the Project site was also considered for longer return period events of 5,000 and 10,000 years, representing earthquakes of up to about magnitude 7.5 along Coastal B.C.

1.6 History

Limited exploration activity, on what is now the Project site, was first recorded in 1973. Granges Inc. completed geophysical and geochemical surveys and limited drilling between 1973 and 1994. Following some further drilling from 2005 to 2007, the Project was acquired by Richfield Ventures Corp. (Richfield) in early 2009. During the second half of 2009, throughout 2010 and the first five months of 2011, Richfield continued its exploration drilling program at Blackwater.

New Gold purchased Richfield in May 2011 and thereby acquired a 75% interest in the Davidson claims and 100% interests in each of the Dave and Jarrit claims.

Subsequently, New Gold acquired Geo Minerals Ltd. and Silver Quest Resources Ltd., which resulted in full control of the Project.

New Gold continued with a major exploration drilling, metallurgical testwork, and feasibility-level engineering program upon which this Report is based. In parallel with engineering and exploration activities New Gold has advanced efforts to secure the necessary approvals, permits, and agreements to begin construction and operations.

No production has occurred from the Project area.

1.6.1 Geological Setting

The Blackwater deposit is considered to be an example of an intermediate sulphidation epithermal-style gold–silver deposit.

Given the lack of outcrop, geological interpretation has been based primarily on drill information plotted on section and plan views.

Mineralization at Blackwater is hosted within felsic to intermediate composition volcanic rocks that have undergone extensive silicification and hydrofracturing in association with pervasive stockwork veined and disseminated sulphide mineralization.

Mineralization is strongly controlled by northwest–southeast-trending structures characterized by zones of tectonic brecciation and chloritic gouge. Northeast-trending structural discontinuities also appear to have a major control on alteration and mineralization, but do not appear to be affected by recent movement. A major north–south-trending fault dissects the orebody and east–northeast-trending faults along UTM easting 375,600E. This fault represents a well-defined disruption in lithology, alteration, and mineralization patterns and was used to subdivide the resource block model into two structural domains, one to the east of it and one to the west.

The alteration minerals most commonly identified included muscovite, high- and low temperature illite, ammonium bearing illite, smectite, silica, biotite, and chlorite.

Gold–silver mineralization is associated with a variable assemblage of pyrite–sphalerite–marcasite–pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite).

In the opinion of the QPs, knowledge of the deposit setting, lithologies, structural and alteration controls on mineralization are sufficient to support Mineral Resource estimation.

1.7 Exploration

Given the lack of bedrock exposures in the immediate Blackwater deposit area, geologic information has been obtained primarily by exploration drilling. New Gold mapping of pits and road-cut exposures over the deposit has recently supported the geological interpretation of the deposit in the subsurface.

New Gold carried out soil and stream geochemical surveys over parts of the Blackwater Project area between late May and mid-September 2012. A total of 4,517 samples were collected. The results of the soil survey indicated numerous areas displaying multi-element anomalies including gold, zinc, silver, copper, bismuth, and molybdenum, many of which merit follow-up investigation. Results of a restricted stream silt sampling program of 43 samples indicated anomalous copper and zinc values from streams to the northwest and southeast of the Blackwater deposit.

During 2010, Richfield contracted Quantec Geoscience Ltd. of Toronto to conduct a Titan 24 DC resistivity and IP chargeability geophysical survey. The results of the survey indicate good correspondence between known mineralization and the Titan IP-resistivity results. In general, zones of significant gold mineralization correlate positively to zones of moderate resistivity and moderate IP chargeability.

Polished section petrographic analysis, X-ray diffraction analysis and whole-rock lithogeochemical analyses have been conducted on selected drill samples. A two-phase alteration study was also completed to develop the alteration model for the deposit.

1.8 Mineralization

Disseminated Au–Ag mineralization is defined by an east–west-trending tabular–conical-shaped deposit with a lateral extent of up to 1,300 m east–west x 950 m north–south. Mineralization remains open at depth in the southwestern part of the deposit as well as to the north and northwest. The centre of the deposit has an average thickness of 350 m and, where open, a vertical extension of up to 600 m. The mineralized zone plunges shallowly to the north and northwest with inferred steep, north-plunging higher-grade mineralized shoots, measuring tens of metres thick, likely influenced by near-vertical structural intersections.

1.9 Drilling

A total of 1,149 core drill holes (357,507 m) have been drilled in the Project area between 2009 and January 2013. Of this total, 134 were completed by Richfield, and 1,015 by New Gold. Drilling by parties other than Richfield and New Gold is referred to as legacy drilling; this drilling, completed between 1981 and the end of 2006 consists of 81 holes totalling 7,633 m. The legacy drilling is not used in resource estimation.

The exploration drilling carried out since 2009 has been predominantly HQ diameter (63.5 mm) diamond drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths. Drilling for metallurgical has used PQ diameter (85 mm) core. Some of the condemnation drilling was undertaken using reverse circulation (RC) methods.

Geological logging includes geotechnical, magnetic susceptibility, and specific gravity measurements taken at regular intervals. Lithology is logged and the core prepared for systematic sampling at regular 1 m intervals. The lithological nomenclature at the Project has undergone revision on two occasions since New Gold took control of the Project in June of 2011, and currently six principal rock lithology types are defined. Magnetic susceptibility and conductivity data were measured at 10 cm increments along the core with a hand-held conductivity and magnetic susceptibility meter. Recovery and rock quality designation (RQD) data were measured and recorded in LogChief™.

Core recovery for the 2009, 2010, 2011, and 2012 drilling programs averaged 92%, and the median core recovery was 96%.

Planned drill hole collar locations were measured in the field using hand held global positioning system (GPS) instruments. Locations were subsequently confirmed by Trimble differential GPS. Of the 1,040 holes, 1,025 were then professionally surveyed by All North Consulting using a Real Time Kinematic (RTK) technique to enhance the precision of the location data. Elevations for the drill collars were determined by draping collar coordinates over the topography measured by an aerial light detection and ranging (LiDAR) survey.

Down-hole surveys are performed using Reflex survey equipment, and dip angle and azimuth are recorded. A +18.8° magnetic declination correction factor is applied to the magnetic azimuth record. Data are entered into LogChief™ in tables designed specifically for the Project.

Thirteen specific geotechnical HQ holes were drilled; in addition, 10 hydrological pilot holes (also at HQ size) were drilled to serve as monitoring stations, where a piezometer is installed to measure the level of the aquifer in the deposit area. Twenty-seven specific metallurgical holes were drilled, four of which were HQ in size; the remaining 23 holes were drilled at PQ. Fourteen waste rock characterisation holes (HQ) were drilled, and 91 RC holes and 18 core holes comprised the condemnation drill program.

Gold and silver mineralization occurs within an irregularly-shaped system of stockwork and disseminated sulphides that strikes approximately east–west and dips moderately to the north. Depending on the inclination of an individual drill hole, and the local dip of mineralization, drill intercept widths are approximately equivalent to true widths. Drill sampling is representative of the gold–silver grades in the deposits.

In the QP's opinion the quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the exploration and infill drill programs from 2009 to 2013 are sufficient to support Mineral Resource estimation. There are no known sampling or recovery factors with these programs that could materially impact the accuracy and reliability of the results.

1.10 Sampling and Analysis

New Gold and predecessor company personnel conducted the drill core handling and sampling on the property. Drill core was transported in core boxes from the drill site to camp by four-wheel drive vehicle and logged in a specially built core handling facility. Samples were taken systematically on 1 m long sawn half-core sample intervals, then tagged and bagged. Four sample bags were placed into a larger rice bag labelled with the sample numbers and sealed with a numbered banker's security tag. Between preparation and shipment, a period of up to four days, the rice bags containing the samples were stored in a secure area behind the core cutting area. The remaining half cores were archived in core sheds on the property. New Gold personnel drove trucks containing the samples to Prince George, and from there the samples were delivered to the laboratories by bonded couriers.

Certified reference standards (CRMs), blanks, and duplicates are inserted into the sample stream, and four sample bags are then placed into a larger rice bag labelled with the sample identification (ID) numbers and sealed with a numbered banker's security tag. The drill hole database is supported by about 80,000 QA/QC check assays.

Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops and ALS Mineral Laboratories (ALS) in Vancouver, Vanderhoof, Terrace, Reno, and Elko were used for sample preparation. Eco Tech in Kamloops and ALS in North Vancouver were used as the primary assay laboratories. Both laboratories were accredited and are independent of New Gold.

Drill core samples were prepared using standard crush, split, and pulverise sample preparation procedures. Pulverized samples were analysed for gold by fire assay atomic absorption spectrometry (FA ASS). Preparation and FA AA procedures varied between the laboratories but were generally similar.

The Eco Tech samples were initially assayed for silver by aqua-regia digestion (AR) and AAS finish, and later by AR and induction-coupled plasma spectrometry atomic emission spectrometry (ICP AES) finish. The ALS samples were analyzed for silver by four acid digestion ICP AES finish until July 2012, after which time silver was analyzed by a four acid digestion AAS. Eco Tech overlimit results (>30 g/t Ag) were reassayed by AR/AAS method. ALS overlimit results (>100 g/t) were reassayed by a four acid digestion with AAS finish with a higher detection limit.

Assay procedures also include a multi-element package (28 elements at Eco Tech, 33 elements at ALS) by AR digestion and ICP AES finish. Overlimit analysis was completed on samples returning greater than 1% Cu, Pb, or Zn.

Metallurgical samples were selected from the designated metallurgical holes, and samples from numerous resource holes across the deposit. The samples were collected and despatched from site to laboratories under the supervision of the Exploration Manager. Sample security protocols used were the same as the exploration sample protocols.

Specify gravity measurements were made in the field for more than 32,000 samples using a water immersion method without a wax coating. ALS verified the field measurements by analyzing 154 samples using a water immersion method without a wax coating and 55 samples using a wax-coat water immersion method. The results showed no bias between the field and laboratory methods for all but overburden samples.

The current drill hole and assay database for the project is administered from the New Gold Vancouver office using the same Maxwell GeoServices software products, LogChief™ and DataShed™, which were used during the Richfield drilling programs.

In the QP's opinion the sample preparation, security, and analytical procedures used by New Gold for the Blackwater Project have ensured the validity and integrity of samples taken. Quality control procedures implemented in 2012 for silver analysis shows acceptable levels of precision and accuracy for silver results. Previous concerns regarding the accuracy and precision of pre-2012 silver results due to lack of comprehensive silver QC is mitigated by the 2012 QC results. Data from holes drilled between 1981 and 1994 have no documented QA/QC information, and they are not deemed acceptable for use in resource estimation.

1.11 Data Verification

Data verification programs have been completed by Mr. Ronald G. Simpson, Principal of GeoSim, and AMEC. The QP has reviewed the New Gold and AMEC reports and independently audited the sample database for interval errors and missing sample intervals. Four site inspections have been carried out since 2010. The QP concluded that the QA/QC with respect to the results received for the 2009, 2010, 2011, and 2012 exploration programs is acceptable, the protocols have been well documented and that the drill hole database is adequate to support the geological interpretations and Mineral Resources estimated in this report.

1.12 Security of Samples

Samples are transported to Prince George by truck, where the driver waits with the samples in the truck until pick-up for onward shipment by a bonded courier. Before July 2011, the Richfield samples, including the standards, blanks, and duplicates, were shipped to Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops, B.C. Since the acquisition of Richfield by New Gold in June 2011, and the subsequent acquisition of Eco Tech by the ALS Group in July 2011, samples have been shipped to ALS Minerals (ALS) in North Vancouver, B.C.

Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility. Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory. Current sample storage procedures and storage areas are consistent with industry standards.

1.13 Mineral Reserve and Mineral Resource Estimates

1.13.1 Mineral Resource Estimate

The Mineral Resource Estimate is based upon a geologic block model that incorporates 286,966 individual assays from 309,516 m of core from 1,003 drill holes at a nominal drill hole spacing ranging from 25 m to 50 m. Assay data density is sufficient to classify the mineral resource at the Measured and Indicated confidence levels as necessary to support the estimation of a Mineral Reserve. The drill hole database was supported by about 80,000 quality assurance/quality control (QA/QC) check assays. Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for geologic modelling, statistical analysis, or grade estimation.

Block models were created in Geovia-Surpac Vision© software using a block size with dimensions of 12 m x 12 m x 12 m. Models included geological, structural, alteration, sulphide, base of weathering, and pyrite content (shell). A grade shell domain was generated for gold within the mineralized pyrite shell to better delineate areas of very low grade material. A second grade shell domain was created to constrain a high-silver/low-gold zone in the upper north area of the deposit that was mostly outside the pyrite shell.

A gold grade capping threshold of 45 g/t Au was selected within the gold grade shell. Gold assays outside of the gold grade shell were capped at 5 g/t Au. Silver assays falling within the pyrite shell were capped using a 150 g/t Ag threshold. Silver assays falling within the silver grade shell were not capped. Gold assays falling within the silver grade shell were capped at 1 g/t Au. The amounts of metal removed are considered to be reasonable for a project at the feasibility study stage of development.

Gold and silver grades within the corresponding zone domains were estimated in three passes within the pyrite shell using ordinary kriging (OK). Grades within the north silver zone were estimated in two passes using OK. A single-pass, nearest-neighbour (NN) estimate using 12 m composites was also carried out for use in model validation.

Blocks were assigned preliminary resource classifications based on drill hole spacing. Blocks falling within the 25 m x 25 m drill hole spacing pattern were assigned a tentative 'Measured' classification. Blocks not meeting these conditions were classified as 'Indicated' if they were within the area drilled with a 50 m x 50 m spacing. All other estimated blocks were assigned to the 'Inferred' category. For Measured Mineral Resources, blocks falling outside of a 0.2 g/t gold equivalent grade shell were downgraded to Indicated as continuity of economic grades outside of the grade shell cannot be demonstrated. Blocks falling within the high-grade silver grade shell were classified to the Indicated and Inferred categories using a nominal drill hole spacing of 50 m.

To assess reasonable prospects for eventual economic extraction a Lerchs–Grossmann (LG) optimized pit was used to constrain reporting of Mineral Resources. These economic assumptions are almost identical to the economic assumptions used for the Mineral Reserve pit optimization with the notable exception of metal prices, which are higher for the Mineral Resource estimate.

Gold equivalent (AuEq) values were calculated using variable metallurgical recoveries depending on the mineralization type. Mineralization types were assigned to oxide, transition or sulphide depending on the oxidation state. New Gold selected a 0.3 g/t AuEq cut-off grade to report the Mineral Resource estimate for stockpile material and a 0.4 g/t AuEq cut-off for estimating Mineral Resources considered amenable to direct processing. The marginal cut-off grade is likely to be slightly lower than 0.3 g/t AuEq for the transition and sulphide ore types.

1.13.2 Mineral Resource Statement

The Qualified Person for the estimate is Mr Ronald G. Simpson of Geosim. Mineral Resources have an effective date of 31 March, 2013. Mineral Resources are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources in Table 1-1 are reported inclusive of Mineral Reserves and do not include dilution. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

Table 1-1: Mineral Resource Tabulation (Effective date March 31, 2013 Ronald G. Simpson P.Geo)

Resource Category	Tonnes & Grade: Tonnes (kt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	Contained Metal: Gold (Moz)	Silver (Moz)
Measured & Indicated Resources						
Direct processing material						
Measured	116,955	1.04	5.6	1.10	3.90	21.06
Indicated	189,044	0.78	6.0	0.84	4.73	36.47
M&I (direct processing)	305,999	0.88	5.8	0.94	8.62	57.52
Stockpile material						
Measured	26,521	0.30	4.1	0.35	0.26	3.50
Indicated	64,382	0.30	4.4	0.35	0.62	9.11
M&I (stockpile)	90,904	0.30	4.3	0.35	0.87	12.60
Total M&I	**396,903**	**0.74**	**5.5**	**0.81**	**9.50**	**70.13**
Inferred Resources						
Inferred (direct processing)	13,815	0.76	4.1	0.80	0.34	1.82
Inferred (stockpile)	3,785	0.31	3.6	0.35	0.04	0.44
Total Inferred	**17,600**	**0.66**	**4.0**	**0.71**	**0.38**	**2.26**

Notes to accompany Mineral Resource Table

1. Mineral Resources are reported inclusive of Mineral Reserves
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual open pit shell based on metal prices of $1,400/oz gold, $28.00/oz silver, and average metallurgical recoveries of 88.0% gold and 64.0% silver for oxide mineralization, 85.0% gold and 58.0% silver for transitional oxide / sulphide mineralization, and 85.0% gold and 44.0% silver for sulphide mineralization. The pit shell also considers a mining cost of $1.64/t for mineralized material, waste mining cost of $1.94/t; ore processing cost of $6.85/t; sustaining capital for the mill of $0.18/t; G&A cost of $1.25/tonne; allocation for the tailings facility costs of $0.60/t; royalties at 1.5% of revenue; refining costs of 0.1% of Revenue, and pit slopes that range from 23 to 43º.
4. Total contained metal calculated on the basis of Tonnes * Grade / 31.10348 grams per troy ounce
5. Gold-equivalent grade estimate based on $1,400/oz gold, $28.00/oz silver, and differential metallurgical recoveries (refer to footnote 3)
6. Direct processing material is defined as mineralization above a 0.4 g/t AuEq cutoff that is likely to be mined and processed directly
7. Stockpile material defined as mineralization above a 0.3 g/t AuEq and below a 0.4 g/t AuEq cutoff that is suitable for stockpiling and future processing based on average metallurgical recoveries of 79.0% gold and 37.0% silver. The 0.3 g/t AuEq lower cutoff grade is considered adequate to cover mining, processing, and additional handling costs
8. Tonnages are rounded to the nearest 1,000 tonnes, grades and metal content are rounded to two decimal places;
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content;
10. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces.

The following factors, among others, could affect the Mineral Resource estimate: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the LG pit shell, including metal recoveries, and mining and process cost assumptions.

1.13.3 Mineral Reserve Estimate

A proposed mining production schedule was developed through the design of an ultimate open pit within the Mineral Resource model. The open pit shell is based on metal prices of $1,300/oz gold and $22.00/oz silver, with variable recoveries by grade and ore type averaging 86.6% for gold and 49.1% for silver.

An elevated cut-off grade strategy has been selected to minimize the Project payback period and maximize the NPV. During the first 10 years of the Project, where a surplus of ore will be mined, the highest-grade ore will be sent to the mill and the rest stockpiled for processing at the end of the mine life. Mineral Reserves to undergo direct mill processing are defined as mineralization above a lower cut-off grade that varies by year between 0.26 g/t and 0.38 g/t AuEq. Mineral Reserves described as stockpiled material consist of ore tonnage above a 0.32 g/t AuEq cut-off grade that is mined and stockpiled before being sent to the mill. This stockpiled tonnage includes ore mined before mill start-up, lower grade ore mined during preproduction and commercial production, and ore tonnage misclassified or misallocated during the mining process.

The AuEq value used to determine which material goes to the mill, and which is stockpiled, is based on $1,400/oz gold, $28.00/oz silver, and average metallurgical recoveries of 88.0% gold and 64.0% silver for oxide mineralization, 85.0% gold and 58.0% silver for transitional oxide / sulphide mineralization, and 85.0% gold and 44.0% silver for sulphide mineralization.

There are two primary dilution and loss scenarios. The first scenario sees a surplus of ore being mined and being sent to both the mill and the low-grade stockpile. In the second scenario, all ore mined is sent to the mill with no surplus sent to the low-grade stockpile. Dilution and losses vary for these two scenarios due to the different cut-off grades used, resulting in different ore / waste contact block configurations. As such, the resulting average dilution for periods where both the mill and the stockpile are fed is 5% at a grade of 0.16 g/t Au and 3.19 g/t Ag. For periods where all ore is sent to the mill directly, dilution is 4% at a grade of 0.12 g/t Au and 2.90 g/t Ag. In addition, all isolated ore blocks—ore blocks with waste on all four adjacent sides—will be mined as waste and all isolated waste blocks—waste blocks with ore on all four adjacent sides—will be mined as ore.

Beyond the dilution factor, a misallocation factor is also applied when calculating the ore tonnes. This factor accounts for ore that is intended for the plant, based on grade, but is sent to the stockpile, or vice versa. A factor of 15% of the total material sent to the stockpile was applied to determine the misallocated quantities. The misallocated stockpile ore is made up with the average mill feed ore for the period. This misallocated material would average about 700kt per year, or just under 4% of the total mill feed.

Inherent rehandle has been assumed to average 3% over the life of the mine and this factor has been applied to all material mined.

Mineral Reserves were estimated assuming US$1,300/oz gold and US$22/oz silver at an exchange rate of 0.95 US$/CAD$.

1.13.4 Mineral Reserves Statement

Mineral Reserves are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. The Qualified Person for the estimate is Mr Jay Horton, P.Eng., of Norwest. Mineral Reserves have an effective date of 2 December, 2013.

Mineral Reserves are summarized in Table 1-2. Factors that may affect the estimate include, among others, gold and silver prices, US$ exchange rate assumptions, geotechnical assumptions, the ability of the mining operation to meet the projected annual production rate, capital and operating cost estimates, and process plant recoveries.

1.14 Mining Operations

1.14.1 Metallurgical Testwork

An extensive metallurgical testwork program was carried out over the period 2008 to 2013 on samples that were composited to represent process plant feed in the mine development plan. These samples were obtained from two primary sources: a dedicated metallurgical drilling program, and composites from exploration drill holes.

Material classified as Oxide for metallurgical purposes is characterized as having little to no sulphur present as sulphide. The Transition material represents incomplete oxidation of the sulphide, and Sulphide material is characterized by little oxidation of the sulphide or the host material. It is estimated that 3% of the mineralization is Oxide, 11% is Transition, and 86% is Sulphide.

During the 2012 Preliminary Economic Assessment (2012 PEA), it was determined that whole ore leach (WOL) was the most promising flowsheet. Mineralogical and diagnostic leach testing indicated that the primary areas of further investigation required to optimize WOL processing were primary grind size, reagent addition, and leach retention time.

Table 1-2: Mineral Reserve Estimate (effective date 2 December 2013, Jay Horton, P.Eng.)

Reserve Category	Tonnes & Grade			Contained Metal	
	Tonnes (Mt)	Au g/t	Ag g/t	Gold (Moz)	Silver (Moz)
Direct processing material					
Proven	124.5	0.95	5.5	3.79	22.1
Probable	169.7	0.68	4.1	3.73	22.3
Sub-Total Direct Processing	294.3	0.79	4.7	7.51	44.4
Stockpiled material					
Proven	20.1	0.50	3.6	0.33	2.3
Probable	30.1	0.34	14.6	0.33	14.1
Sub-Total Stockpiled	50.2	0.40	10.2	0.65	16.4
Total Direct Processing + Stockpiled					
Proven	144.6	0.88	5.3	4.11	24.4
Probable	199.8	0.63	5.7	4.05	36.4
Total Reserves	**344.4**	**0.74**	**5.5**	**8.17**	**60.8**

Footnotes to Accompany the Mineral Reserves Table:

1. Mineral Reserves are reported within an open pit design based on metal prices of $1,300/oz gold, $22.00/oz silver, with variable recoveries by grade and ore type averaging 86.6% for gold and 49.1% for silver
2. Contained metal calculated on the basis of Tonnes * Grade / 31.10348 grams per troy ounce
3. Mineral Reserves that are classified as amenable to direct processing are defined as mineralization above a lower cut-off grade that varies by year between 0.26 g/t and 0.38 g/t gold equivalent (AuEq) and represents ore that is to be mined and processed directly
4. Mineral Reserves noted as stockpiled material consist of ore tonnage above a 0.32 g/t AuEq cut-off grade that is mined and stockpiled before being sent to the mill. This stockpiled tonnage includes ore mined before mill start-up, lower grade ore mined during preproduction and commercial production, and ore tonnage misclassified or misallocated during the mining process. No stockpiles currently exist at site
5. The gold-equivalent grade estimate used for cut-off grades only is based on $1,400/oz gold, $28.00/oz silver, and average metallurgical recoveries of 88.0% gold and 64.0% silver for oxide mineralization, 85.0% gold and 58.0% silver for transitional oxide / sulphide mineralization, and 85.0% gold and 44.0% silver for sulphide mineralization
6. Cut-off grade values are based on a gold price of $1,300/oz. The cut-off grade calculation includes the following costs: minimum profit; operating cost (ore mining, hauling cost, processing, G&A); sustaining capital cost for mining, tailings storage facility and the mill; royalty and refining cost; reduced metallurgical gold recovery for stockpiled ore (79%)
7. There are two primary dilution and loss scenarios. The first scenario sees a surplus of ore being mined and being sent to both the mill and the low-grade stockpile. In the second scenario, all ore mined is sent to the mill with no surplus sent to the low-grade stockpile. Average dilution for periods where both the mill and the stockpile are fed is 5% at a grade of 0.16 g/t Au and 3.19 g/t Ag. For periods where all ore is sent to the mill directly, dilution is 4% at a grade of 0.12 g/t Au and 2.90 g/t Ag
8. Tonnages grades and metal content are rounded
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
10. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces.

Subsequent to the issue of the PEA-level report, a total of 21 trade-off studies and 21 metallurgical programs were performed to further process definition and to provide variability testing across the deposit. Information from these studies was used to update the gold and silver recovery models for the 2013 Feasibility Study and to develop the recovery equations used within the mining model.

Mineralogical studies completed have indicated that a significant amount of gold occurs as fine but liberated native gold and electrum particles. This fraction comprises between 20% and 60% of the total gold content depending on the oxidative state and lithology. A second fraction is likely to be in the form of high-silver electrum particles, gold associated with sulphides, and low-exposure particles. The balance of the precious metals are present either as finely disseminated values locked in sulphides, as slow-leaching species (tellurides), or encapsulated in silicates and carbonate gangue particles. No evidence of carbonaceous material was reported in the mineralogical reports.

Based on the JKTech global database of ore breakage characteristics, the Blackwater material is hard, with the Oxide and Transition materials being somewhat softer than the Sulphide. The BMWi and RMWi data showed a slight tendency toward medium hardness, Ai values showed tendency to mild abrasion, and Axb results suggested medium hardness. As a result, the base case flowsheet for the grinding circuit scenario of 60 kt/d at a P_{80} of 150 µm and 75% percentile hardness is two parallel lines, each with one SAG mill, pebble crusher, and ball mill.

A relationship was constructed relating tonnage as a function of ore hardness as measured by the Axb value but being constrained directly by ball mill limitations. This relationship provides a model that allows for optimization of the pit to maximize throughput and ultimately gold production. It is important to understand that these models all assume that production equipment is operating efficiently and that process control is optimized. Based on the annual average values for Axb, an optimal mill throughput was developed in conjunction with the use of a low-grade stockpile.

Confirmatory tests were performed for grind size, oxygen and lead nitrate addition, and cyanide and lime consumption. In addition, carbon-in-leach and variability work was undertaken. The response of samples to the WOL circuit has been very uniform, indicating a consistent form of gold mineralization within the Blackwater deposit. Grinding was noted to have a greater effect on gold leaching than previously observed, particularly at the coarser grind end. Silver extraction showed less sensitivity to grind.

The effect of processing varying blends of Oxide, Transition, and Sulphide composites in the WOL circuit was analyzed by comparing their measured and calculated gold and silver recoveries. Results indicate that there should be no metallurgical constraints on blending different oxidative types in the feed to the process plant.

Some parts of the Blackwater deposit, including an area to the north, contain higher levels of silver than the main zone. Additional work performed to determine the leaching response of high grade silver ore and to determine the best strategy for processing this material. Results indicated that the low gold / high silver grade material in the northern area will leach in similar fashion as the rest of the ore. A Transition composite sample was also tested and reported slightly higher gold and silver recoveries, again consistent with the main Transition ore behaviour.

An evaluation of throughput rates indicated that the optimum throughput is around 60,000 t/d and return diminishes rapidly with increasing tonnage.

Variability through the deposit was assessed via a number of methods. Overall metallurgical results show that mini-composite variability samples, including the four Sulphide ore types, were readily amenable to WOL at 125 µm grind and 36 hours' leach time. Cyanide consumption and lime requirement averages for the Transition mini-composites were notably higher than for the rest of the samples tested, as observed in previous programs. Oxidative state remains the single most important source of variability in the deposit. No discrete comminution behaviour units were identified from the mapping of ore hardness parameters Axb and BWi, which supports the interpretation that the grinding characteristics are very consistent (hard to very hard Axb and hard BWi) throughout the pit shell. Spatial mapping of gold and silver recovery identified no problem areas or significant variations within the pit shell.

For the purpose of plant recovery estimation and extraction equations, the samples are classified by oxidative state, as this has proven to be the primary driver of the metallurgical performance. LOM recoveries, assuming a WOL flowsheet, are estimated as:

- Au recovery: 86.6%
- Ag recovery: 49.1%

No elements that could be considered deleterious in the proposed process were identified from the testwork.

1.14.2 Geotechnical

Open pit geotechnical conditions are based on a review of geomechanical information, stability analyses results, the mining equipment to be utilized, and experience from similar open pit operations in the Project region.

Three geotechnical domains were defined for the purposes of the slope stability analyses. The slope stability analyses for each pit design sector found few geotechnical controls for the pit slope design, other than rock mass failure through the broken zone and associated slope depressurization requirements.

Inter-ramp slope angles of 27° are recommended for surficial material; bench face angles of 60° and inter-ramp slope angles of 40° should be achievable in broken material; and bench face angles of 70° and inter-ramp slope angles of 48° should be achievable in competent rock. Modifications to these general slope recommendations were made to three of the pit design sectors to ensure safe operations.

1.14.3 Hydrology

Water inflows to the Blackwater open pit will include both groundwater and surface water runoff. The contributions from groundwater will progressively increase as the pit extends below the groundwater table.

The 1-in-100 year return period storm has been used to size the pit surface water dewatering system and was estimated to be approximately 142,000 m^3. The estimated runoff coefficient inside the open pit surface area was conservatively assumed to be 100%.

The pit dewatering design is based on lowering the groundwater table within the highly permeable zone to approximately 15 m below the pit base elevation, considering both removal of groundwater from storage and from recharge. A combination of in-pit and perimeter pumping wells will be used for slope depressurization and pit dewatering.

1.14.4 Production Schedule

Large-scale open pit mining will provide process plant feed at a nominal rate of 60,000 t/d, or 21.9 million tonnes per year (Mt/a). Annual mine production of ore and waste will peak at 91.7 Mt/a. The average life-of-mine (LOM) stripping ratio including pre-production is 2.00:1. The operational stripping ratio, excluding waste stripping during the development phase, is 1.88:1. The production schedule is based on Proven and Probable Mineral Reserves; Inferred Mineral Resources within the pit shell have been set to waste.

A total of four phases were developed and incorporated into the mine production schedule. Some of these phases exist only for a short time, usually to increase ore grade or to produce material suitable for the tailings storage facility (TSF) construction.

The production schedule is based on 24-hour, year-round mining operations with limited weather delays and shutdowns. The total annual schedule is 355 operating days, or 8,520 hours. The mine will operate on two 12-hour shifts per day.

Key design factors in the mine operations plan include:

- Supply of pre-stripping overburden and delivery of designated waste rock for the construction of the TSF embankment
- Ore grade is maximized in the early years of the Project to reduce the payback period. Lower-grade ore is stockpiled and processed in later years
- Delivery of potentially acid generating (PAG) and non acid generating (NAG) metal leaching waste rock to the TSF for underwater disposal.

The Blackwater pit is designed with 12 m bench heights for both ore and waste.

Loading will be carried out by a combination of hydraulic and cable shovels backed up by a large front-end loader. Haul distances for both ore and waste will vary significantly, and therefore truck fleet requirements vary; 290 t capacity trucks have been selected. Drilling and blasting of approximately 7 Mt (dry) per month will be required to maintain production levels.

Mining operations will commence during the construction phase and by Year 1 the equipment fleet will comprise four 200/250 millimetre (mm) diesel blast hole drills, two 40 m^3 hydraulic shovels, one 28 m^3 front-end loader, and fourteen 290 tonne trucks. These will be supplemented with backup graders, and track and rubber-tired dozers.

The total tonnage mined increases to approximately 90 Mt/a in Years 3 through 10, and the mining fleet increases with the addition of four blast hole drills, one electric cable shovel, and 13 more haul trucks.

Table 1-3 summarizes the planned LOM production schedule.

1.14.5 Process

The 60,000 t/d process plant facility will consist of a primary crushing plant, a coarse ore stockpile (COS), a SAG/ball mill/crusher (SABC) grinding circuit, pre-leach thickening, whole ore cyanide leaching , carbon-in-pulp (CIP) recovery of precious metals from solution, elution of precious metals from carbon, and recovery of precious metals by electrowinning followed by smelting to doré. The plant will also have facilities for carbon regeneration, tailings thickening, and cyanide destruction.

Table 1-3: Mine Production Schedule

Duration	Mill Feed (Mt)	Head Grade		Recovery		Average Annual Metal Produced	
		Au (g/t)	Ag (g/t)	Au (%)	Ag (%)	Au (koz)	Ag (koz)
Year 1 – 9	183.4	0.85	5.6	87.1	50.1	485	1,842
Year 1 – 14	292.9	0.79	4.7	86.8	48.5	463	1,531
Year 15 – 17	48.9	0.40	10.2	84.4	50.6	177	2,726
LOM	**341.8**	**0.74**	**5.5**	**86.6**	**49.1**	**413**	**1,742**

Note: Table excludes 2.65 Mt of material mined and milled in the pre-production period.

Run-of-mine (ROM) ore will be crushed and ground to 80% passing 150 µm in a conventional dual-train milling circuit. Ground ore will be directed to a leach feed thickener, then to a leaching and CIP extraction circuit. Extracted gold and silver will be released from carbon in the stripping columns and recovered by electrowinning before being smelted into gold-silver doré. Stripped carbon will be treated in a regeneration kiln.

Plant design criteria are based on JKTech testwork, JKSimMet grinding simulations, and laboratory leaching, settling, and carbon absorption tests carried out by various laboratories, consultants, and vendors. Where no data are available, process design criteria were developed in consultation with New Gold based on industry averages and modelling by equipment suppliers and independent consultants.

Key process equipment will consist of the following:

- A primary crushing plant with a 1,520 x 2,870 mm (60" x 113") gyratory crusher
- A SAG/ball mill/crusher grinding circuit:
  - two 11.0 m diameter x 6.7 m (36' x 21.5') EGL, 17 MW SAG mills
  - two 8.2 m diameter x 12.8 m (27' x 42') EGL, 17 MW ball mills
  - two 1,000 kW pebble crushers
- Whole ore leaching and CIP circuit:
  - 24 leach tanks of 18 m diameter
  - two trains of seven 400 m^3 capacity CIP tanks
  - two 80 m diameter thickeners
- Three cyanide destruction vessels using the SO_2/air process on leach CIP residue before transfer in a single stream to the TSF.

The bulk of the water requirements for the process plant will be met with reclaim water recovered from air compressors, column heat exchangers, the thickeners, and the

TSF. Bulk and packaged reagents will be trucked to site and stored for use in the process plant.

1.14.6 Onsite Infrastructure

An 880-person construction camp will be erected on site, which, together with the expansion of the existing camp from 250 to 426 persons, will provide accommodations for contractors and construction management staff. It will not be used in the operations phase. For operations a high-quality modular camp with a capacity of 420 persons will be constructed on site.

Onsite infrastructure to support mining and milling activities will include a primary crusher, reclaim conveyors, mill building, elution and refinery building, whole ore leach tanks, main truck shop, administration and emergency services buildings, explosives storage facility and fuel farm.

Power will be supplied to the Blackwater site by connection to the BC Hydro grid. A 139.5 km long 230 kV transmission line will be constructed from the BC Hydro Glenannan Substation to the Blackwater site. The transmission line has been routed to make use of existing access and to cross recently logged areas as much as practical along its alignment. The incoming transmission line will terminate at the site main substation adjacent to the main process facilities. The anticipated electrical load for the Blackwater site is as follows:

- Peak load 107 MW
- Average load 98.4 MW
- Power factor 98%

Emergency power will be available from a standby power station that will consist of a minimum of two modular gensets rated at a nominal 3.0 MW.

A fibre optic cable will be installed along with the main transmission line to provide high bandwidth telecommunications access to the site.

Wells will be developed near the new camp area to supply water for the temporary and operations camps. The water will be treated and distributed around the camp site for domestic use.

Fresh water for the Project will be sourced from Tatelkuz Lake, approximately 20 km northeast of the mine site. The fresh water will be used to mitigate flow reductions in Davidson Creek downstream of the TSF and for Project operations as required.

1.14.7 Waste Rock Characterisation

Some of the waste rock and the tailings are classified as potentially acid generating (PAG) and/or metal leaching (ML). Waste rock was classified based on its Neutralizing Potential Ratio (NPR) and ML potential as follows:

- PAG1 – NPR ≤ 1.0 (PAG)
- PAG2 - 1.0 < NPR ≤ 2.0 (PAG)
- NAG3 – NPR > 2.0 and Zn ≥ 1,000 ppm (NAG-ML)
- NAG4 – NPR > 2.0 and 600 ≤ Zn <1,000 ppm (NAG)
- NAG5 – NPR > 2.0 and Zn < 600 ppm (NAG).

PAG1 and PAG2 waste rock will be stored and submerged in the TSF within one year of mining to prevent the formation of acid rock drainage (ARD). NAG3 waste rock will be stored and submerged in the TSF within three to five years of mining to reduce metal leaching. NAG4 waste rock will be used for construction on site and in the downstream shell of TSF Dam D as required. NAG5 will be used for construction and in the downstream shell of TSF Dam D.

Overburden is classed as non acid generating and will be used for construction on and off site. Sulphide and transition ore tailings are classified as potentially acid generating and will be kept saturated or submerged within the TSF during operations to prevent ARD. Oxide tailings exhibit low ARD and ML potential and will be placed as the upper layer in TSF Site C.

1.14.8 Waste Rock Storage Facilities

The East waste facility, to be sited east of the pit, is the primary storage location for overburden and NAG5 waste not required for TSF construction. The overall dump has a capacity of approximately 50 Mt and will be constructed in a series of lifts to enhance stability and minimize resloping requirements at closure.

The West waste facility, to be located west of the pit, is the primary storage location for overburden and NAG4 waste not required for TSF construction. The West waste facility has a capacity of approximately 87 Mt.

The mined overburden will be distributed between the two waste facilities based on the ratio of rock to overburden placed in a given period. A minimum of 10% of waste rock will be placed with the overburden to provide plating and maintain the overall integrity of the waste facilities.

1.14.9 Tailings Storage Facility

The TSF was designed to permanently store tailings, PAG1 and PAG2 waste rock, and NAG3 waste rock generated during the operation of the mine. The TSF has two adjacent sites, Site C and Site D. The design of the TSF is supported by extensive geotechnical site investigations. The facility has been designed to contain 453 Mm^3 of tailings and waste rock material and will require 73.9 Mm^3 of construction material, about 65.8 Mm^3 or 89% of which will be waste rock and overburden from the open pit.

The TSF embankments will be engineered, water retaining, zoned earthfill/rockfill dams with a compacted low-permeability core zone and appropriate filter/transition zones. A total of three embankments will be constructed across the two sites. The TSF Site C and Site D dams will be expanded using the downstream and centreline construction methods, respectively.

Site runoff water will be stored on site within the TSF, and the supernatant will be recycled back to the process plant. Additional water may be required to maintain a water cover over PAG waste rock and tailings in the TSF facility during some years of operation if conditions are drier than average. Fresh water for the mill or additional makeup water for the TSF to support operations will be pumped from Tatelkuz Lake.

1.14.10 Water Management

All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure. The following strategies are used in the tailings and mine water management plan:

- Manage sediment mobilization and erosion by installing sediment controls prior to land disturbance, limiting land disturbance to the minimum practicable extent. Install appropriate temporary erosion and sediment control measures or Best Management Practices (BMP) prior to, and during, initiation of land disturbance
- Use the water within the proposed Project area to the maximum practicable extent by collecting and managing site runoff from disturbed areas, maximizing the recycle of process water, and storing water within the TSF
- Pump water as required from Tatelkuz Lake through a water supply pipeline to a water reservoir downstream of the TSF main dam to provide fresh water and makeup water for the process plant and to maintain in-stream fish needs in Davidson Creek
- Operate the TSF with no surface water discharge during operations

- Collect all (except minimal) TSF seepage downstream of the main dam during operations and post closure until the pit lake overflows or the water is acceptable for direct discharge to Davidson Creek
- Monitor surface water and groundwater quality, maintain fish habitat, develop compensatory fish habitat, and reclaim disturbed areas.

During operations, drainage from the low-grade and coarse ore stockpiles may become acidic with elevated metals content; the drainage will be collected and neutralized with lime to increase the pH and precipitate metals before disposal in the TSF. Pit water is predicted to be of neutral pH with relatively low metals content during operations; it will be pumped to a small holding/monitoring pond, which will overflow to the TSF.

Tailings slurry from the mill will be treated using the SO_2/air process. Laboratory testwork showed the process will be effective in lowering cyanide and metal concentrations.

1.14.11 Environment

New Gold has conducted extensive environmental baseline studies and is preparing comprehensive environmental management plans for the Project. Completed studies have provided baseline information on the Project's atmospheric, terrestrial, and aquatic environments, the socio-economic setting and heritage.

Two areas have been identified that will require careful environmental management:

- Potential effects on fish due to a flow reduction in Davidson Creek downstream of the TSF; the Project footprint avoids those lower reaches and proposes to mitigate for flow reductions in Davidson Creek immediately downstream of the mine site by pumping water from Tatelkuz Lake in sufficient volumes to avoid a harmful alteration destruction or disruption (HADD) of fish habitat
- Effects on whitebark pine (*Pinus albicaulis*); a management plan will be developed to mitigate the Project effects, and the closure and reclamation plan will incorporate measures to restore whitebark pine in the mine site area.

The Project lies outside the key overwintering habitat of the Tweedsmuir-Entiako caribou herd. The site is also entirely within the Nechako River basin and is outside the Blackwater River basin that is an important fishery and heritage river.

New Gold has designed and will operate the Project in accordance with the International Cyanide Management Code.

1.14.12 Closure Plan

The closure plan has simplified water management requirements resulting from the compact Project layout and integrated waste management strategy.

The closure plan employs proven practices and is not dependent on long-term active treatment and monitoring. All Project components will be decommissioned and reclaimed according to best industry practices and provincial and federal regulations. Proposed end land use objectives for mine closure are wildlife habitat and return of the land for traditional use by Aboriginal groups.

The estimated closure and reclamation cost for the Project, discounted to the last year of operations, is approximately $86 million, including progressive reclamation conducted during the mine life. The estimated salvage value of the Project is about $78.4 million.

1.14.13 Permitting

The environmental assessment process for the Project officially started in October 2012 with the acceptance of the Project Description by both the BC EAO (*BCEAA*) and the CEAA.

A large number of federal or provincial permits are required for mine construction and subsequent operations. Some of the provincial permits have legislated timelines, while others do not. The federal permits and authorizations do not have legislated timelines, and some permits originating from federal agencies may not be issued within the review period set out in the B.C. legislation. Key permits, licences, and authorizations required to construct the mine include:

- BC Mines Act permit
- BC Environmental Management Act permits for discharges from the Project to surface waters and for air emissions
- Schedule 2 amendment under the federal Fisheries Act, which is required to place mine waste into a natural water body that is frequented by fish
- Section 35(2) authorization under the federal Fisheries Act for a harmful alteration, disruption, or destruction of fish habitat
- Section 23 application or exemption under the federal Navigable Waters Protection Act for effects on navigable waters
- Licences or approvals under the BC Water Act for the use of water

- Mining Lease.

1.14.14 Social Considerations and Sustainability

New Gold is actively consulting with First Nations, government, and other stakeholders that could potentially be affected by the Project. The intent of the consultation is to increase the mutual awareness and understanding of the Project and its potential effects, and to explore potential strategies to mitigate negative effects and enhance positive ones.

The Project will create approximately 595 permanent jobs. The construction work force will be roughly 1,200 on average, peaking at 1,500. New Gold is committed to maximizing local employment and contracting opportunities. New Gold will work collaboratively with community partners, use existing training programs to prepare local workers, and establish new programs for specific training where necessary.

1.14.15 Capital Cost Estimate

The capital cost estimate for the Blackwater Project was developed to provide an estimate suitable for the 2013 Feasibility Study phase, including costs to design, procure, construct, and commission the facilities. In consideration of the current state of design and procurement, this estimate falls under the AACE Class 3 Estimate classification and its accuracy is expected to be within +15%/-10% of final Project cost. Capital costs will be capitalized until commercial production, which is defined as 30 days at an average of 60% of production capacity.

The estimate covers the direct field costs of executing the Project, indirect costs associated with the design, construction, and commissioning of the new facilities, Owner's costs, including head office charges, Project expenses, and mine site ramp-up training and engineering, and contingency.

The cost estimate is based on a combination of material take-off (MTO) data, design drawings, vendor quotes, manufacturers' information, and industry standards and rates. All costs are expressed in Q3 2013 Canadian dollars. For the purposes of estimation of Mineral Reserves, these costs have been brought forward to Q4 2013, with no escalation applied.

The total estimated development capital cost for the Project is $1,963 million (US$1,865 million) inclusive of a $200 million (US$190 million) contingency (Table 1-4). The development capital cost equates to $280 (US$266) per recoverable gold ounce over the life of the Project. The total LOM sustaining capital is estimated to be $681 million (US$647 million), equivalent to an average of $97 (US$92) per recoverable gold ounce.

Table 1-4: Summary of Project Capital Costs

Description	Cost ($M)
Direct Costs	
Mining & Preproduction Development	286
On-Site Infrastructure	166
Process Plant	632
Tailing Facilities and Water Reclaim	90
Access Corridor	12
Off-Site Infrastructure	127
Total Direct Costs	1,314
Owner's and Indirect Costs	
Owner's Costs	78
EPCM	113
Other Indirects	258
Total Owner's and Indirect Costs	449
Subtotal	1,763
Contingency (11%)	200
Total Project	**1,963**

1.14.16 Operating Cost Estimate

For operating cost estimation purposes the Project has been divided in three areas: Mining, Processing, and General and Administrative (G&A). The costs for each department include labour, operating and maintenance supplies, freight, and utilities as appropriate. The average life-of-mine (LOM) operating cost estimate is based on a total of 341.8 Mt of ore milled over the mine life at an average rate of 21.9 Mt per year. All costs are expressed in Q3 2013 Canadian dollars with no allowance for contingency. Costs exclude royalties, GST, depreciation, escalation, and product transport costs, which are handled separately in the Blackwater economic model.

LOM operating costs are estimated at $14.49/t of ore milled (Table 1-5), and $12.48/t of ore milled after accounting for silver credits. Total LOM all-in sustaining cash costs are estimated at $14.47/t of ore milled. The average LOM cost after silver credits and royalty is US$578/oz Au produced (Table 1-6). The average LOM all-in sustaining cash cost is US$670/oz Au produced.

Table 1-5: Summary of Project Operating Costs

Area	LOM Unit Cost ($/t milled)	(CAD$/oz Au produced)	(US$/oz Au produced)
Mining	5.33	260	247
Processing	7.20	351	333
General and Administrative	1.43	70	67
Refining Costs	0.21	10	10
Transport and Insurance	0.05	2	2
Total Royalties	0.27	13	12
Total Cash Costs	**14.49**	**707**	**671**
Silver Credits	(2.01)	(98)	(93)
Total Cash Costs incl. silver credit and royalties	**12.48**	**609**	**578**
Sustaining Capital	1.99	97	92
All-In Sustaining Cash Costs	**14.47**	**706**	**670**

Table 1-6: Cash Costs per Ounce of Gold Produced during Periods of Mine Operation

Cost Item	Years 1 – 9	1 – 14	15 – 17	LOM
Cash Costs – $/oz including royalty	673	683	996	707
Cash Costs – $/oz including Ag credit, royalty	585	606	639	609
All-in Sustaining Cash Costs - $/oz	721	709	664	706
Cash Costs – US$/oz including royalty	639	649	946	671
Cash Costs – US$/oz including Ag credit, royalty	555	576	607	578
All-in Sustaining Cash Costs - US$/oz	685	674	631	670

1.14.17 Financial Analysis

The results of the economic analysis discussed in this section represent forward-looking information as defined under Canadian securities law. Actual results may differ materially from those expressed or implied by forward-looking information. The reader should refer to Section 1.3 of this Report for more information regarding forward-looking statements, including material assumptions (in addition to those discussed in this section and elsewhere in this Report) and risks, uncertainties and other factors that could cause actual results to differ materially from those expressed or implied in this section (and elsewhere in the Report).

The financial analysis was carried out using a discounted cash flow (DCF) model. All cash inflows and outflows throughout the life of the Project are discounted at a given discount rate to a present value and are then added to yield a net present value (NPV). The DCF model for the Blackwater Project assumes 100% equity financing (no debt) as a simplifying assumption, and all figures are expressed in Q3 2013 constant dollars or real terms. No inflation or real cost escalation was assumed. Cash flows are assumed to occur mid-period. All monetary figures are expressed in Canadian dollars unless otherwise specified. The exchange rate assumption used is $1 Canadian equals $0.95 US.

The Project value is determined on a pre-tax and after-tax basis with discounting to the start of Year -2 (2015), which marks the first year of Project construction. Project expenses incurred in Year -3 are treated in the cash flow model as occurring at the start of Year -2. Mining pre-production starts in Year -2, and Year 1 marks the first year of commercial production resulting in revenue.

1.14.18 Base-Case Pre-Tax Evaluation

For the pre-tax evaluation, the following Base Case financial parameters were calculated:

- $1,044 million (US$991 million) NPV (pre-tax, Year -2 (2015)) at 5.0% discount rate.
- 11.3% IRR
- 6.2-year payback on $1,963 million (US$1,865 million) capital cost

Base Case metal prices were:

- Gold – US$1,300/oz
- Silver – US$22/oz
- Exchange – $0.95 (US$/CAD$).

Four gold price, silver price, and exchange rate scenarios were used in the pre-tax model to evaluate the sensitivity on pre-tax NPV, IRR, and payback. The Base, Low, Moderate, and High gold price cases and results are shown in Table 1-7. This table demonstrates sensitivities to gold price, silver price, and exchange rate for the four selected scenarios. All other parameters, including capital costs and operating costs, are fixed.

Annual gold production and cash costs, net of silver credits and royalty, are shown in Figure 1-1. Figure 1-2 shows the sensitivity of pre-tax NPV to gold price, exchange rate, capital costs, and operating costs. The Project is most sensitive to gold prices and foreign exchange rate assumptions, followed by operating and capital costs. The cash flow model includes payments of underlying NSR royalties to parties that retain a royalty interest.

Table 1-7: Commodity Price and Exchange Rate Scenarios, Pre-Tax

Item	Unit	Case Base	Low	Moderate	High
Gold Price	US$/oz	1,300	1,150	1,450	1,600
Silver Price	US$/oz	22	20	24	26
Exchange Rate	US$/CAD$	0.95	0.93	0.97	1.00
NPV, CAD$	$M	1,044	432	1,631	2,120
IRR	%	11.3	7.8	14.4	16.8
Payback Period	years	6.2	7.5	5.1	4.5
NPV, US$	$M	991	402	1,582	2,120
IRR	%	11.3	7.8	14.4	16.8
Payback Period	years	6.2	7.5	5.1	4.5

Note: Pre-tax NPV at Year -2 (2015) and 5% discount rate.

Figure 1-1: Base Case Pre-Tax Gold Production and Cash Costs

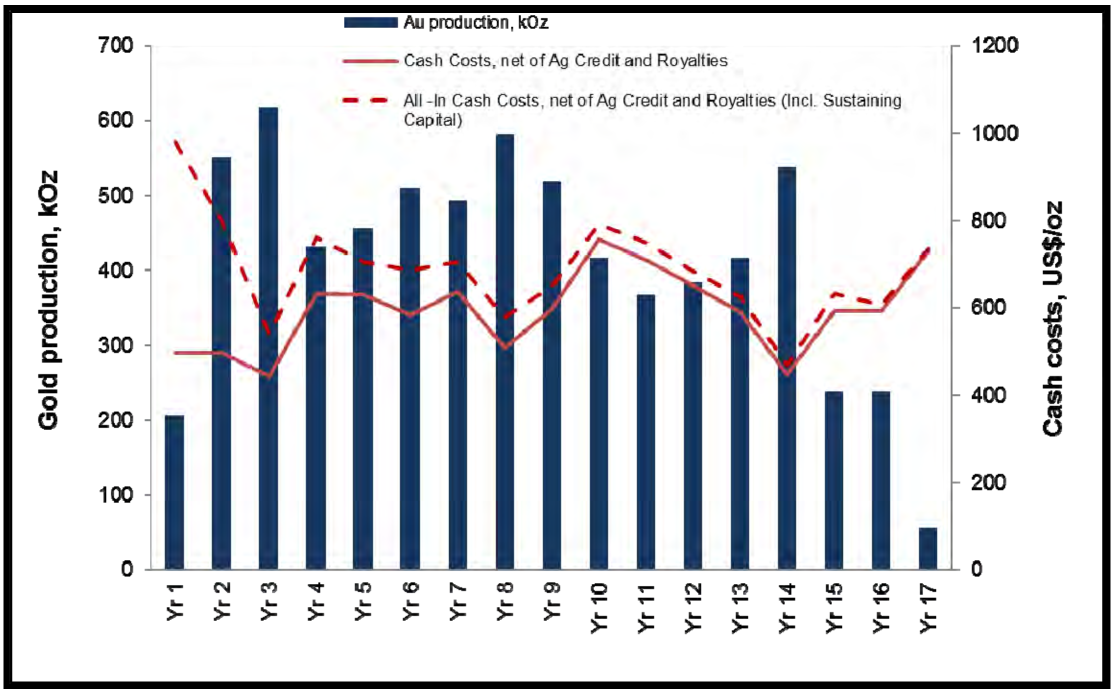
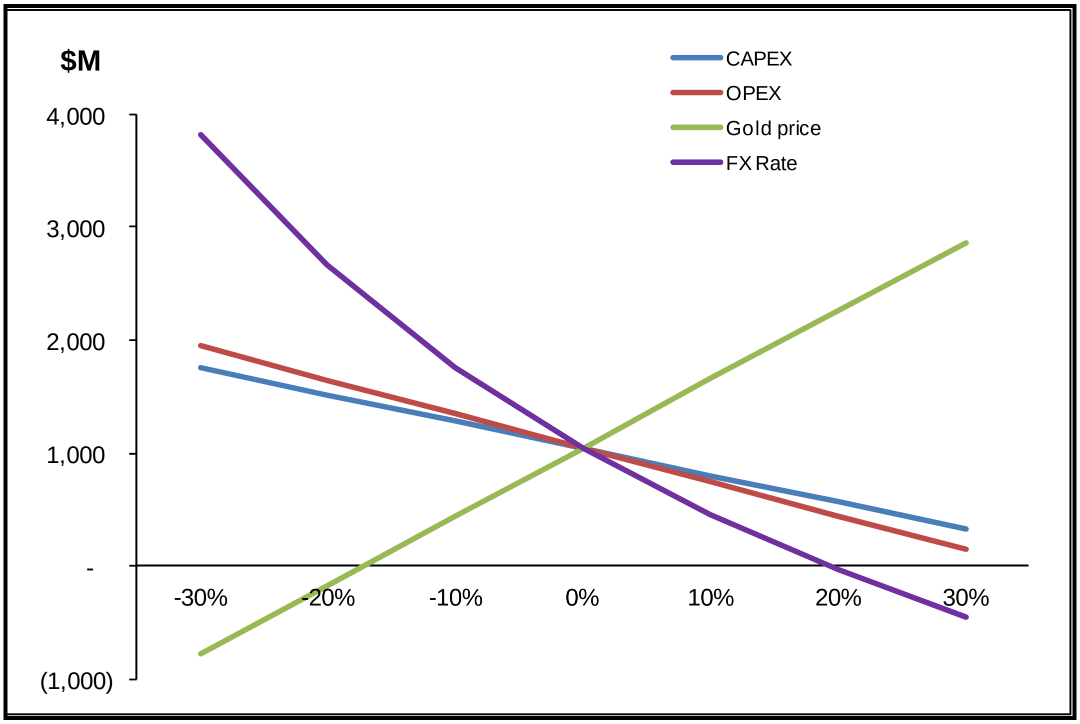


Figure 1-2: Pre-tax NPV Sensitivity to Economic Input Assumptions



1.14.19 Base-Case After-Tax Evaluation

For the after-tax evaluation, the following Base Case financial parameters were calculated:

- $616 million (US$585 million) NPV (post-tax, Year -2 (2015)) at 5.0% discount rate.
- 9.3% IRR
- 6.4-year payback on $1,963 million (US$1,865 million) capital cost

The after-tax financial model was calculated at the same four gold price, silver price, and exchange rate scenarios that were used in the pre-tax model. The Base, Low, Moderate, and High gold price cases and after-tax results are shown in Table 1-8.

Table 1-8: Commodity Price and Exchange Rate Scenarios, After-Tax

Item	Unit	Case Base	Low	Moderate	High
Gold Price	US$/oz	1,300	1,150	1,450	1,600
Silver Price	US$/oz	22	20	24	26
Exchange Rate	US$/CAD$	0.95	0.93	0.97	1.00
NPV, CAD$	$M	616	199	1,008	1,329
IRR	%	9.3	6.5	11.9	13.8
Payback Period	years	6.4	7.7	5.3	4.6
NPV, US$	$M	585	185	978	1,329
IRR	%	9.3	6.5	11.9	13.8
Payback Period	years	6.4	7.7	5.3	4.6

Note: Post-tax NPV at Year -2 (2015) and 5% discount rate.

An important consideration when calculating the after-tax economics for Blackwater is that the Project is held within a corporate entity that also includes New Gold's New Afton operation located in British Columbia and the company's two Canadian-based corporate offices, one of which is in Vancouver, British Columbia and the other in Toronto, Ontario. As such, New Gold should be able to realize tax synergies between different assets by utilizing tax attributes interchangeably amongst its portfolio of assets, all with the goal of maximizing New Gold's overall profitability rather than that of any one operation or project. For purposes of the base case after-tax economics, only the deductions, allowances and credits that are specifically related to the Blackwater Project have been included. No allocation of potential attributes related to New Gold's current, or future, corporate administrative expenses, interest expenses or capital costs at the company's other Canadian-based assets has been made in the after-tax analysis. In addition, the after-tax economics for Blackwater do not take into account any future corporate re-organizations or tax planning that New Gold may undertake to maximize its overall profitability.

1.14.20 Sensitivity Analysis

Sensitivity analysis was performed on the Project using metal price, exchange rate, operating costs and capital costs. The Project is more sensitive to changes in the gold price and the USD:CAD exchange rate than to changes in capital or operating costs.

1.14.21 Risks and Opportunities

New Gold developed and implemented a comprehensive risk and opportunities register during the Feasibility Study and tracked progress in addressing and advancing the items.

The major risks to the Project were identified as:

- Changes to metal prices and exchange rate assumptions
- Capital cost growth
- Increases in operating costs
- Productivity assumptions
- Dilution control
- Presence of high-grade silver in the mill feed
- Integration of mining operations and the TSF construction
- Permitting delays
- Lack of social licence affecting permit grant

Project opportunities included:

- Delineation of additional mineralization that could support higher-confidence resource categories through additional drilling
- Use of a trolley assist system later in the mine life
- Assessment of methods to reduce waste mining costs
- Use of oxygen rather than compressed air for cyanide leaching and cyanide detoxification
- Pre-crushing the SAG mill feed to a finer feed size
- Value engineering initiatives.

1.15 Exploration and Development

The Blackwater Project area offers good potential for the discovery of additional mineralization that may support mineral resource estimation. Work to develop this potential is ongoing and involves a combination of detailed interpretation of deposit geology from the growing body of exploration drill hole information in conjunction with expanded geologic mapping, geochemical sampling, airborne and ground-based geophysical methods, and exploration drilling, both beyond the limits of the Blackwater deposit as they are currently known, and within the greater Project area.

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a mining operation on the Project is at the discretion of New Gold.

1.16 Recommendations

There are no meaningful recommendations arising from the feasibility study. The decision to proceed with a mining operation on the Project is at the discretion of New Gold.

New Gold is intending to continue with activities that will support the permitting process for the Project. These activities form a two-phase work program, which will cost approximately $8–10 million in support of environmental studies and long-lead items required for Project permitting. The phases will be conducted concurrently, with the aim of achieving full Project permitting at the end of the program. Each phase is estimated to be $4–5 million to complete.

The work is envisaged as:

- Phase 1: Environmental Impact Assessment – Completion and filing of the EIS including ongoing environmental baseline work
- Phase 2: Provincial and Federal permitting – Completion of engineering site investigations and design to enable completion of key Provincial (e.g. Mines Act) and Federal (e.g. Fisheries Act) permit applications.

2.0 INTRODUCTION

AMEC Americas Ltd. (AMEC), Allnorth Consultants Limited (Allnorth), Geosim Services Inc. (Geosim), Norwest Corporation (Norwest), and Knight Piésold Ltd (Knight Piésold) prepared a technical report (the Report) for New Gold Inc (New Gold) on the Blackwater gold project, located 112 km southwest of Vanderhoof in British Columbia, Canada (Figure 2-1).

2.1 Terms of Reference

The Report was prepared in support of the New Gold press release entitled "New Gold Announces Blackwater Feasibility Study Results" dated 12 December 2013.

All measurement units used in this Report are metric, and currency is expressed in Canadian dollars unless stated otherwise. The Report uses Canadian English.

For the purposes of the report, two terms are used for the mine production: life-of-mine (LOM) refers to the life of mine including the pre-production period; the operational period refers to the mine life excluding the pre-production duration.

2.2 Qualified Persons

The following serve as the qualified persons (QPs) for this Technical Report as defined in National Instrument 43-101, *Standards of Disclosure for Mineral Projects*, and in compliance with Form 43-101F1:

- Ronald G. Simpson, P.Geo., Principal, Geosim
- Ignacy (Tony) Lipiec, P.Eng., Director, Process Engineering, AMEC Vancouver
- Gary Christie P.Eng., Project Manager, AMEC Vancouver
- Jay Horton, P.Eng. Manager, Technical Services, Norwest
- Bruno Borntraeger, P.Eng., Specialist Engineer and Project Manager, Knight Piésold, Vancouver.

Figure 2-1: Blackwater Project Location Map

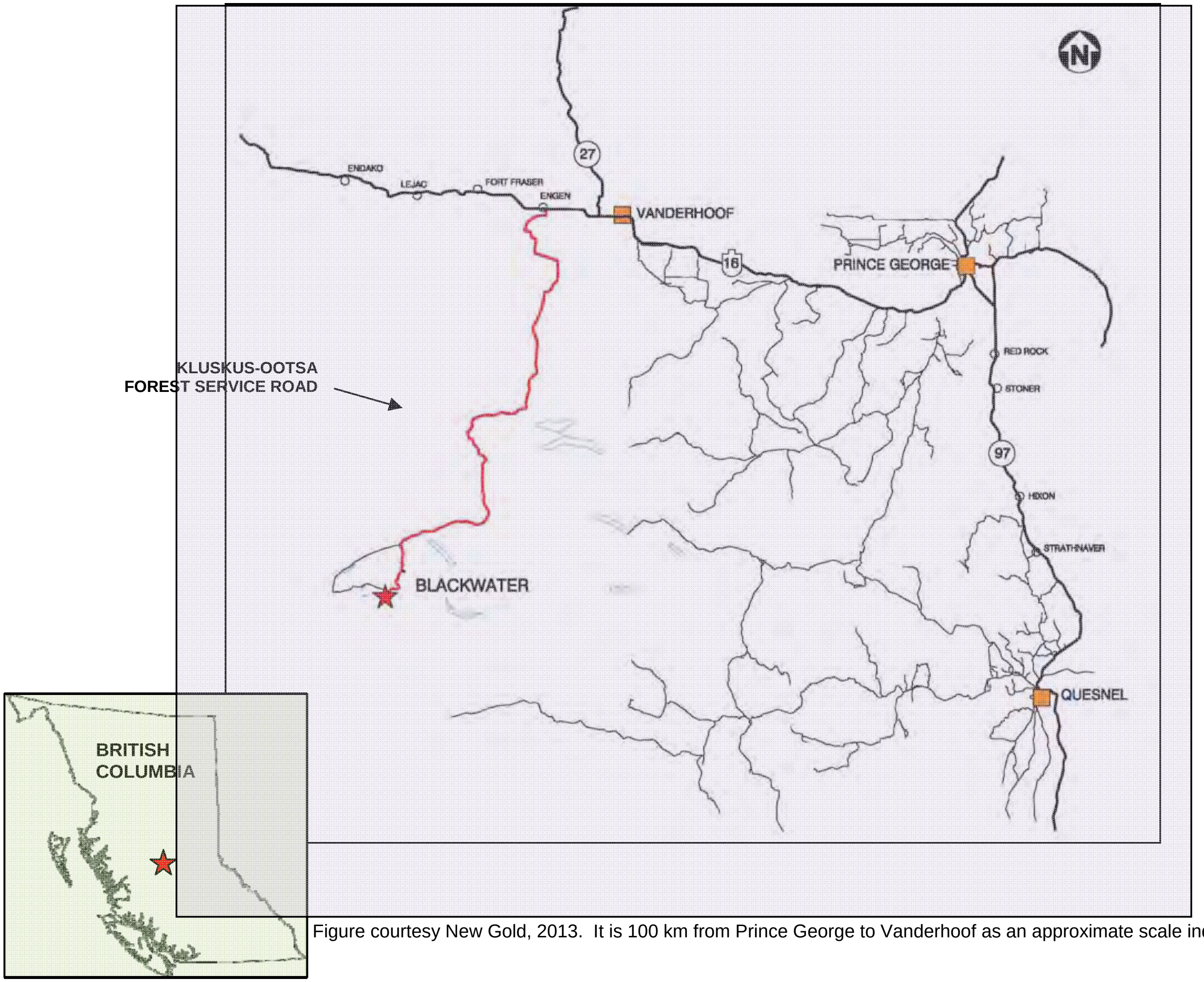


Figure courtesy New Gold, 2013. It is 100 km from Prince George to Vanderhoof as an approximate scale indicator.

2.3 Site Visits and Scope of Personal Inspection

Site visits were performed as follows.

Mr Gary Christie visited the site from 22–23 May, 2013. During his site visit, Mr Christie viewed the proposed locations of the site facilities and access road.

Mr Jay Horton visited the site on 22 May, 2013. During his visit, Mr. Horton viewed the proposed location of the pit and waste dumps.

Mr Ronald G. Simpson visited the site on December 13, 2010; September 8, 2011; November 28; 2011, and September 20, 2012. The purpose of the visits was to review the drilling, sampling, and quality assurance/quality control procedures. The geology and mineralization encountered in the drill holes completed to date were also reviewed. A detailed description of the site visit findings is included in Section 12.1.

Mr Bruno Borntraeger visited the site on September 28, 2011, December 20, 2011, February 21, 2012 and April 24, 2012. The purpose of these visits was to view proposed locations of site facilities, potential construction material borrow areas and review geotechnical site investigation progress.

Mr Tony Lipiec did not visit the site, but completed review and audit visits to the laboratories involved in the flowsheet development and recovery determinations for the Blackwater deposit. These laboratory visits included Dawson, McClelland and Metsolve. During these visits procedures in the handling of samples, testing and analysis of products were reviewed. Mr Lipiec has reviewed the testwork performed under the supervision of New Gold since 2011. Mr Lipiec considers that the test work performed is appropriate to support the selection and development of the flowsheet used as the project basis.

2.4 Effective Dates

The Report has a number of effective dates as follows:

- Date of supply of last assay data used in resource estimation: 16 January 2013
- Date of Mineral Resource estimate: 31 March, 2013
- Date of Mineral Reserve estimate: 2 December 2013
- Date of financial analysis: 2 December 2013
- Date of supply of latest information on mineral tenure, surface rights and Project ownership: 14 January 2014

The overall effective date of the Report is taken to be the date of the supply of the latest information on mineral tenure and surface rights, and is 14 January 2014.

2.5 Information Sources and References

The key information source for the Report was the 2013 Feasibility Study, entitled:

- AMEC, 2013: Blackwater Project, B.C. Feasibility Study: unpublished report prepared by AMEC, Knight Piésold, Geosim, Norwest Corp, and Allnorth, December 2013, 17 vols.

Information used to support this Report was also derived from previous technical reports on the Project, and from the reports and documents listed in the References section. Additional information was sought from New Gold personnel where required.

2.6 Previous Technical Reports

New Gold has previously filed technical reports on the Project as follows:

- Simpson, R., 2011a: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date March 2, 2011, re-addressed June 6, 2011.
- Simpson, R., 2011b: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date September 19, 2011.
- Simpson, R., 2012: Technical Report, Blackwater Gold Project, Omineca Mining Division, British Columbia, Canada: report prepared for New Gold Inc. and Silver Quest Resources Ltd., effective date March 7, 2012.
- Simpson, R.G., Welhener, H.E., Borntraeger, B., Lipiec T., and Mendoza, R., 2012: Blackwater Project British Columbia, Canada NI 43-101 Technical Report on Preliminary Economic Assessment: report prepared for New Gold Inc. by GeoSim Services Inc, Independent Mining Consultants Inc, Knight Piésold Ltd. and AMEC, effective date 28 August, 2012.

3.0 RELIANCE ON OTHER EXPERTS

The QP authors of this Report state that they are qualified persons for those areas as identified in the "Certificate of Qualified Person" for each QP, as included in this Report. The QPs have relied, and believe there is a reasonable basis for this reliance, upon the following other expert reports, which provided information regarding mineral rights, surface rights, and environmental status in sections of this Report as noted below.

3.1 Mineral Tenure

The QPs have not reviewed the mineral tenure, nor independently verified the legal status, ownership of the Project area or underlying property agreements. The QPs have fully relied upon, and disclaim responsibility for, information supplied by New Gold experts and experts retained by New Gold for this information through the following documents:

- Lawson Lundell, LLP, 2013: opinion letter addressed to New Gold from Lawson Lundell LLP dated 24 December 2013
- Lawson Lundell LLP, 2014a: Mineral Tenure Opinion Letter addressed to New Gold from Lawson Lundell LLP dated 14 January 2014
- Lawson Lundell, LLP, 2014b: Report Summarizing Certain Agreements Pertaining to the Blackwater Project addressed to New Gold from Lawson Lundell LLP dated 13 January 2014

This information is used in Section 4.2, Section 4.3, Section 4.4 and Section 4.6 of the Report, and in support of the Mineral Resource estimate in Section 14 and the Mineral Reserve estimate in Section 15.

- New Gold 2014a: letter addressed to AMEC Americas Ltd. from New Gold dated 17 January 2014

This information is used in Section 4.4 and Section 4.6 of the Report, and in support of the Mineral Resource estimate in Section 14 and the Mineral Reserve estimate in Section 15.

3.2 Surface Rights

The QPs have fully relied upon, and disclaim responsibility for, information supplied by experts retained by New Gold for information relating to the status of the current Surface Rights as follows:

- Lawson Lundell LLP, 2014c: Report on Certain Surface Rights in the Vicinity of the Blackwater Project addressed to New Gold from Lawson Lundell LLP dated 14 January, 2014.

This information is used in Section 4.5 of the Report and in support of the Mineral Resource estimate in Section 14 and the Mineral Reserve estimate in Section 15.

3.3 Taxation

The QPs have fully relied upon, and disclaim responsibility for, information supplied by New Gold experts for information relating to the taxation assumptions used in the after-tax financial model as follows:

- New Gold, 2014b: Blackwater Project – NI 43-101 Technical Report on Feasibility Study: letter provided to Mr Gary Christie, AMEC, regarding the reasonability of the tax calculations and tax assumptions in the after-tax life of mine model dated 15 January 2014.

This information is used in Section 22 of the Report.

4.0 PROPERTY DESCRIPTION AND LOCATION

4.1 Introduction

For the purpose of this report the Blackwater property (the Property) refers to the entire Blackwater claim block. The Blackwater Project (the Project) refers to exploration and development activity related to the Blackwater deposit.

The Property lies in central British Columbia, approximately 112 km southwest of Vanderhoof and 446 km northeast of Vancouver. The Property is within NTS map sheet 93F/02 and is centred at 5893000 N and 375400 E (UTM NAD83).

The Property is one of four contiguous claim blocks in the area held by New Gold (Blackwater, Capoose, Auro and Key) (Figure 4-1). The Capoose, Auro and Key claim blocks are discussed in this Report for convenience only and do not form any part of the Blackwater Property or Project.

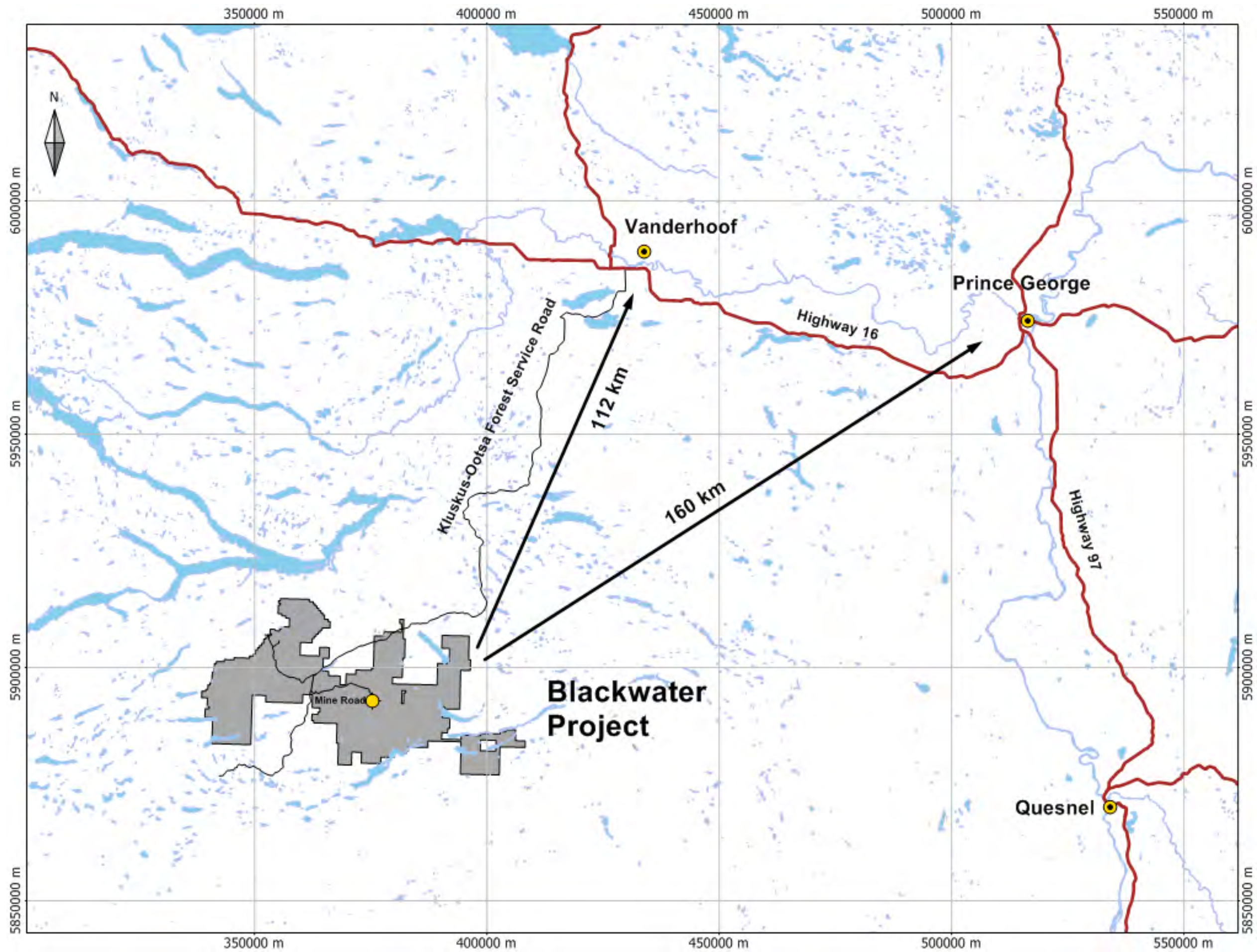
4.2 Property and Mineral Title in British Columbia

Prior to 1 June 1991, recordations in respect of a mineral claim or mining lease in British Columbia were manually recorded on, or attached to, the original application document for a mineral claim or the original lease document for a mining lease. From June 1991 to 11 January 2005, all records were entered into a computer database, maintained by the Gold Commissioner's Office. On 12 January 2005, the British Columbia mineral titles system was converted to an online registry system, MTO, and ground-staking of claims was eliminated in favour of map-staking based on grid cells.

Claims recorded prior to 12 January 2005 are referred to as legacy claims; Claims acquired through map staking are referred to as cell claims. From and after the date of changeover to map-staking, claim holders could convert legacy claims to cell claims, or maintain the original legacy claim. Legacy claims vary in size and shape, depending on the regulations that were in force at the time of staking and recordation. Cell claims comprise from 1 to 100 cells which range from 21 hectares in southern British Columbia to 16 hectares in the north.

Mineral title may also be held as part of Crown grants or freehold tenure issued under separate grant, such as a railway grant. Crown-granted mineral rights originate from staked mineral claims that were surveyed then granted from the Crown to private individuals or corporations under the legislation in effect at the time of grant.

Figure 4-1: Location Plan Blackwater, Capoose, Auro and Key Claim Blocks



Note: Figure courtesy New Gold, 2014

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There can be instances where there may be more than one type of mineral tenure in existence over the same land area; examples are where a Crown-granted mineral title is overlapped by a mineral tenure granted under the Mineral Tenure Act (British Columbia) (the MTA). In this case, the holder of the MTA mineral tenure is entitled only to those minerals not covered in the Crown-granted mineral title.

To keep claims in good standing in accordance with the MTA, a minimum value of work or cash-in-lieu is required annually. The minimum value of work required to maintain a legacy or cell mineral claim for one year is currently set at $5 per hectare for the first and second anniversary years, $10 per hectare for the third and fourth anniversary years, $15 per hectare in the fifth and sixth anniversary years, and $20 per hectare for each subsequent anniversary year. The cash-in-lieu required to maintain a mineral claim for an anniversary year is double the value of the work commitment requirement.

The holder of a mineral claim or mining lease issued under the MTA does not have exclusive possession of the surface or exclusive right to use the surface of the land. However, the holder of such claims and leases does have the right to access the lands for the purpose of exploring for minerals and to use the surface for mining activities (exploration, development, and production).

The surface of a mineral claim or mining lease may either be privately owned or owned by the Crown.

The MTA provides for a recorded claim holder to use, enter and occupy the surface of a claim for the exploration and development or production of minerals, including the treatment of ore and concentrates, and all operations related to the exploration and development or production of minerals and the business of mining, subject to production limits. Permits are required before undertaking most exploration or mining activity.

A mining lease is required if the claim holder wishes to produce more than 1,000 tonnes of ore in a year from each unit in a legacy claim (typically 25 hectares) or each cell in a cell claim. The holder of a mineral claim may obtain a mining lease for that claim if certain requirements are met (surveying if required, payment of fees, and posting of notices). A mining lease allows the lessee to hold Crown mineral lands for up to 30 years initially, and is renewable if certain conditions are met. A recorded claim holder must give surface owners of private land and leaseholders of Crown land notice before entering for any mining activity. A recorded holder is liable to compensate the surface owner for loss or damage caused by the entry, occupation or use of the area for exploration and development or production of minerals.

4.3 Project Ownership History

The Project area in the vicinity of the Property was initially explored by Granges Inc. from 1973. In 2005, Silver Quest Resources Ltd. (Silver Quest) acquired an interest in the area, and entered into a joint venture with Richfield Ventures Corp. (Richfield) in 2009.

In 2011, New Gold acquired Richfield and Silver Quest, and a third company, Geo Minerals Limited (Geo), to consolidate the ground holdings in the Property and the Capoose claim block.

All mineral claims comprising the Property are cell claims.

In 2012, New Gold signed an option agreement to earn a 100% interest in a single Capoose area mineral claim from a private corporation.

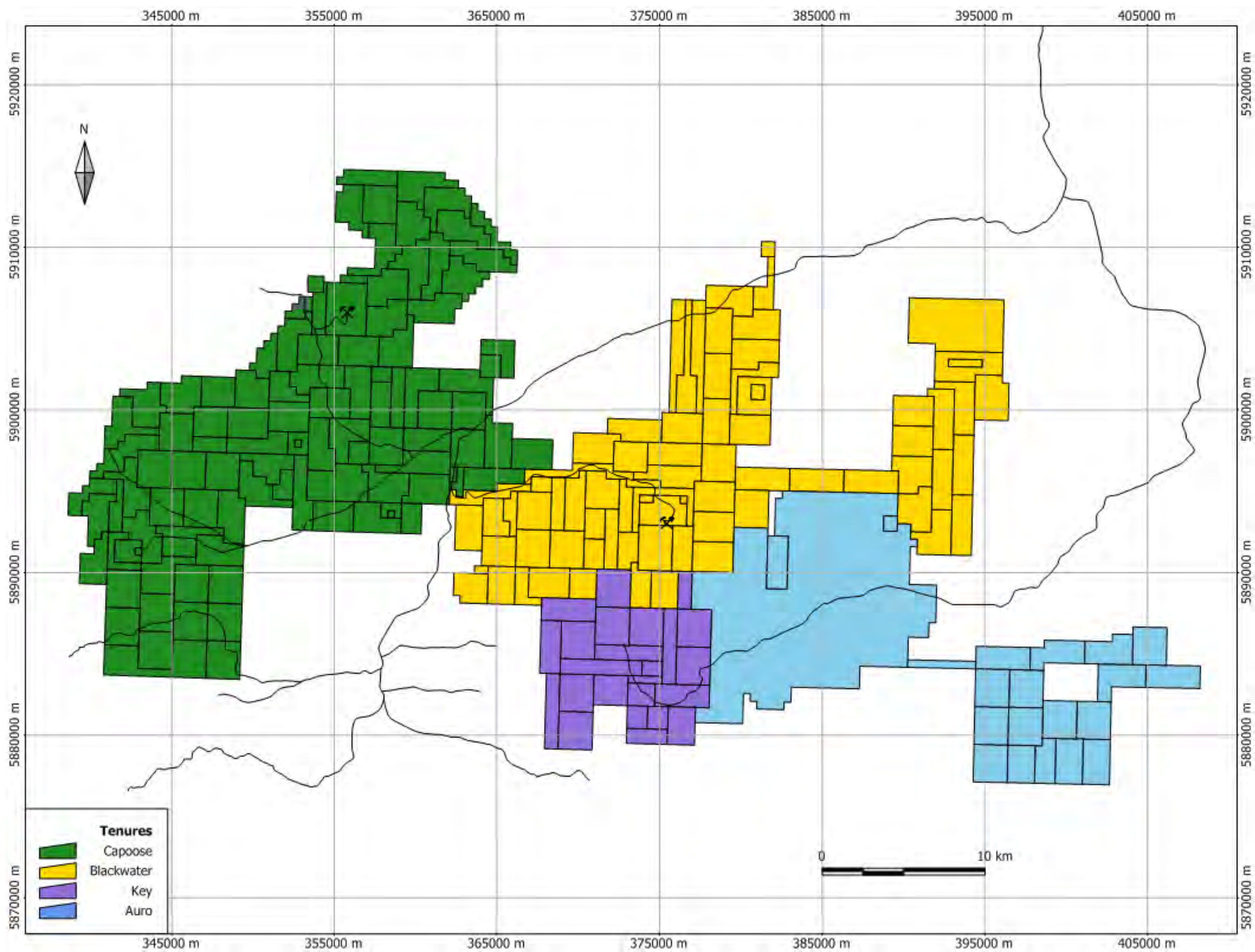
Also during 2012, New Gold acquired the Auro properties from Gold Reach Resources Ltd. (Gold Reach) and added a further mineral claim to form the Auro claim block. An additional infill claim was recorded and added to the Auro claim block in 2013. Also in 2013, the Company acquired the Key claim block from Troymet Exploration Corporation (Troymet).

In January 2014 New Gold recorded four additional mineral claims, which were added to the Blackwater claim block.

4.4 Mineral Tenure

New Gold holds 100% recorded interest in 227 mineral claims covering an area of 104,678 ha distributed among the Property and the Capoose, Auro, and Key claim blocks (Figure 4-2).

Figure 4-2: Mineral Claims Blocks



Note: Figure courtesy New Gold, 2014

4.4.1 Blackwater Claim Block

The Property comprises 75 mineral cell claims totalling 30,578 ha (Figure 4-3, Table 4-1). All Blackwater claims are 100% held in the name of New Gold. Sixty-nine claims expire in 2022. Two claims expire in October 2014 and four claims expire in January 2015. New Gold holds both the recorded and beneficial interest in these claims

None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves. The Blackwater deposit spans the Davidson claim (509273), the Dave claim (515809) and the Jarrit claim (515810).

4.4.2 Capoose Claim Block

The Capoose claim block is situated west of the Blackwater claim block and consists of 107 mineral claims totalling 42,655 ha (Figure 4-4). All Capoose claims are 100% held in the name of New Gold. Capoose mineral claims are summarized in Table 4-1.

One hundred and one of the Capoose claims expire in 2022. The other claims expire in September or October of 2014.

All claims in the Capoose claim block, excepting claim 238045, are cell claims. Capoose claims 706597 and 645063 partially overlap portions of legacy claims held by a third party.

None of the Capoose claims are known to overlap with areas of any Crown granted mineral claims.

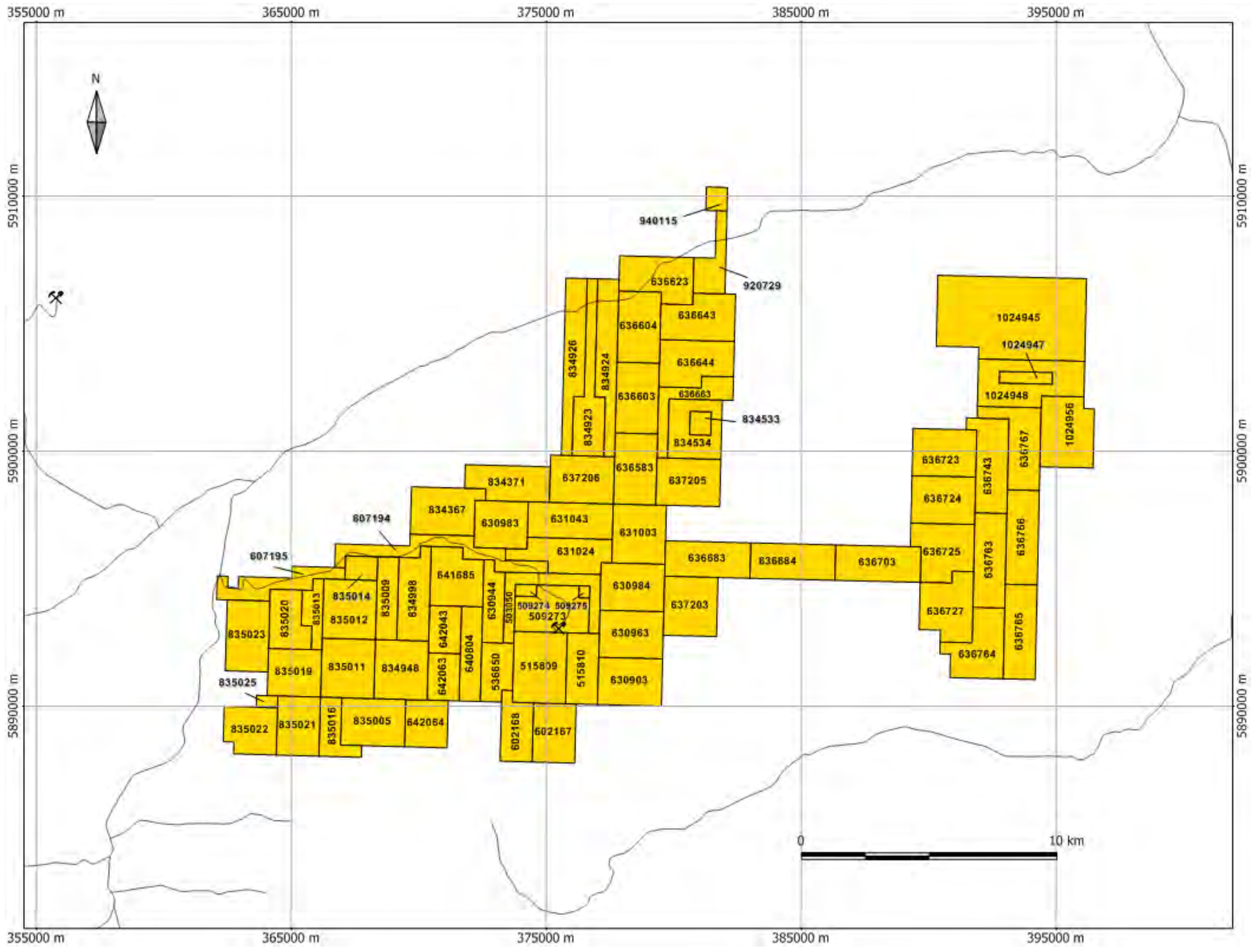
Three Capoose claims partially overlap no-staking reserves, and 16 claims partially overlap the Entiako Provincial Park.

4.4.3 Auro Claim Block

The Auro claim block lies southeast of the Blackwater claim block and contains 21 mineral claims totalling 22,591 ha (Figure 4-5). All Auro claims are 100% held in the name of New Gold. The Auro mineral claims are summarized in Table 4-1

Twenty Auro claims expire in 2022. Claim 1018105 expires in March 2014.

Figure 4-3: Blackwater Claim Block



Note: Figure courtesy New Gold, 2014

Table 4-1: Claims Listing

Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
646683	PRINCESS	03/10/2009	29/08/2022	Mineral	Claim	093F	407.18	Auro
745822	NG1	12/04/2010	29/08/2022	Mineral	Claim	093F	485.50	Auro
745842	NG2	12/04/2010	29/08/2022	Mineral	Claim	093F	485.50	Auro
745862	NG3	12/04/2010	29/08/2022	Mineral	Claim	093F	465.90	Auro
745882	NG4	12/04/2010	29/08/2022	Mineral	Claim	093F	465.87	Auro
745902	NG5	12/04/2010	29/08/2022	Mineral	Claim	093F	465.84	Auro
745922	NG6	12/04/2010	29/08/2022	Mineral	Claim	093F	485.68	Auro
745942	NG7	12/04/2010	29/08/2022	Mineral	Claim	093F	485.22	Auro
745962	NG8	12/04/2010	29/08/2022	Mineral	Claim	093F	485.68	Auro
745982	NG9	12/04/2010	29/08/2022	Mineral	Claim	093F	485.42	Auro
746002	NG10	12/04/2010	29/08/2022	Mineral	Claim	093F	485.73	Auro
746022	NG11	12/04/2010	29/08/2022	Mineral	Claim	093F	485.73	Auro
746042	NG12	12/04/2010	29/08/2022	Mineral	Claim	093F	466.50	Auro
746062	NG13	12/04/2010	29/08/2022	Mineral	Claim	093F	485.96	Auro
746082	NG14	12/04/2010	29/08/2022	Mineral	Claim	093F	466.50	Auro
746102	NG15	12/04/2010	29/08/2022	Mineral	Claim	093F	465.99	Auro
746182	NG15	12/04/2010	29/08/2022	Mineral	Claim	093F	388.77	Auro
746202	NG16	12/04/2010	29/08/2022	Mineral	Claim	093F	330.44	Auro
831124	AURO PROPERTY	05/08/2010	29/08/2022	Mineral	Claim	093F	14025.98	Auro
982702	BW BRIDGE	26/04/2012	29/08/2022	Mineral	Claim	093F	194.14	Auro
1018105		27/03/2013	27/03/2014	Mineral	Claim	093F	77.52	Auro
503050	WHITEWATER	13/01/2005	29/08/2022	Mineral	Claim	093F	348.76	Blackwater
509273	GOT	19/03/2005	29/08/2022	Mineral	Claim	093F	484.45	Blackwater
509274	got2	19/03/2005	29/08/2022	Mineral	Claim	093F	38.75	Blackwater
509275	got3	19/03/2005	29/08/2022	Mineral	Claim	093F	19.38	Blackwater
515809		01/07/2005	29/08/2022	Mineral	Claim	093F	581.60	Blackwater
515810		01/07/2005	29/08/2022	Mineral	Claim	093F	348.96	Blackwater
536650	NIGHT FLIGHT	06/07/2006	29/08/2022	Mineral	Claim	093F	271.42	Blackwater
602167	BWD	05/04/2009	29/08/2022	Mineral	Claim	093F	387.94	Blackwater
602168	BWD2	05/04/2009	29/08/2022	Mineral	Claim	093F	310.35	Blackwater
607194	BLACKWATER 2	08/07/2009	29/08/2022	Mineral	Claim	093F	464.86	Blackwater
607195	BLACKWATER 1	08/07/2009	29/08/2022	Mineral	Claim	093F	348.74	Blackwater

Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
630903	BW1	09/09/2009	29/08/2022	Mineral	Claim	093F	465.33	Blackwater
630944	BW2	09/09/2009	29/08/2022	Mineral	Claim	093F	251.90	Blackwater
630963	BW3	09/09/2009	29/08/2022	Mineral	Claim	093F	465.15	Blackwater
630983	BW4	09/09/2009	29/08/2022	Mineral	Claim	093F	387.29	Blackwater
630984	BW5	09/09/2009	29/08/2022	Mineral	Claim	093F	464.98	Blackwater
631003	BW6	09/09/2009	29/08/2022	Mineral	Claim	093F	484.14	Blackwater
631024	BW7	09/09/2009	29/08/2022	Mineral	Claim	093F	445.49	Blackwater
631043	BW8	09/09/2009	29/08/2022	Mineral	Claim	093F	464.73	Blackwater
636583	KASSY 1	18/09/2009	29/08/2022	Mineral	Claim	093F	464.53	Blackwater
636603	KASSY 2	18/09/2009	29/08/2022	Mineral	Claim	093F	464.27	Blackwater
636604	KASSY 3	18/09/2009	29/08/2022	Mineral	Claim	093F	464.01	Blackwater
636623	KASSY 4	18/09/2009	29/08/2022	Mineral	Claim	093F	463.82	Blackwater
636643	KASSY 5	18/09/2009	29/08/2022	Mineral	Claim	093F	483.30	Blackwater
636644	KASSY 6	18/09/2009	29/08/2022	Mineral	Claim	093F	483.47	Blackwater
636663	KASSY 7	18/09/2009	29/08/2022	Mineral	Claim	093F	290.18	Blackwater
636683	RIGHT STUFF 1	18/09/2009	29/08/2022	Mineral	Claim	093F	464.86	Blackwater
636684	RIGHT STUFF 2	18/09/2009	29/08/2022	Mineral	Claim	093F	464.87	Blackwater
636703	RIGHT STUFF 3	18/09/2009	29/08/2022	Mineral	Claim	093F	464.87	Blackwater
636723	RIGHT STUFF 4	18/09/2009	29/08/2022	Mineral	Claim	093F	464.45	Blackwater
636724	RIGHT STUFF	18/09/2009	29/08/2022	Mineral	Claim	093F	464.63	Blackwater
636725	RIGHT STUFF 6	18/09/2009	29/08/2022	Mineral	Claim	093F	484.18	Blackwater
636727	RIGHT STUFF 7	18/09/2009	29/08/2022	Mineral	Claim	093F	484.41	Blackwater
636743	RIGHT STUFF 8	18/09/2009	29/08/2022	Mineral	Claim	093F	483.84	Blackwater
636763	RIGHT STUFF 9	18/09/2009	29/08/2022	Mineral	Claim	093F	464.85	Blackwater
636764	RIGHT STUFF 10	18/09/2009	29/08/2022	Mineral	Claim	093F	484.56	Blackwater
636765	RIGHT STUFF 11	18/09/2009	29/08/2022	Mineral	Claim	093F	465.12	Blackwater
636766	RIGHT STUFF 12	18/09/2009	29/08/2022	Mineral	Claim	093F	464.76	Blackwater
636767	RIGHT STUFF 13	18/09/2009	29/08/2022	Mineral	Claim	093F	464.42	Blackwater
637203	OZZY	19/09/2009	29/08/2022	Mineral	Claim	093F	484.42	Blackwater
637205	BABY JANE	19/09/2009	29/08/2022	Mineral	Claim	093F	464.57	Blackwater
637206	DAVID DALE	19/09/2009	29/08/2022	Mineral	Claim	093F	464.57	Blackwater
640804	PUREANDY	25/09/2009	29/08/2022	Mineral	Claim	093F	310.16	Blackwater
641685	RICHFIELDADJACENTCC	26/09/2009	29/08/2022	Mineral	Claim	093F	445.58	Blackwater

Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
642043	BW	27/09/2009	29/08/2022	Mineral	Claim	093F	232.57	Blackwater
642063	BW 2	27/09/2009	29/08/2022	Mineral	Claim	093F	232.66	Blackwater
642064	BW3	27/09/2009	29/08/2022	Mineral	Claim	093F	310.34	Blackwater
834367	RICH 1	27/09/2010	29/08/2022	Mineral	Claim	093F	484.06	Blackwater
834371	DAVIDSON	27/09/2010	29/08/2022	Mineral	Claim	093F	425.87	Blackwater
834533	DAVIDSON 1	29/09/2010	29/08/2022	Mineral	Claim	093F	77.39	Blackwater
834534	DAVIDSON 2	29/09/2010	29/08/2022	Mineral	Claim	093F	406.33	Blackwater
834923	DAVIDSON 3	02/10/2010	29/08/2022	Mineral	Claim	093F	483.59	Blackwater
834924	DAVIDSON 4	02/10/2010	29/08/2022	Mineral	Claim	093F	483.45	Blackwater
834926	DAVIDSON 5	02/10/2010	29/08/2022	Mineral	Claim	093F	483.45	Blackwater
834948		03/10/2010	29/08/2022	Mineral	Claim	093F	484.69	Blackwater
834998	RICH 2	04/10/2010	29/08/2022	Mineral	Claim	093F	426.28	Blackwater
835005		04/10/2010	29/08/2022	Mineral	Claim	093F	465.50	Blackwater
835009		04/10/2010	29/08/2022	Mineral	Claim	093F	271.27	Blackwater
835011		04/10/2010	29/08/2022	Mineral	Claim	093F	484.69	Blackwater
835012		04/10/2010	29/08/2022	Mineral	Claim	093F	484.46	Blackwater
835013		04/10/2010	29/08/2022	Mineral	Claim	093F	174.41	Blackwater
835014	DAVE	04/10/2010	29/08/2022	Mineral	Claim	093F	116.23	Blackwater
835016		04/10/2010	29/08/2022	Mineral	Claim	093F	232.77	Blackwater
835019		04/10/2010	29/08/2022	Mineral	Claim	093F	387.77	Blackwater
835020		04/10/2010	29/08/2022	Mineral	Claim	093F	329.47	Blackwater
835021	BW WEST	04/10/2010	29/08/2022	Mineral	Claim	093F	387.94	Blackwater
835022	BW WEST2	04/10/2010	29/08/2022	Mineral	Claim	093F	368.56	Blackwater
835023		04/10/2010	29/08/2022	Mineral	Claim	093F	465.19	Blackwater
835025	BW WEST2	04/10/2010	29/08/2022	Mineral	Claim	093F	38.79	Blackwater
920729	JONBLK	21/10/2011	21/10/2014	Mineral	Claim	093F	251.20	Blackwater
940115	NOREADD	06/01/2012	21/10/2014	Mineral	Claim	093F	77.25	Blackwater
1024945	BW NE	09/01/2014	09/01/2015	Mineral	Claim	093F	1817.21	Blackwater
1024947	BW-N 1	09/01/2014	09/01/2015	Mineral	Claim	093F	96.71	Blackwater
1024948	BW-N 2	09/01/2014	09/01/2015	Mineral	Claim	093F	599.6	Blackwater
1024956	BW-N 3	09/01/2014	09/01/2015	Mineral	Claim	093F	561.13	Blackwater
238045	CAP	18/09/1978	18/09/2014	Mineral	Claim	093F	100.00	Capoose
512838		17/05/2005	29/08/2022	Mineral	Claim	093F	811.88	Capoose

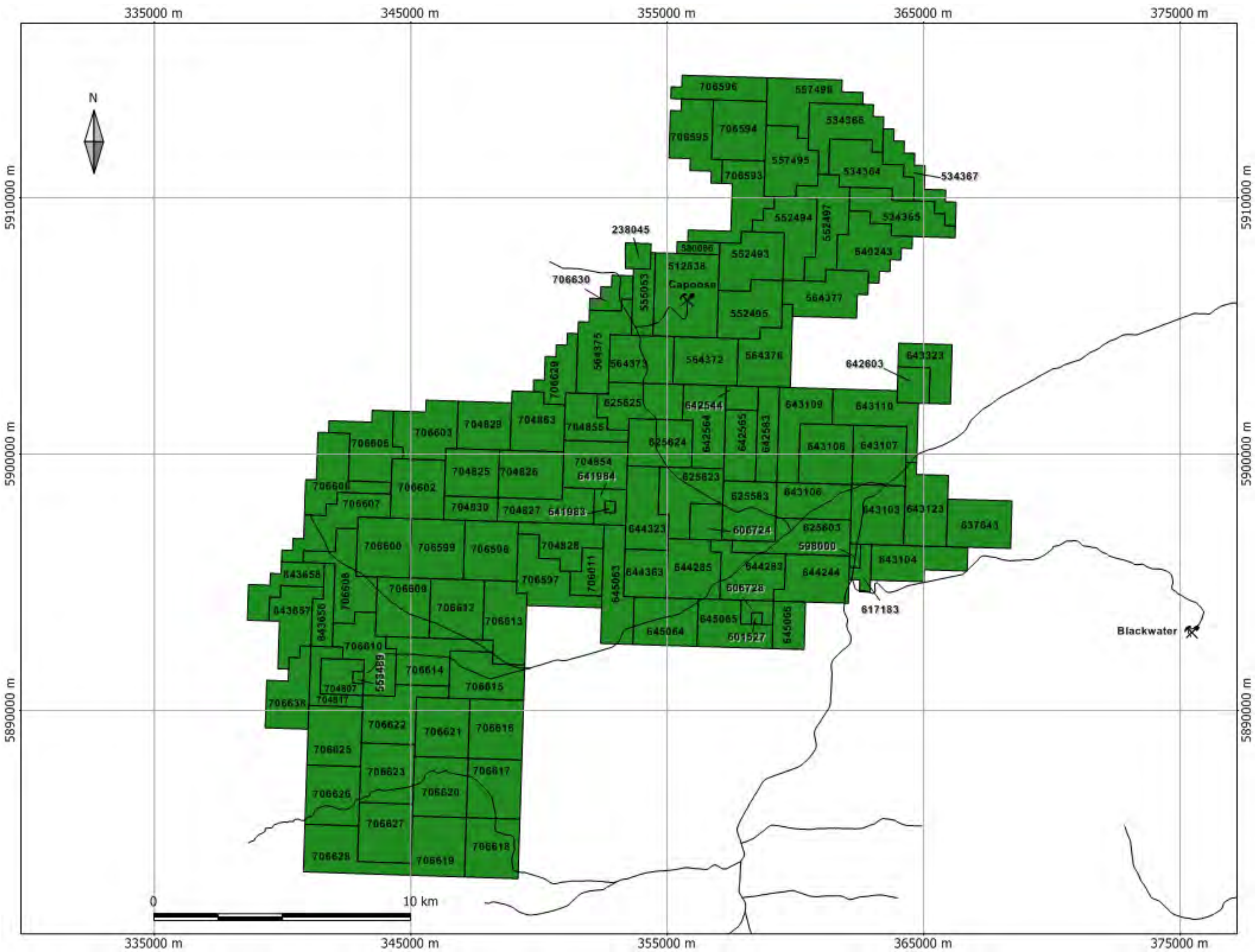
Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
534364	JAG-1	24/05/2006	29/08/2022	Mineral	Claim	093F	482.75	Capoose
534365	JAG-2	24/05/2006	29/08/2022	Mineral	Claim	093F	482.92	Capoose
534366	JAG-3	24/05/2006	29/08/2022	Mineral	Claim	093F	482.60	Capoose
534367	JAG-4	24/05/2006	29/08/2022	Mineral	Claim	093F	289.67	Capoose
552493	NE CAPOOSE	22/02/2007	29/08/2022	Mineral	Claim	093F	483.12	Capoose
552494	NE CAPOOSE 2	22/02/2007	29/08/2022	Mineral	Claim	093F	483.00	Capoose
552495	E CAPOOSE	22/02/2007	29/08/2022	Mineral	Claim	093F	483.31	Capoose
552497	NE CAPOOSE3	22/02/2007	29/08/2022	Mineral	Claim	093F	482.97	Capoose
553489	PAW	03/03/2007	29/08/2022	Mineral	Claim	093F	19.39	Capoose
555053	CAP	26/03/2007	21/10/2014	Mineral	Claim	093F	251.30	Capoose
557495	JAG-5	23/04/2007	29/08/2022	Mineral	Claim	093F	482.73	Capoose
557496	JAG-6	23/04/2007	29/08/2022	Mineral	Claim	093F	482.49	Capoose
564372	CAPOOSE S	09/08/2007	29/08/2022	Mineral	Claim	093F	464.18	Capoose
564373	CAPOOSE SW	09/08/2007	29/08/2022	Mineral	Claim	093F	464.18	Capoose
564375	CAPOOSE SW2	09/08/2007	29/08/2022	Mineral	Claim	093F	483.48	Capoose
564376	CAPOOSE E2	09/08/2007	29/08/2022	Mineral	Claim	093F	483.49	Capoose
564377	CAPOOSE E3	09/08/2007	29/08/2022	Mineral	Claim	093F	483.24	Capoose
580086	CAPOOSE NORTH	01/04/2008	29/08/2022	Mineral	Claim	093F	77.29	Capoose
598000	BUCK	26/01/2009	21/10/2014	Mineral	Claim	093F	38.74	Capoose
601527	FAWN	23/03/2009	29/08/2022	Mineral	Claim	093F	19.38	Capoose
606724	FAWN	27/06/2009	21/10/2014	Mineral	Claim	093F	174.29	Capoose
606728	MALAPUT E-W	27/06/2009	21/10/2014	Mineral	Claim	093F	96.90	Capoose
617183	BUCK 2	10/08/2009	21/10/2014	Mineral	Claim	093F	96.86	Capoose
625583	M-1	29/08/2009	29/08/2022	Mineral	Claim	093F	484.09	Capoose
625603	M-2	29/08/2009	29/08/2022	Mineral	Claim	093F	484.19	Capoose
625623	M-3	29/08/2009	29/08/2022	Mineral	Claim	093F	484.03	Capoose
625624	M-4	29/08/2009	29/08/2022	Mineral	Claim	093F	464.48	Capoose
625625		29/08/2009	29/08/2022	Mineral	Claim	093F	483.68	Capoose
637643	EMMA	20/09/2009	29/08/2022	Mineral	Claim	093F	464.75	Capoose
641983	FAWN	27/09/2009	29/08/2022	Mineral	Claim	093F	19.36	Capoose
641984	FAWN 2	27/09/2009	29/08/2022	Mineral	Claim	093F	154.91	Capoose
642544	FAWNIE DOME	28/09/2009	29/08/2022	Mineral	Claim	093F	116.08	Capoose
642564	FD 2	28/09/2009	29/08/2022	Mineral	Claim	093F	464.40	Capoose

Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
642565	FD 3	28/09/2009	29/08/2022	Mineral	Claim	093F	348.36	Capoose
642583	FD 4	28/09/2009	29/08/2022	Mineral	Claim	093F	309.62	Capoose
642603	TOP LAKE	28/09/2009	29/08/2022	Mineral	Claim	093F	174.09	Capoose
643103	BUCK 1	29/09/2009	29/08/2022	Mineral	Claim	093F	484.09	Capoose
643104	BUCK 2	29/09/2009	29/08/2022	Mineral	Claim	093F	445.52	Capoose
643106	BUCK 3	29/09/2009	29/08/2022	Mineral	Claim	093F	406.59	Capoose
643107	BUCK 4	29/09/2009	29/08/2022	Mineral	Claim	093F	483.85	Capoose
643108	BUCK 5	29/09/2009	29/08/2022	Mineral	Claim	093F	483.85	Capoose
643109	BUCK 6	29/09/2009	29/08/2022	Mineral	Claim	093F	483.74	Capoose
643110	BUCK 7	29/09/2009	29/08/2022	Mineral	Claim	093F	483.69	Capoose
643123	BUCK 8	29/09/2009	29/08/2022	Mineral	Claim	093F	484.06	Capoose
643323	TOP	29/09/2009	29/08/2022	Mineral	Claim	093F	309.45	Capoose
644244	CAPOOSE M6	30/09/2009	29/08/2022	Mineral	Claim	093F	484.35	Capoose
644283	CAPOOSE M7	30/09/2009	29/08/2022	Mineral	Claim	093F	484.33	Capoose
644285	CAPOOSE M8	30/09/2009	29/08/2022	Mineral	Claim	093F	464.95	Capoose
644323	CAPOOSE M9	30/09/2009	29/08/2022	Mineral	Claim	093F	464.74	Capoose
644363	CAPOOSE M10	30/09/2009	29/08/2022	Mineral	Claim	093F	309.98	Capoose
645063	CAPOOSE M11	30/09/2009	29/08/2022	Mineral	Claim	093F	465.04	Capoose
645064	CAPOOSE M12	30/09/2009	29/08/2022	Mineral	Claim	093F	465.15	Capoose
645065	CAPOOSE M13	30/09/2009	29/08/2022	Mineral	Claim	093F	426.40	Capoose
645066	CAPOOSE M14	30/09/2009	29/08/2022	Mineral	Claim	093F	232.57	Capoose
649243	JAG-8	08/10/2009	29/08/2022	Mineral	Claim	093F	483.05	Capoose
704807	PAWING	26/01/2010	29/08/2022	Mineral	Claim	093F	213.30	Capoose
704817	PAWS	26/01/2010	29/08/2022	Mineral	Claim	093F	213.31	Capoose
704825	FAWN WEST	26/01/2010	29/08/2022	Mineral	Claim	093F	387.18	Capoose
704826	FW 2	26/01/2010	29/08/2022	Mineral	Claim	093F	464.61	Capoose
704827	FW 3	26/01/2010	29/08/2022	Mineral	Claim	093F	406.65	Capoose
704828	FW 4	26/01/2010	29/08/2022	Mineral	Claim	093F	387.38	Capoose
704829	FW 5	26/01/2010	29/08/2022	Mineral	Claim	093F	387.03	Capoose
704830	FW 6	26/01/2010	29/08/2022	Mineral	Claim	093F	193.64	Capoose
704854	FW 7	27/01/2010	29/08/2022	Mineral	Claim	093F	464.57	Capoose
704855	FW 8	27/01/2010	29/08/2022	Mineral	Claim	093F	309.60	Capoose
704863	FW 9	27/01/2010	29/08/2022	Mineral	Claim	093F	445.07	Capoose

Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
706011	FW 10	11/02/2010	29/08/2022	Mineral	Claim	093F	193.74	Capoose
706593	CPN1	19/02/2010	29/08/2022	Mineral	Claim	093F	482.89	Capoose
706594	CPN2	19/02/2010	29/08/2022	Mineral	Claim	093F	482.61	Capoose
706595	CPN3	19/02/2010	29/08/2022	Mineral	Claim	093F	444.03	Capoose
706596	CPN4	19/02/2010	29/08/2022	Mineral	Claim	093F	328.07	Capoose
706597	CPW1	19/02/2010	29/08/2022	Mineral	Claim	093F	484.38	Capoose
706598	CPW2	19/02/2010	29/08/2022	Mineral	Claim	093F	484.28	Capoose
706599	CPW3	19/02/2010	29/08/2022	Mineral	Claim	093F	484.28	Capoose
706600	CPW4	19/02/2010	29/08/2022	Mineral	Claim	093F	484.27	Capoose
706602	CPW5	19/02/2010	29/08/2022	Mineral	Claim	093F	484.04	Capoose
706603	CPW6	19/02/2010	29/08/2022	Mineral	Claim	093F	483.81	Capoose
706605	CPW7	19/02/2010	29/08/2022	Mineral	Claim	093F	483.88	Capoose
706606	CPW8	19/02/2010	29/08/2022	Mineral	Claim	093F	484.00	Capoose
706607	CPW9	19/02/2010	29/08/2022	Mineral	Claim	093F	484.18	Capoose
706608	CPW9	19/02/2010	29/08/2022	Mineral	Claim	093F	484.40	Capoose
706609	CPW10	19/02/2010	29/08/2022	Mineral	Claim	093F	484.51	Capoose
706610	CPW11	19/02/2010	29/08/2022	Mineral	Claim	093F	484.69	Capoose
706612	CPW12	19/02/2010	29/08/2022	Mineral	Claim	093F	484.51	Capoose
706613	CPW13	19/02/2010	29/08/2022	Mineral	Claim	093F	484.55	Capoose
706614	CPW14	19/02/2010	29/08/2022	Mineral	Claim	093F	484.70	Capoose
706615	CPW15	19/02/2010	29/08/2022	Mineral	Claim	093F	484.77	Capoose
706616	CPW16	19/02/2010	29/08/2022	Mineral	Claim	093F	484.97	Capoose
706617	CPW17	19/02/2010	29/08/2022	Mineral	Claim	093F	485.20	Capoose
706618	CPW18	19/02/2010	29/08/2022	Mineral	Claim	093F	485.43	Capoose
706619	CPW19	19/02/2010	29/08/2022	Mineral	Claim	093F	485.43	Capoose
706620	CPW20	19/02/2010	29/08/2022	Mineral	Claim	093F	485.20	Capoose
706621	CPW21	19/02/2010	29/08/2022	Mineral	Claim	093F	484.97	Capoose
706622	CPW22	19/02/2010	29/08/2022	Mineral	Claim	093F	484.92	Capoose
706623	CPW23	19/02/2010	29/08/2022	Mineral	Claim	093F	485.15	Capoose
706625	CPW24	19/02/2010	29/08/2022	Mineral	Claim	093F	485.01	Capoose
706626	CPW25	19/02/2010	29/08/2022	Mineral	Claim	093F	485.25	Capoose
706627	CPW26	19/02/2010	29/08/2022	Mineral	Claim	093F	485.39	Capoose
706628	CPW27	19/02/2010	29/08/2022	Mineral	Claim	093F	485.47	Capoose

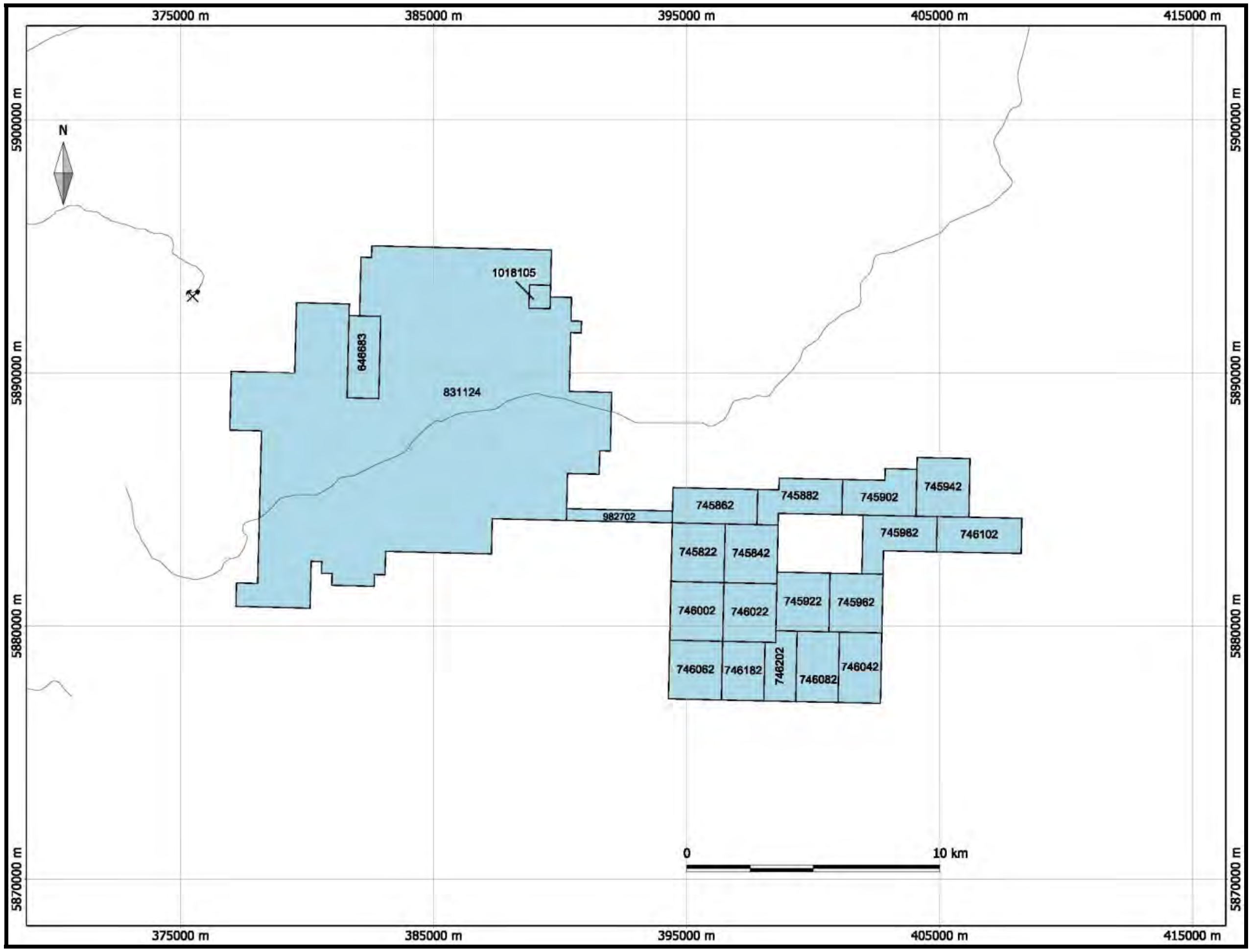
Tenure Number	Claim Name	Issue Date	Good To Date	Tenure Type	Tenure Sub Type	Map Number	Area (ha)	Block
706629	CPNW1	19/02/2010	29/08/2022	Mineral	Claim	093F	290.15	Capoose
706630	CPNW2	19/02/2010	29/08/2022	Mineral	Claim	093F	154.65	Capoose
706638	PAWS 2	19/02/2010	29/08/2022	Mineral	Claim	093F	407.28	Capoose
843656	JOHNNY NORTH	20/01/2011	29/08/2022	Mineral	Claim	093F	213.19	Capoose
843657	JOHNNY NW	20/01/2011	29/08/2022	Mineral	Claim	093F	407.04	Capoose
843658	JOHNNY W	20/01/2011	29/08/2022	Mineral	Claim	093F	310.04	Capoose
564994	KEY 1	24/08/2007	31/08/2020	Mineral	Claim	093F	485.25	Key
564995	KEY 2	24/08/2007	31/08/2020	Mineral	Claim	093F	485.16	Key
564996	KEY 3	24/08/2007	31/08/2020	Mineral	Claim	093F	485.16	Key
564997	KEY 4	24/08/2007	31/08/2020	Mineral	Claim	093F	466.00	Key
564998	KEY 5	24/08/2007	31/08/2020	Mineral	Claim	093F	388.26	Key
564999	KEY 6	24/08/2007	31/08/2020	Mineral	Claim	093F	388.22	Key
565000	KEY 7	24/08/2007	31/08/2020	Mineral	Claim	093F	116.50	Key
565001	KEY 8	24/08/2007	31/08/2020	Mineral	Claim	093F	97.10	Key
589167	LOCK 1	30/07/2008	31/08/2020	Mineral	Claim	093F	485.57	Key
589177	LOCK 2	30/07/2008	31/08/2020	Mineral	Claim	093F	485.80	Key
589183	LOCK 3	30/07/2008	31/08/2020	Mineral	Claim	093F	484.93	Key
589231	LOCK 4	30/07/2008	31/08/2020	Mineral	Claim	093F	485.16	Key
589232	LOCK 5	30/07/2008	31/08/2020	Mineral	Claim	093F	485.39	Key
589234	LOCK 6	30/07/2008	31/08/2020	Mineral	Claim	093F	388.44	Key
589236	LOCK 7	30/07/2008	31/08/2020	Mineral	Claim	093F	466.15	Key
589238	LOCK 8	30/07/2008	31/08/2020	Mineral	Claim	093F	233.08	Key
589241	LOCK 9	30/07/2008	31/08/2020	Mineral	Claim	093F	407.65	Key
589242	LOCK 10	30/07/2008	31/08/2020	Mineral	Claim	093F	465.66	Key
589243	LOCK 11	30/07/2008	31/08/2020	Mineral	Claim	093F	193.97	Key
589244	LOCK 12	30/07/2008	31/08/2020	Mineral	Claim	093F	388.55	Key
642003	YELLOW & BLACK	27/09/2009	31/08/2020	Mineral	Claim	093F	174.86	Key
642004	BLACK & YELLOW	27/09/2009	31/08/2020	Mineral	Claim	093F	174.86	Key
642023	BLACK	27/09/2009	31/08/2020	Mineral	Claim	093F	388.61	Key
642024	YELLOW	27/09/2009	31/08/2020	Mineral	Claim	093F	233.20	Key

Figure 4-4: Capoose Mineral Claims



Note: Figure courtesy New Gold, 2014.

Figure 4-5: Auro Mineral Claims



Note: Figure courtesy New Gold, 2013

No Auro claims are known to overlap any legacy or Crown Granted mineral claims. One claim partially overlaps a no staking reserve.

4.4.4 Key Claims

The Key claim block comprises 24 mineral claims immediately south of the Blackwater and west of the Auro claim blocks (Figure 4-7). The claim block totals 8,854 hectares. All Key claims are 100% held in the name of New Gold. The Key mineral claims are summarized in Table 4-1.

All claims in the Key claim block are cell claims with no predecessors. All claims have a common expiry date of August 31, 2020.

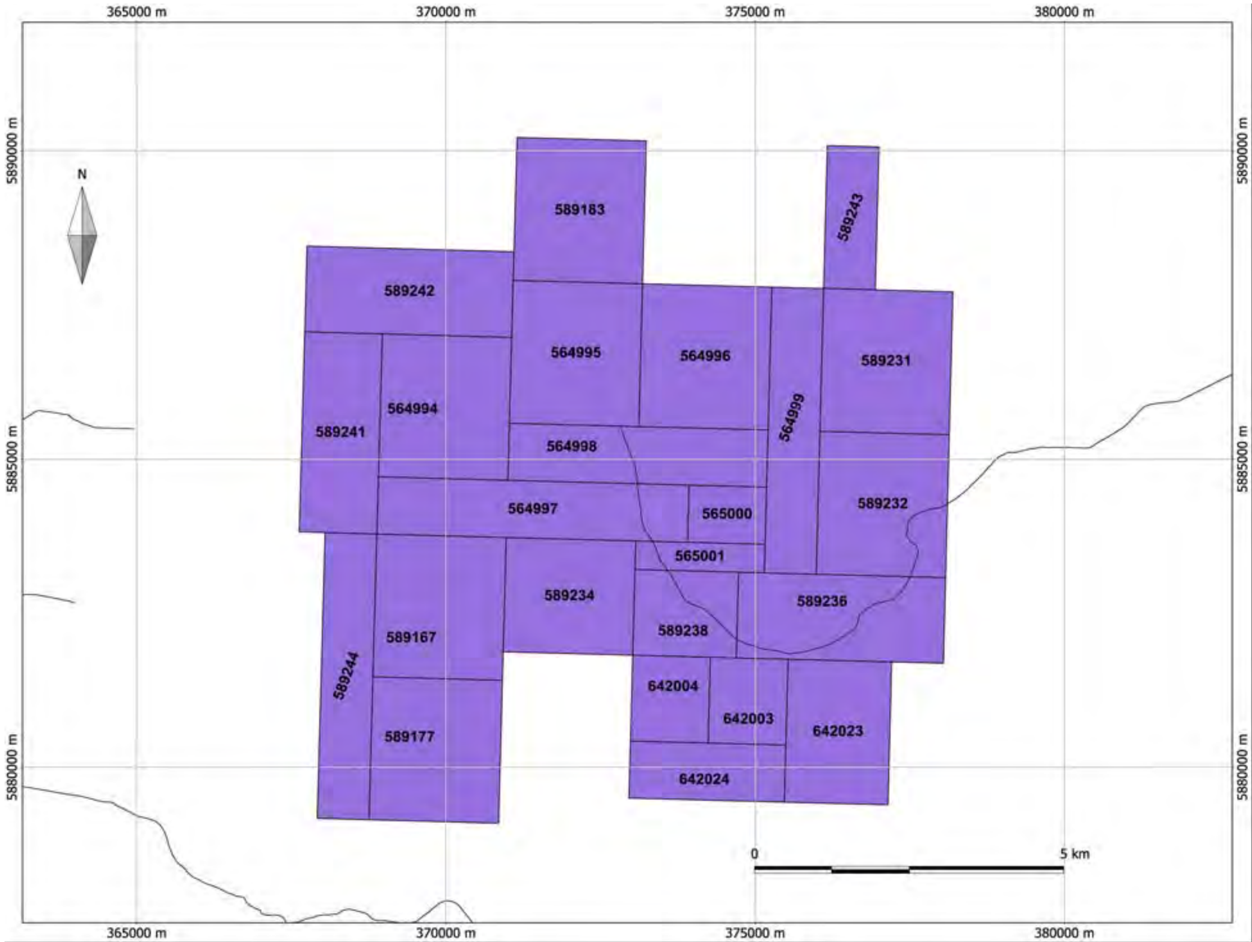
No Key claims are known to overlap any legacy or Crown Granted mineral claims. No Key claims overlap any mineral reserves or parks.

4.5 Surface Rights

New Gold, as the holder of mineral claims (or of a mining lease, once it is obtained), does not have exclusive possession of the surface or exclusive right to use the surface of those lands. However, the holder of a mineral claim or a mining lease does have the right to access those lands for the purpose of exploring for minerals and to use the surface for mining activities (exploration, development, and production) and there is no legal requirement to obtain a surface lease (issued pursuant to the *Land Act*) or other surface tenure to undertake such activities within the area covered by New Gold's claim blocks. Therefore, any mine infrastructure does not require additional land tenures if it is located on a mining lease and the surface is owned by the Crown. In the case of the Project, the Crown owns the surface in the area of the proposed mine site. If New Gold requires exclusive possession of certain areas, then additional rights under the *Land Act*, such as a surface lease or acquisition of the fee simple may be required.

Once New Gold holds a mining lease that is on unreserved land owned by the government (if it is not lawfully occupied for a purpose other than for mining, and is not protected heritage property), then New Gold could apply for certification by the Minister of Mines and Energy that the surface rights over that area are required by New Gold for the purposes of a mining activity. If that certification is made, then New Gold will have the right to obtain a surface tenure under the *Land Act* on the terms and conditions set by the minister responsible for the *Land Act*.

Figure 4-6: Key Mineral Claims



Note: Figure courtesy New Gold, 2014

In addition, the Project will require the construction and operation of certain infrastructure, parts of which are to be located outside of the Property. The key components of this infrastructure are an airstrip, an airstrip access road, a mine access road, a water pipeline, and an electric transmission line (the Linear Infrastructure). As currently planned, the proposed routes and locations of this infrastructure appear to be located almost entirely on provincial Crown land. As such, the Linear Infrastructure will require New Gold to obtain additional surface rights from the Crown. In most cases, a surface tenure under the *Land Act* will be required. There are various types of tenures that can be obtained, including temporary permits, licences, leases, rights of way, and fee simple interests. The type of surface tenure desired for each component will be determined on a case by case basis.

Because of the size of the Project, before it can proceed to permitting and production it must first successfully undergo environmental assessment under both the *BC Environmental Assessment Act* and the *Canadian Environmental Assessment Act, 2012*. Final permitting for construction and operation will not occur until a positive decision is made under each of these processes. As a result, New Gold has not yet commenced the process to obtain certain permits, approvals and rights that it will ultimately require for the Project, including certain surface tenures under the *Land Act*, and a provincial *Mines Act* permit to construct the Project and enter production. The Project will require additional provincial and federal authorizations and permits, based on the characteristics of the Project. Environmental assessment and Project permitting is addressed in section 20 of this Report.

A review of overlapping surface rights in the vicinity of the Property was undertaken in December 2013 and January, 2014. The review utilized searches of the MTO system, the Integrated Land and Resources Registry (ILRR) system, and the Land Title Office, all of which are maintained by the Government of British Columbia. The majority of the Blackwater mineral claims comprising the Property are located on Crown lands. The review identified an overlapping private parcel, land reserves/notations, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitter areas, and an Ungulate Winter Range.

A review of surface rights in the vicinity of the Linear Infrastructure was undertaken in December, 2013. The review utilized searches of the MTO system, the Integrated Land and Resources Registry (ILRR) system, and the Land Title Office, all of which are maintained by the Government of British Columbia. This review identified private parcels; a Land Act licence, rights of way, reserves/notations and a transfer of administration/control area; grazing tenures; forest tenures; a forest recreation site; traplines; guide outfitter areas; a wildlife management area; an agriculture land reserve; and third-party mineral tenures overlapping or in close proximity to the proposed electrical transmission line route. The review also identified grazing tenures, forest tenures, traplines, and guide outfitter areas overlapping all elements of the Linear Infrastructure; a forest recreation site overlapping the proposed water pipeline route; and third party mineral tenures overlapping the airstrip, airstrip access road, and the water pipeline route.

The review found no Indian reserves, provincial or federal parks, Ecological Reserves, Protected Areas, Wildlife Habitat Areas, placer mineral tenures, coal tenures, geothermal resource tenures, petroleum and natural gas tenures, or Land Act agreements, inclusions, inventories, leases, permits or permissions, overlapping the Property or the Linear Infrastructure.

4.6 Royalties and Encumbrances

New Gold's 100% interest in the Property is subject to four NSR agreements. No other material encumbrances that are recorded against the Blackwater claims and are still active have been identified. Terms of the NSR agreements relating to the Property are described in Section 4.6.1. Terms of the agreements relating to the Capoose, Auro and Key claim blocks are provided in Sections 4.6.2, 4.6.3 and 4.6.4 are for reference only and do not impact the 2013 Feasibility Study or the Project.

4.6.1 Blackwater Claim Block Agreements and Encumbrances

Dave Option

New Gold, through Richfield, acquired a 100% interest in mineral claim 515809 (Dave Claim) obtained in 2010 from five individuals. The optionors retained a 2.5% NSR. New Gold has purchased 40% of the Dave NSR royalty for $1,000,000, and a 1.5% NSR royalty remains. The claim covers a portion of the Blackwater deposit.

Jarrit Option

New Gold, through Richfield, acquired a 100% interest in mineral claim 515810 (Jarrit Claim) obtained in 2011 from four individuals. The optionors retained a 2% NSR royalty. New Gold has purchased half of the Jarrit NSR royalty for $1,200,000, and a 1% NSR royalty remains. The claim covers a portion of the Blackwater deposit.

JR Option

In January 2011, Richfield acquired an option to earn a 100% interest in the JR Claims, 637203, 637205, and 637206, from the same optionors as the Jarrit Option. The optionors retained a 3% NSR royalty. The terms of the agreement transferred to New Gold upon its acquisition of Richfield. New Gold exercised the option to acquire a 100% interest in the JR Claims in January 2013. New Gold may purchase two-thirds of the JR Claims NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain.

PS Claim

New Gold, through Geo, acquired a 100% interest in mineral claim 835014, which had been obtained in 2011 from a third-party individual. The third-party individual retained a 2% NSR royalty, of which New Gold may purchase half for $1,000,000.

4.6.2 Capoose Claim Block Agreements and Encumbrances

JAG Option

In December 2011, Silver Quest exercised an option to earn a 100% interest in the JAG Option claims, 534364, 534365, 534366, 534367, 557495, 557496, 552497 and 649243, from a third-party individual. New Gold inherited the ownership and terms of the agreement with its acquisition of Silver Quest. The optionor retained a 2% NSR royalty, of which New Gold may purchase half for $1,000,000. New Gold has an obligation to pay an advance royalty to the optionor of $30,000 per annum, to be credited against the NSR royalty.

Emma Option

In November 2012, New Gold was granted an option to earn a 100% interest in mineral claim 637643 (Emma Claim) from a private corporation. New Gold has until November 2016 to exercise the option by paying the optionor a total of $1,500,000 and satisfying a $500,000 work commitment. On exercise by New Gold, the optionor will retain a 1.2% NSR royalty, of which New Gold may purchase half (0.6% NSR) for $750,000.

Buck Option

New Gold, through Silver Quest, acquired a 100% interest in the Buck Option claims, 643103, 643104, 643106, 643107, 643108, 643109, 643110, and 643123, obtained in December 2011from Paget Minerals Corporation. The optionor retains a 1.5% NSR royalty. New Gold may purchase two-thirds of the Buck NSR royalty for $2,000,000, such that a 0.5% NSR royalty would remain.

Capoose Property Option

New Gold, through Silver Quest, acquired a 100% interest in the following claims, 641983, 641984, 704825, 704826, 704827, 704828, 704829, 704830, 704854, 704855, 704863, 706011, 642544, 642564, 642565, 642583, 553489, 704807, 704817, and 706638, obtained prior to December 19, 2011 from an individual. The optionor retained a 1.5% NSR royalty. New Gold may purchase two-thirds of the NSR royalty for $1,300,000.

Capoose Option and Joint Venture

New Gold, through Silver Quest, acquired a 100% interest in claim no. 512838, 60% of which was obtained from Bearclaw Capital Corp. in 2009, and the remaining 40% of which was obtained from Bearclaw by November 2010. Through an addendum to the original agreement, Silver Quest's mineral claim nos. 552493, 552494, 552495, 564372, 564373, 564375, 564376 and 564377 became part of the subject property upon Silver Quest's exercise of the 60% option in October 2009. The optionor retained a 2.25% NSR royalty on the entire property. There is a buy-down right contained in the agreement which may entitle New Gold to purchase four-ninths of the NSR royalty for $1,500,000, such that a 1.25% NSR royalty would remain.

4.6.3 Auro Claim Block Agreements and Encumbrances

In March 2012, New Gold acquired a 100% interest in the Auro Properties (claims 646683, 745822, 745842, 745862, 745882, 745902, 745922, 745942, 745962, 745982, 746002, 746022, 746042, 746062, 746082, 746102, 746182, 746202, and 831124) from Gold Reach. The vendor retained a 2% NSR royalty with no buy-down provision. New Gold is required to spend $1,500,000 on exploration by December 2014 to advance the properties.

4.6.4 Key Claim Block Agreements and Encumbrances

Key Agreement

In December 2013, New Gold purchased a 100% interest in the Key Property (claims 564994, 564995, 564996, 564997, 564998, 564999, 565000, 565001, 589167, 589177, 589183, 589231, 589232, 589234, 589236, 589238, 589241, 589242, 589243, 589244, 642003, 642004, 642023, and 642024) from Troymet Exploration Corporation. Troymet retained a 2% NSR royalty. New Gold may purchase half of the Key NSR royalty for $2,000,000, such that a 1% NSR royalty would remain. New Gold is required to spend $1,500,000 on the Key Property on or before year end 2018, with a minimum of $500,000 of said expenditures to occur by year end 2014.

4.6.5 Underlying Agreements

In October 2010, Troymet acquired a 100% interest in four claims (642003, 642004, 642023, and 642024) comprising part of the Key property from a third-party individual.

The third-party individual retained a 2% NSR royalty, three-quarters of which (1.5% NSR) can be purchased at any time for $750,000. New Gold confirmed this royalty with its December 2013 acquisition of Troymet's Key property.

In August 2007, Troymet acquired a 100% interest in 20 claims (564994, 564995, 564996, 564997, 564998, 564999, 565000, 565001, 589167, 589177, 589183, 589231, 589232, 589234, 589236, 589238, 589241, 589242, 589243, and 589244) comprising part of the Key Property. These claims are subject to a 3% NSR royalty in favour of an individual. Two-thirds of the royalty (2% NSR) may be purchased for $1,000,000 in cash or stock at any time. New Gold confirmed this royalty with its December 2013 acquisition of Troymet's Key property.

4.7 Environment, Environmental Liabilities and Social Licence

The Project is currently subject to liabilities limited to exploration activities conducted since 1973 by New Gold, its predecessors, and previous explorers. New Gold has posted a reclamation security sufficient to reclaim the site in accordance with British Columbia *Mines Act*.

Environmental and social base line studies and potential environmental and social effects of development and operation of the Blackwater Project are discussed in Section 20 of this Report.

4.8 Permits

The status of current exploration permits required for the Blackwater Project, the approvals required under the British Columbia *Environmental Assessment Act* (BCEAA) and the *Canadian Environmental Assessment Act, 2012* (CEAA 2012), and a summary of communities and aboriginal groups consultation and approvals and permits required to construct and operate the Project are provided in Section 20 of this report.

4.9 Comments on Section 4

In the opinion of the QPs, the information discussed in this section supports the declaration of Mineral Resources and Mineral Reserves, based on the following:

- Information from legal experts and New Gold experts provided to AMEC indicated that New Gold holds 100% of the Property
- Information from legal experts supports that the mining tenure comprising the Property is valid and is sufficient to support declaration of Mineral Resources and Mineral Reserves

- Information from legal experts noted that the majority of the Property is located on Crown lands. Surface rights in the vicinity of proposed electrical transmission lines, water pipeline, airstrip and access roads were also reviewed. The review identified overlapping private parcels, a Land Act licence, rights of way, land reserves/notations, a transfer of administration/control area, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitter areas, a wildlife management area, an agricultural land reserve and an ungulate winter range were associated with the Property or the Linear Infrastructure or both
- New Gold's 100% interest in the Property is subject to four net smelter return (NSR) agreements
- New Gold will need to apply for additional permits as appropriate under local, Provincial, and Federal laws to allow mining operations
- Notwithstanding the information contained above in this Section 4, there is no guarantee that title to any of the Property will not be challenged or impaired, and third parties may have valid claims affecting the Property, including prior unregistered liens, agreements, transfers or claims, including aboriginal land claims, and title may be affected by, among other things, undetected defects. As a result, there remains a risk that there may be future constraints on New Gold's ability to operate the Property or New Gold may be unable to enforce rights with respect to the Property.

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

5.1 Accessibility

The Project site is readily accessible by vehicle from the Kluskus-Ootsa Forest Service Road originating approximately 100 km west of Prince George and 10 km west of Vanderhoof, off Highway 16 (Figure 5-1). At kilometre 146.5 along the Kluskus-Ootsa road, an 18 km mine road built in 1986 by Granges and improved by Richfield provides direct access to the property and camp location. Driving time from Vanderhoof to the property is about 2.5 hours.

Helicopter access is from bases in Vanderhoof, Prince George, or Quesnel.

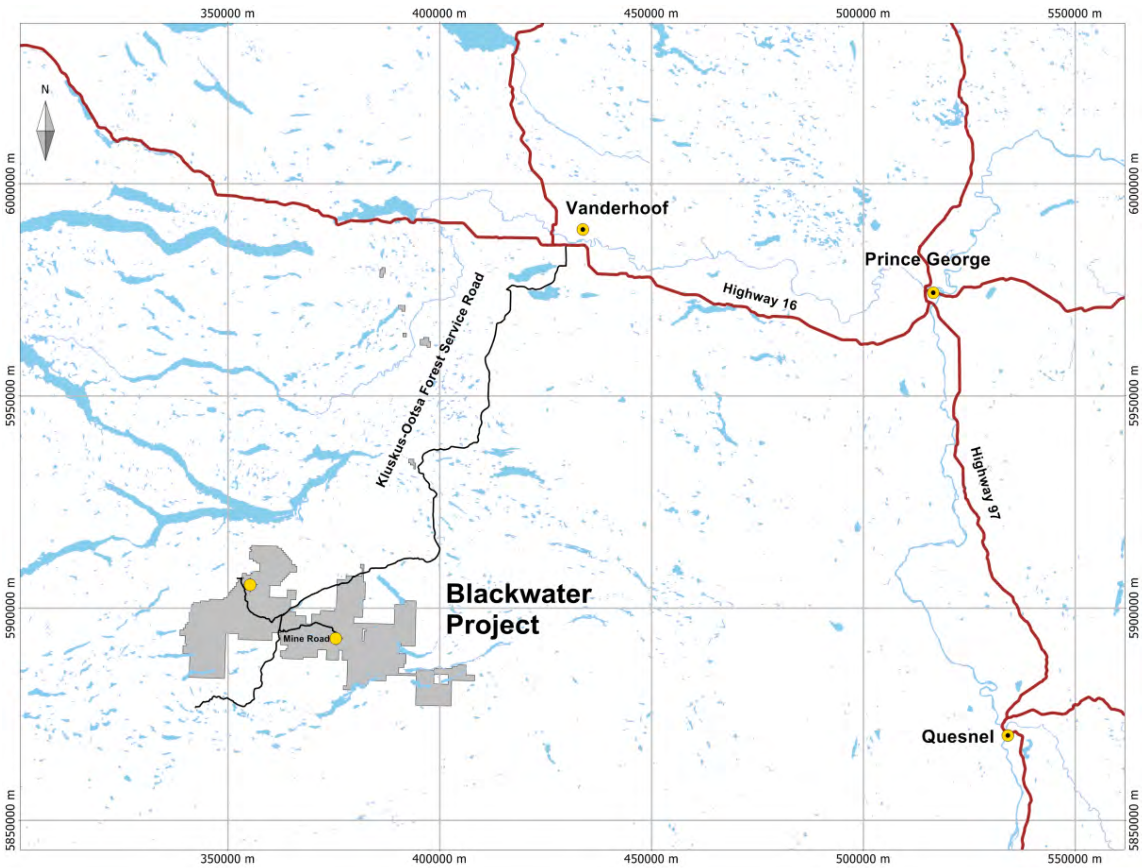
5.2 Climate

The climate in the Project area is sub-continental, characterized by brief warm summers and long cold winters resulting from the influence of cold arctic air. The climate is also influenced by moisture-laden weather systems moving east by way of the low Kitimat Ranges. Temperatures range from a minimum of -40°C in winter to a maximum of 32°C in summer. The mean annual precipitation for the site is estimated to be 636 mm with 49% falling as rain and 51% as snow. The weather allows for a reasonable mapping, prospecting and geochemical sampling season. Except in the case of extreme situations drilling may be conducted year round. The weather is not expected to present any unusual difficulties for mining operations.

5.3 Local Resources and Infrastructure

The Project area is very sparsely inhabited; three ranches are found within a 20 km radius of the Project site. Some services are available in Vanderhoof, but Prince George is the regional hub with air service from major centres.

Figure 5-1: Project Access



Note: Figure courtesy New Gold, 2013

There is no grid-connected power in the direct vicinity of the Project. The main BC Hydro 500 kV transmission lines supplying western B.C. are approximately 100 km to the north. Several interconnection points from the 500 kV lines to existing 230 kV substations and transmission lines are possible in an area between Fraser Lake and Vanderhoof. Power for the current Blackwater exploration camp is provided by generators.

Personnel to support development and operation of the mine can be drawn from British Columbia's well-developed mining industry.

The deposit is located on the north slope of Mt. Davidson, and the proposed Project infrastructure including the mill, waste and tailings storage will be sited predominantly in the Davidson Creek watershed.

Precipitation run-off, ground water from pit dewatering, and supplemental fresh water from a nearby lake are water sources for mineral processing. A ground water well field will supply fresh water for the camp.

Infrastructure required for Project development and operation is discussed in more detail in Section 18 of this Report.

5.4 Physiography

The elevation of the Blackwater Project ranges from just over 1,000 m in low-lying areas northeast of the proposed mine site to 1,800 m at the summit of Mt. Davidson on the southwest side of the property.

Outcrop on the property is limited. Most of the area is covered with thick glacial deposits of 2 m or more, except for the upper 150 m of Mt. Davidson. The slopes of Mt. Davidson are covered by till and colluvium, but the claim area is underlain by lodgement and melt-out till deposits.

The study area falls within the Fraser Plateau biogeoclimatic region and more specifically within the Nazsko Upland subregion. The Fraser Plateau biogeoclimatic region is a broad, rolling plateau of basaltic lava flows that was entirely overridden by ice sheets moving eastward in the north and northward in the south during the last glaciation. The most prominent ice flow directions determined from glacial fluting are northeast. The Fraser Plateau region contains ten subregions, with only the Nazko Upland occurring within the Project area. Five biogeoclimatic (BGC) units represent the Project area, as follows:

- Sub-Boreal Spruce Moist Cold Babine Variant
- Sub-Boreal Spruce Moist Cold Kluskus Variant
- Engelmann Spruce – Subalpine Fir Moist Very Cold Nechako Variant
- Engelmann Spruce – Subalpine Fir Moist Very Cold Parkland
- Boreal Altai Fescue Alpine Undifferentiated Subzone.

The Nazko Upland subregion is an undulating upland area that contains the Fawnie and Nechako ranges near the study area in the north-northeast. Low-elevation valley bottoms are dominated by stands of lodgepole pine. Hybrid white spruce tends to dominate on moist to wet sites below 1,500 m, while subalpine fir and Englemann spruce are dominant above 1,500 m. Lodgepole pine is a major species on dry, fire-prone sites at most elevations. The recent pine beetle epidemic has infested almost all of the lodgepole pine forests within this subregion. The Nazko Upland subregion also contains an extensive network of lakes, rivers, and wetland complexes.

Atmospheric heating of these water bodies can result in convective activity and sporadic summer showers.

5.5 Regional Seismicity

A seismicity assessment was carried out for the Project, including a review of the regional seismicity and a probabilistic seismic hazard analysis, to provide seismic design parameters for the TSF and other facilities at the Project site, including mine waste dumps and water management dams. Design ground motion parameters provided by the seismic hazard analysis include peak ground acceleration, spectral acceleration (defining the uniform hazard spectrum), and earthquake magnitude.

The Project is situated within central B.C., where the level of recorded historical seismic activity has been low. The maximum earthquake magnitude for this region of central B.C. is estimated to be about magnitude 7.0, based on historical earthquake data and the regional tectonics (Adams and Halchuk, 2003). There is potential for larger earthquakes of up to about magnitude 7.5 along the coastal region of mainland western B.C., including the Coast Mountains.

The high seismicity offshore and west of the Project site is associated with the Queen Charlotte fault system, which defines the boundary between the Pacific and North American tectonic plates. There is a potential for earthquakes of up to about magnitude 8.5 along this fault system (Adams and Halchuk, 2003). The amplitude of ground motions experienced at the Project site from earthquakes along this fault system would be very low due to attenuation over such a large distance, with a peak ground acceleration (PGA) expected to be less than 0.05 *g*.

The seismic hazard along the southwest coast of B.C. is significant because of subduction zone earthquakes along offshore faults and within the subducting oceanic tectonic plate (Juan de Fuca plate). There is potential for very large earthquakes of magnitude 8.0 to 9.0+ along this Cascadia subduction zone. The amplitude of ground motions experienced at the site are expected to have been around 0.05 *g*, based on estimates of the last significant earthquake, which occurred 400 km to the south, about 300 years ago.

There is also potential for intraslab (inslab) earthquakes, to occur deep within the subducted Juan de Fuca plate. These events, which have potential to be as large as magnitude 7.0 to 7.5, would likely be more than 300 km to the southwest, at a depth of over 35 km. Ground motions experienced at the Project site for this type of subduction earthquake are also expected to be less than 0.05 *g*.

A design earthquake magnitude 8.5 has been selected for earthquake return periods of 500, 5,000, and 10,000 years, based on the review of regional tectonics and historical seismicity, and the findings of deaggregation of the probabilistic seismic hazard. This represents large magnitude earthquakes along the Queen Charlotte fault system and Cascadia subduction zone. The potential for shallow crustal earthquakes closer to the Project site was also considered for longer return period events of 5,000 and 10,000 years, representing earthquakes of up to about magnitude 7.5 along Coastal B.C.

5.6 Comments on Section 5

In the opinion of the QP:

- Mining activities should be capable of being conducted year-round
- There is sufficient suitable land available for any future tailings disposal, mine waste disposal, and related mine infrastructure within the mineral claims.

6.0 HISTORY

Mineralization on the property was discovered in 1973 during a regional silt geochemical survey by Granges. The survey located anomalous zinc and other metals in stream sediments east and north of the Project area. Between 1973 and 1985 a range of ground and airborne geophysical and geochemical surveys was conducted to locate and delimit mineralization.

Drilling began in 1985 and continued to 1994 completing a total of 6,300 m in 75 holes. A further 1,333 m of drilling in seven holes was completed by Silver Quest Resources Ltd. in the winter of 2005–2006. The focus of the work during this period was on high-grade vein deposits, which at the time were considered the only known gold–silver targets in the Nechako Plateau.

Richfield began work on the Project in 2009 when the company recognized the bulk gold potential on the property. In 2009, Richfield optioned the Davidson Claims from Silver Quest and the Dave Claim and Jarrit Claim from the Rozek family. Richfield subsequently completed all of the earn-in requirements for these agreements, resulting in its holding a 75% interest in the Davidson Claims and 100% interests in each of the Dave and Jarrit Claims.

In June 2011, New Gold acquired all of the issued and outstanding common shares of Richfield pursuant to a court-approved plan of arrangement. In December 2011, New Gold acquired all of the issued and outstanding common shares of Silver Quest and a third company, Geo Minerals Limited. New Gold subsequently amalgamated with Richfield, Geo and Silver Quest effective January 1, 2012 and became the direct operator of the Project.

Exploration work completed by all operators is summarized in Table 6-1.

Table 6-1: Work History

Year	Operator	Work
1973	Granges	Regional silt survey located anomalous silver, zinc, and lead in the Mt. Davidson area. This was followed by a wide-spaced soil survey northeast of Mt. Davidson.
1976	Granges	Soil sample and ground magnetometer surveys to follow up 1973 soil results.
1977	Granges	Pem claim staked covering most of the presently defined mineral deposit. Pulse EM survey on the Pem claim (12.5 km)
1979	Granges	Vector Pulse EM survey on the Pem claim (7 km).
1981	Granges	Helicopter EM and magnetometer survey.
1981	Granges	Horizontal Loop EM survey on the Deb #1 claim.
1981	Granges	Reconnaissance mapping of the Mt. Davidson area.
1982	Granges	Soil geochemistry (220 samples) and ground magnetometer survey (20.8 line km) on the Pem claim.
1983	Granges	Hammer seismic survey.
1984	Granges	Hand-trenching (30 trenches for total of 66 m) and VLF survey (4.8 line km) on the Pem claim. Only 1 trench intersected bedrock.
1985	Granges	Winkie drilling (8 holes for total of 507 m) on the Pem claim. Holes DAV 1-8.
1986	Granges	Construction of access road from km 146.5 on the Kluskus Haulage road, east 18 km to the Pem grid.
1986	Granges	Percussion drilling (34 holes for a total of 1,524 m) on the Pem claim. RC 1-34.
1987	Granges	Diamond drilling (23 holes for total of 2,617 m) on the Pem claim. Holes DAV 9-31.
1992	Granges	Line Cutting (58.8km); collection of 955 soil samples and 35, stream silt samples; geological mapping over 6000 hectares at a scale of 1:10,000; 50 km of geophysical surveys (IP, mag, VLF); , and diamond drilling of 5 holes BD92-32 to BD92-36, for total of 785 m.
1994	Granges	Linecutting (48.2 km); 29 rock, samples, 1598 soils samples, 23 silt samples and 4 lake sediments were collected; Dighem airborne geophysical survey (881 line km of EM, Mag and radiometrics); 20km IP survey over the Dave claim; Diamond drilling (5 holes for total of 761.68 m). Holes DAV 94-37 to DAV 94-41.
1997	Kennecott Canada	4 km of linecutting and an Induced Polarization survey on the Dave claim.
2005	Silver Quest	Diamond drilling (5 holes for total of 939 m). Holes DAV05-01 to DAV05-05.
2006	Silver Quest	Diamond drilling (2 holes for total of 394 m). Holes DAV06-06 and DAV06-07.
2007	Silver Quest	Soil sampling program, 335 samples.
2009	Richfield	Diamond drilling (18 holes for total of 3,621 m) BW0042 to BW0059
2010	Richfield	Diamond drilling (57 holes for total of 21,336 m). Holes BW0060 to BW0116.
2011	Richfield	Diamond drilling (59 holes for total of 19,727 m). Holes BW0117 to BW0175.
2011	New Gold	Diamond drilling (125 holes for total of 49,316 m). Holes BW0176 TO BW0295,297,298 (+50R). Additional 7 metallurgical test holes for a total of 2,282 m.
2012	New Gold	Diamond drilling (716 holes for total of 207,333 m). Holes BW0299 to BW1013, 296. In addition: 13 geotechnical holes for a total of 5,003 m 20 metallurgical test holes for a total of 1,816 m, 14 waste rock characterisation holes for a total of 2,952 m, and 7 holes drilled for hydrological monitoring (pilot holes) for a total of 2,265 m.
2013	New Gold	Diamond drilling (1 hole for total of 420 m). Hole BW1014. Additional 3 holes drilled for hydrological monitoring (pilot holes) for a total of 1,146 m.

Seven mineral resource estimates have been prepared for the Blackwater Gold Project since March 2011.

- 2 March 2011: Initial mineral resource estimate for Richfield Ventures
- 1 September 2011: Resource update
- 31 December 2011: Resource update
- 7 March 2012: Resource update
- 18 July 2012: Resource update
- 31 December 2012: Resource update
- 31 March 2013: Resource update

New Gold disclosed results of a Preliminary Economic Analysis (PEA) on the Blackwater Project in September 2012. A list of the technical reports prepared on the Project are included in Section 2.6 of this Report.

The March 2013 resource update documented in this report forms the basis for the 2013 Feasibility Study. This estimate includes data for all drilling completed by Richfield and New Gold between August 1, 2009, and January 16, 2013.

No production has occurred from the Project area.

7.0 GEOLOGICAL SETTING AND MINERALIZATION

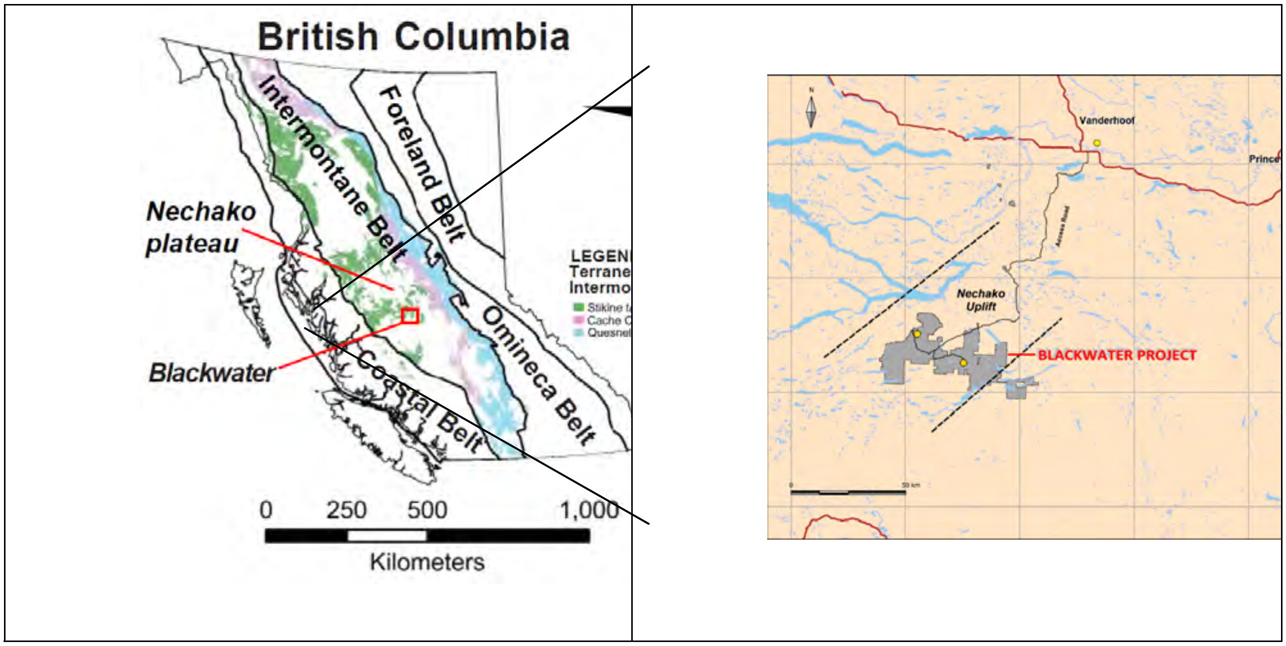
The Blackwater Project is located on the Nechako Plateau near the geographic centre of British Columbia. The plateau is part of the Intermontane Belt superterrane situated between the Coast Belt to the west and the Omineca Belt to the east (Figure 7-1). Topographic relief for the plateau is moderate with elevations ranging from 1,000 to 1,800 m above sea level. The Intermontane Belt consists of an assemblage of three accreted tectonostratigraphic terranes: Stikine, Cache Creek and Quesnel (Riddell, 2011). The Project area is underlain by rocks of the Stikine terrane, comprising an assemblage of magmatic arc and related sedimentary rocks that span Jurassic to early Tertiary time. These rocks have been exposed within an easterly-trending structural high termed the Nechako uplift.

The Nechako uplift is bounded to the north and south by the northeast-striking Natalkuz and Blackwater faults, respectively (Diakow and Levson, 1997; Diakow et al., 1997). The latest extensional displacement along these faults juxtaposes older Mesozoic and Tertiary rocks in the central part of the uplift against younger Cretaceous and Tertiary volcanic rocks to the north and south (Diakow and Webster, 1994; Diakow and Levson, 1997; Friedman et al., 2001). Though the Natalkuz and Blackwater faults are poorly defined due to scarce bedrock exposures, a feature characteristic of the Nechako Plateau in general, strong linear trends marking the traces of these structures are evident in the available gravity and airborne magnetics data for the region. The eastern and western limits of the uplift are not clearly defined by current geologic mapping coverage. The northwesterly-trending Chedakuz fault and adjacent Nechako range transect the uplift and mark the eastern limit of the Project area. To the west the Nechako uplift extends into a provincial park that is well beyond the area currently being explored by New Gold.

7.1 Local and Property Geology

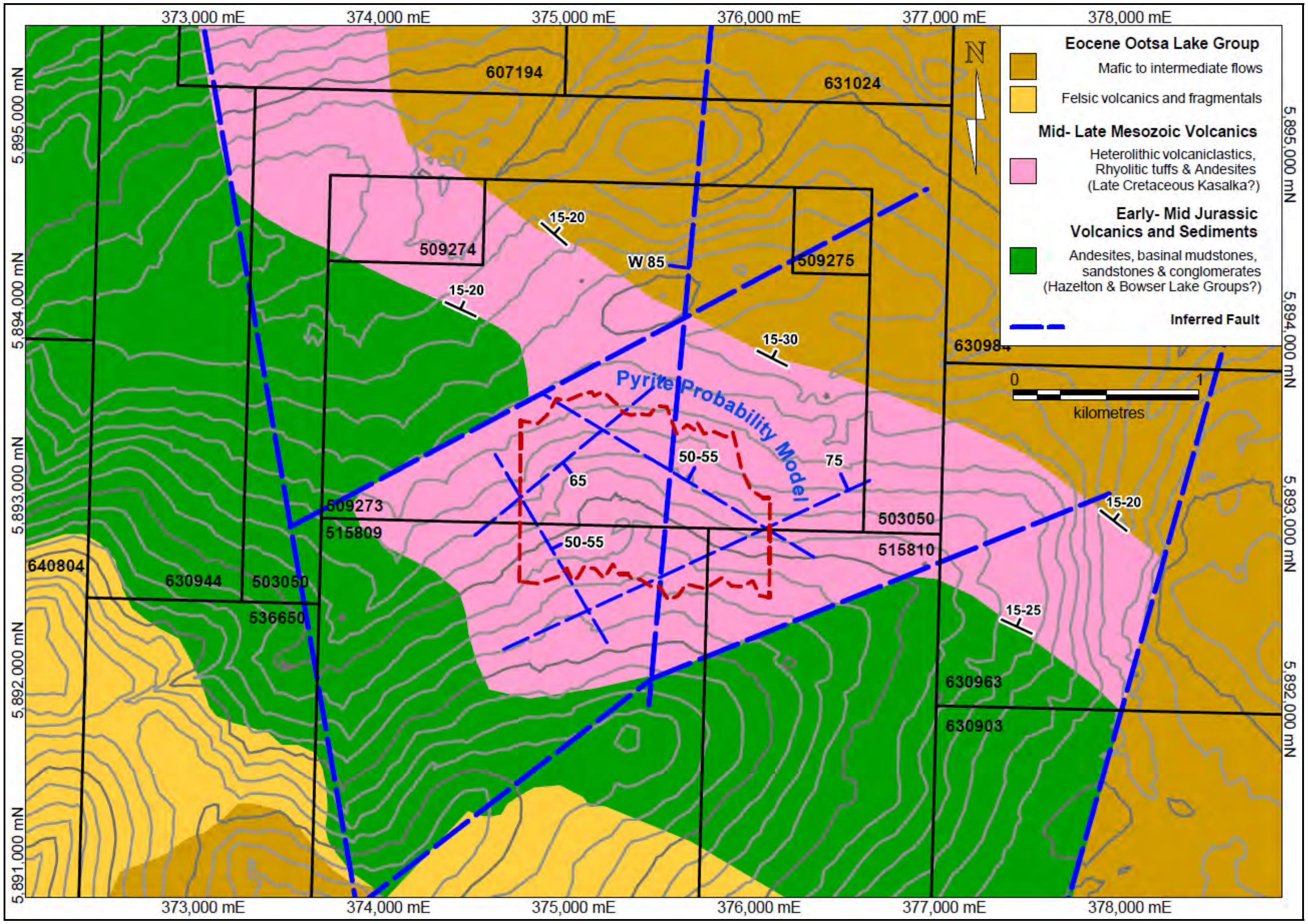
Quaternary glacial overburden, colluvial, and fluvial deposits mask the majority of bedrock within the Project area. Outcrop is sparse and limited peak and ridges and streams draining of the flanks of Mt. Davidson. Property geology is based on interpretations derived from observations and interpretation of geologic field mapping by New Gold in conjunction with core and reverse circulation drilling data collected between 2009 and 2013. Figure 7-2 is a sketch map of the top-of-bedrock geology for the Blackwater Project development area. The red dashed line shown in Figure 7-2 delineates the outer limits of the pyrite probability shell generated for the mineral resource estimate discussed in Section 14.

Figure 7-1: Blackwater Project Location and Tectono-Stratigraphic Setting



Note: Figure courtesy New Gold, 2013

Figure 7-2: Top of Bedrock Geology in Vicinity of Blackwater Deposit



Note: Red dashed line delineates outer limits of mineral resource discussed in Section 14. Figure courtesy New Gold, 2013.

The Project site is underlain by a sequence of volcanic units consisting of heterolithic breccias, rhyolitic tuff, and andesite. The local volcanic section is further subdivided as follows: a lower sequence of andesite, felsic volcaniclastic rock, heterolithic breccias, and tuff, which host the Blackwater deposit, and an upper sequence of post-mineral Eocene age felsic volcanic and fragmental rocks and mafic to intermediate flows belonging to the Ootsa Lake Group. The felsic volcaniclastic rocks and tuff of the lower sequence are late Cretaceous in age based on U-Pb geochronologic dating of zircons which yielded ages ranging from 72.4 ± 1.0 Ma and 74.1 ± 2.2 Ma (Mortensen, 2011). The adjacent andesites have been interpreted to conformably underlie the felsic volcaniclastic rocks and thus belong to the late Cretaceous Kasalka Group.

Additional work is required to fully constrain the age of the andesite in the lower volcanic sequence.

Together the lower and upper volcanic sequences comprise a gently northeasterly dipping section underlain by basinal mudstones, fine sandstones, and conglomerates interpreted as belonging to the late Jurassic Bowser Lake Group. These units are cross-cut by well-developed systems of northeast-, northwest- , and northerly-striking faults that define a polygonal structural fracture pattern at all scales.

Host rocks within the Blackwater deposit area are pervasively hydrofractured, pyritized, and altered to a mixture of silica and sericite. Locally the amount of silica introduced through hydrofracturing and silicification may affect 25% or more of the total volume of altered host rocks. At the deposit scale, brittle-style tectonic deformation affects all rock units. Interpretation and correlation of clearly recognizable faults are made difficult by the intense hydrofracturing and multiple fault sets. Instead, extensive zones of broken rocks cross-cut the mineralized zone and grade laterally into unbroken rock with no obvious bounding fault surfaces.

Within the Blackwater deposit and surrounding area, the Kasalka volcanics commonly contain dark reddish-brown garnet crystal fragments up to a centimetre in diameter as an accessory in the heterolithic breccias, locally making up 1% to 2% of the rock. XRF data on the garnets indicate they are Mn-rich spessartine.

Outcrops of massive felsic lapilli tuff assigned to the Ootsa Lake Group are found along the uppermost elevations of Mt. Davidson to the south of the deposit. The Ootsa rocks comprise felsic and andesitic units that are distinguished from those hosting the Blackwater deposit by their darker gray colour, larger lithic clasts, plagioclase phyric content and the presence of fresh, black, stubby euhedral doubly-terminated quartz crystals up to 1 mm across, which commonly make up a few percent of the rock.

The lithological codes used in the Blackwater drill hole database have been defined according to observed descriptive criteria only. The codes do not include assignment of individual rock units to formally defined regional stratigraphic units. The lithological codes are summarized in Table 7-1.

7.2 Structure

Mineralization is strongly controlled by northwest–southeast-trending structures characterized by zones of tectonic brecciation and chloritic gouge. Northeast-trending structural discontinuities also appear to have a major control on alteration and mineralization, but do not appear to be affected by recent movement. A set of east–northeast-trending graben-forming faults bound the mineralization and fragmental package to the north and south. A major north–south-trending fault dissects the orebody and east–northeast-trending faults along UTM easting 375,600E. This fault represents a well-defined disruption in lithology, alteration, and mineralization patterns and was used to subdivide the block model in Section 14 into two structural domains, one to the east of it and one to the west.

7.3 Alteration

The alteration minerals most commonly identified included muscovite, high- and low temperature illite, ammonium bearing illite, smectite, silica, biotite, and chlorite. Alteration assemblages were defined as follows:

- Potassic Hornfels: biotite ± K-feldspar “flooding” or replacement by biotite and/or garnet with pyrrhotite ± actinolite ± alkali Feldspar (albite, orthoclase)
- Sericite-Chlorite: illite, Fe-chlorite ± interlayered illite-smectite, carbonate (commonly siderite)
- Quartz–sericite: fine-grained sugary greenish-grey to buff-coloured quartz, muscovite, or highly crystalline illite + pyrite, black sphalerite, dendritic black sulphide (DBS), and lesser pyrrhotite, rare tourmaline
- Silica–sericite: silica, illite ± pyrite, red sphalerite, pyrrhotite
- Massive Silica: grey glassy massive finely crystalline silica, sulphide-destructive
- Ammonium: ammonium-bearing micas and rare buddingtonite (NH4-bearing feldspar).

The six alteration assemblages were subsequently consolidated into three principal categories: ammonium-bearing illite overprint, texture-destructive quartz–mica ‘sericitic’, and potassic.

Table 7-1: Drill Database Lithological Codes

Code	Description
OB	Overburden
AND	Andesite
FT	Felsic tuff
FLPT	Felsic lapilli tuff
VC	Volcaniclastic
EC	Epiclastic
SED	Argillite / Sandstone / Conglomerate

The alteration model indicates the presence of two centres of texture destructive sericitic alteration cored by the ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite.

7.4 Mineralization

Core drilling has defined a zone of continuous Au mineralization that extends at least 1,300 m along its longest dimension east-west and at least 950 m north–south. The vertical thickness of the zone ranges up to 600 m, remaining open at depth in a few locations.

In general, all rocks at Blackwater are mineralized, with trace pyrite-pyrrhotite-sphalerite in outboard andesite flows and volcaniclastics, or as gold-bearing polymetallic sulphide mineralization within the fragmental felsic unit of the deposit. The only exceptions are Eocene (?) dacite porphyry dykes intersected along the southern and northwestern part of the drilling grid, and amygdaloid mafic intermediate flows in the northern part of the grid, possibly related with the Eocene Ootsa Group.

Mineralized rocks within the main Blackwater resource area can be broadly divided into a thick succession of felsic to intermediate pyroclastic and volcaniclastic rocks, volcanic flows and breccias, and related volcanic and lithic-derived sedimentary units (fine to coarse epiclastic rocks). Whole-rock analysis indicates that these units range from rhyolite to dacite to andesite in composition. Detailed age relationships between the mineralized host rocks at Blackwater are not entirely understood, but the vertical succession and locally observed progressive inter-bedding of these units suggest the andesite to be oldest, followed by the felsic tuffs and subsequently the felsic volcaniclastic rocks.

Disseminated Au–Ag mineralization is defined by an east–west-trending tabular–conical-shaped deposit with a lateral extent of up to 1,300 m east–west x 950 m north–

south. Mineralization remains open at depth in the southwestern part of the deposit as well as to the north and northwest. The centre of the deposit has an average thickness of 350 m and, where open, a vertical extension of up to 600 m. The mineralized zone plunges shallowly to the north and northwest with inferred steep, north-plunging higher-grade mineralized shoots, measuring tens of metres thick, likely influenced by near-vertical structural intersections.

Gold-silver mineralization is associated with a variable assemblage of pyrite-sphalerite-marcasite-pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite).

Sulphide mineralization at Blackwater can be divided into the following types:

- Disseminated
  - as pinhead to coarse blebby sulphide grains and aggregates typically ranging from 1% to 5% total volume of the rock, but locally exceeding this volume. Disseminations may be uniform or irregular, with sulphides displaying an anhedral to euhedral crystal form.
  - disseminations of a dark-grey, very fine grained sulphide material called "dendritic-black-sulphide" (DBS) is common at Blackwater and may form as fine disseminations to coarse clusters, as thicker coatings to fractures, or as an irregular network of "dendritic" micro cracks within the rock mass.
- Porosity infill
  - sulphides that fill, rim, or replace devitrified pyroclasts, tephra, and juvenile pumiceous material. Sulphides also commonly form parallel to compositional layering and laminations within felsic pyroclastic flows and laminated tuff units. Mineralized amygdules and altered feldspars are also observed in the andesite flow units.
- Vein
  - polymetallic, anhedral to euhedral sulphide assemblages in sub-millimetre to centimetre-scale polymetallic veinlets-veins of quartz-sericite-chlorite-clay (illite) ± (iron) carbonate ± tourmaline ± vivianite.
- Hydrothermal brecciation and related silicification –
  - centimetre- to metre-scale zones of hydrothermal brecciation, alteration, and elevated sulphide content. These breccia zones are typically healed with silica-sericite-sulphide cement and cut by a micro stockwork of vitric quartz ± sulphide veinlets.

- Structure-related (late?) –
  - sulphides crushed to comminuted in brittle fault breccia and gouge.

Hydrothermal alteration (and possibly contact metamorphism) has produced several superimposed alteration assemblages, including pervasive silica–sericite–clay (illite) ± biotite alteration and veinlet / fracture-controlled silica–sericite–chlorite–clay ± iron carbonate ± tourmaline. An early (?) biotite–silica–albite ± chlorite/actinolite hornfelsing event may have been significant, although mineralization in these rocks appears to be lower than in units without evident hornfelsing. Visible native gold has been noted in some drill holes.

Secondary quartz occurs in several modes:

- Pervasive, amorphous to translucent silicification with associated illite ± sericite. Commonly holes display intense silicification of felsic units, epiclastics, and more intermediate volcaniclastic rocks with biotite alteration of the matrix (hornfels).
- Cryptocrystalline silica replacements in felsic ash-tuff layering
- Silica cement/matrix to local hydrothermal brecciation
- Sub-millimetre vitric quartz veinlets in zones of intense silicification; commonly as a micro-stockwork.

Given the lack of outcrop, geological interpretation has been based primarily on drill information plotted on section and plan views.

The current Blackwater geological model is based on three principal components: lithology and structure, alteration, and mineralization. The lithological and structural component includes andesite, volcanic fragmentals, and laminated volcanics. Gold and silver mineralization is hosted predominantly within a central core of felsic tuffs and volcaniclastic breccias that are enveloped by a sequence of massive and more-cohesive andesitic flows and tuffs. The deposit is roughly rhombohedral in plan, bounded by near vertical northwest- and northeast trending faults. The fragmental package is funnel-shaped, elongated to the west–northwest–east–southeast, and open to the southwest at depth. The alteration component indicates the presence of two centres of texture destructive sericitic alteration cored by an ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite. The mineralization component has been built through a combined “Pyrite + DBS” simulation, which identified the pyritic mineralization domain and independently confirmed the presence of key faults seen in the lithological and structural model.

The Blackwater geologic model is discussed in more detail in Section 14.

Figure 7-3 is a control plan for the drill sections included as Figure 7-4, Figure 7-5 and Figure 7-6. The sections present gold and silver grade as histogram bars representing 5 m down-hole composites. The gold composites are constrained by the pyritic mineralisation domain. Lengths and average grades of some representative intervals are shown on the sections.

7.5 Comments on Section 7

In the QP's opinion, knowledge of the deposit settings, lithologies, mineralization style and setting, and structural and alteration controls on mineralization are sufficient to support Mineral Resource estimation:

- Given the lack of outcrop, geological interpretation has been based primarily on drill information plotted on section and plan views
- The current Blackwater geological model is based on three principal components: lithology and structure, alteration, and mineralization. The lithological and structural component includes andesite, volcanic fragmentals, and laminated volcanics
- Gold and silver mineralization is hosted predominantly within a central core of felsic tuffs and volcaniclastic breccias that are enveloped by a sequence of massive and more-cohesive andesitic flows and tuffs.

Figure 7-3: Drill Hole Plan Showing Location of Referenced Cross-Sections

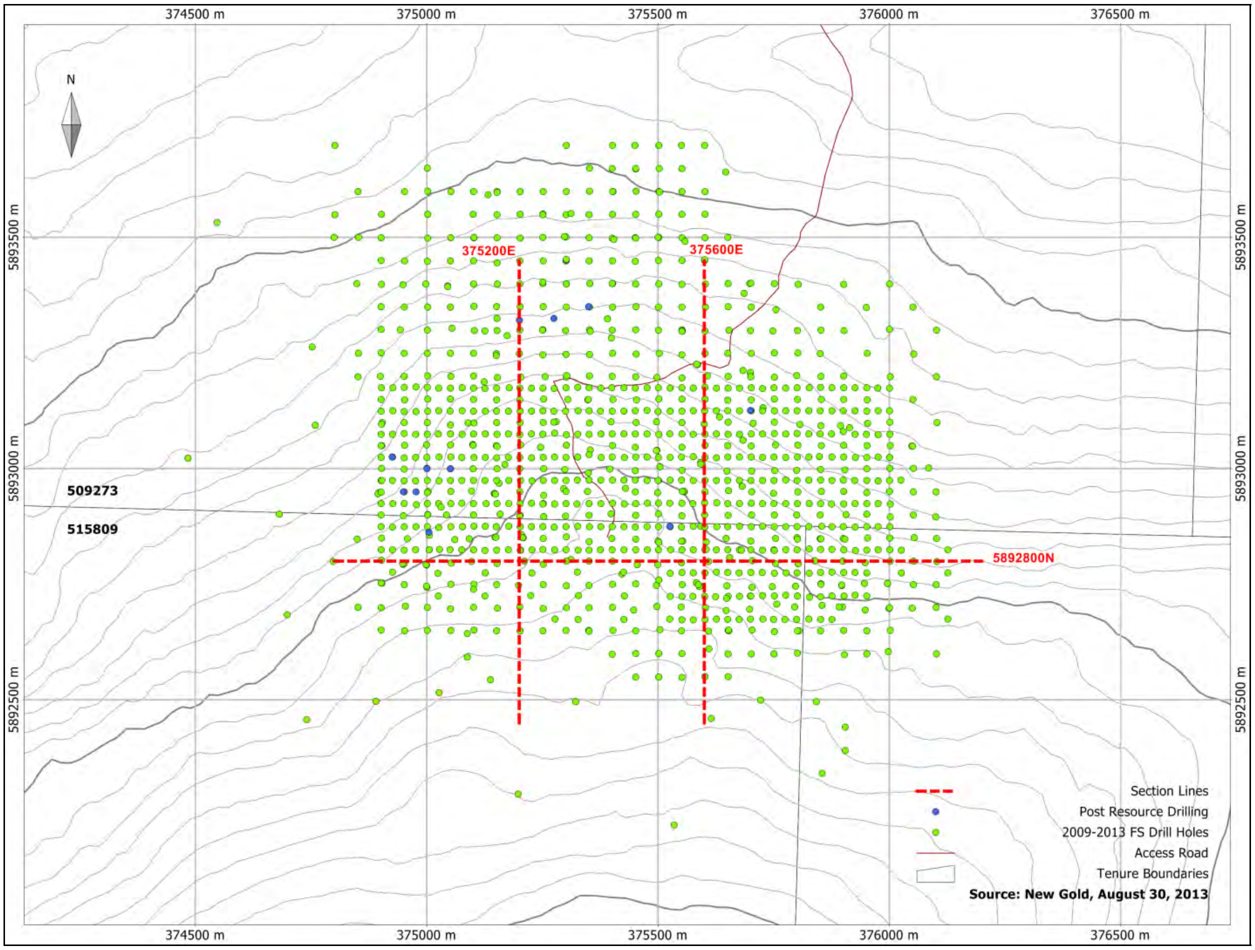
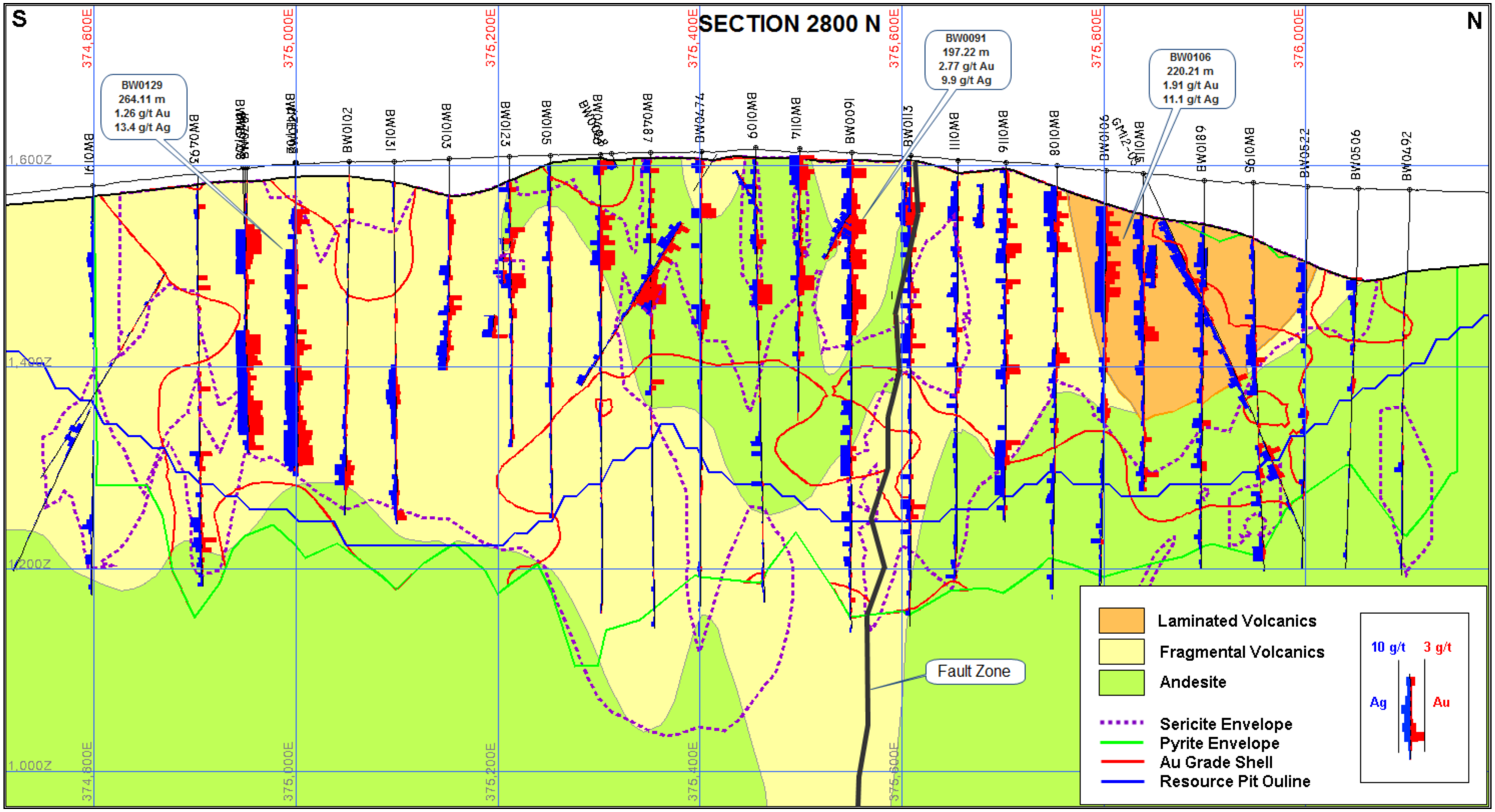
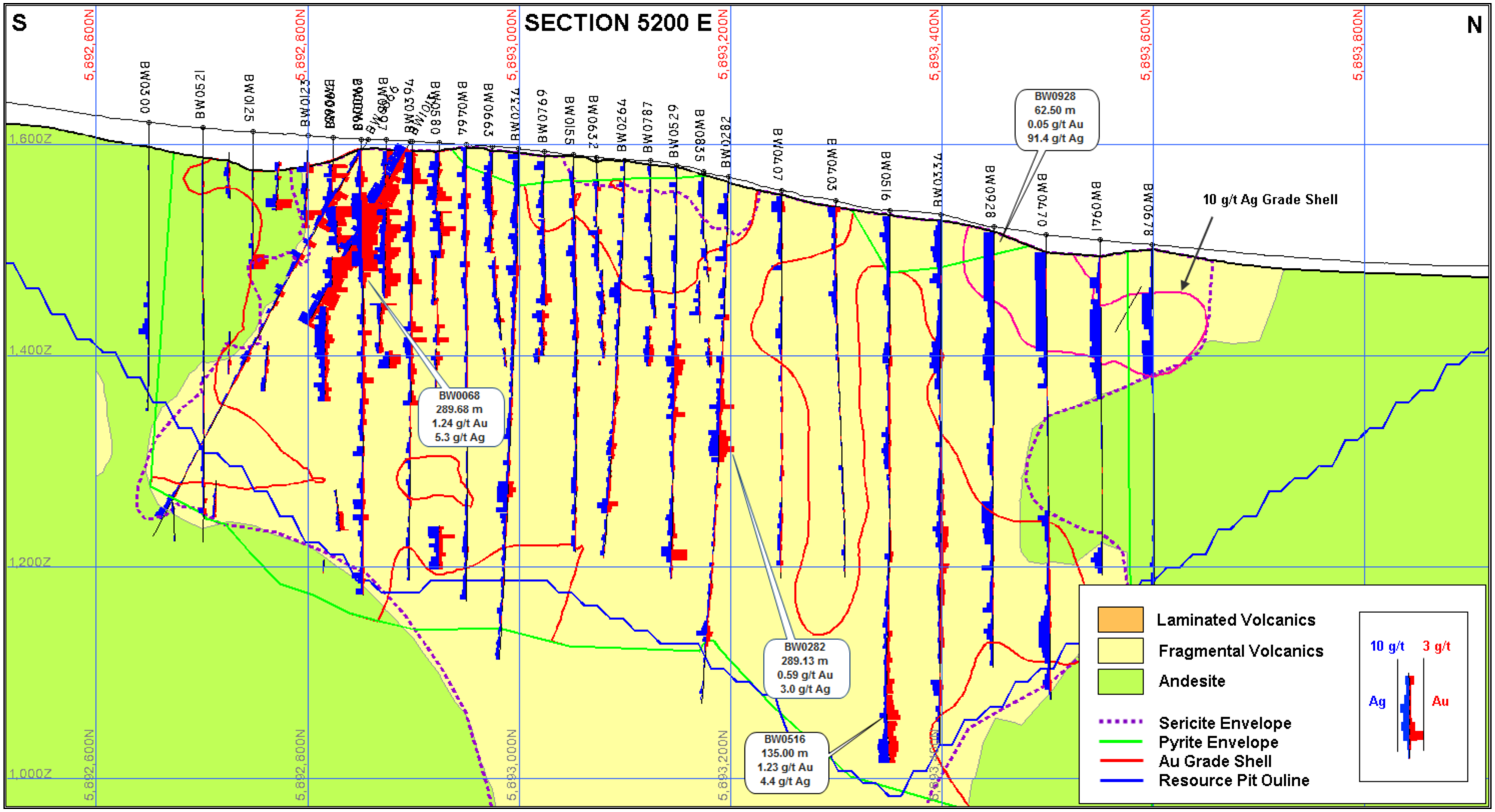


Figure 7-4: Cross-Section 2800 N



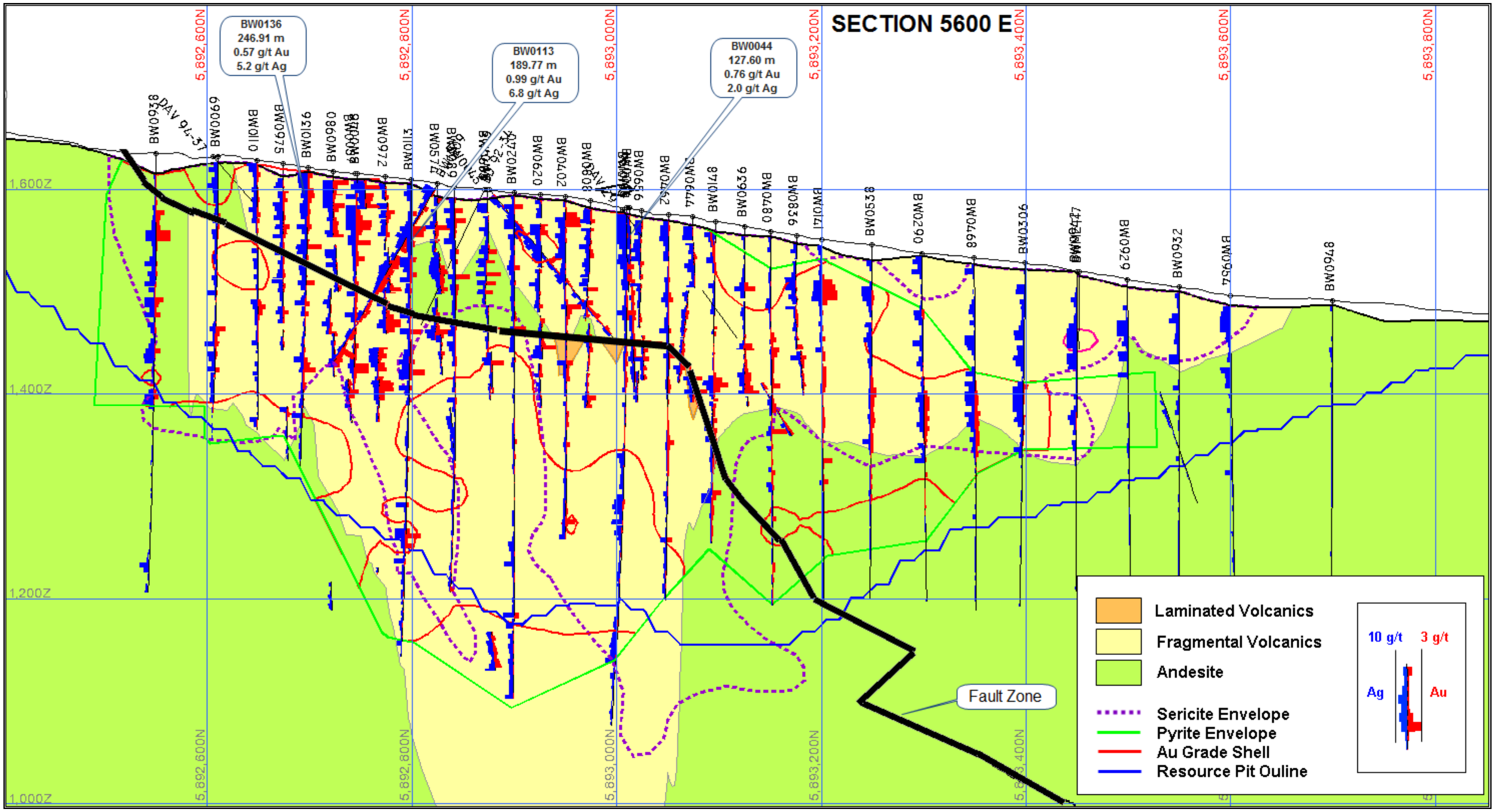
Note: Figure courtesy New Gold, 2013

Figure 7-5: Cross-Section 5200 E



Note: Figure courtesy New Gold, 2013

Figure 7-6: Cross-Section 5600 E



Note: Figure courtesy New Gold, 2013

- The deposit is roughly rhombohedral in plan, bounded by near vertical northwest- and northeast trending faults. The fragmental package is funnel-shaped and elongated west–northwest to east–southeast and open to the southwest at depth.
- The alteration component indicates the presence of two centres of texture-destructive sericitic alteration cored by an ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite.
- The mineralization component has been built through a combined “Pyrite + dendritic black sulphide or DBS” simulation, which identified the pyritic mineralization domain and independently confirmed the presence of key faults seen in the lithological and structural model. The Blackwater geologic model is discussed in more detail in Section 14 of this Report.

8.0 DEPOSIT TYPES

The Blackwater deposit is considered to be an example of a volcanic-hosted, epithermal-style gold-silver deposit.

Pervasive stockwork veined and -disseminated sulphide mineralization at Blackwater is hosted within felsic to intermediate composition volcanic rocks that have undergone extensive silicification and hydrofracturing.

The geological setting, style of gold-silver mineralization, and associated alteration assemblage for the Blackwater deposit share the characteristics of both low and intermediate sulphidation epithermal deposit types, according to the classification system of Sillitoe and Hedenquist (2003). Gold-silver mineralization is associated with a variable assemblage of pyrite-sphalerite-marcasite-pyrrhotite ± chalcopyrite ± galena ± arsenopyrite (± stibnite ± tetrahedrite ± bismuthite). Sulphide and gangue mineralogy is reasonably characteristic of an intermediate sulphidation regime as defined by Sillitoe and Hedenquist (2003). However, the massive fine-grained silicification present at Blackwater is more typical of high-sulphidation deposits and minor carbonate gangue of a low-sulphidation environment.

A typical section showing the main features of calc-alkaline volcanic arc setting and associated epithermal and related mineralization is included as Figure 8-1.

Key features of these deposit styles seen at Blackwater are summarized in Table 8-2. Figure 8-2 illustrates the hypothesized relationship of the mineralized volcanic rocks to surrounding strata at Blackwater.

8.1 Comments on Section 8

In the QP's opinion the deposit type used for exploration targeting is appropriate to the mineralization identified and the regional setting.

Figure 8-1: Schematic Section of Calc-Alkaline Volcanic Arc Setting and Associated Epithermal and Related Mineralization

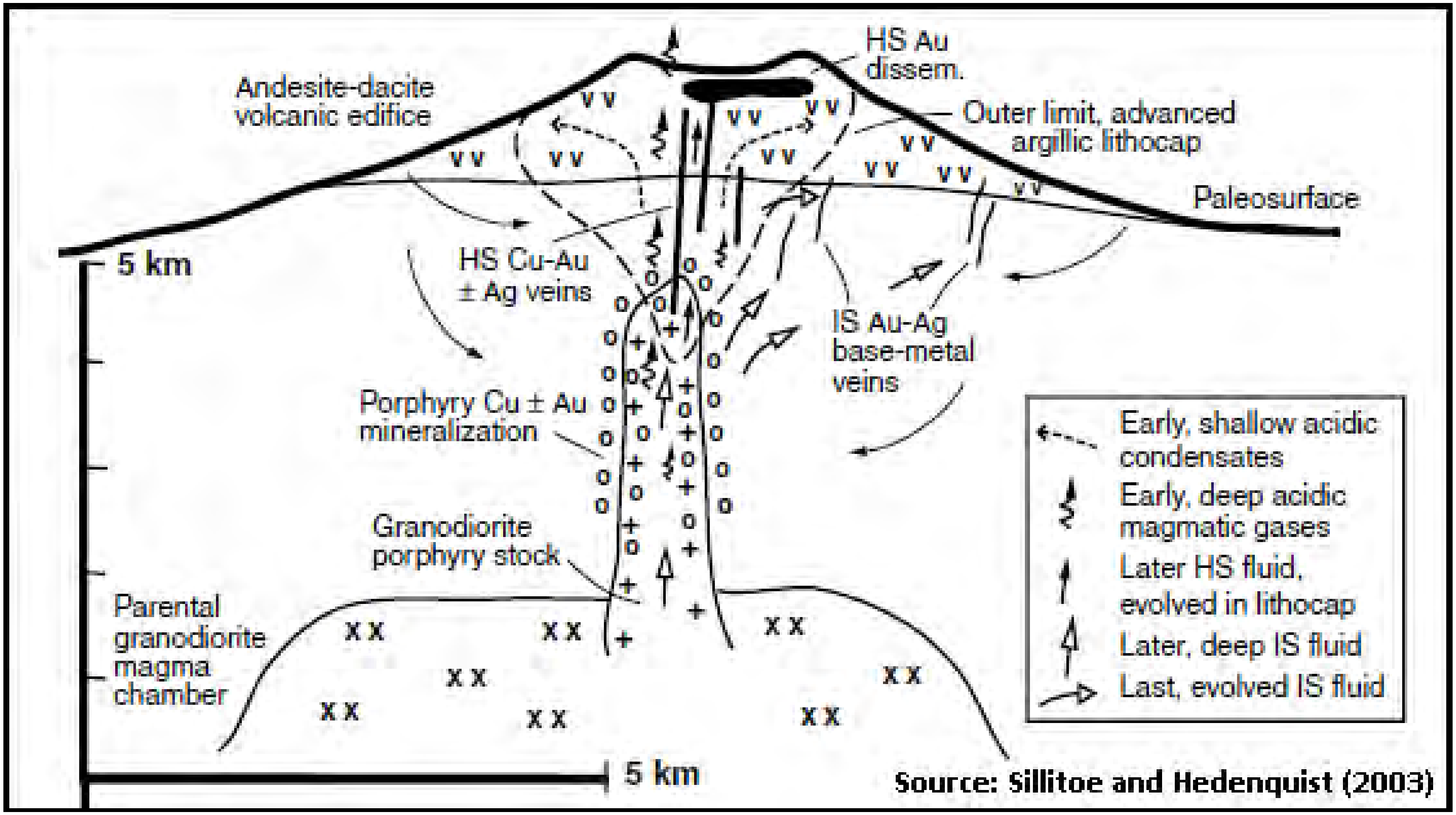


Table 8-1: Epithermal Gold Deposit Types as defined by Sillitoe and Hedenquist (2003); modified for Blackwater (2013)

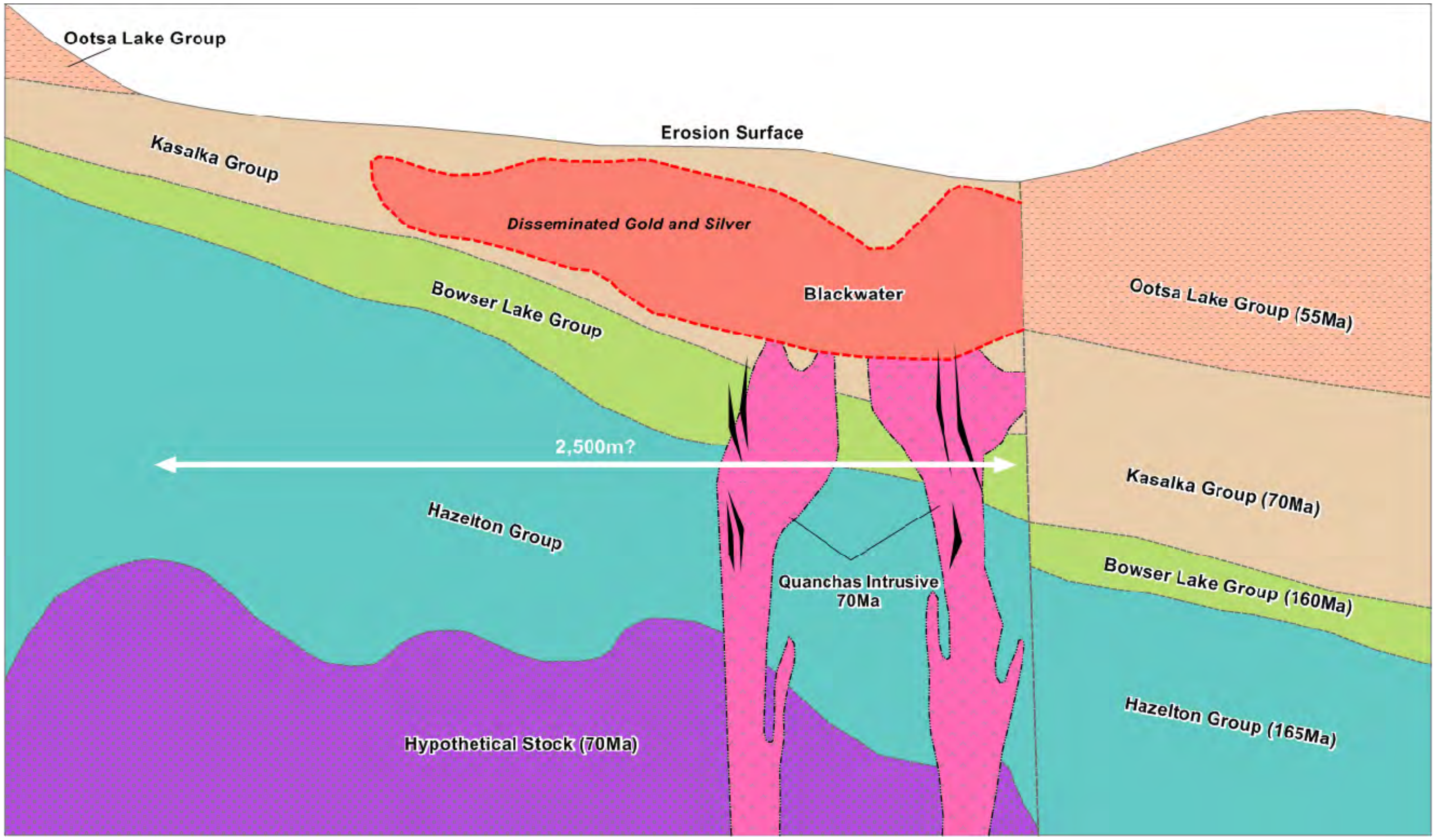
	High sulfidation		Intermediate sulfidation	Low sulfidation	
	Oxidized magma	(Reduced magma)[1]		Subalkaline magma	Alkaline magma
Type example	El Indio, Chile (vein); Yanacocha, Peru (disseminated)	Potosí, Bolivia	Baguio, Philippines (Au-rich); Fresnillo, Mexico (Ag-rich)	Midas, Nevada	Emperor, Fiji
Genetically related volcanic rocks	Mainly andesite to rhyodacite	Rhyodacite	Principally andesite to rhyodacite but locally rhyolite	Basalt to rhyolite	Alkali basalt to trachyte
Key proximal alteration minerals	Quartz-alunite/APS; quartz-pyrophyllite/dickite at depth	Quartz-alunite/APS; quartz-dickite at depth	Sericite; adularia generally uncommon	Illite/smectite-adularia	Roscoelite-illite-adularia
Silica gangue	Massive fine-grained silicification and vuggy residual quartz		Vein-filling crustiform and comb quartz	Vein-filling crustiform and colloform chalcedony and quartz; carbonate-replacement texture	Vein-filling crustiform and colloform chalcedony and quartz; quartz deficiency common in early stages
Carbonate gangue	Absent		Common, typically including manganiferous varieties	Present but typically minor and late	Abundant but not manganiferous
Other gangue	Barite common, typically late		Barite and manganiferous silicates present locally	Barite uncommon; fluorite present locally	Barite, celestite, and/or fluorite common locally
Sulfide abundance	10–90 vol %		5–>20 vol %	Typically <1–2 vol % (but up to 20 vol % where hosted by basalt)	2–10 vol %
Key sulfide species	Enargite, luzonite, famatinite, covellite	Acanthite, stibnite	Sphalerite, galena, tetrahedrite-tennantite, chalcopyrite	Minor to very minor arsenopyrite ± pyrrhotite; minor sphalerite, galena, tetrahedrite-tennantite, chalcopyrite	
Main metals	Au-Ag, Cu, As-Sb	Ag, Sb, Sn	Ag-Au, Zn, Pb, Cu	Au ± Ag	
Minor metals	Zn, Pb, Bi, W, Mo, Sn, Hg	Bi, W	Mo, As, Sb	Zn, Pb, Cu, Mo, As, Sb, Hg	
Te and Se species (Not determined)	Tellurides common; selenides present locally	None known but few data	Tellurides common locally; selenides uncommon	Selenides common; tellurides present locally	Tellurides abundant; selenides uncommon

APS = aluminum-phosphate-sulfate minerals

Characteristic of Blackwater

Note: Figure courtesy New Gold, 2013.

Figure 8-2: Cross-Section of Conceptual Blackwater Model



Note: Figure courtesy New Gold, 2013.

9.0 EXPLORATION

9.1 Geological Mapping

Given the lack of bedrock exposures in the immediate Blackwater deposit area, geologic information has been obtained primarily by exploration drilling. In 1992, Granges carried out 1:10,000 scale geologic mapping to the north of the deposit and, in an earlier 1984 program, excavated a total of 30 hand trenches; only one, however, in the northwestern part of the resource area, reached bedrock and returned an anomalous silver grade from a grab sample. More recently, New Gold mapping of pits and road-cut exposures over the deposit has confirmed the geologic interpretation of the deposit in the subsurface.

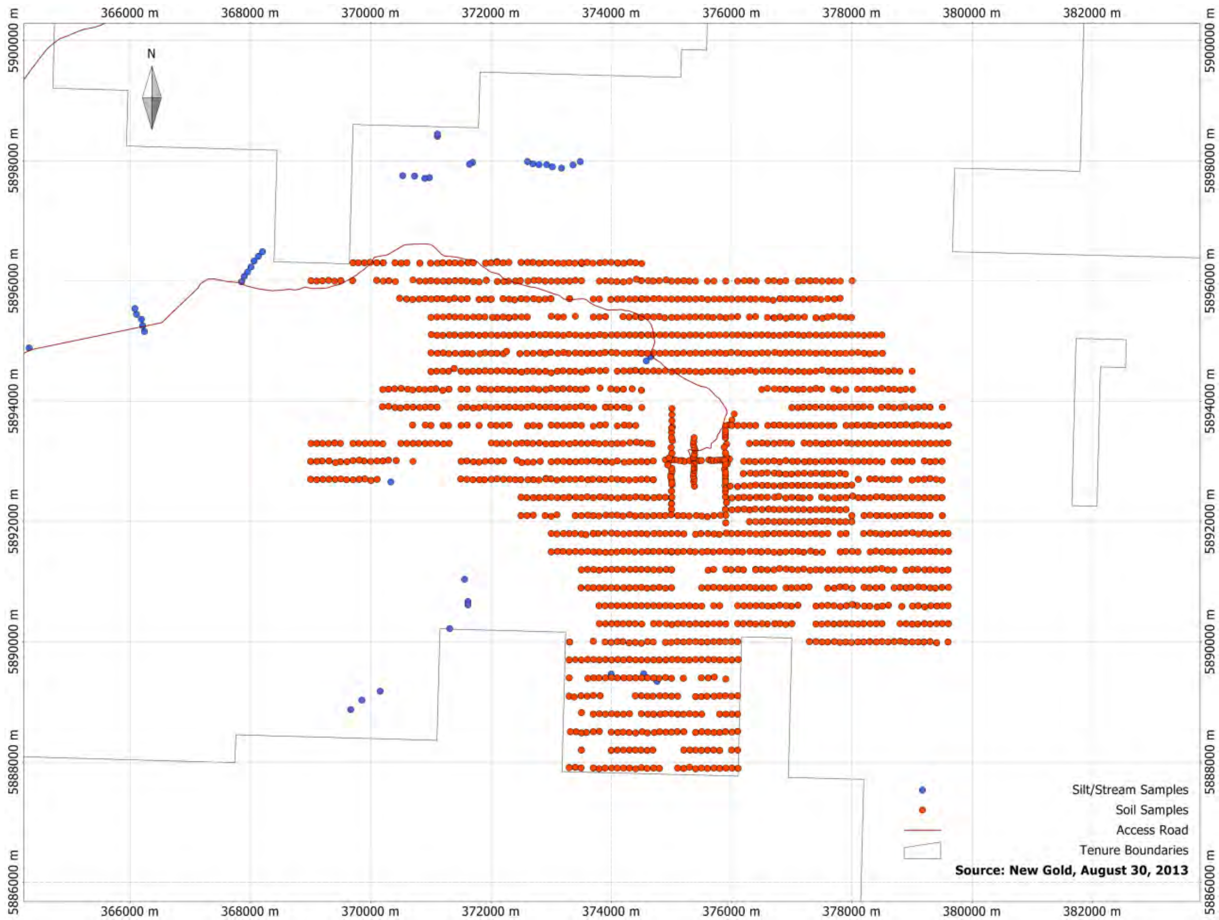
Results from recent drilling by Richfield and New Gold indicate areas of shallow overburden cover near the centre of the deposit that may be potential targets for future bulk sampling or trench mapping / sampling programs.

9.2 Geochemical Sampling

New Gold carried out soil and stream geochemical surveys over parts of the Blackwater property between late May and mid-September 2012. The sampling programs were conducted by a mixed crew of New Gold staff and contracted personnel from UTM Exploration Consultants, with the work supervised by New Gold Exploration Management.

The purpose of the geochemical surveys was two-fold: to conduct a soil orientation survey over the known Blackwater deposit; and to investigate the potential for additional areas of mineralization in the Blackwater area by testing surface soils and silts in streams draining regions of higher relief. Figure 9-1 shows both the soil survey sample sites on east–west grid lines and the silt sample locations on streams within the property.

Figure 9-1: Soil and Silt Survey Locations



The soil samples were collected at 100 m stations along grid lines spaced 300 m apart. Based on known or estimated overburden depths, one or more soil samples were collected at each station—Ah, Upper-B, and MMI. For geochemical analysis, the Ah samples were sent to Acme Analytical Laboratories in Vancouver, B.C., the Upper-B samples to SGS Laboratories in Vancouver, B.C., and the MMI samples to SGS in Toronto, Ontario. A total of 4,517 samples were collected: 1,415 from Ah, 1,379 from Upper-B, and 1,723 from MMI. The results of the soil survey indicated numerous areas displaying multi-element anomalies including gold, zinc, silver, copper, bismuth, and molybdenum, many of which merit follow-up investigation.

Additionally, in 2012 a total of 43 stream silts were collected in key drainage areas around Blackwater. The samples were sent to SGS Laboratories in Vancouver, B.C., for analysis. The results indicated anomalous copper and zinc values from streams to the northwest and southeast of the Blackwater deposit. As summarized in Table 6-1 in Section 6, previous operators performed extensive soil geochemistry testing between 1982 and 2007.

9.3 Geophysics

During 2010, Richfield contracted Quantec Geoscience Ltd. of Toronto to conduct a Titan 24 DC resistivity and IP chargeability geophysical survey. The objective of the study was to determine the relationship between IP chargeability and resistivity and zones of known gold mineralization within the mineral resource area to aid in geologic interpretation and drill targeting. The survey was carried out along five 3.5 km long north-south lines spaced 400 m apart with dipole length of 100 m. In October 2011, Quantec carried out a second-phase survey, consisting of eleven 2 km north-south lines with dipole length of 50 m, on behalf of New Gold.

The results of the survey indicate good correspondence between known mineralization and the Titan IP-resistivity results. In general, zones of significant gold mineralization correlate positively to zones of moderate resistivity and moderate IP chargeability.

Figure 9-2 shows the survey line locations and extent, and Figure 9-3 shows the results along line 5500E, where one of the prominent chargeability features correlated well with a medium- to high-resistivity gradient and most drill holes intersected significant gold mineralization.

Figure 9-2: Titan-24 DC-IP Survey Lines

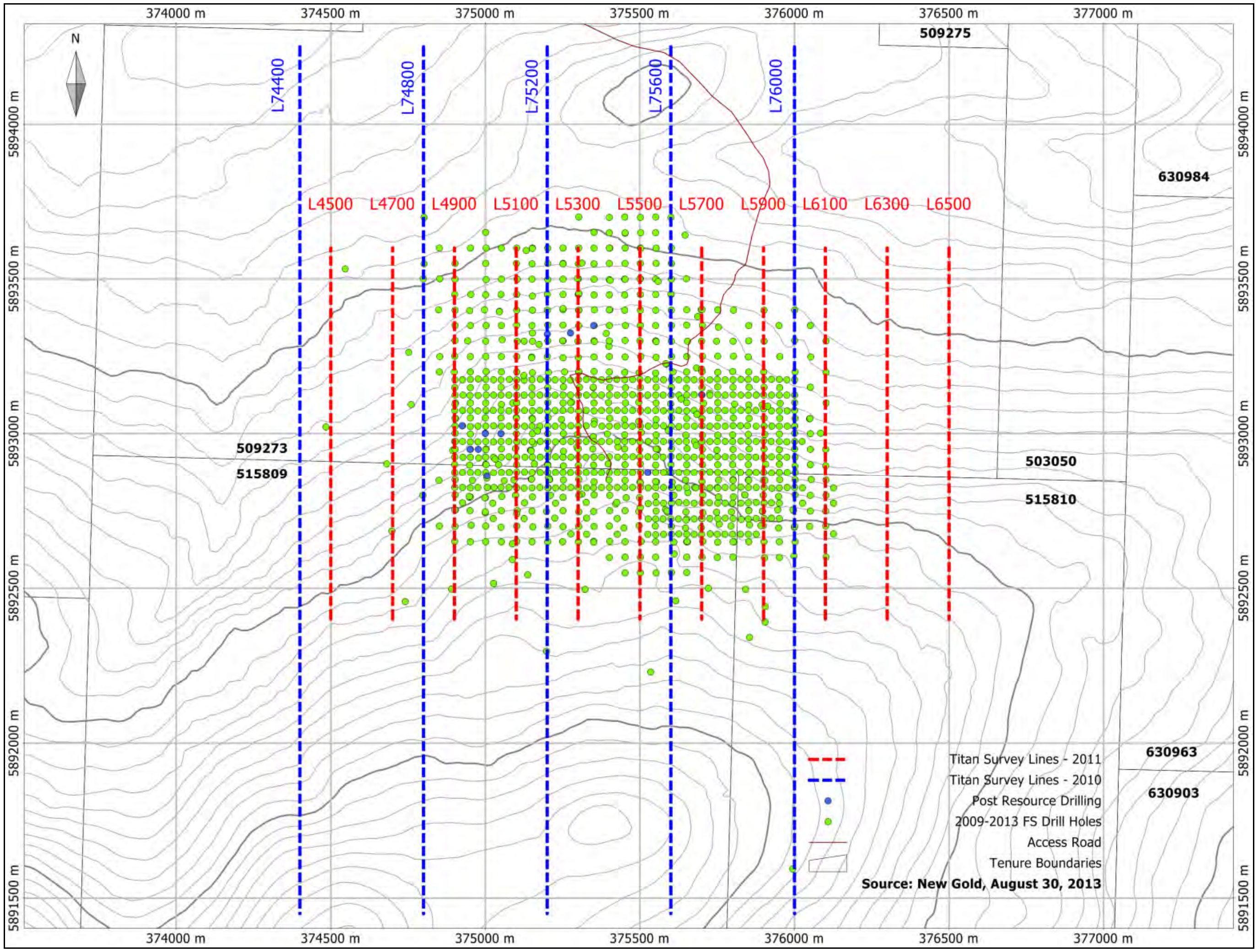
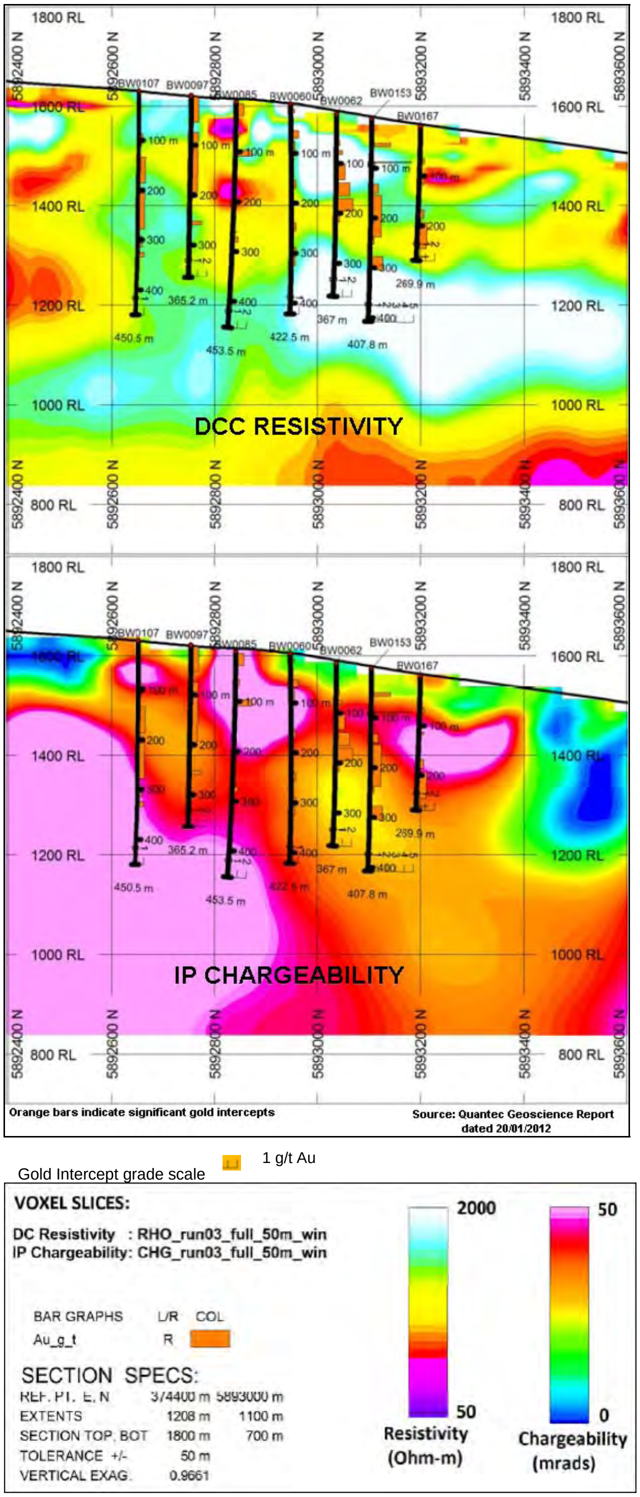


Figure 9-3: Quantec 3D Results on Line 5500E



9.4 Other Surveys and Investigations

9.4.1 Topographical Grids and Surveys

Eagle Mapping Ltd. generated detailed topography in August 2010 from an aerial survey flown on July 7 of the same year. Topography was generated at 2 m contour intervals over an area of 5 km^2 and at 5 m contours over an area of 56 km^2.

Eagle Mapping performed an aerial light detection and ranging (LiDAR) survey of the Project area on August 8 and 9, 2011. Although the area of interest (AOI) for this survey was 412 km^2 in size, the survey actually covered approximately 500 km^2 to buffer the true AOI for quality assurance purposes.

The LiDAR topographic data were collected using a Riegl VQ-480 laser scanner and airborne GPS/IMU. Data was collected in one to two pulses/m^2 with a ±0.25 m vertical accuracy and ±0.35 m horizontal accuracy based on ground control points. Topographic data were provided to New Gold as contours in DWG format, ArcASCII elevation grids, and orthophotography in GeoTiff format.

9.4.2 Petrology, Mineralogy, and Research Studies

Polished section petrographic analysis has been conducted on selected drill samples. In 2009 and 2010, sample suites were selected for the purpose of understanding the nature of the host volcanic and volcaniclastic rocks, and the gold and silver mineralization. Sample descriptions were performed by Vancouver Petrographics Ltd.

In 2009, Eco Tech Laboratories performed whole-rock lithogeochemical analyses with the aim of constraining the geochemical fingerprint of the host volcanic rocks by providing insight into the tectonic affinity, geochemical classification, and petrological evolution.

The Metallurgical Division of Inspectorate Laboratories completed an analysis of a drill composite from drill hole BW0059. Opaque phases identified from X-ray diffraction analysis included quartz, micas, orthoclase, clays, and minor calcium sulphates and carbonates. Pyrite, iron oxides (limonite, hematite, magnetite, goethite), and pyrrhotite were the main iron-bearing phases.

Mineralization identified included sphalerite, chalcopyrite, cubanite, and traces of tetrahedrite, chalcocite, and dioptase. In some samples, the chalcopyrite and cubanite were observed to be tightly intergrown. Other minerals such as rutile, ilmenite, and traces of graphite were also observed.

9.4.3 Alteration Study in Support of Geological Modelling

A two-phase alteration study was completed to develop the alteration model for the deposit. For the first phase some 20 widely spaced drill holes were re-logged in detail across sections 5892900N, 375050E, and 375700E and analyzed by SWIR spectrometer at approximate 10 m spacing down hole. Subsequently, the second phase involved the selection of an additional 135 representative holes, which were collected on approximately 100 m centres for re-logging and spectral analysis at a nominal 20 m down hole sample spacing.

Analysis of spectral data and identification of mineral species was done using the SPECMIN™ spectral reference libraries from Spectral International, Inc. (SII) and The Spectral Geologist (TSG™) data processing software developed by the Commonwealth Scientific and Industrial Research Organisation (CSIRO).

An alteration database was created in Microsoft Access to capture spectral parameters, sample descriptions, and minerals identified. Spectral parameters include the wave length position of important absorption features and illite crystallinity.

Two spectral readings were collected from each sample to ensure reproducibility and to test for heterogeneity of the samples. Measurements were collected on fresh broken surfaces, with the exception of select analyses done on fracture fill. Each sample collected during the first phase of the study was photographed and described for lithology, colour, grain size, texture, and hardness. The samples were marked for future reference and analysis before being returned to the core boxes on site. Samples collected during the second phase are labelled and stored in separate core boxes on site.

The alteration minerals most commonly identified included muscovite, high- and low temperature illite, ammonium bearing illite, smectite, silica, biotite, and chlorite. Relative proportions of alteration mineral species were quantified by intensity, grouped into alteration assemblages, and plotted on down hole spectral strip logs.

9.5 Exploration Potential

The Blackwater Project area offers good potential for the discovery of additional mineralization that may support mineral resource estimation. Work to develop this potential is ongoing and involves a combination of detailed interpretation of deposit geology from the growing body of exploration drill hole information in conjunction with expanded geologic mapping, geochemical sampling, airborne and ground-based geophysical methods, and exploration drilling, both beyond the limits of the Blackwater deposit as they are currently known, and within the greater Project area.

9.6 Comments on Section 9

In the QP's opinion, the exploration programs completed to date are appropriate to the style of the known mineralization within the Project area.

Given the lack of bedrock exposure, no detailed surface geologic mapping has been carried out over the main deposit or surrounding area by New Gold, and geological information has been obtained primarily by core drilling. Areas of shallow overburden near the centre of the deposit are potential targets for bulk sampling or trench mapping/sampling programs.

Geophysical surveys have proven useful to assist in interpreting deposit geology and identifying drill targets for future exploration.

The resolution and accuracy of the surface topography as interpreted from the 2011 LiDAR survey are considered sufficient to support detailed Project studies.

10.0 DRILLING

A total of 1,149 core drill holes (357,507 m) have been drilled in the Project area between 2009 and January 2013 (Table 10-1). Of this total, 134 were completed by Richfield, and 1,015 by New Gold.

A drill hole location plan showing all drilling in the Project area is included as Figure 10-1. Figure 10-2 shows the Richfield and New Gold drilling in the Blackwater deposit area and Figure 10-3 shows the approximate locations of the drill holes completed by other predecessor companies in the immediate area of the Blackwater deposit.

Representative cross sections showing drill holes and grades were included in Section 7 (refer to Figures 7-4 to 7-6).

Drilling by parties other than Richfield and New Gold, referred to as legacy drilling, is summarized in Table 6-1 and in the following bullet points

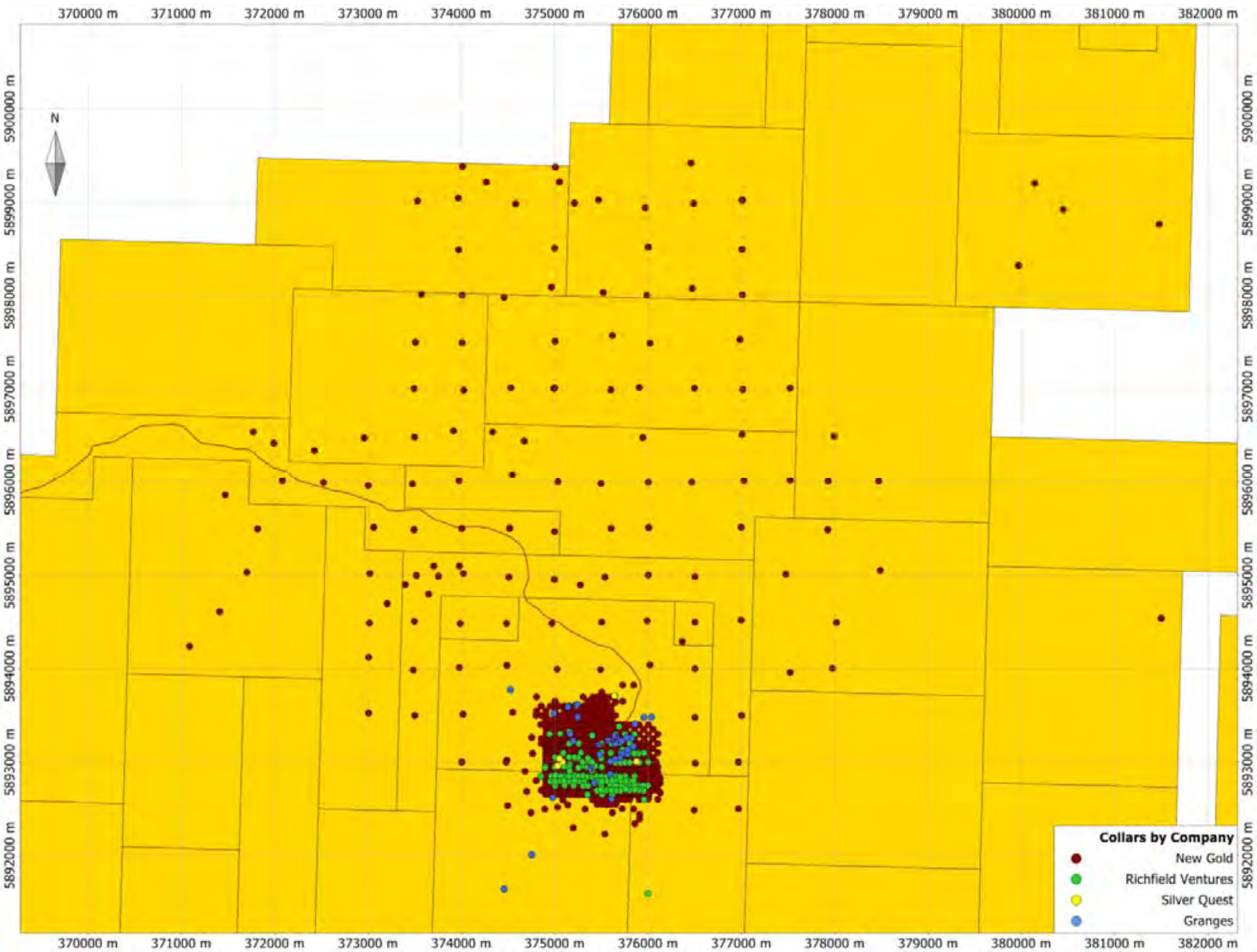
- Between 1981 and 1986, Granges drilled eight diamond drill core holes for a total 507 m. The helicopter-supported Winkie drill had difficulty with the ground conditions, and some holes did not reach target depths. Drilling focused on northern parts of the property. Results of this early drilling, including drill logs, assays, and core, have not been located.
- During the 1986 season, Granges drilled 34 reverse circulation (RC) drill holes totalling 1,524 m in an area it named the Silver Zone. As with data for the earlier drilling, and despite thorough search, the drill logs and assays for the RC drilling could not be located.
- During 1987, Granges completed 22 diamond drill core holes totalling 2,725 m on the Silver Zone and the Gold Zone to the east. These areas are located over what is now recognized to be the northern part of the Blackwater mineral resource. Drill logs and assay data were filed for assessment, and drill core is stored on the property.
- In 1992, Granges drilled in five core holes totalling 785 m focused mainly southwest of the Gold Zone and on the Dave claim to the southwest. Logs and assays were filed for assessment and core is stored on the property.

Table 10-1: 2013 Feasibility Study Drill Hole Summary Table

Series	Year	Company	Holes Drilled	Total metres	Intervals Assayed	Metres Assayed
BW0042 TO BW0059	2009	RVC	18	3,621.23	3,408	3,413.62
BW0060 TO BW0116	2010	RVC	57	21,335.92	20,034	20,219.03
BW0117 TO BW0175	2011	RVC	59	19,727.37	18,243	18,840.15
	Subtotal	RVC	134	44,684.52	41,685	42,472.80
BW0176 TO BW0295,297,298 (+50R)	2011	New Gold	125	49,315.78	45,577	46,230.03
BWMET01 TO BWMET07	2011	New Gold	7	2,281.91	1,347	1,364.76
BW0296 and BW0299 to BW1013	2012	New Gold	716	207,333.15	192,848	195,943.88
BWMET08 TO BWMET27	2012	New Gold	20	1,816.50	0	0.00
BWWR01 TO BWWR14	2012	New Gold	14	2,952.50	1,120	2,697.45
Pilot Holes	2012	New Gold	7	2,265.27	1,879	1,928.00
GM12-01 TO GM12-13	2012	New Gold	13	5,003.00	4,555	4,772.99
Condemnation - DDH	2012	New Gold	18	7,036.53	4,551	4,634.28
Condemnation - RC	2012	New Gold	91	33,252.00	16,493	33,004.50
BW1014	2013	New Gold	1	420.00	413	414.00
Pilot Holes	2013	New Gold	3	1,145.74	0	0.00
	Subtotal	New Gold	1,015	312,822.38	268,783	290,989.89
	Total	**All**	**1,149**	**357,506.90**	**310,468**	**333,462.69**

Note: +50R represents BW050R – BW050 was re-entered and drilled deeper

Figure 10-1: Project Drill Hole Location Plan



Note: Figure courtesy New Gold, 2013

Figure 10-2: Drill Hole Location Plan, Blackwater Deposit Area

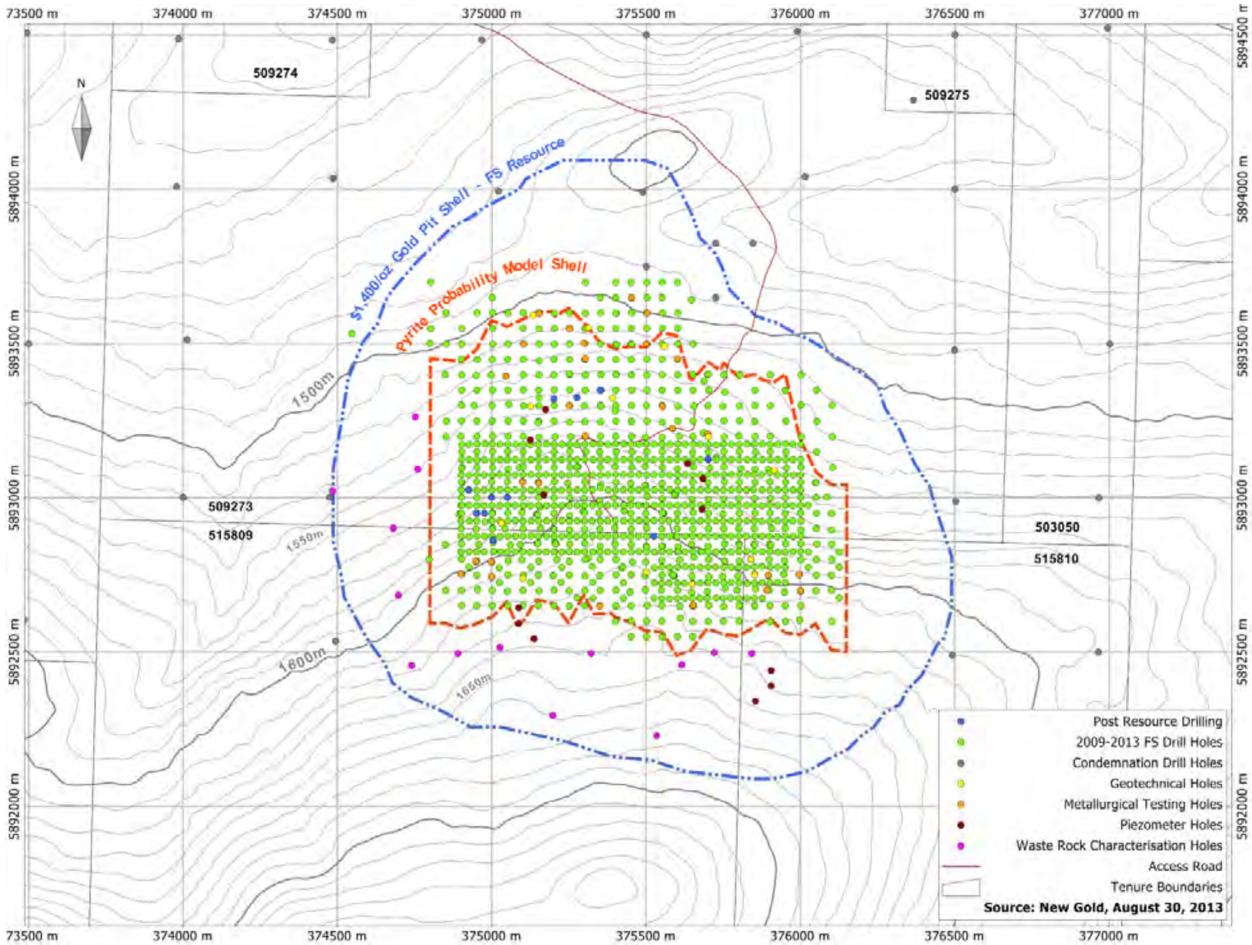
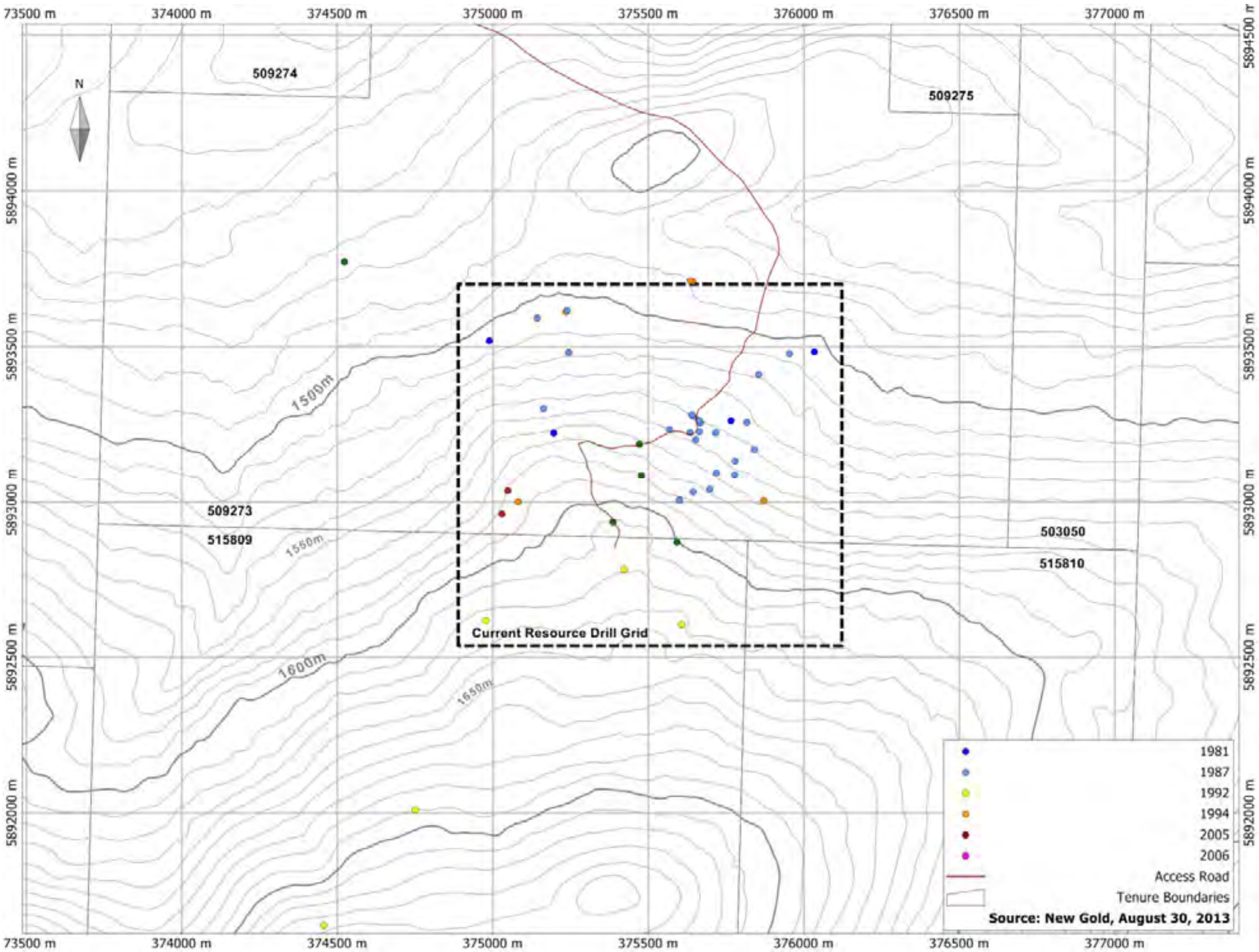


Figure 10-3: Drill Hole Location Plan, Historic Drilling



- In 1994 Granges drilled five core holes were totalling 759 m. The data were filed for assessment, and the core was stored.
- No exploration drilling was conducted at Blackwater between 1995 and 2005.
- Silver Quest drilled five holes in 2005 for a total 939 m and another two holes in early 2006 for a total of 394 m. The holes were widely dispersed. The later drilling focused on the Quest Zone.

Cumulative drilling completed between 1981 and the end of 2006 was 81 holes totalling 7,633 m. In 2010 Richfield re-logged the Granges drill core and reviewed the data to help guide its exploration efforts.

10.1 Drill Methods

The exploration drilling carried out since 2009 has been predominantly HQ diameter (63.5 mm) diamond drill core except where a reduction to NQ diameter (47.6 mm) was required to attain target depths. Twenty-three metallurgical holes (BWMET05 – BWMET27), and one deep hole (BW0364) were PQ diameter (85 mm) core. Ninety-one reverse circulation holes were drilled as part of a condemnation program. Contractors and rig types used on the Project for the Richfield and New Gold drill programs are summarized in Table 10-2. Drill core is transported from drill to camp by four-wheel drive vehicle for core logging.

10.2 Geological Logging

Drill core is logged in a specially built core handling facility at the Project site. Logging includes geotechnical, magnetic susceptibility, and specific gravity measurements taken at regular intervals. Lithology is logged and the core prepared for systematic sampling at regular 1 m intervals. Core sawing and sampling are the last steps in core handling. Core is cut in half using a diamond blade rock saw, with one half of the sample interval submitted for assay and geochemical analysis and the other half returned to the core box and stored at the Project site for future reference.

Logged data are entered into LogChief™ tables by Project geologists.

Magnetic susceptibility and conductivity data were measured at 10 cm increments along the core with a hand-held conductivity and magnetic susceptibility meter (GDD MPP-EM2S+Probe) and stored internally for future use.

Recovery and rock quality designation (RQD) data were measured and recorded in LogChief™. An RQD measurement is the cumulative length of core pieces longer than 10 cm in a given core run divided by the total length of that run. Recovery and RQD measurements were performed by company geotechnical staff.

Table 10-2: Drill Contractor and Rig Type Summary Table

Year	Company	Drill Rig Type
2009	Falcon Drilling	F-2000
2010	Falcon Drilling	F-2000
2011	Falcon Drilling	F-2000, F-5000, F-6000
	Paycore Drilling	TITAN
2012	Falcon Drilling	F-2000, F-5000, F-6000
	Paycore Drilling	TITAN, Discovery
	Hy-Tech Drilling	S-F Tech 5000
	Boart Longear	Ingersol Rand TH100
2013	Paycore Drilling	TITAN, Discovery
	Hy-Tech Drilling	S-F Tech 5000

The lithological nomenclature at the Project has undergone revision on two occasions since New Gold took control of the Project in June of 2011. These steps were taken in an effort to simplify, standardize, and clarify the nomenclature in a way that will facilitate consistency in logging, geologic interpretation, and ultimately resource modelling. As a result, the following six principal rock lithology types have been defined: Overburden (OB), Felsic Tuff (FT), Felsic Lapilli Tuff (FLPT), Volcaniclastic (VC), Andesite (AND), and Sediments (SED). Based on these rock types a thorough re-logging of each hole drilled before June 2011 has been completed and the new nomenclature applied.

10.3 Recovery

Core recovery for the 2009, 2010, 2011, and 2012 drilling programs averaged 92%, and the median core recovery was 96%. Poor core recovery often occurs in zones of faulting and fracturing. The average RQD value for this period was 44%.

10.4 Collar Surveys

Planned drill hole collar locations were measured in the field using hand held global positioning system (GPS) instruments. Locations were subsequently confirmed by Trimble differential GPS. Of the 1,040 holes, 1,025 were then professionally surveyed by All North Consulting using a Real Time Kinematic (RTK) technique to enhance the precision of the location data. Elevations for the drill collars were determined by draping collar coordinates over the topography measured by the LiDAR survey.

10.5 Downhole Surveys

Down-hole surveys are performed using Reflex survey equipment, and dip angle and azimuth are recorded. A +18.8° magnetic declination correction factor is applied to the magnetic azimuth record. Data are entered into LogChief™ in tables designed specifically for the Project.

10.6 Geotechnical and Hydrogeological Drilling

Thirteen specific geotechnical HQ holes were drilled for a total length of these geotechnical holes was 5,003 m.

Ten specific hydrogeological pilot holes (HQ) were drilled to serve as monitoring stations, where a piezometer is installed to measure the level of the aquifer in the deposit area. These are numbered PH12-2-1, PH12-3-1, PH12-3-2, PH12-3-3, PH12-4-1, PH12-4-2, PH12-4-3, PH13-1-1, PH13-2-2, and PH13-2-3. The total length of these hydrogeological holes is 3,411 m. Locations of these geotechnical and hydrogeological holes are shown in Figure 10-2.

10.7 Metallurgical Drilling

Twenty-seven specific metallurgical holes were drilled, four HQ holes, BWMET01–04, and 23 PQ holes, BWMET05–27. The total length of these metallurgical holes was 4,098 m. Locations of these metallurgical drill holes are shown in Figure 10-2.

10.8 Waste Rock Characterisation Drilling

Fourteen specific waste rock characterisation holes (HQ) were drilled, BWWR01–14. The total length of these holes was 2,952.5 m. Locations of these waste rock drill holes are shown in Figure 10-2.

10.9 Condemnation Drilling

Eighteen diamond drill holes (HQ; 7,036.53 m) and 91 reverse circulation (RC; 33,252 m) holes were drilled to condemn potential site facility areas surrounding the Blackwater deposit.

10.10 Sample Length/True Thickness

Typical drill hole orientations are as indicated in the example cross-sections in Section 7 (refer to Figures 7-4 to 7-6). The sections show examples of summary assay (or mineralization intensity) values. The grade variations encountered in the drilling are illustrated by colour-coded down-hole histograms for gold and silver and indicate areas of higher grades, low grades, and intervals of higher grades in lower-grade zones. The sections confirm that sampling is representative of the gold–silver grades in the deposits.

Gold and silver mineralization occurs within an irregularly-shaped system of stockwork and disseminated sulphides that strikes approximately east–west and dips moderately to the north. Depending on the inclination of an individual drill hole, and the local dip of mineralization, drill intercept widths are approximately equivalent to true widths.

10.11 Comments on Section 10

In the QP's opinion the quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the exploration and infill drill programs from 2009 to 2013 are sufficient to support Mineral Resource estimation. There are no known sampling or recovery factors with these programs that could materially impact the accuracy and reliability of the results.

- Core logging meets industry standards for gold–silver exploration.
- Collar surveys and down-hole surveys have been performed using industry-standard instrumentation
- Recovery from core drill programs is acceptable to allow reliable sample data for Mineral Resource estimation
- Gold and silver mineralization occurs within an irregularly shaped system of stockwork and disseminated sulphides that strikes approximately east–west and dips moderately to the north. Depending on the inclination of an individual drill hole, and the local dip of mineralization, drill intercept widths are approximately equivalent to true widths
- Drill orientations are generally appropriate for the mineralization style
- Drill orientations are shown in the example cross-sections included in Section 7 and can be seen to appropriately test the mineralization. The sections display typical drill-hole orientations for the deposits, show examples of summary assay (or mineralization intensity) values using colour ranges for assay intervals that include areas of non-mineralized and very low grade mineralization, and outline areas where higher-grade intercepts can be identified within lower-grade sections. The sections confirm that sampling is representative of the gold–silver grades in the deposits, reflecting areas of higher and lower grades
- Sampling methodologies are discussed in Section 11.0, and comments on the sampling protocols are outlined in Section 11.9.

- Metallurgical recoveries are discussed in Section 13.0, and comments on the recoveries are outlined in Section 13.6.

11.0 SAMPLE PREPARATION, ANALYSES, AND SECURITY

The sample preparation, security, and analytical procedures used by New Gold for the Blackwater Project have ensured the validity and integrity of samples taken. The procedures and results have been described in GeoSim (2011a, b; 2012) and AMEC (2012c).

New Gold reviewed the control sample results when received from the laboratory (New Gold, 2012a to 2012j). AMEC has reviewed the final control sample results before each resource estimate since November 2011. The AMEC reviews are summarized in AMEC (2012a, b, d, e, f; 2013a, b).

Quality control procedures implemented in 2012 for silver analysis show acceptable levels of precision and accuracy for silver results. The 2012 QC results mitigate previous concerns about the accuracy and precision of pre-2012 silver results resulting from a lack of comprehensive silver QC.

Data from holes drilled between 1981 and 1994 have no documented QA/QC information and were not used in resource estimation.

11.1 Sampling Methods

New Gold and predecessor company personnel conducted the drill core handling and sampling on the property. Drill core was transported in core boxes from the drill site to camp by four-wheel drive vehicle and logged in a specially built core handling facility. Samples were taken systematically on 1 m long sawn half-core sample intervals, then tagged and bagged. Four sample bags were placed into a larger rice bag labelled with the sample numbers and sealed with a numbered banker's security tag. Between preparation and shipment, a period of up to four days, the rice bags containing the samples were stored in a secure area behind the core cutting area.

The remaining half cores were archived in core sheds on the property. New Gold personnel drove trucks containing the samples to Prince George, and from there the samples were delivered to the laboratories by bonded couriers.

11.2 Analytical and Test Laboratories

Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops and ALS Mineral Laboratories (ALS) in Vancouver, Vanderhoof, Terrace, Reno, and Elko were used for sample preparation. Eco Tech in Kamloops and ALS in North Vancouver were used as the primary assay laboratories. Both laboratories were accredited and are independent of New Gold.

11.3 Sample Preparation and Analysis

Drill core samples were prepared using standard crush, split, and pulverise sample preparation procedures. Pulverized samples were analysed for gold by fire assay atomic absorption spectrometry (FA ASS). Preparation and FA AA procedures varied between labs but were generally similar.

The Eco Tech samples were initially assayed for silver by aqua-regia digestion (AR) and AAS finish, and later by AR and induction-coupled plasma spectrometry atomic emission spectrometry (ICP AES) finish. The ALS samples were analyzed for silver by four acid digestion ICP AES finish until July 2012, after which time silver was analyzed by a four acid digestion AAS. Eco Tech overlimit results (>30 g/t Ag) were reassayed by AR/AAS method. ALS overlimit results (>100 g/t) were reassayed by a four acid digestion with AAS finish with a higher detection limit.

Assay procedures also include a multi-element package (28 elements at Eco Tech, 33 elements at ALS) by AR digestion and ICP AES finish. Overlimit analysis was completed on samples returning greater than 1% Cu, Pb, or Zn

11.4 Metallurgical Sampling

Metallurgical samples were selected from the designated metallurgical holes, and samples from numerous resource holes across the deposit. The samples were collected and despatched from site to laboratories under the supervision of the Exploration Manager. Sample security protocols used were the same as the exploration sample protocols described above.

11.5 Density Determinations

Specify gravity measurements were made the field for more than 32,000 samples using a water immersion method without a wax coating. ALS verified the field measurements by analyzing 154 samples using a water immersion method without a wax coating and 55 samples using a wax-coat water immersion method. The results showed no bias between the field and laboratory methods for all but overburden samples.

11.6 Quality Assurance and Quality Control

Quality assurance / quality control (QA/QC) protocols included “blind” insertion of certified reference material (CRM) standards, blanks, quarter-hole (field duplicate), coarse reject, and pulp duplicates. The drill hole database was supported by some 80,000 QA/QC check assays.

11.6.1 Standards

Three CRM standards with best values approximating the low, medium, and high Au grades expected at Blackwater were used to monitor laboratory accuracy. The CRMs were initially purchased from CDN Resource Laboratories Ltd in Langley, Canada, then from WCM Minerals in Burnaby, Canada, and then from Geostats Pty, in WA, Australia. No high-grade silver CRM standards were used. Accuracy was measured in terms of bias from the expected CRM grades. Acceptable bias was observed for the range of gold grade expected at the deposit. Marginally acceptable high bias was observed for silver grades less than 5 ppm. Figure 11-1 shows how standard G310-6 from Geostats Pty. performed between November 1st 2012 and February 2nd 2013, excluding fails, which were re-assayed by ALS.

11.6.2 Blank Samples

Upon receipt of the assay data the blank results were examined and compared with accepted values. Assay results for blanks have been near or at detection limit for gold and silver.

11.6.3 Duplicates

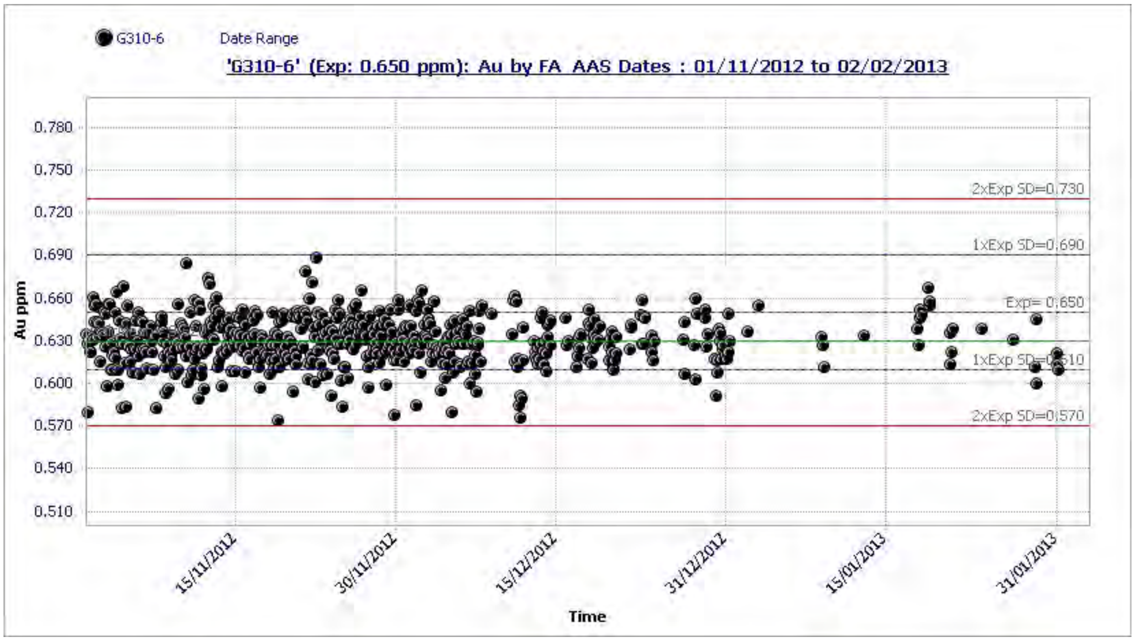
Four types of duplicates are run at Blackwater to assess the precision of the assay analyses; R1= Repeat, D1= Pulp Duplicate, D2= Coarse Duplicate, and E1= External Check. The insertion rates are 1/10, 1/20, 1/20, and 1/50 respectively.

New Gold uses the coefficient of variation (CVAVR) (Stanley and Lawie, 2007) and reduced major axis (RMA) plots to assess the performance of duplicates pairs. Visualization of CV and RMA plots also helps to identify outliers and certain sampling errors that statistical tests may not find.

Pulp and coarse reject duplicates (R1, D1, and D2) performed within acceptable CVARV and RMA limits at ALS. Figure 11-2 shows the RMA plot for Au repeats (R1), and Figure 11-3 shows the CVARV distribution for pulp duplicates (D1).

External pulp duplicates (E1) run at SGS in Vancouver show a 2% low bias compared with the original ALS analyses. This bias is within acceptable limits.

Figure 11-1: G310-6 Performance Chart



Note: Figure courtesy New Gold, 2013

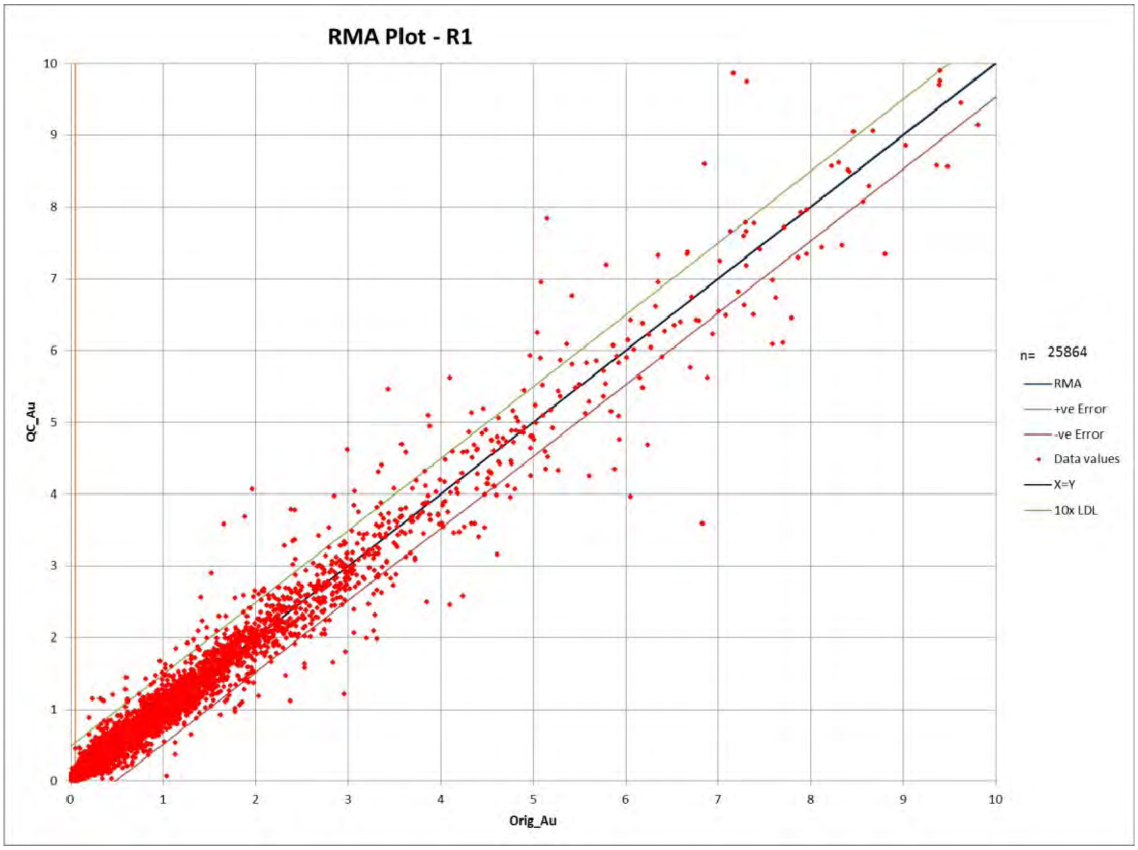
11.6.4 Field Duplicates

Assay results for 2,479 field duplicate pair results from 2009 to March with grade values of >0.2 g/t and <10 g/t Au were compared using absolute relative difference cumulative frequency plots. At the 90% cumulative frequency level, the values for Eco Tech and ALS are both around 70% for Au and 60% for Ag, indicating a high level of variability between field duplicates. A commonly accepted value is 90% of paired duplicates with <30% absolute relative difference. As of October 2011, quarter-core field duplicates are no longer inserted.

11.6.5 Sampling Procedure Optimisation

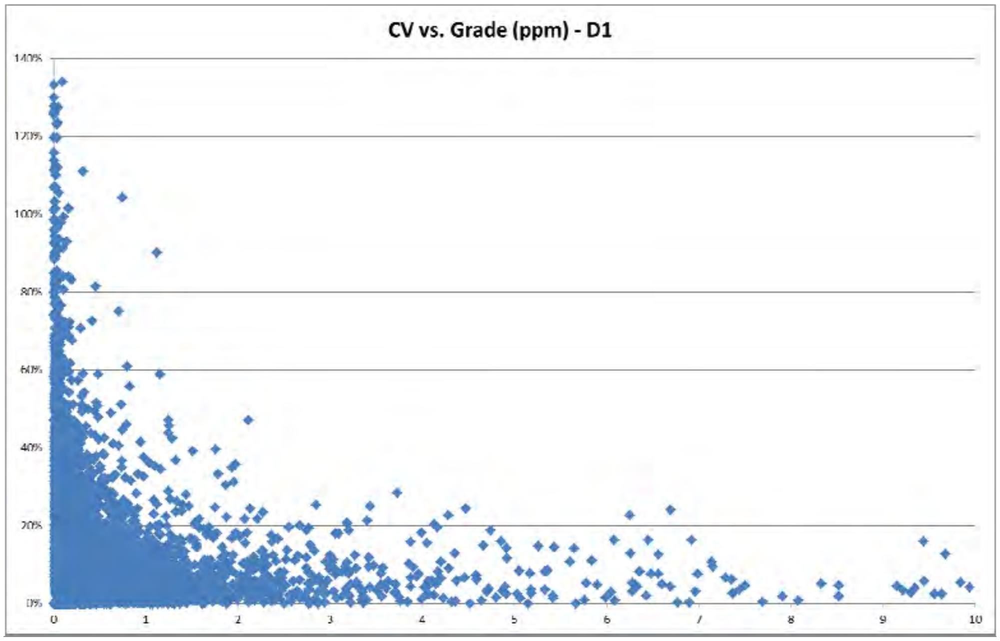
During 2012, check programs were run on different stages of the sampling procedure to try to optimize the level of precision achieved at ALS. The programs included drying the original sample for a longer time to remove extra moisture to see if this could improve the homogeneity achieved during milling; pulverizing samples to different particle size specifications to test for any impact on achievable precision; and assaying different sample aliquot sizes. All the programs undertaken confirmed the procedures already in place are the optimum specifications to prepare and analyze Blackwater samples.

Figure 11-2: RMA plot for Au Repeats at ALS



Note: Figure courtesy New Gold, 2013

Figure 11-3: CV% plot for Au Pulp Duplicates at ALS



Note: Figure courtesy New Gold, 2013

11.7 Databases

The current drill hole and assay database for the Project is administered from the New Gold Vancouver office using the same Maxwell GeoServices products, LogChief™ and DataShed™ that were used during the Richfield drilling programs.

Drill hole data logged in the field are entered into a LogChief™ database specifically tailored to the Project. LogChief™ validates the data as they are entered, and the final logs are exported and transferred to the database administrator in Vancouver for import to DataShed™, the master database. DataShed™ performs additional data validation checks, and the administrator re-checks all logs.

The assay certificates received from both Eco Tech and ALS are delivered in a format that allows instant import to DataShed™.

Access permission for entering and editing data into the database is restricted to the New Gold Corporate Exploration Manager and the Project Database Administrator. The database is hosted on the New Gold server, which routinely backs up every day for protection from data loss due to potential drive failures or other technical issues.

11.8 Sample Security

Samples are transported to Prince George by truck, where the driver waits with the samples in the truck until pick-up for onward shipment by a bonded courier. Before July 2011, the Richfield samples, including the standards, blanks, and duplicates, were shipped to Eco Tech Stewart Group Laboratories (Eco Tech) in Kamloops, B.C. Since the acquisition of Richfield by New Gold in June 2011, and the subsequent acquisition of Eco Tech by the ALS Group in July 2011, samples have been shipped to ALS Minerals (ALS) in North Vancouver, B.C.

11.9 Comments on Section 11

In the QP's opinion the sample preparation, security, and analytical procedures used by New Gold for the Blackwater Project have ensured the validity and integrity of samples taken. Quality control procedures implemented in 2012 for silver analysis shows acceptable levels of precision and accuracy for silver results. Previous concerns regarding the accuracy and precision of pre-2012 silver results due to lack of comprehensive silver QC is mitigated by the 2012 QC results.

The quality of Au and Ag analytical data collected during the Richfield and New Gold programs are sufficiently reliable.

- Data are collected following industry-standard sampling protocols
- Sample collection and handling of core were undertaken in accordance with industry-standard practices, with procedures to limit potential sample losses and sampling biases
- Sample intervals in core are at 1 m intervals; the sample intervals are considered to be adequately representative of the mineralization
- Bulk density determination procedures are consistent with industry-standard procedures, and there are sufficient bulk density determinations to support tonnage estimates
- Sample preparation for samples that support Mineral Resource estimation has followed similar procedures for the Richfield and New Gold programs, and both laboratories used the same pulverization sizing
- Core drill programs were analyzed by independent laboratories using industry-standard methods
- Drill programs included insertion of blank, duplicate and CRM samples

- Data that were collected were subject to validation, using in-built program triggers that automatically checked data on upload to the database
- Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards
- Sample security has relied upon the fact that the samples were always attended or locked in the on-site sample preparation facility
- Chain-of-custody procedures consist of filling out sample submittal forms that are sent to the laboratory with sample shipments to make certain that all samples are received by the laboratory
- Current sample storage procedures and storage areas are consistent with industry standards.

Data from holes drilled between 1981 and 1994 have no documented QA/QC information, and they are not deemed acceptable for use in resource estimation.

12.0 DATA VERIFICATION

12.1 Site Visit

Mr. Ronald G. Simpson, Principal of GeoSim Services Inc. and an independent Qualified Person engaged by New Gold, visited the site on December 13, 2010; September 8, 2011; November 28, 2011; and September 20, 2012 and reviewed the drilling, sampling, and quality assurance/quality control procedures. The geology and mineralization encountered in the drill holes completed to date were also reviewed.

During the site visits the Mr. Simpson verified:

- Collar locations are reasonably accurate by comparing six drill hole database collar locations with hand-held GPS readings.
- Drill hole collars are clearly marked with sturdy wooden fence posts, and the drill hole identity, orientation, and depth are inscribed onto a metal tag
- Down-holes surveys are routinely taken at approximately 50 m intervals using a Reflex single-shot unit.
- Drill logs compare well with observed core intervals.
- Core recoveries were generally high through the mineralized zones, with some core loss noted in zones of more intense fracturing and faulting.
- Specific gravity is determined using a water immersion method where the weight of the sample in air and in water is measured with a balance beam scale.
- Assay results for four samples of drill core collected in 2010 by Mr. Simpson and submitted for assay were consistent with database values at the same intervals.

12.1.1 Drill Hole Database Verification

Mr. Simpson independently audited the sample database for interval errors and missing sample intervals. AMEC reviewed the drill hole database records for transcription errors in 2011 and 2012. New Gold implemented database transcription error checks as part of the QA/QC program in 2012. Results of these reviews were reported in QA/QC reports prepared by New Gold and AMEC, both of which conclude acceptable levels of transcription errors were achieved.

12.2 Other Data Verification

Verification of metallurgical, hydrological, environmental baseline and geotechnical data is discussed in the relevant sections of this Report. The data are concluded to be adequate to support the Feasibility Study.

12.3 Comments on Section 12

The QP has reviewed the New Gold and AMEC reports and independently audited the sample database for interval errors and missing sample intervals. Four site inspections have been carried out since 2010. The QP concluded that the QA/QC with respect to the results received for the 2009, 2010, 2011, and 2012 exploration programs is acceptable, the protocols have been well documented and that the drill hole database is adequate to support the geological interpretations and Mineral Resources estimated in this Report.

13.0 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Metallurgical Testwork

13.1.1 Introduction

The industry recognized metallurgical laboratories that performed the testwork from 2008 and 2013 are listed in Table 13-1. The testwork programs were extensive and included the preparation of metallurgical composite samples for process development and variability response testing. The range of testing activities is summarized in Table 13-2, and the actual testwork programs are listed in Table 13-3.

13.1.2 Sample Selection for Process Development

Composite selection was based on the following primary considerations:

- Oxidative state
- Lithological domain
- Spatial representation (x/y/z) within the constraints of the FS pit shell
- Infilling the spatial representation on the main part of the orebody, with particular emphasis on the first five years of production while avoiding duplication with respect to previous programs
- Variable Au grades above a cut-off of ≥0.2 g/t of Au
- Ag grades in the approximate ratio of 5:1 to 8:1 Ag:Au on average, as well as specialized programs on high Ag material
- Available drill core inventory.

The following main changes have been made to the earlier metallurgical domain classifications:

- The andesite (AND) was split into two lithologies: AND (andesite) and ABX (andesite breccia).
- The FT (felsic tuff) was split back, and the new nomenclatures, LV (laminated volcanic) and FT relate to RHY and SIL5, respectively. RHY and SIL5 were tested independently under earlier programs.

Master composites of high Ag Transition and high Ag Sulphide were made up from samples from drill holes north of mine grid parallel 5,893,350 m N (Figure 13-1). Master composite of oxide material was not created as this oxidative state did not exist at these depths.

Table 13-1: Laboratories and Testwork

Laboratory	Testwork
Inspectorate	Preliminary metallurgical testing
G&T	Preliminary metallurgical testing, Comminution
SGS (Lakefield & Vancouver)	Whole Ore Leaching (WOL), Flotation Concentrate Leach (FCL), CN Detox
Dawson Metallurgical Labs	FCL
McClelland Laboratories	WOL, Heap Leach,
Pocock	Solid / Liquid Separation (SL)
MetSolve	WOL

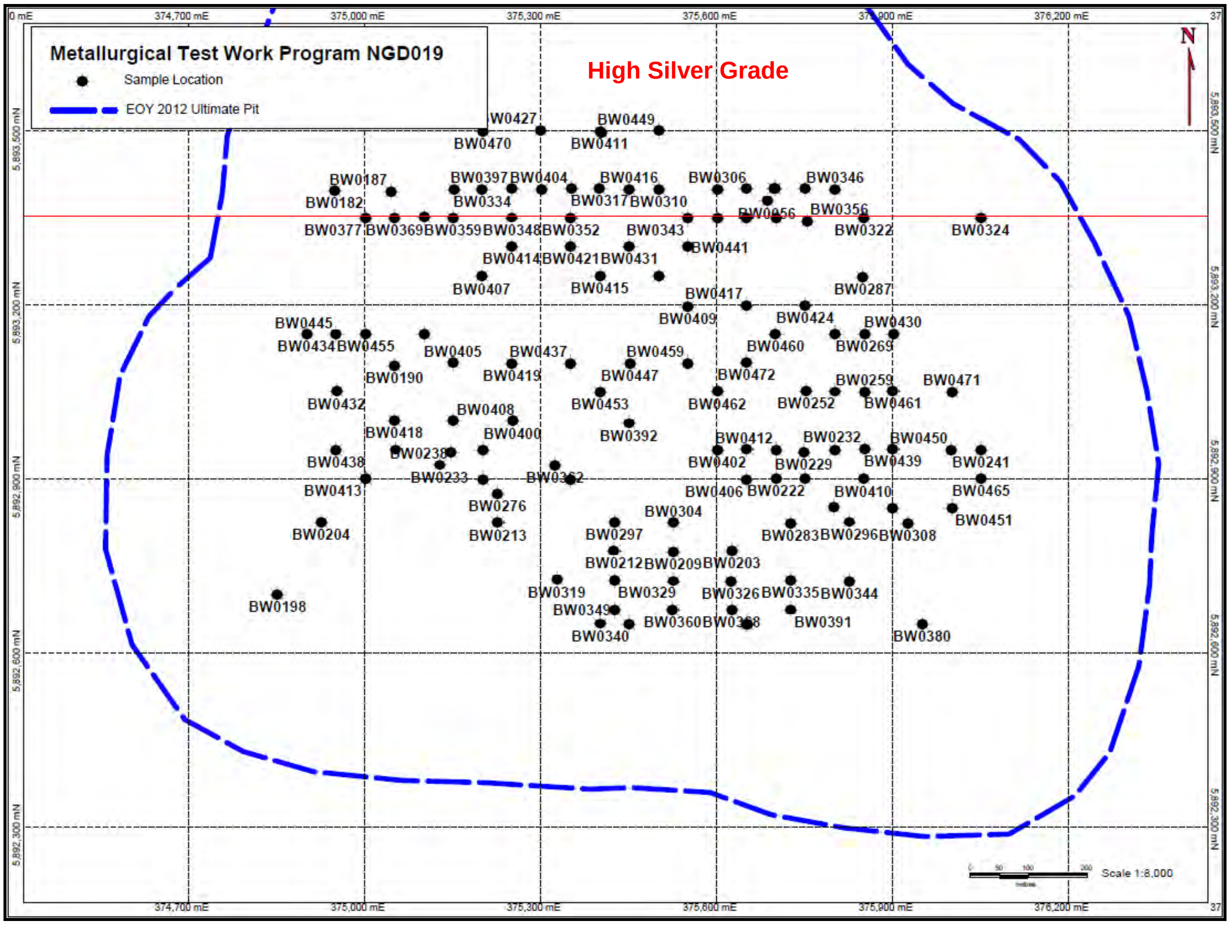
Table 13-2: Summary of Testwork on Metallurgical Composite Samples

Testwork	No. of Tests	Description	Notes
Grinding	6	Full DWT (drop weight tests)	
	140	Abbreviated SMC drop tests	
	144	BWI determinations	
	23	Bond low energy impact crushing work index	
	132	Bond abrasion index	
	1	Concentrate regrinding	
Flotation (181 tests)	51	Development work	Scoping tests, reagent screening, pH, slurry density
	26	Grind optimization	Nominal P_{80} between 75 and 212 µm
	12	Reagent optimization	Series of frothers, collectors, and activators
	66	Variability	On main lithologies
	10	Blends	By oxidative state
	16	Flotation concentrate regrind and leaching	Regrind between 17 and 75 µm Cyanide dosage (1 to 3.5 g/L)
Leaching (231 tests)	24	Development work	Scoping tests, aeration, oxygen, slurry density
	64	Grind optimization	Nominal P_{80} between 75 and 212 µm
	27	Cyanide dosage	Between 0.25 and 3.5 g/L NaCN addition
	106	Variability	On main lithologies
	10	Blends	By oxidative state
Heap Leach (168 tests)	151	Bottle roll leach tests	
		• Crush size	At 9.3 mm, 6.3 mm, 4.8 mm, 3.4 mm, 1.7 mm
		• Cyanide dosage	At 0.2, 0.3, 0.4, 0.5 g/L NaCN
		• High silver grade ore	At 6.3 mm, 4.8 mm, 3.4 mm, 1.7 mm, 212 µm, 75 µm
	17	Column tests	
Gravity Concentration (32 tests)	9	Gravity ahead of flotation tests	
	21	Gravity ahead of leach tests	
	2	E-GRG tests	LaPlante procedure on Transition and Sulphide master composites
Environmental	7	Cyanide destruction tests	Transition and Sulphide master composites

Table 13-3: Metallurgical Test Programs

Program No. (Report Date)	Laboratory	Description
RVC001 (2010)	Inspectorate, Richmond, BC, Canada	Whole ore leach, gravity, flotation characterization, and mineralogy
RCV002 (2010)	G&T Kamloops, BC, Canada	Whole ore leach, gravity, flotation characterization, and mineralogy
NGD001 (2011)	G&T Kamloops, BC, Canada	Bottle roll extractions on crushed feed for heap leach characterization
NGD002 (2011)	McClelland Laboratories Sparks, NV, USA	Bottle roll extractions on crushed and ground feed for whole ore leach and heap leach characterization
NGD003 (2011)	Dawson Metallurgical Labs, Salt Lake City, UT, USA	Flotation parameter optimization and variability Mineralogy of flotation products
NGD004 (2011)	G&T Kamloops, BC, Canada	Comminution parameters
NGD005 (2012)	Pocock Ind Salt Lake City, UT, USA	Thickening and filtration on flotation products
NGD006 (2011)	SGS Lakefield, Canada	Grinding parameters
NGD007 (2011)	G&T Kamloops, BC, Canada	Gravity and whole ore leach parameter optimization and variability. Flotation and regrind concentrate leaching optimization
NGD008 (2011)	SGS Vancouver, BC, Canada	Whole ore leach parameter optimization and variability
NGD009 (2012)	Dawson Metallurgical Labs, Salt Lake City, UT, USA	Flotation Concentrate Leach (FCL) Optimization, Recovery Definition, Variability
NGD010 (2013)	SGS Lakefield, Canada	Whole Ore Leach (WOL) Cyanide Destruction Optimization, Variability
NGD011 (2013)	SGS Lakefield, Canada	Environmental Program – water treatment, sample preparation, and environmental tests
NGD012 (2012)	Metso York, PA, USA	Bulk Flotation Concentrate – Special Jar Mill Grindability Test
NGD013 (2012)	G&T Kamloops, BC, Canada	Comminution parameters
NGD014 (2011)	McClelland Laboratories Sparks, NV, USA	Confirm heap leach extractions and agglomerate preparation requirements and strength characteristics
NGD015 (2012)	McClelland Laboratories Sparks, NV, USA	Whole Ore Leach (WOL) CIP optimization, recovery, variability
NGD016	SGS Lakefield, Canada	WOL pre-aeration, oxygen sparging, lead nitrate addition, grinding in cyanide tests. FCL confirmatory tests
NGD018 (2012)	FLSmidth Knelson Langley, B.C., Canada	E-GRG type gravity concentration test
NGD019 (2013)	McClelland Laboratories Sparks, NV, USA	Heap leach recovery definition and expanded reserve north to include high silver grade ore
NGD020 (2013)	G&T Kamloops, BC, Canada	Grinding (Infill and first 5 years)
NGD021	MetSolve Laboratories Inc. Langley, B.C., Canada	Whole Ore Leach (WOL) variability repeats

Figure 13-1: Drill Hole Locations for High Silver Grade Sampling



Note: Figure courtesy New Gold, 2013

Diagnostic leach tests on Transition and Sulphide high silver grade samples showed silver leach extractions consistent with the low silver grade samples, at about 65% leach extraction. Gold extraction was not significantly affected by the high silver content.

13.1.3 Mineralogy

The deposit is characterized as a strongly silicified, low gold grade, sulphide-hosted deposit with pyrite, pyrrhotite, and sphalerite present in the mineralization and with some oxidation near surface. Material classified as Oxide is characterized as having little to no sulphur present as sulphide. The Transition material represents incomplete oxidation of the sulphide, and Sulphide material is characterized by little oxidation of the sulphide or the host material. It is estimated that 3% of the mineralization is Oxide, 11% is Transition, and 86% is Sulphide.

The implications for plant recoveries, as derived from the mineralogical work carried out on the Blackwater samples, are as follows:

- A significant amount of gold occurs as fine but liberated native gold and electrum particles. This fraction is expected to leach completely within 24 hours or less. This fraction comprises between 20% and 60% of the total gold content depending on the oxidative state and lithology.
- A single occurrence of coarse gold (200 µm particle) was reported, which suggests that the coarse gold is not statistically abundant enough to warrant a dedicated processing strategy like gravity concentration.
- A second fraction—likely high-silver electrum particles, gold associated with sulphides, and low-exposure particles—exhibits considerably slower kinetics. Extraction becomes asymptotic after 24 hours, and cumulative extraction with the fine liberated gold is expected to reach up to 90% in 48 hours. This can be considered the theoretical limit for direct cyanide extraction for a primary grind with at P_{80} 180 µm and a 48 hour cyanide leach under ideal conditions.
- The balance of the precious metals are present either as finely disseminated values locked in sulphides, as slow-leaching species (tellurides), or encapsulated in silicates and carbonate gangue particles.
- No evidence of carbonaceous material was reported in the mineralogical reports.

13.1.4 Comminution

Testwork for Equipment Sizing

The Blackwater material can be considered fairly hard, and initial work in the PEA indicated the need for a comprehensive testing program to define not only the comminution process but also the applicability of the chosen comminution equipment packages to the deposit. As a result, a large number of samples, representative of both spatial distribution and domain, were selected and tested in four specific comminution characterization programs: NGD004, NGD006, NGD013, and NGD020. The four programs produced comprehensive comminution data. The tests included:

- Six full drop weight tests (DWT)
- 140 abbreviated SAG mill comminution (SMC) tests
- 144 ball mill work index (BWI) tests

- 23 Bond low energy impact crushing work index (CWi) tests
- 132 Bond abrasion Index tests.

Based on the JKTech global database of ore breakage characteristics, the Blackwater material is hard, with the Oxide and Transition materials being somewhat softer than the Sulphide.

Calculations and simulations were performed to select the size of the comminution equipment required to achieve the target nominal tonnage. This work used the grinding characteristics at the 75th percentile of the database for the Sulphide material (Table 13-4).

The JKSimMet and Bond equation methods were used to generate the recommended design and to determine the energy requirements for equipment selection. However, Morrell's equation, JKSimMet without phantom cyclone, and SAGDesign were also used to compare and adjust the selected methodology.

Simulation work was performed for all stages of comminution: gyratory crushing, SAG milling, pebble crushing, and ball milling. The programs utilized to simulate the circuit configurations and sizes include the Metso Bruno crushing program, Sandvik's PlantDesigner crusher selection program, JKSimMet, Molycop Tools 2.0, internal AMEC methodology, and the Bond method.

The base case flowsheet for the grinding circuit scenario of 60 kt/d at a P_{80} of 150 µm and 75% percentile hardness is two parallel lines, each with one SAG mill, pebble crusher, and ball mill. The methodology for sizing equipment incorporated the following formulas and constraints:

- SAG and ball mill power using the JKTech charge formula
- Ball mill power using Bond equation
- Ball mill residence time >1.2 minutes
- SAG mill pebble discharge rate (<0.17742 m^2(t/h) pebbles)
- SAG mill pulp discharge rate <366.12 $m^3/h/m^2$.

Primary Crusher Simulations and Determination of SAG Mill F_{80}

The primary crusher target was set at a P_{80} = 133 mm, with the open site setting (OSS) and feed size distribution adjusted through trial and error in the model until the desired product size distribution was achieved. New Gold then modelled the conditions required during blasting to achieve this desired ROM feed size distribution to the primary crusher.

Table 13-4: Ore Hardness Characteristics – Sulphide Ore

Characteristic	Unit	Minimum	Average	75th Percentile	Maximum
SG	-	2.63	2.74	2.78	2.90
Wic	kWh/t	10.80	13.51	14.9	15.88
Mic	kWh/t	6.1	9.9	11.0	12.9
DWi	kWh/m^3	5.3	9.0	10.0	11.9
Mih	kWh/t	11.8	19.1	21.3	25.0
A	-	54.2	78.8	94.8	100.00
b	-	0.74	0.40	0.29	0.23
Axb	-	49.6	30.4	27.5	23.0
ta	-	0.49	0.30	0.26	0.22
Mia	kWh/t	16.7	24.2	26.5	30.1
Bond BMWi	kWh/t	13.5	18.3	19.7	22.1
Bond RMWi	kWh/t	13.3	17.8	19.5	23.2
Ai	-	0.05	0.19	0.25	0.46

The Bond crusher power equation was used to determine the power requirements. A crusher work index of 15.9 kWh/t and a crusher factor of 75% were used. This factor takes into account the tendency of the fines to pass unimpeded through the crusher.

The sizing of the primary crusher and the feed size distribution curve generated by trial and error were validated using Metso's Bruno simulation program. This showed that a 60x89 gyratory crusher would operate at 69% load with a power draw of 445 kW and have a top capacity of 3,945 t/h.

JKSimMet Simulations for SAG Mills

Given the antecedents of JKSimMet modelling underestimating the SAG mill size and power requirements for hard ores, JKTech specifically recommended the use of the default JKSimMet variable rates for the AG/SAG model used for the Blackwater SAG mill. Outputs from this model are expected to be conservative.

JKTech recommends setting the reference F_{80} value to 100, always noting that results are expected to be conservative for soft ores but optimistic for hard ores. Therefore AMEC opted for using the Reference F_{80} value calculated from the DWi values using the following equation:

- F_{80} (mm) = 0.2*CSS*DWi$^{0.7}$

For the comminution design criterion of 75th percentile hardness, the calculated Reference F_{80} was 125 µm, resulting in two 36 ft SAG mills with 17 MW power requirement per mill. The SAG mill size was set to ensure that its power was sufficient and that the pebble and pulp discharge rates were within the recommended parameters.

Pebble Crusher

The pebble crusher power was calculated using the Bond equation with a modified crusher work index of 27.7 kWh/t (=CWi*(1+pebble crusher factor)) to take into account the hardness of the pebbles from SAG, which is generally greater than that of the average for the ore. The crusher model was then selected to provide the required 1,500 kW power draw.

JKSimMet Simulations for Ball Mills

In performing the ball mill sizing calculations, AMEC recommended excluding the phantom cyclone credit from the ball mill design calculation because:

- There is a good potential that the large amount of fines pre-requisite for the use of the phantom cyclone credit would not be present because the design incorporates large pebble ports (89 mm), which inevitably coarsens the grind, to prevent overloading the SAG mill.
- For the Blackwater design criterion of 60 kt/d at a P_{80} of 150 µm, there is no change in the ball mill sizing whether the phantom cyclone credit is taken or not.

The ball mill size was adjusted to ensure that the power draw for the charge is greater than the power draw required for size reduction, based on the Bond power equation, and that the ball mill retention time is within the recommended parameter of >1.2 minutes.

Maximum Tonnage

Simulations indicate that the maximum tonnage achievable for the average ore hardness is 67.7 kt/d when processing 100% Sulphide ore.

Finest Grind

For processing 60 kt/d of Sulphide ore at the average ore hardness, P_{80} 120 µm is the minimum achievable.

Hardness Model

A relationship was constructed relating tonnage as a function of ore hardness as measured by the Axb value but being constrained directly by ball mill limitations. For ore softer than the 50th percentile hardness, ball mill retention time was found to be the limiting factor. At the 50th percentile ore hardness, the constraint was ball mill power. For ores harder than the 50th percentile, the SAG mill is the limiting factor.

Based on the annual average values for Axb, an optimal mill throughput was developed—in conjunction with the use of a low-grade stockpile.

13.1.5 Heap Leach and Column Tests

The amenability of material in the Blackwater deposit to heap leach was tested in four programs, including bottle-roll tests on various crush sizes and column leach tests on selected samples.

The column testing results showed that the Blackwater mineralization was amenable to heap leach cyanidation at the evaluated feed sizes of 80% 6.3 mm, 80% 4.3 mm, and 80% 3.4 mm. In general, extraction was highest at the finer 3.4 mm crush, with the Sulphide LTFF and RHY lithologies displaying a slightly higher sensitivity to feed size than the AND lithology. Tests performed on low gold / high silver grade samples followed suit.

Gold recovery rates were moderate over the leach cycle, and no improvements are expected with leach cycles longer than the tested 168 hours. Cyanide consumptions were also moderate, 0.72 to 1.01 kg/t, but would be substantially lower in actual plant practice.

After an assessment of the capital and operating costs in combination with the expected metallurgical performance, heap leach cyanidation was rejected as the base flowsheet for the 2013 Feasibility Study.

13.1.6 Flotation Testwork

Flotation testwork was performed during the PEA and 2013 Feasibility Study to examine the applicability of this process route. The initial flotation testwork established the ability to produce a bulk flotation concentrate with gold recoveries averaging 90.5%, despite the variability in test conditions and resulting mass pulls. At 14 minutes' flotation time, a concentrate grade of approximately 12% sulphur at 90% sulphur recovery would be expected, resulting in a mass pull of approximately 9% at the average sulphur feed grades (1.3%) present in the mineralization.

After the initial PEA flotation testwork, program NGD009 was undertaken to test three master composites of Oxide, Transition, and Sulphide materials. The testing included evaluating the effects of primary grind, flotation kinetics, regrind size, and concentrate leach kinetics. An additional 50 Sulphide mini-composites were tested under conditions identified from testing the master composites for variability analysis. The tested Sulphide mini-composites correspond to felsic tuff, felsic lapilli tuff, volcaniclastic, and andesite lithologies.

A duplicates test format was used to determine the flotation kinetics of each composite. Once the concentrates were produced from each ore type, leach tests were performed in duplicate after regrinding to P_{80} 30 µm to provide leach rates and cyanide consumption parameters.

The information derived from the flotation concentrate leach testing program is summarized below.

- The Oxide master composite did not respond well to concentration by flotation compared to the other two oxidative states. Dawson Metallurgical Labs reported an average flotation recovery of 60% for gold and 28.8% for silver in a mass pull of 4.85%. Gold losses were reported to be evenly distributed throughout the size fractions. The gold that was recovered to concentrate did respond well to cyanide leaching, with over 95% Au and 93% Ag extracted by 72 hours. This, however, only translates to average recoveries of 59.7% and 26.6% for Au and Ag, respectively.
- The response of the Transition master composite to concentration by flotation was an average recovery of 78.1% for Au and 60.8% for Ag into a 12.2% mass pull. Reground flotation concentrate (P_{80} 30 µm) responded very well to leaching, with extractions of 97% for Au and 69% for silver after 24 hours. The overall recoveries were 75.8% Au and 42.0% Ag.
- The Sulphide master composite flotation response was tested at three nominal primary grinds: 80% passing 210 µm, 150 µm, and 75 µm. Increasing the grind fineness was not a significant factor in increasing Au recovery in the range tested, but the flotation of silver-bearing minerals improved with decreasing particle size. Flotation recovery at the nominal 180 µm grind averaged 89.1% for Au and 87.7% for Ag. The resulting flotation concentrate leach was also tested at three nominal regrind sizes: as produced, 40 µm, and 15 µm. At nominal regrind size of 30 µm, overall, the leach extraction of gold reached up to 95% in 72 hours, while silver extraction reached only 58%. Overall recoveries were 82.9% Au and 51.4% Ag.
- The leach kinetics and overall recovery were affected by the concentration of cyanide for both gold and silver, especially the latter. The optimum cyanide concentration was between 1 and 2 g/L NaCN.
- The results of variability testing on four Sulphide ore lithologies—andesite, felsic lapilli tuff, felsic tuff, and volcaniclastic—followed the general trend shown by the

Sulphide master composite. The main variation was observed in the mass pull trend for each lithology, with felsic lapilli tuff (FLPT) producing an average of 19% mass pull to concentrate while the other lithologies averaged 12%. This is likely associated with the slightly higher sulphide content reported for the FLPT lithology. Flotation and leach recoveries also followed the general trend shown by the Sulphide master composite, averaging 90% flotation recovery and 94% leach recovery. The volcaniclastic (VC) lithology showed the lower average flotation recovery, at 85%, while the andesite (AND) lithology showed the lower average leach extraction, at 90.5%.

After an assessment of capital and operating costs in combination with the expected metallurgical performance, flotation pre-concentration together with cyanidation was rejected as the basis for the 2013 Feasibility Study.

13.1.7 Leach Testwork

The standard conditions used for the FS leach testwork programs were established from the results of the testwork performed for the PEA. The conditions included a nominal grind size of P_{80} 125 µm, 50% solids, 36 hours' leach time, and a base cyanide concentration of 1 g/L. For the FS, confirmatory tests were performed for grind size, oxygen and lead nitrate addition, and cyanide and lime consumption. In addition, carbon-in-leach and variability work was undertaken.

Effect of Head Grade

Although the head grades of the Transition and Sulphide composites for the post-PEA testwork were considerably lower than those tested for the PEA, the impact on gold extraction—a decrease from 86.6% to 85.4% by whole ore leach (WOL)—is considered to be minor and validates the choice of WOL for the process. The response of samples to the WOL circuit has been very uniform, indicating a consistent form of gold mineralization within the Blackwater deposit. Additional evidence of uniform mineralization is the similarity in the shapes of the leach kinetic curves. Further, results of the independent FCL programs have shown the same consistency among high- and low-grade samples

Effect of Grind

Grind has been shown to have a consistent effect on the level of recovery.

Sulphide composite samples under program NGD015 indicate that increasing the coarseness of the material results in lower recoveries at standardized conditions of 30 hours' leach time and 1,000 g/L NaCN addition.

These non-kinetic leach tests indicate that grinding has a greater effect on gold leaching than previously observed, particularly at the coarser grind end. While grinding finer than 150 µm produces marginal extraction increments, the curve suggests a higher degree of variance and recovery in leach extraction for coarser grinds, possibly falling below 80%. Silver extraction showed less sensitivity to grind, with less than 4% difference in extraction between grinds of 212 µm and 75 µm.

Aeration

Testwork performed on Sulphide ore samples during program NGD016, which included pre-conditioning with air, resulted in lower cyanide consumption, but leach extractions at 30 hours remained consistent with the base line tests where no aeration was done. In spite of the potential improvement in cyanide consumption rates by pre-aeration, implementation is not practical given the scale of the Project.

The use of oxygen during cyanide leaching was also investigated. The results of the test showed potential for improving leach kinetics that warrants further investigation vis-a-vis oxygen addition costs and increased cyanide consumption rates.

Lead Nitrate

Scoping testwork at 200 g/t lead nitrate addition performed on Sulphide composite samples indicated no measurable increase in leach extraction or a decrease in cyanide consumption, despite improved kinetics. Hence, this reagent is not being considered for the process. Peroxide addition in combination with lead nitrate was also tested and resulted in faster leaching, but again this combination effected no change on leach extraction or reagent consumption.

Blends

The effect of processing varying blends of Oxide, Transition, and Sulphide composites in the WOL circuit was analyzed by comparing their measured and calculated gold and silver recoveries. There should be no metallurgical constraints on blending different oxidative types in the feed to the process plant. Measured and estimated leach extraction for gold correlate well on the unitary slope. The silver recovery dispersion reflects the wide response range of the silver mineralization in different oxidative states but does not present significant deviation from the unitary slope. The kinetic curves of the different blends were also checked, and no potential issues were identified. Cyanide and lime consumption remained within the master composites ranges.

Cyanide and Lime Consumption

Initial investigations of cyanide dosage at concentrations between 1 and 3.5 g/L of NaCN indicated no metallurgical advantage to adding more than 1 g/L. Gold extraction did not improve, and the increased silver extraction did not pay for the extra cyanide consumed. Additional testwork was performed for the 2013 Feasibility Study at cyanide additions in the range of 0.25 to 1 g/L.

The analysis of gold and silver recovery data from bottle-roll leaching of Oxide and Sulphide composites indicates that at a feed size of P_{80} 150 µm, a cyanide concentration of 0.5 to 1 g/L NaCN is the optimum range. For the Transition composite, the optimum cyanide concentration is estimated to be 0.75 to 1 g/L NaCN.

Allowing cyanide concentration to decrease through consumption, rather than maintaining it at 1 g/L, did not decrease gold recovery but decreased silver recovery significantly, indicating that the silver was more sensitive to the cyanide concentration.

High-Grade Silver Ore (North Silver Zone)

Additional work was performed to determine the leaching response of high-grade silver ore and to determine the best strategy for processing this material. Under program NGD019, scoping bottle-roll tests were performed on high-grade silver Transition and Sulphide samples for nominal P_{80} sizes of 6.3 mm, 4.8 mm, 1.7 mm, 212 µm, and 75 µm. The samples correspond to low gold / high silver grade material.

For the Sulphide material at 75 µm and 212 µm grinds, WOL at 30 hours' leach time using 0.5 g/L NaCN reported gold extractions around 85% and silver extractions of between 35% and 40%. This indicates that the low gold / high silver grade material in the northern area will leach in similar fashion as the rest of the ore. A Transition composite sample was also tested and reported slightly higher gold and silver recoveries, again consistent with the main Transition ore behaviour.

In addition to the WOL work, bottle-roll tests were performed on the coarser fractions, 6.3 mm, 4.8 mm, and 1.7 mm. This work showed moderate amenability to simulated heap leach for the Sulphide ore.

13.1.8 Flowsheet Selection

The three ore processing options considered in the PEA study—heap leaching, flotation with concentrate leaching (FCL), and whole ore leaching (WOL)—were assessed and compared further during multiple testing programs from 2008 to 2012 and in capital and operating cost estimates developed for each option. The results of these programs and analyses confirmed that WOL was the most appropriate option to pursue in the 2013 Feasibility Study.

Selection of the final flowsheet option to be carried into the 2013 Feasibility Study also considered inputs for mining, the TSF, environmental impact mitigation costs, and schedule and permitting risks.

After selecting the WOL flowsheet, a throughput optimization study comparing rates of 30 kt/a, 45 kt/a, and 60 kt/a was conducted. The highest NPV was for a mill feed rate of 60 kt/a. Outputs from the financial model also confirmed that economical recovery of precious metals from the Blackwater Project would require high throughput and high process availability while minimizing operating costs.

The calculated NPVs for the scenarios analyzed showed that the optimum throughput is around 60,000 t/d and return diminishes rapidly with increasing tonnage. Consequently, any throughput expansion will only be economically feasible if it comes with a moderate increase in capital and operating costs. An example is the addition of secondary crushing, whereas additional mill lines would likely not be viable.

13.2 Recovery Estimates

13.2.1 Basis of Estimate

In the 2013 Feasibility Study, recovery is estimated on the basis of the WOL process flowsheet. As a general rule in mineral processing plant operation, the head grade is expected to be the statistically dominant variable affecting overall gold recovery. It is also generally expected that recovery increases with head grade.

For Blackwater, precious metal values are characteristically finely disseminated throughout the ore body. This is consistent with the extraction versus head grade relationship over the grade range 0.2 to 1.2 g/t Au for the main lithologies tested, which showed no statistically significant dependency. The flat slope of the trend line curves suggests an average recovery through the head grade range, while the low correlation factors (R2) suggest that unmeasured factors like the ratio of free-milling gold to slow-leaching gold (electrum, telluride associated) may be a source of variability, but not at a large scale.

For the purpose of plant recovery estimation and extraction equations, the samples are classified by oxidative state, as this has proven to be the primary driver of the metallurgical performance.

13.2.2 Methodology

AMEC and New Gold agreed on the following methodology to generate a normalized database for the correlation of head grade–leach extraction and for plant recovery estimates:

- Use data from test programs NGD003, NGD007, NGD008, and NGD015 within a master database
- Ensure that any test selected for the database meets the following criteria for each oxidative state:
  - Nominal grind between 125 and 150 µm
  - Head grade >0.3 g/t
  - Residue grade >0.05 g/t Au to remove high-recovery outliers due to laboratory error and proximity to FAA assay limits
- Normalize recoveries to 125 µm grind and 36 hours' leach time to account for the effect of grinding in cyanide, residence time in the pre-leach thickener, and the finer size distribution of the Sulphide minerals due to the grinding circulating load
- Use the resulting sets of data to generate histograms and confidence intervals, applying a 2 times standard deviation (2δ) criterion to remove most of the high- and low-end outliers
- Use the data within the defined confidence interval to calculate the weighted gold and silver leach extractions for each oxidative state
- Populate the 2013 Feasibility Study mass balance with estimated leach extractions, providing the ability to determine the impact of recycles.
- Use the normalized database generated by the above criteria to generate gold and silver recovery equations.

Equations for gold and silver leach extractions are listed in Table 13-5. These equations relate the extractions to head grade as provided by the test results and were used to optimize the design of the pit for the determinations of Mineral Resources and Mineral Reserves.

Within the block model, no cap was set for gold extraction for Sulphide and Transition mineralization because there is no indication that higher gold grades are associated with higher electrum or slow-leaching gold minerals. For these mineralization types, gold deportment is expected to be uniform through the gold head grade range. Given the small number of samples available for Oxide mineralization, a cap is required at 95% extraction, equivalent to a gold head grade of 1.92 g/t Au.

Table 13-5: Leach Extraction Equations for Gold and Silver by Oxidative State

Oxidative State	Leach Extraction Equation Au	Leach Extraction Equation Ag
Oxide	y=7.2183ln(x)+90.295	y=4.7597ln(x)+57.03
Transition	y= -1.656ln(x)+84.469	y=11.069ln(x)+42.141
Sulphide	y=3.9438ln(x)+87.92	y=2.1334ln(x)+41.419

For silver, capping was implemented because the testing for higher silver grade material did not indicate improved extraction.

13.2.3 Plant Loss Assumptions

Total metallurgical losses, mainly due to solution losses and attrited carbon losses, were estimated at 1.1% for gold and 3.3% for silver.

The solution loss estimate is based on CIP carbon loading simulations at reported solution tails of 0.002 g/t of gold and 0.068 g/t of silver for a seven-tank CIP circuit moving 40 t/d of carbon.

Carbon losses are based on a carbon consumption of 27 g/t and gold loadings of 50 g/t and 180 g/t of gold and silver, respectively, as indicated by the carbon loading simulations on the seventh tank of the series.

13.2.4 Mass Balance Assumptions

Table 13-6 and Table 13-7 summarize the leach extraction and the efficiencies used in the mass balance for estimation of the plant recoveries to be carried in the 2013 Feasibility Study.

The metal recovery equations relating recovery to head grade by ore type and the plant losses were loaded into the geological resource block model in order to estimate the overall metal recovery over the LOM (Table 13-8). Table 13-9 shows projected feed to the mill, composition by oxidative state, and precious metal grades and recoveries by year.

Table 13-6: 2013 Feasibility Study Plant Recovery Estimate for Gold

Parameter	Unit	Oxide	Transition	Sulphide	LG Stockpile
Leach Extraction	%	88.7	86.0	86.1	80.3
Grind in Cyanide Extraction	%	25.0	25.0	25.0	25.0
Leach Extraction, calculated	%	85.0	81.3	81.5	73.7
Carbon Adsorption Efficiency	%	99.4	99.4	99.4	99.4
Elution Efficiency	%	99.5	99.5	99.5	99.5
Grind – P_{80}	µm	150	150	150	150
Leach time	h	30	30	30	30
CIP Time	h	1.5	1.5	1.5	1.5
Average Head Grade	g/t	0.65	0.58	0.57	0.2
Estimated Plant Recovery	%	88.5	85.7	85.8	80.0

Table 13-7: 2013 Feasibility Study Plant Recovery Estimate for Silver

Parameter	Unit	Oxide	Transition	Sulphide	LG Stockpile
Leach Extraction	%	65.5	58.3	45.0	37.6
Grind in Cyanide Extraction	%	10.0	10.0	10.0	10.0
Leach Extraction, calculated	%	61.7	53.6	38.9	30.7
Carbon Adsorption Efficiency	%	97.2	97.2	97.2	97.2
Elution Efficiency	%	99.5	99.5	99.5	99.5
Grind – P_{80}	µm	150	150	150	150
Leach Time	h	30	30	30	30
CIP Time	h	1.5	1.5	1.5	1.5
Average Head Grade	g/t	5.6	4.4	3.2	2.7
Estimated Plant Recovery	%	65.0	57.7	44.6	37.3

Table 13-8: LOM Recovery Estimate from Resource Block Model

	Gold		Silver	
Material Type	LOM Head Grade (g/t)	LOM Recovery (%)	LOM Head Grade (g/t)	LOM Recovery (%)
Oxide	0.55	86.5	10.1	67.5
Transition	0.73	83.1	7.3	64.8
Sulphide	0.81	87.1	4.2	43.9
Low Grade Stockpile	0.41	84.4	10.2	50.8

Table 13-9: Recovery Estimate from Mine Schedule

Project Year	Oxide (%)	Transition (%)	Sulphide (%)	Mill Feed (with Stockpile) (t)	Au Grade (g/t)	Ag Grade (g/t)	Gold Recovery (%)	Silver Recovery (%)
Preproduction	-	-	-	-	-	-	-	-
	-	-	-	-	-	-	-	-
Year 1	11	65	24	10,803,900	0.90	5.5	84.8	57.6
Year 2	6	37	57	21,898,800	0.91	5.4	86.4	53.1
Year 3	11	20	69	21,898,800	1.00	5.8	87.8	53.7
Year 4	8	17	75	21,898,800	0.71	5.4	86.1	53.9
Year 5	5	11	84	21,898,800	0.75	5.0	86.3	50.9
Year 6	6	13	81	21,903,000	0.83	6.2	87.4	51.1
Year 7	2	2	96	21,903,000	0.80	4.3	87.2	45.5
Year 8	0	3	97	21,903,000	0.94	6.7	88.3	45.6
Year 9	0	2	98	21,903,000	0.84	6.1	87.8	44.9
Year 10	0	0	100	21,903,000	0.69	3.8	86.0	43.7
Year 11	0	0	100	21,902,400	0.61	3.4	85.7	43.6
Year 12	0	0	100	21,902,400	0.63	3.0	85.8	43.4
Year 13	0	0	100	21,902,400	0.69	3.3	86.2	43.4
Year 14	0	0	100	21,902,400	0.88	2.4	86.9	43.0
Year 15	7	21	72	21,900,000	0.40	10.3	84.4	50.6
Year 16	7	21	72	21,900,000	0.40	10.3	84.4	50.6
Year 17	7	21	72	5,121,071	0.40	10.3	84.4	50.6
Total / LOM Average				344,444,771	0.74	5.5	86.6	49.1

13.3 Metallurgical Variability

The samples used for variability testing in metallurgical test program NGD015 were selected by New Gold personnel. A total of 86 mini-composites were generated, including ores representing Oxide, Transition, and four main lithological Sulphide types.

Overall metallurgical results show that all the mini-composite variability samples, including the four Sulphide ore types, were readily amenable to WOL at 125 µm grind and 36 hours' leach time.

Cyanide consumption and lime requirement averages for the Transition mini-composites were notably higher than for the rest of the samples tested, as observed in previous programs.

Oxidative state remains the single most important source of variability in the deposit, as observed in the gold and silver head grade versus tail grade charts (Figure 13-2 and Figure 13-3).

Ore hardness parameters Axb and BWi were mapped in relation to the pit shell. No discrete comminution behaviour units were identified from the mapping, which supports the assertion that the grinding characteristics are very consistent (hard to very hard Axb and Hard BWi) throughout the pit shell (Figure 13-4 and Figure 13-5).

Spatial mapping of gold and silver recovery was also undertaken. No problem areas or significant variation through the pit shell are observed (Figure 13-6 and Figure 13-7).

13.4 Deleterious Elements

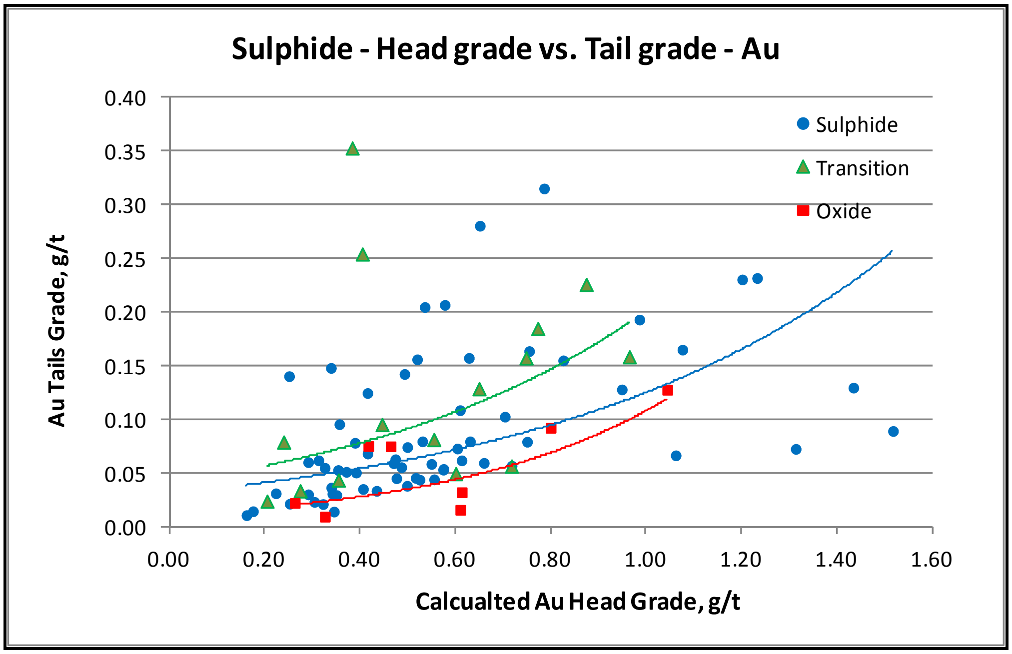
No elements that could be considered deleterious in the proposed process were identified from the testwork.

13.5 Comments on Section 13

The QPs note the following:

- An extensive metallurgical testwork program was carried out over the period 2008 to 2013 on samples that were composited to represent process plant feed in the mine development plan
- The selected process route is WOL
- Mineralogical and diagnostic leach testing indicated that the primary areas of investigation required to optimize WOL processing were primary grind size, reagent addition, and leach retention time
- Testwork results were used to assess the relationship between oxidative state, head grade projection for Au and Ag, and recovery, and to determine the following LOM metallurgical recoveries for WOL:
  - Au recovery – 86.6%
  - Ag recovery – 49.1%.

Figure 13-2: Variability Tests Results for Gold by Oxidative State – NGD015



Note: Figure prepared by AMEC, 2013

Figure 13-3: Variability Test Results for Silver by Oxidative State – NGD015

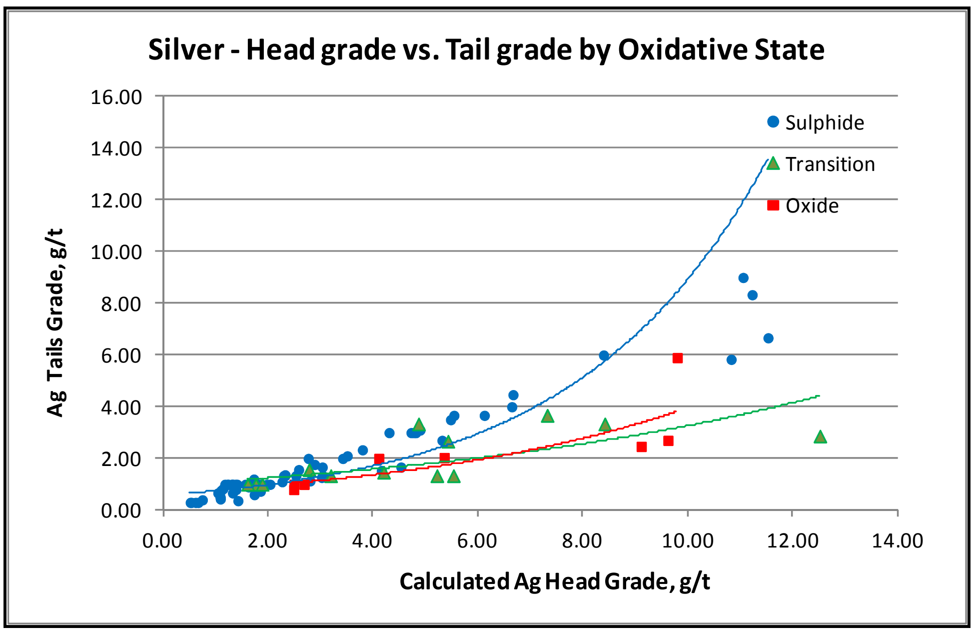


Figure prepared by AMEC, 2013

Figure 13-4: Axb Spatial Mapping

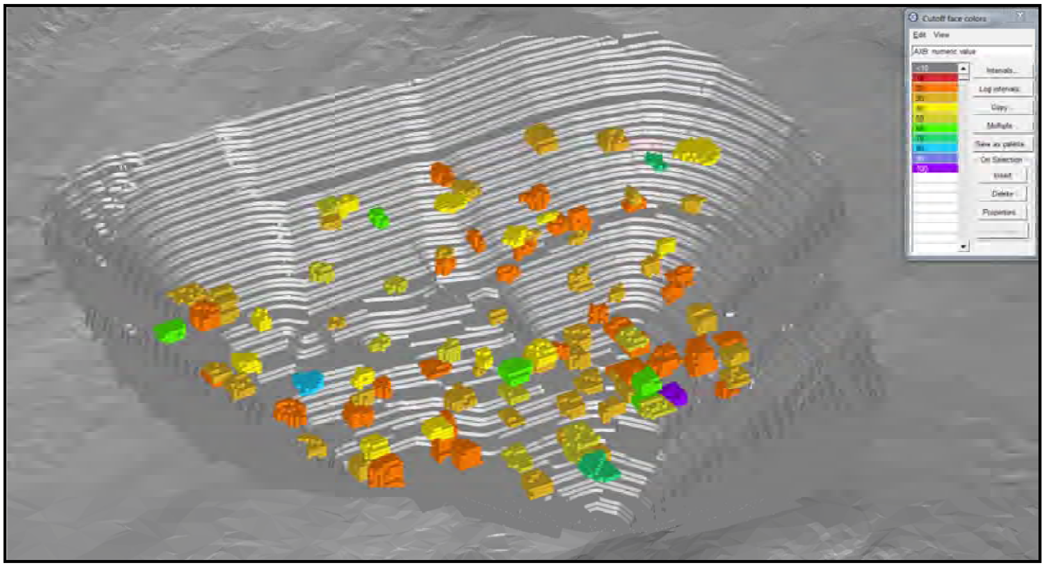


Figure prepared by AMEC, 2013

Figure 13-5: BWi Spatial Mapping

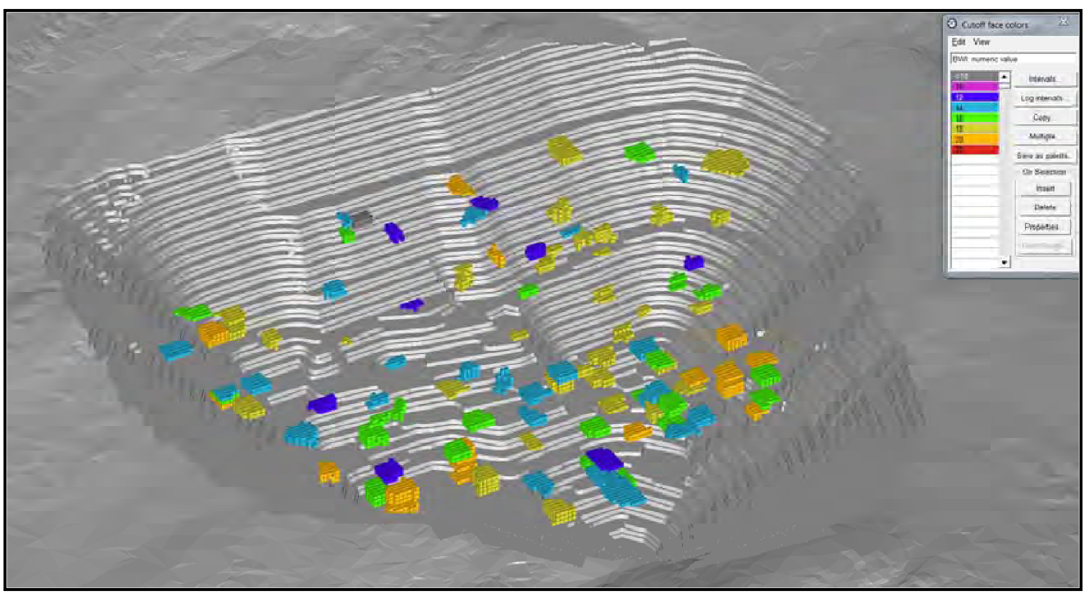


Figure prepared by AMEC, 2013

Figure 13-6: Gold Recovery Spatial Mapping

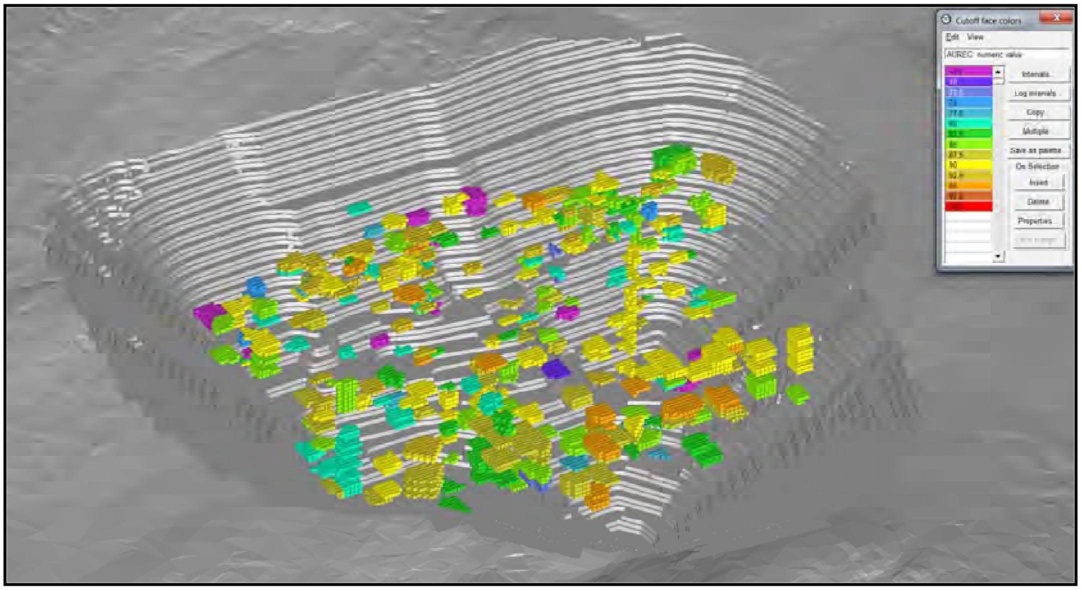


Figure prepared by AMEC, 2013

Figure 13-7: Silver Recovery Spatial Mapping

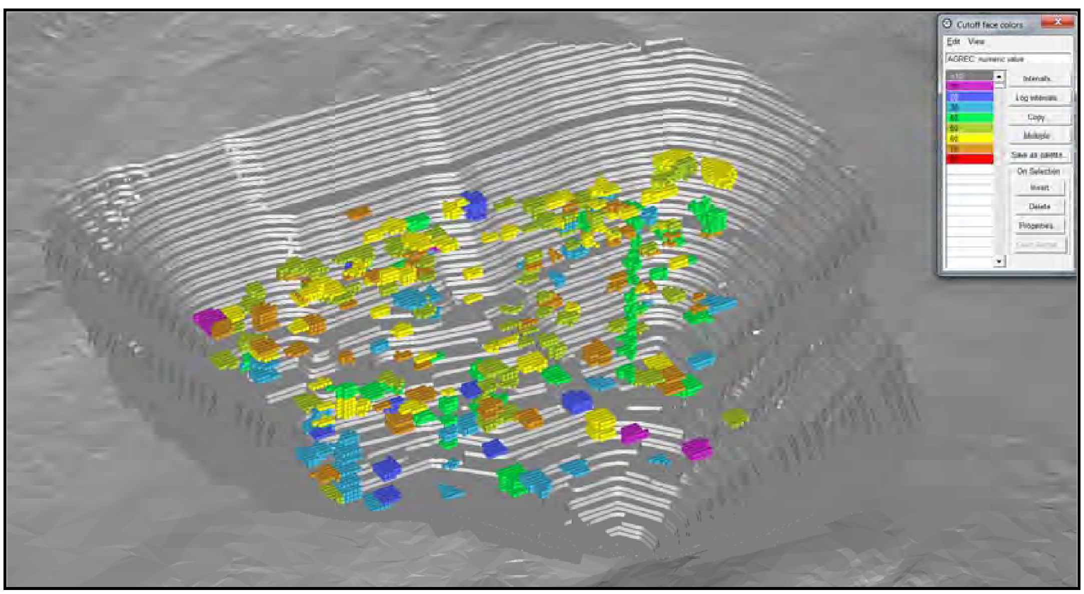


Figure prepared by AMEC, 2013

14.0 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The 2013 Feasibility Study resource estimate includes data for all drilling completed by Richfield and New Gold between August 1, 2009, and January 16, 2013. The estimate was prepared using Geovia-Surpac Vision© software by Ronald G. Simpson of GeoSim.

The total sample database for the Blackwater Gold Project contains results from 1,046 core holes totalling 315,319 m drilled between January 1987 and January 15, 2013. Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for geologic modelling, statistical analysis, or grade estimation.

Of these, 1,010 holes were drilled since the start of 2009, including 974 holes drilled to delineate the Blackwater Mineral Resource, four holes drilled for metallurgical testing in 2011, and 11 geotechnical holes completed in 2012. Fourteen waste rock characterization holes (BWWR series) were also completed in 2012. Portions of these holes occurring within the resource area were analyzed and used in the resource estimation. Data from seven pilot (PH series) holes were also included.

A block model was created in Geovia-Surpac Vision© software using a block size with dimensions of 12 m x 12 m x 12 m. This is an increase from the 10 m x 10 m x 10 m block size used in the PEA. The 12 m bench height for the 2013 Feasibility Study model was selected based on dilution, equipment productivity and costs for an assumed production rate of 60,000 t/d.

14.2 Models

14.2.1 Geological Model

The geological model used for this study was developed by New Gold geologists between March 2012 and January 2013. Modelling involved the re-logging of 901 core holes and the use of a specTERRA™ field-portable short wave infrared (SWIR) spectrometer to identify alteration assemblages.

Lithologies were consolidated into six categories:

- Overburden (OB)
- Laminated volcanics (LV)
- Fragmentals (FRAG)

- Andesite (AND)
- Sediments (SED)
- Intrusive phases (INT).

Indicator kriging was used for preliminary modelling to guide a sectional interpretation honouring drill hole intercepts. Final wireframe models of the principal units were generated using the sectional interpretation. The INT unit was not modelled because few drill intercepts were available.

14.2.2 Structural Model

A selection of 462 drill holes was re-logged by core photographs to identify major fault intervals used to create the structural model. A major north south trending fault dissects the orebody and east–northeast-trending faults along UTM easting 375,600E. This fault represents a well-defined disruption in lithology, alteration, and mineralization patterns and was used to subdivide the model into two structural domains, one to the east of it and one to the west.

14.2.3 Alteration Model

The alteration model was initially developed using the methodology outlined in Section 9.6.

An additional 607 drill holes were re-logged from core photos, and a continuous down hole alteration interpretation domain table was constructed for each drill hole using the POT versus SER altered categories. Any interval that was visibly bleached and altered in excess of approximately 50 vol% was categorized as SER and all others as POT. The NH4 alteration assemblage could not be used in the photographic re-logs because those minerals can only be identified by spectroscopy.

The alteration model indicates the presence of two centres of texture destructive sericitic alteration cored by the ammonium-bearing overprint and haloed by early potassic alteration and hornfelsed andesite. Statistical analysis shows that the NH4 and SER alteration domains closely coincide, and they were therefore combined as a single domain (SER) for resource modelling. A wireframe encompassing the distribution of sericitic alteration was generated from a sectional interpretation of detailed drill hole information. An indicator model of alteration types, simulations of logged silica and sericite intensity, and block estimates of alkali cation percentages, particularly aluminum, provided additional support to the modelled alteration domains.

Statistical comparisons of the modelled sericite and silica between gold and silver, respectively, demonstrate the relationships between mineralization and alteration.

Similar comparisons also demonstrate the relationships between gold and logged sulphides, particularly pyrite and dendritic black sulphide (DBS) mineralization.

14.3 Sulphide Model

Information from drill logs relating to visually estimated logged sulphide mineralization was compiled into fields representing each particular species abundance and style, using four key mineral species: pyrite, sphalerite, pyrrhotite, and DBS. Simulation models of pyrite and the combined mineral percentage of pyrite and DBS, sphalerite, logged sericite, and logged silica were completed.

It was found that the combination of pyrite and DBS had the strongest and most direct correlation to gold mineralization. The lower simulated mineral abundance of 0.5 vol% pyrite + DBS was useful for outlining the overall limits of gold and silver mineralization, while the higher mineral abundances of 1 vol% and 2 vol%, respectively, were found to highlight structures and fault intersections where sulphide “hot spots” occur. These zones often correspond to long, high grade gold intersections in drill core.

An attempt was also made to simulate the distribution of sulphide veins but was not successful because the Project lacks a uniform dataset of logged veining percentages.

A sectional model was interpreted based on the probability model of the 0.5 vol% pyrite + DBS distribution and wire-framed to create a “pyrite shell.” This domain was used as an outer constraint of gold estimation in the model.

New Gold’s current interpretation is that controls on silver mineralization are different from those for gold, but further work on refining the understanding of the silver mineralization is required. The present understanding is limited to an apparent association between silver, sphalerite, sericite, and silica, which may be useful in developing a better model of the controls on silver mineralization in the future.

14.4 Grade Shell Model

A grade shell domain was generated for gold within the mineralized pyrite shell to better delineate areas of very low grade material. The domain was created by indicator kriging of 5 m composites using a threshold of 0.1 g/t Au.

A second grade shell domain was created to constrain a high-silver/low-gold zone in the upper north area of the deposit that was mostly outside the Pyrite Shell. This domain was created by indicator kriging of 5 m composites using a threshold of 10 g/t Ag and is referred to as the “North Silver Zone.” This zone extends approximately 500 m east west and 150 m north-south and is approximately 75 m thick. The zone extends to surface and dips gently to the north.

The east and west structural blocks were further subdivided into five structural domains to model changes in orientation of the mineralized zone and continuity trends for gold. These domains were grouped together for variography where grade continuity showed consistent trends.

Figure 14-1 shows the structural domains in relation to the pyrite shell. Figure 14-2 shows the final grade shell models.

14.4.1 Weathered Surface Models

A bedrock surface was modelled by creating profiles based on drill hole intercepts and generating a digital elevation model. Surfaces representing the base of the weathered oxide and oxide-sulphide transition zones were also generated and used to code the blocks.

14.5 Exploratory Data Analysis

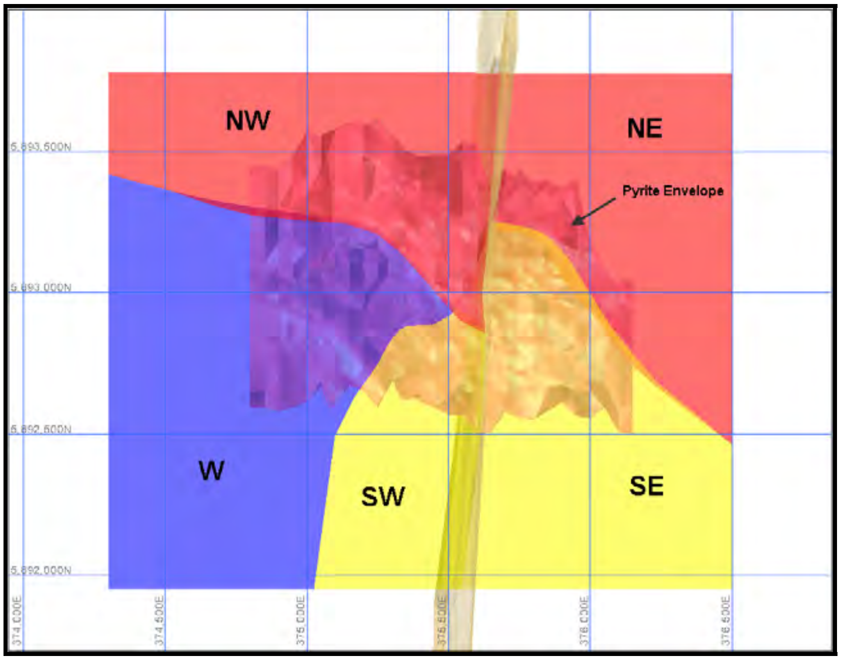
Within the pyrite shell, gold and silver show a weak positive correlation (R^2 or correlation coefficient = 0.22) and a linear regression yields a very low R^2 value of -0.03. Consequently, geostatistics and block grade estimation for silver was not constrained by the pyrite shell but only by the outer structural domain boundaries with the exception of the North Silver Zone lying on the north side of the deposit close to surface (refer to Figure 14-2).

Cumulative probability curves, frequency plots and a basic statistical evaluation of the raw Au and Ag grades were undertaken. Frequency distributions are highly skewed, approaching log normal with no significant bimodality evident for gold.

Contact profiles showed significant changes between the east and west structural domains and the alteration domains. Profiles across lithological boundaries between the andesite, fragmental volcanics and laminated volcanics did not show rapid changes close to contacts.

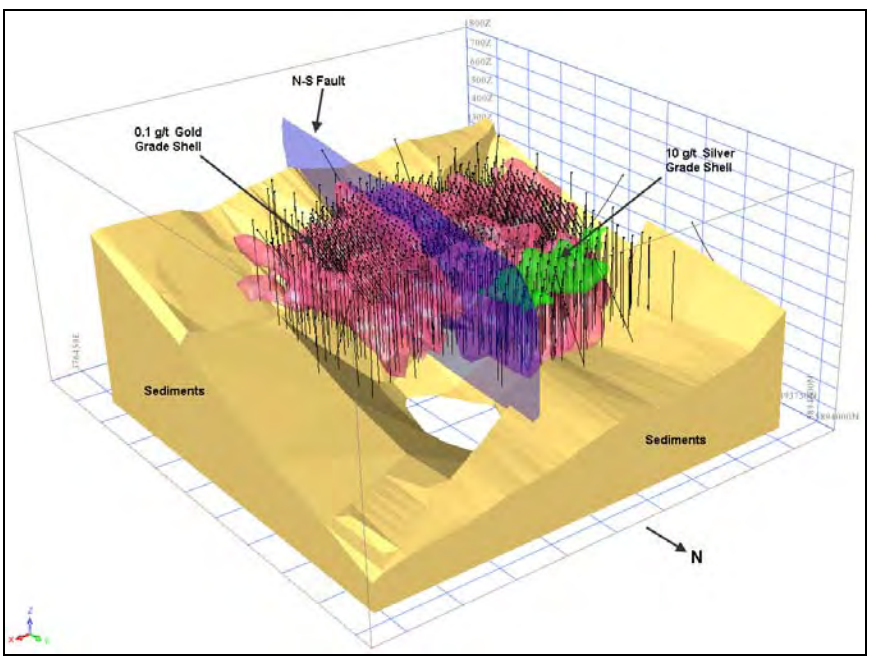
It was concluded that lithological contacts were not as strong a control on gold mineralization as alteration boundaries and it was decided not to impose hard boundaries between lithologies. Semi-hard boundaries were imposed between the sericite envelope and the potassic alteration zone and between the east and west structural domains.

Figure 14-1: Structural Domains showing Pyrite Envelope



Note: Figure prepared by Geosim, 2013.

Figure 14-2: Grade Shell Models



Note: Figure prepared by Geosim, 2013.

14.6 Grade Capping/Outlier Restrictions

Outlier grades were assessed by examining probability plots of the Au and Ag assays within the pyrite shell. Separate probability plots were created for assays subdivided inside and outside of the sericite envelope, inside and outside of the gold grade shell and located in the west or east structural blocks.

No significant differences were observed between east and west, therefore a gold grade capping threshold of 45 g/t Au was selected within the gold grade shell. Gold assays outside of the gold grade shell were capped at 5 g/t Au (Table 14-1).

Silver assays falling within the pyrite shell were capped using a 150 g/t Ag threshold. Silver assays falling within the silver grade shell were not capped. Gold assays falling within the silver grade shell were capped at 1 g/t Au.

The amount of gold metal removed using the variable capping thresholds of 45 g/t Au inside the gold grade shell and 5 g/t Au outside the gold grade shell is 3.46%. The amount of silver metal removed by capping at 150 g/t Ag inside the pyrite shell threshold is 2.03%. The amount of gold metal removed using the capping threshold from the previous mineral resource estimate (50 g/t Au) would be 2.4%.

It was concluded that the amount of metal to remove by capping has not changed significantly from the 2012 end-of-year mineral resource estimate.

The indicated amounts of metal removed are considered to be reasonable for a project at the feasibility study stage of development.

14.7 Composites

14.7.1 Gold and Silver

Samples were composited into 5 m lengths for gold within the individual domains. The silver composites that were created used hard boundaries between the two major structural domains outside the 10 g/t Ag grade shell. The composites were not constrained to the pyrite shell, as this appeared to have no significant influence on silver grades. A separate composite data set was generated inside the 10 g/t Ag grade shell. Samples outside the grade shell were capped prior to compositing.

Table 14-1 Capping Statistics by Domain

Domain	No. of Assays	No. Capped	Cap Level g/t	% Capped	Uncapped Mean (g/t)	Capped Mean (g/t)	% Metal Removed
All Au in Pyrite Shell	233,130	136	45/5	0.06%	0.462	0.446	-3.46%
Au in Gold Grade Shell - East	48,206	45	45	0.09%	0.656	0.617	-5.95%
Au in Gold Grade Shell - West	105,671	48	45	0.05%	0.655	0.645	-1.53%
Inside Pyrite Shell outside Gold Grade Shell - East	26,107	24	5	0.09%	0.084	0.080	-4.76%
Inside Pyrite Shell outside Grade Shell - West	53,146	19	5	0.04%	0.086	0.081	-5.81%
Au in North Silver Zone	2,454	5	1	0.20%	0.054	0.050	-7.41%
Ag outside North Silver Zone	286,799	127	150	0.04%	3.45	3.38	-2.03%

14.7.2 Acid-Rock-Drainage

Secondary elements associated with Au–Ag mineralization including As, Ca, Cd, Cu, Ni, Pb, Total S, Sb and Zn were estimated by the inverse distance to the second power (ID^2) method in a single domain using 10 m down-hole composites. Results were used to develop the acid-rock-drainage (ARD) model described in Section 18.

14.8 Density Assignment

A total of 31,699 density measurements made on core sampled between 2009 and the end of 2012 have been used for density modeling. The samples were statistically analyzed by weathered zone and outliers removed.

Separate ID^2 interpolations were carried out within the oxide, transition and sulphide zones using hard boundaries. Hard boundaries were also used between the three main lithological domains. An isotropic search ellipsoid was used with a maximum search distance of 100 m. A minimum of four and maximum of 16 samples were required to estimate a block. A maximum of three samples was allowed from a single drill hole.

Blocks that were not estimated were assigned the mean value for the corresponding lithology and material type. The density of the overburden was assigned an assumed value of 2.0 g/cm^3.

14.9 Variography

Nested spherical directional semi-variograms for Au were modeled in Geovia–Surpac Vision© software using 5 m capped composites falling within three of the grouped structural domains.

For gold, composites were further restricted to the pyrite shell with a separate population modeled within the north silver zone. For Ag, the composites were combined into two populations, one inside the north silver zone and one including all composites outside of this domain. Variogram model parameters for Au and Ag are shown in Table 14-2 and Table 14-3.

14.10 Estimation/Interpolation Methods

Gold and silver grades within the corresponding zone domains were estimated in three passes within the pyrite shell using ordinary kriging (OK). Grades within the north silver zone were estimated in two passes using OK. A single-pass, nearest-neighbour (NN) estimate using 12 m composites was also carried out for use in model validation.

Grade estimation for gold within the pyrite shell was carried out using a combination of hard and soft boundaries between fault domains, alteration types, grade shells, and structural domains. The area drilled on a 35 m x 35 m drill spacing was estimated using soft boundaries between alteration and structural domains for the first two search passes, with the only hard boundary being the gold grade shell. In pass three, hard boundaries were imposed for the sericite envelope and the north-south fault as well as the gold grade shell.

For the area outside the 35 m x 35 m drill spacing, soft boundaries were used for the first pass only, and hard boundaries were used between alteration, fault, and grade shell domains for passes two and three.

The maximum and minimum number of composites used for block gold grade estimation and the maximum search distances for the interpolations were adjusted for each structural zone to compensate for differences in sample density and to assist in smoothing the grade distribution to match a target model.

Silver grades were estimated by OK within the pyrite shell using the same search distance parameters as those used for gold. The silver composites were not limited to the Pyrite Shell but did exclude those falling within the north silver zone grade shell.

Table 14-2: Variogram Model Parameters – Au in Pyrite Shell

Domain	Azimuth	Dip	co	c1	a1	c2	a2
	315	-80	0.304	0.45	20	0.246	190
West 1000	30	3	0.304	0.45	17	0.246	175
	300	10	0.304	0.45	20	0.246	105
	300	-40	0.19	0.62	13	0.19	300
North 2000/3000	36	-8	0.19	0.62	9	0.19	200
	315	49	0.19	0.62	13	0.19	150
	90	-45	0.32	0.532	15	0.147	203
South 4000/5000	90	45	0.32	0.532	15	0.147	160
	180	0	0.32	0.532	15	0.147	115

Table 14-3: Isotropic Variogram Model Parameters – Ag

Domain	Item	co	c1	a1	c2	a2	c3	a3
North Ag Zone	Ag Uncapped	0.2	0.4	12	0.3	40	0.1	100
North Ag Zone	Au cap 1 g/t	0.15	0.27	17.5	0.25	31	0.33	140
Outside Ag Zone	Ag Cap 150 g/t	0.3	0.41	23.3	0.14	75	0.15	164

Composites falling within the north silver zone in the northern part of the deposit were used to estimate the blocks within this domain. All blocks were estimated in the first two passes, so a third pass was not necessary.

Table 14-4 provides the parameters for the estimation of gold inside the pyrite shell. Table 14-5 contains the interpolation parameters outside the north silver zone; Table 14-6 summarizes the parameters inside the zone.

14.11 Block Model Validation

Model verification was initially carried out by visual comparison of blocks and sample grades in plan and section views. The estimated block grades showed reasonable correlation with adjacent composite grades.

A global comparison of OK, NN, and composite mean values showed a reasonably close relationship.

Swath plots were generated to assess the model for global bias by comparing kriged, inverse distance weighting to the second power (ID^2) and NN estimates within slices 36 m in width through the deposit. Results show a reasonable comparison between the interpolation methods, particularly in the main portions of the deposit.

Table 14-4: Grade Interpolation Plan for Gold inside Pyrite Shell

First Pass (or within 50 x 25 m drill pattern)						First Pass						Second Pass (50 x 25 m drill pattern constraint)					
						Search Distance			No. of Composites			Search Distance			No. of Composites		
Variogram Subdomain	Structural Domain	Grade Shell	Alter Code	Domain Code	Comp Code	Major	Semi-Major	Minor	Min	Max	Max / hole	Major	Semi-Major	Minor	Min	Max	Max / hole
1000	300	10	3	1313	313	30	27.5	16.6	3	8	2	60	55	33.1	3	8	2
2000	300	10	3	2313	313	50	33.3	25	7	24	6	100	66.7	50	7	24	6
3000	300	10	3	3313	313	50	33.3	25	7	24	6	100	66.7	50	7	24	6
4000	300	10	3	4313	313	30	23.6	16.9	3	8	2	60	47.2	33.9	3	8	2
5000	300	10	3	5313	313	30	23.6	16.9	3	8	2	60	47.2	33.9	3	8	2
1000	300	20	3	1323	323	30	27.5	16.6	3	8	2	60	55	33.1	3	8	2
2000	300	20	3	2323	323	50	33.3	25	7	24	6	100	66.7	50	7	24	6
3000	300	20	3	3323	323	50	33.3	25	7	24	6	100	66.7	50	7	24	6
4000	300	20	3	4323	323	30	23.6	16.9	3	8	2	60	47.2	33.9	3	8	2
5000	300	20	3	5323	323	30	23.6	16.9	3	8	2	60	47.2	33.9	3	8	2
Second or Third Pass outside 50 x 25 m drill pattern						**Second Pass**						**Third Pass**					
1000	100	10	1	1111	111	60	55	33.1	3	8	2	150	137.6	82.9	3	12	3
2000	100	10	1	2111	111	100	66.7	50	7	24	6	150	100	75	7	24	7
5000	100	10	1	5111	111	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
1000	100	10	2	1112	112	60	55	33.1	3	8	2	150	137.6	82.9	3	12	3
2000	100	10	2	2112	112	100	66.7	50	7	24	6	150	100	75	7	24	7
5000	100	10	2	5112	112	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
3000	200	10	1	3211	211	100	66.7	50	7	24	6	150	100	75	7	24	7
4000	200	10	1	4211	211	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
3000	200	10	2	3212	212	100	66.7	50	7	24	6	150	100	75	7	24	7
4000	200	10	2	4212	212	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
1000	100	20	1	1121	121	60	55	33.1	3	8	2	150	137.6	82.9	3	12	3
2000	100	20	1	2121	121	100	66.7	50	7	24	6	150	100	75	7	24	7
5000	100	20	1	5121	121	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
3000	200	20	1	3221	221	100	66.7	50	7	24	6	150	100	75	7	24	7
4000	200	20	1	4221	221	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
1000	100	20	2	1122	122	60	55	33.1	3	8	2	150	137.6	82.9	3	12	3
2000	100	20	2	2122	122	100	66.7	50	7	24	6	150	100	75	7	24	7
5000	100	20	2	5122	122	60	47.2	33.9	3	8	2	150	118.1	84.7	3	12	3
3000	200	20	2	3222	222	60	40	30	3	8	2	150	100	75	3	12	3
4000	200	20	2	4222	222	100	78.7	56.5	7	24	6	150	118.1	84.7	7	24	7

Table 14-5: Grade Interpolation Plan for Silver Outside North Silver Zone

First pass or within 50 x 25 m drill pattern											
First Pass						Second Pass					
Search Distance			No. of Composites			Search Distance			No. of Composites		
Major	Semi-Major	Minor	Min	Max	Max / hole	Major	Semi-Major	Minor	Min	Max	Max / hole
35	35	35	5	16	4	75	75	75	5	16	4
Second or third pass outside of 50 x 25 m drill pattern											
Second Pass						**Third Pass (50 x 25 m drill pattern constraint)**					
Search Distance			**No. of Composites**			**Search Distance**			**No. of Composites**		
Major	**Semi-Major**	**Minor**	**Min**	**Max**	**Max / hole**	**Major**	**Semi-Major**	**Minor**	**Min**	**Max**	**Max / hole**
75	75	75	5	16	4	150	150	150	4	16	4
75	75	75	5	16	4	150	150	150	4	16	4

Table 14-6: Grade Interpolation Plan for North Silver Zone

First Pass						Second Pass					
Search Distance			No. of Composites			Search Distance			No. of Composites		
Major	Semi-Major	Minor	Min	Max	Max / hole	Major	Semi-Major	Minor	Min	Max	Max / hole
35	35	35	5	16	4	75	75	75	5	16	4

14.11.1 AMEC Reviews

Checks on the estimated metal-at-risk in the final model indicated that 3.49% of the gold had been removed, which was in good agreement with the estimate of 3.46% expected in the capping analysis. The metal loss in individual domains closely matched that predicted by the sample statistics with the exception of the western structural domain outside of the grade shell where 17.7% of the metal was removed. This was due primarily to the influence of two extreme outliers from drill holes BW0437 (114 g/t Au) and GM12-01 (99 g/t Au) which were capped at 5 g/t for the final estimate.

AMEC conducted a change of support, model selectivity check on Measured and Indicated OK block gold grade estimates within the gold grade shell. The results show that the grade-tonnage curve of the OK model closely matches the Herco corrected grade-tonnage curve, assuming a 12 m x 12 m x 12 m SMU size. At cut-off grades between 0.2 g/t and 0.4 g/t Au, and assuming a 12 m x 12 m x 12 m SMU size, the OK model reports a difference of between +2.0% and -3.3% in the tonnes compared to the Herco-corrected model.

14.12 Classification of Mineral Resources

Blocks were assigned preliminary classifications based on drill hole spacing. Blocks falling within the 25 m x 25 m drill hole spacing pattern were assigned a tentative ‘Measured’ classification. Blocks not meeting these conditions were classified as ‘Indicated’ if they were within the area drilled with a 50 m x 50 m spacing. All other estimated blocks were assigned to the ‘Inferred’ category.

For Measured Mineral Resources, blocks falling outside of a 0.2 g/t gold equivalent grade shell were downgraded to Indicated as continuity of economic grades outside of the grade shell cannot be demonstrated.

Blocks falling within the high-grade silver grade shell were classified to the Indicated and Inferred categories using a nominal drill hole spacing of 50 m.

A smoothing algorithm was used to remove isolated Measured blocks within areas of Indicated blocks, isolated Indicated blocks within areas of Measured, Indicated blocks within areas of Inferred, and to remove isolated blocks of Inferred within areas of Indicated.

14.13 Reasonable Prospects of Economic Extraction

To assess reasonable prospects for eventual economic extraction a Lerchs–Grossmann optimized pit, prepared using the general economic and technical assumptions listed in Table 14-7, was used to constrain reporting of Mineral Resources. These economic assumptions are almost identical to the economic assumptions used for the Mineral Reserve pit optimization with the notable exception of metal prices which are higher (see Section 15).

14.14 Gold Equivalency Grades

Gold equivalent (AuEq) values were calculated using variable metallurgical recoveries depending on the mineralization type. Mineralization types were assigned to oxide, transition or sulphide depending on the oxidation state. The following calculations were used to calculate gold equivalent values:

- Oxide material type:
  - AuEq = Au + (Ag * (28.00 / 31.1035 * 0.64) / (1400 / 31.1035 * 0.88)
- Transition material type:
  - AuEq = Au + (Ag * (28.00 / 31.1035 * 0.58) / (1400 / 31.1035 * 0.85))
- Sulphide material type:
  - AuEq = Au + (Ag * (28.00 / 31.1035 * 0.44) / (1400 / 31.1035 * 0.85))

New Gold selected a 0.3 g/t AuEq cut-off grade to report the Mineral Resource estimate for stockpile material and a 0.4 g/t AuEq cut-off for estimating Mineral Resources considered amenable to direct processing. Using the gold equivalent values and costs as shown in Table 14-7, the marginal cut-off grade is likely to be slightly lower than 0.3 g/t AuEq for the transition and sulphide ore types.

Table 14-7: Pit Optimization Parameters

	Parameter
Pit Slope	Variable (23° to 43°)
Ore Mining Cost	C$1.64/tonne
Waste Mining Cost	C$1.94/tonne
Ore Processing Cost	C$6.85/tonne
Sustaining Capital Mill	C$0.18/tonne
G&A Cost	C$1.25/tonne
Tailings Facility	C$0.60/tonne
Royalties	1.5% (of Revenue)
Refining Costs	0.1% (of Revenue)
Gold Recovery	88.0% (Oxide), 85.0% (Transition and Sulphide). Stockpile material 0.3-0.4 g/t AuEq 79%
Silver Recovery	64.0% (Oxide), 58.0% (Transition), 44.0% (Sulphide). Stockpile material 0.3-0.4 g/t AuEq 37%
Gold Price	US$1,400/oz
Silver Price	US$28/oz
Exchange Rate USD:CAD	0.95

14.15 Mineral Resource Statement

The Qualified Person for the estimate is Mr Ronald G. Simpson of Geosim. Mineral Resources have an effective date of 31 March, 2013.

Mineral Resources are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves. Mineral Resources are reported inclusive of Mineral Reserves and do not include dilution. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.

The Mineral Resource estimate segregates the portion of mineralized material that should be processed directly versus that which the company would likely stockpile and process toward the end of the mine life of the Project.

In order to complete this segregation, the New Gold elected to utilize a dual cut-off strategy, whereby all material with an AuEq cut-off of greater than 0.4 g/t AuEq is considered for direct processing, while all material with an AuEq grade of between 0.3 and 0.4 g/t AuEq is planned to be stockpiled.

The Mineral Resource estimate, inclusive of the Mineral Reserves reported in Section 15, is presented in Table 14-8.

Table 14-8: Mineral Resource Tabulation (Effective date 31 March, 2013 Ronald G. Simpson P.Geo)

Resource Category	Tonnes & Grade: Tonnes (kt)	Au (g/t)	Ag (g/t)	AuEq (g/t)	Contained Metal: Gold (Moz)	Silver (Moz)
Measured & Indicated Resources						
Direct processing material						
Measured	116,955	1.04	5.6	1.10	3.90	21.06
Indicated	189,044	0.78	6.0	0.84	4.73	36.47
M&I (direct processing)	305,999	0.88	5.8	0.94	8.62	57.52
Stockpile material						
Measured	26,521	0.30	4.1	0.35	0.26	3.50
Indicated	64,382	0.30	4.4	0.35	0.62	9.11
M&I (stockpile)	90,904	0.30	4.3	0.35	0.87	12.60
Total M&I	**396,903**	**0.74**	**5.5**	**0.81**	**9.50**	**70.13**
Inferred Resources						
Inferred (direct processing)	13,815	0.76	4.1	0.80	0.34	1.82
Inferred (stockpile)	3,785	0.31	3.6	0.35	0.04	0.44
Total Inferred	**17,600**	**0.66**	**4.0**	**0.71**	**0.38**	**2.26**

Notes to accompany Mineral Resource Table

1. Mineral Resources are reported inclusive of Mineral Reserves
2. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
3. Mineral Resources are reported within a conceptual open pit shell based on metal prices of $1,400/oz gold, $28.00/oz silver, and average metallurgical recoveries of 88.0% gold and 64.0% silver for oxide mineralization, 85.0% gold and 58.0% silver for transitional oxide / sulphide mineralization, and 85.0% gold and 44.0% silver for sulphide mineralization. The pit shell also considers a mining cost of $1.64/t for mineralized material, waste mining cost of $1.94/t; ore processing cost of $6.85/t; sustaining capital for the mill of $0.18/t; G&A cost of $1.25/tonne; allocation for the tailings facility costs of $0.60/t; royalties at 1.5% of revenue; refining costs of 0.1% of Revenue, and pit slopes that range from 23 to 43º.
4. Total contained metal calculated on the basis of Tonnes * Grade / 31.10348 grams per troy ounce
5. Gold-equivalent grade estimate based on $1,400/oz gold, $28.00/oz silver, and differential metallurgical recoveries (refer to footnote 3)
6. Direct processing material is defined as mineralization above a 0.4 g/t AuEq cut-off that is likely to be mined and processed directly
7. Stockpile material defined as mineralization above a 0.3 g/t AuEq and below a 0.4 g/t AuEq cut-off that is suitable for stockpiling and future processing based on average metallurgical recoveries of 79.0% gold and 37.0% silver. The 0.3 g/t AuEq lower cut-off grade is considered adequate to cover mining, processing, and additional handling costs
8. Tonnages are rounded to the nearest 1,000 tonnes, grades and metal content are rounded to two decimal places;
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content;
10. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces.

14.16 Factors That May Affect the Mineral Resource Estimate

Factors which could affect the Mineral Resources include:

- Commodity price and exchange rate assumptions
- Pit slope angles and other geotechnical factors
- Assumptions used in generating the LG pit shell, including metal recoveries, and mining and process cost assumptions

There are no other known factors or issues that materially affect the estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing and political factors.

14.17 Comments on Section 14

The QP is of the opinion that the Mineral Resource estimate was appropriately undertaken performed using the 2003 CIM best practice guidelines and meets the 2010 CIM definition standards.

15.0 MINERAL RESERVE ESTIMATES

15.1 Introduction

A proposed mining production schedule was developed through the design of an ultimate open pit within the Mineral Resource model. Large-scale open pit mining will provide process plant feed at a nominal rate of 60,000 t/d, or 21.9 Mt/a. Annual mine production of ore and waste will peak at 91.3 Mt/a with a life-of-mine (LOM) stripping ratio of 2.00:1 including preproduction.

15.2 Pit Slopes

The Blackwater pit is designed for 12 m bench heights based on consideration of the loading equipment capabilities (mining height and reach), production drill configuration, and geo-mining conditions. This may be modified during future detailed planning and equipment selection. The pit slope design parameters were provided by Knight Piésold, and are shown in Figure 15-1.

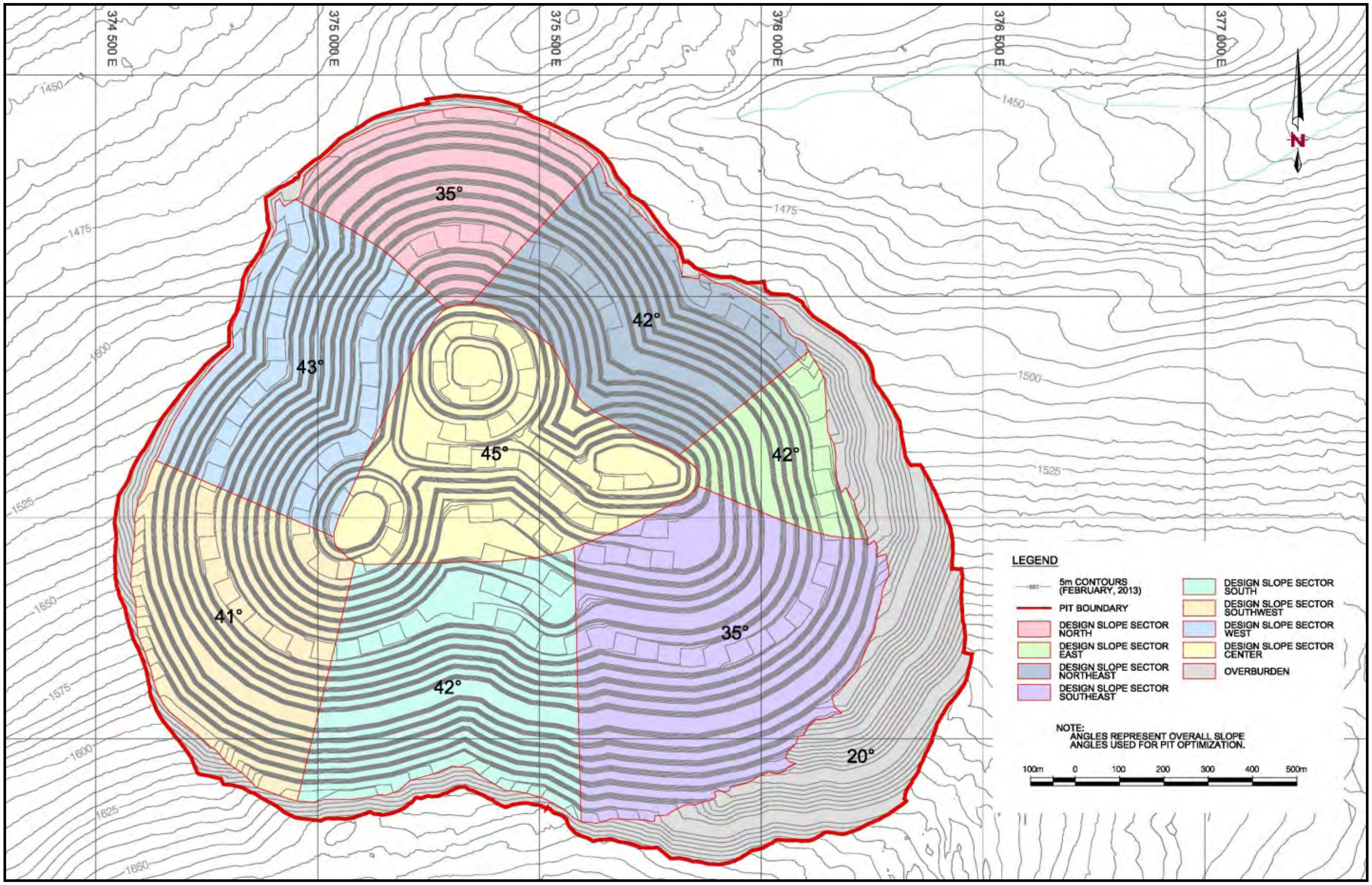
Additional information on the geotechnical setting for the open pit is included in Section 16.

15.3 Dilution and Mining Losses

Given the low-grade nature of the Blackwater deposit, the capability to mine ore and waste selectively to control dilution and ore loss will be a critical concern for the operation. The dilution factor selected for development of the pit designs and tonnage calculations was based on a review of the orebody type as well as practical considerations related to mine operations, including geological modelling, in-pit ore control, and mining equipment selectivity.

Dilution and ore loss will vary depending on the model block size. The feasibility model is based on 12 m x 12 m x 12 m blocks for delineation of ore and waste. The planned 5.2 m x 6 m blast hole spacing for 12 m benches in ore would allow partial blocks to be mined, whereas the geological model would group four of these smaller mining blocks together. This means that while an entire modelling block must be categorized as either ore or waste, better delineation of the two material types will be achievable during actual mining. Therefore, blocks in the block model inherently incorporate some dilution and ore loss. For example, the grade of a block made up of 50% ore and 50% waste will be determined by an average of the two and will be classed as either entirely ore, which would be equivalent to 100% dilution, or entirely waste, equivalent to 100% ore loss. Some additional dilution is expected at the ore / waste boundary to account for over-break / over-dig, and other operational factors.

Figure 15-1: Pit Wall Angles – Plan View



Note: Figure prepared by Knight Piésold, 2013.

All isolated ore blocks—ore blocks with waste on all four adjacent sides—will be mined as waste. Given the scale of the proposed mining equipment and the need to maintain productivity of the major equipment, it will likely prove to be impractical to separate these blocks. All isolated waste blocks—waste blocks with ore on all four adjacent sides—will be mined as ore. Similar to the isolated ore blocks, these blocks are unlikely to be separable at the proposed mining scale and rate.

There are two primary dilution and loss scenarios. The first scenario sees a surplus of ore being mined and being sent to both the mill and the low-grade stockpile. In the second scenario, all ore mined is sent to the mill with no surplus sent to the low-grade stockpile. Dilution and losses vary for these two scenarios due to the different cutoff grades used, resulting in different ore / waste contact block configurations. As such, the resulting average dilution for periods where both the mill and the stockpile are fed is 5% at a grade of 0.16 g/t Au and 3.19 g/t Ag. For periods where all ore is sent to the mill directly, dilution is 4% at a grade of 0.12 g/t Au and 2.90 g/t Ag.

Besides dilution, a misallocation factor is also applied when calculating the ore tonnes. This factor accounts for ore that is intended for the plant, based on grade, but is sent to the stockpile, or vice versa. A factor of 15% of the total material sent to the stockpile was applied to determine the misallocated quantities. The misallocated stockpile ore is made up with the average mill feed ore for the period. This misallocated material would average about 700 kt/a, or just under 4% of the total mill feed.

No additional ore loss factor is assumed because the gradual grade contact means that ore lost is replaced with dilution of nearly the same grade.

15.4 Conversion Factors from Mineral Resources to Mineral Reserves

15.4.1 Specific Gravity

In-situ dry density values for ore and waste rock (based on testwork carried out on samples of rock from site) were included in the geologic block model. The in-situ overburden density in the block model was not used because updated information was available.

The in-situ dry density averages 2.70 t/m³ for rock and 2.10 t/m³ for overburden. A range of swell factors were used depending on the destination or application.

In addition to separating ore and waste based on grade, the waste materials were classified according to their neutralizing potential ratio (NPR) and zinc content. PAG1 and PAG2 are classified as PAG (potentially acid generating) and require special handling, while NAG3 to 5 and overburden are classified as NAG (non-acid generating). NAG3 waste rock is defined as potentially metal leaching and also requires special handling.

15.4.2 Cut-off Grades

Two distinct cut-off grades (COGs) were considered for the Blackwater Project based on the destination of the ore, specifically whether the ore is being sent directly to the plant or to the ore stockpile. A higher COG was used for the stockpiled material to account for the cost of rehandling. The prices and costs for determining the cut-off grade are based on the 2012 PEA updated by New Gold and are summarized in Table 15-1.

An elevated COG strategy has been selected to minimize the Project payback period and maximize the NPV. During the first 10 years of the Project, where a surplus of ore will be mined, the highest-grade ore will be sent to the mill and the rest stockpiled for processing at the end of the mine life.

The cut-off grade calculation was reviewed upon the finalization of the 2013 Feasibility Study costs, gold recoveries and royalty payments. The net impact of these changes is to increase the cut-off grade from 0.26 g/t AuEq to 0.28 g/t AuEq for direct feed ore and 0.32 g/t AuEq to 0.33 g/t AuEq for stockpiled ore. This change is within the range of accuracy expected in a study of this level.

15.4.3 Re-handle Considerations

Two primary components of rehandle have been considered. The first is inherent rehandle as part of the mining process, which includes two components:

- Rehandle of road material around the pit rim, and within the pit limits, as the pit advances
- Rehandle of material internal to the pit, such as temporary ramps and short-term stockpiles.

This inherent rehandle has been assumed to average 3% over the life of the mine and this factor has been applied to all material mined. While loading and dozing this material is specifically accounted for, the additional trucking required to move it is very limited and is assumed to be included in the 5% unproductive time on the primary haul trucks (290 t) and in the utility fleet trucks (36 t).

Table 15-1: Cut-off Grade

Gold Price	U$/oz	Direct Feed	Stockpile
Gold Price	US$/g	41.79	41.79
Exchange Rate (US/CAD)	0.95	-	-
Gold Price – Canadian	CAD$/g	43.99	43.99
Royalty (NSR) %	1.5%	0.66	0.66
Refining	0.1%	0.04	0.04
Net Gold Price	CAD$/g	43.29	43.29
Opex			
Ore Mining	$/t	0.00	1.20
Incremental Haul	$/t	-0.35	0.00
Processing	$/t	6.85	6.85
G&A	$/t	1.25	1.25
Minimum Profit	$/t	1.00	1.00
Capex			
TSF Sustaining	$/t	0.61	0.61
Mill Sustaining	$/t	0.18	0.18
Total Costs (NSR Costs)	$/t	9.54	11.09
Costs Recovered Equivalent Grade (AuEq)	g/t	0.220	0.256
Recovery	%	85.0	79.0
Costs Cut-off Grade Recovery Based Grade (AuEq)	**g/t**	**0.26**	**0.32**

The second class of rehandle is rock hauled within the pit to be used for plating overburden materials to maintain trafficability. The need for plating will vary depending on the nature of the ground encountered and the weather conditions. The imported rock will be used for roads and shovel pocket foundations and will be mined out as part of the overburden mining process. Over the life of the Project, rock imports average 8% of the overburden mined.

15.4.4 Metal Prices

Mineral Reserves were estimated assuming US$1,300/oz gold and US$22/oz silver at an exchange rate of 0.95 US$/CAD$.

15.5 Mineral Reserves Statement

Mineral Reserves are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Mineral Reserves.

The Qualified Person for the estimate is Mr Jay Horton, P.Eng., of Norwest. Mineral Reserves have an effective date of 2 December, 2013.

Mineral Reserves are summarized in Table 15-2.

Table 15-2: Mineral Reserve Estimate (effective date 2 December 2013, Jay Horton, P.Eng.)

Reserve Category	Tonnes & Grade			Contained Metal	
	Tonnes (Mt)	Au g/t	Ag g/t	Gold (Moz)	Silver (Moz)
Direct processing material					
Proven	124.5	0.95	5.5	3.79	22.1
Probable	169.7	0.68	4.1	3.73	22.3
Sub-Total Direct Processing	294.3	0.79	4.7	7.51	44.4
Stockpiled material					
Proven	20.1	0.50	3.6	0.33	2.3
Probable	30.1	0.34	14.6	0.33	14.1
Sub-Total Stockpiled	50.2	0.40	10.2	0.65	16.4
Total Direct Processing + Stockpiled					
Proven	144.6	0.88	5.3	4.11	24.4
Probable	199.8	0.63	5.7	4.05	36.4
Total Reserves	**344.4**	**0.74**	**5.5**	**8.17**	**60.8**

Footnotes to Accompany the Mineral Reserves Table:

1. Mineral Reserves are reported within an open pit design based on metal prices of $1,300/oz gold, $22.00/oz silver, with variable recoveries by grade and ore type averaging 86.6% for gold and 49.1% for silver
2. Contained metal calculated on the basis of Tonnes * Grade / 31.10348 grams per troy ounce
3. Mineral Reserves that are classified as amenable to direct processing are defined as mineralization above a lower cut-off grade that varies by year between 0.26 g/t and 0.38 g/t AuEq and represents ore that is to be mined and processed directly
4. Mineral Reserves noted as stockpiled material consist of ore tonnage above a 0.32 g/t AuEq cut-off grade that is mined and stockpiled before being sent to the mill. This stockpiled tonnage includes ore mined before mill start-up, lower grade ore mined during preproduction and commercial production, and ore tonnage misclassified or misallocated during the mining process. No stockpiles currently exist at site
5. The gold-equivalent value used for cut-off grades only is based on $1,400/oz gold, $28.00/oz silver, and average metallurgical recoveries of 88.0% gold and 64.0% silver for oxide mineralization, 85.0% gold and 58.0% silver for transitional oxide / sulphide mineralization, and 85.0% gold and 44.0% silver for sulphide mineralization
6. Cut-off grade values are based on a gold price of $1,300/oz. The cut-off grade calculation includes the following costs: minimum profit; operating cost (ore mining, hauling cost, processing, G&A); sustaining capital cost for mining, tailings storage facility and the mill; royalty and refining cost; reduced metallurgical gold recovery for stockpiled ore (79%)
7. There are two primary dilution and loss scenarios. The first scenario sees a surplus of ore being mined and being sent to both the mill and the low-grade stockpile. In the second scenario, all ore mined is sent to the mill with no surplus sent to the low-grade stockpile. Average dilution for periods where both the mill and the stockpile are fed is 5% at a grade of 0.16 g/t Au and 3.19 g/t Ag. For periods where all ore is sent to the mill directly, dilution is 4% at a grade of 0.12 g/t Au and 2.90 g/t Ag
8. Tonnages grades and metal content are rounded
9. Rounding as required by reporting guidelines may result in apparent summation differences between tonnes, grade and contained metal content
10. Tonnage and grade measurements are in metric units. Contained gold and silver ounces are reported as troy ounces.

15.6 Factors That May Affect the Mineral Reserve Estimate

The following factors may affect the mineral reserve estimate

- Gold and silver prices
- US dollar exchange rates
- Geotechnical assumptions
- Ability of the mining operation to meet the annual production rate
- Mill recoveries
- Capital and operating cost estimates.

15.7 Comments on Section 15

The current Mineral Reserve estimates are based on the most current knowledge, permit status, and engineering constraints. The QP is of the opinion that the Mineral Reserves have been estimated using industry best practices and confirm to CIM (2010).

16.0 MINING METHODS

16.1 Geotechnical

Open pit geotechnical conditions are based on a review of geomechanical information, stability analyses results, the mining equipment to be utilized, and experience from similar open pit operations in the Project region.

The geotechnical model for the Blackwater pit has five separate components:

- Geology model – intercalated volcanic and volcaniclastic felsic to intermediate lapilli and ash tuff, volcanic breccia, and andesitic flows
- Alteration model – sericite in the centre of the deposit and potassic alteration surrounding the deposit with an oxidized weathered zone encountered below the bedrock surface
- Structure model – several inactive fault zones and smaller-scale rock mass fabric and structural domains in the main, northwest, and southeast structural domains
- Rock mass characteristics model – rock mass rating (RMR) and rock quality designation (RQD) logging parameters
- Hydrogeology model – groundwater levels near surface in the upper southern slopes and artesian flows on the lower northern slopes. The competent rock mass has low to moderate permeability and a moderately steep groundwater drawdown cone. The broken zone areas have the potential to be a large-scale, high-permeability zone.

Three geotechnical domains were defined for the purposes of the slope stability analyses:

- Surficial material – Glacial till deposits predominantly range from 5 m to 20 m thick across the open pit. Surficial material thickness increases up to 110 m along the eastern side of the deposit.
- Broken zone – This domain is recognized in the RQD block model as zones with RQD of less than 40% and was encountered in all deposit rock types.
- Competent rock – Defined as all zones with RQD greater than 40% and a rock mass quality of FAIR to GOOD.

The slope stability analyses for each pit design sector found few geotechnical controls for the pit slope design, other than rock mass failure through the broken zone and associated slope depressurization requirements. The achievable slope geometry is controlled by the location and extent of the broken zone for all phases of the proposed pit development. Site-specific geotechnical information has been used to characterize each geotechnical domain, to identify specific design sectors for both the proposed interim and final pits, and to develop the required slope design parameters.

The pit slope design parameters for all pit phases include:

- Surficial material – Bench face angles of 27º, height of 36 m with 20 m wide step-outs
- Broken zone – Bench face angles of 60° and inter-ramp slope angles of 39° should be achievable. Maximum inter-ramp height should be limited to 120 m. Slope depressurization will be required in these areas
- Competent rock – Bench face angles of 65° and inter-ramp slope angles of 45 to 46° should be achievable. Maximum inter-ramp heights should be limited to 200 m. The competent rock domain will be stable without groundwater depressurization.

Slope designs used to develop the pit are shown in Table 16-1.

Geotechnical investigations supporting the infrastructure design are discussed in Section 18.

16.2 Hydrogeology

Water inflows to the Blackwater open pit will include both groundwater and surface water runoff. The contributions from groundwater will progressively increase as the pit extends below the groundwater table. The contributions from surface water will be direct precipitation into the pit and runoff from the limited contributing catchments around the pit excavation. The inflows from direct precipitation will increase with increasing pit area in conjunction with groundwater inflows as the pit increases in depth.

The 1-in-100 year return period storm has been used to size the pit surface water dewatering system and was estimated to be approximately 142,000 m^3. The estimated runoff coefficient inside the open pit surface area was conservatively assumed to be 100%.

The pit dewatering design is based on lowering the groundwater table within the highly permeable zone to approximately 15 m below the pit base elevation, considering both removal of groundwater from storage and from recharge.

Table 16-1: 2013 Pit Design Parameters

Design Sector	Bench Height (m)	Bench Width (m)	Bench Face Angle (°)	Inter-ramp Angle (°)
North	24	16	60	39
Northeast	24	13	65	45
East	24	13	65	45
Southeast	24	16	60	39
South	24	12	65	46
Southwest	24	13	65	45
West	24	12	65	46
Center	24	12	65	46
Overburden	36	20	27	20

A combination of in-pit and perimeter pumping wells will be used for slope depressurization and pit dewatering. All pumping wells will be installed to a nominal depth of approximately 350 m below ground level.

The in-pit groundwater wells will remove water from storage in the highly permeable zone. This will also draw down water in the surrounding rock mass as the groundwater flows toward the highly permeable zone. Perimeter dewatering wells will be established along the south high wall to lower and extend the cone of depression beyond the pit walls where drainage to the permeable zone is not adequate for slope depressurization. There will be approximately 10 dewatering wells spaced at 200 m to 250 m intervals to achieve an adequate cone of depression to lower the groundwater level.

Hydrogeological investigations supporting the infrastructure design are discussed in Section 18.

16.3 Pit Design

A 3D block model provided by New Gold was used for development and evaluation of the pit shells for the Project.

Initial Pit Shell Optimization

A series of economic pit shells was created using the pit optimization tools in MineSight© by varying the base gold price of US$1,300/oz. For the base case, the selected pit shell was generated at 85% of the base gold price. Only Measured and Indicated Mineral Resources were used in the pit optimization. In addition, a sensitivity analysis was carried out by generating a second suite of pit shells based on varied cost inputs and pit wall angles.

The range of parameters used in pit optimization runs is summarized in Table 16-2. All costs are in Canadian dollars. The operating costs are based on those provided in the PEA, updated by New Gold to include sustaining capital for the mine, mill, and tailings storage facility (TSF). All pit optimization was carried out on a gold equivalent basis.

The parameters shown in Table 16-2 and Table 16-3 were used for pit optimization only. The final feasibility study (FS) costs and parameters are detailed later in this report. The net metal prices quoted in Table 16-2 (CAD$) include deductions for royalty and refining charges. A sample calculation of net gold price is shown in Table 16-4.

Pit optimization was carried out on a gold equivalent basis using the following formulae for the calculation of gold equivalent grade:

- Oxide ore type: AuEq = Au + (Ag * (28.00 / 31.1035 * 0.64) / (1,400 / 31.1035 * 0.88)
- Transition ore type: AuEq = Au + (Ag * (28.00 / 31.1035 * 0.58) / (1,400 / 31.1035 * 0.85))
- Sulphide ore type: AuEq = Au + (Ag * (28.00 / 31.1035 * 0.44) / (1,400 / 31.1035 * 0.85)).

16.3.1 Ultimate Pit Boundaries

Other than the extent of the model, no physical pit boundaries were used for the pit optimization,

16.3.2 Results of Optimization

Net revenue and contained ounces of gold for various pit shells, based on the optimization costs and a gold price of US$1,300/oz, were evaluated. In consultation with New Gold, the pit shell corresponding to 85% (approximately $1,105/oz gold price) of the $1,300/oz gold price was selected as the basis for the detailed pit design. This shell was selected for the following reasons:

- There is only a small increase in ore tonnage and gold ounces beyond this pit shell
- The net revenues for the larger shells show only marginal improvements
- The slightly smaller pit shell provides some margin for the incorporation of increased waste movement in detailed designs.

Table 16-2: Pit Optimization Variables

Parameter	Unit	Base Case	Low Range	Upper Range
Net Gold Price	$/g	36.80	10.82	49.79
Net Gold Price	US$/oz	1,087.37	319.71	1,471.21
Ore Mining	$/t	1.89	1.51	2.27
Processing Cost	$/t	8.90	7.12	10.68
Waste Cost	$/t	2.15	1.72	2.58
Pit Slope Angle	degrees	Variable, see Table 15-1	-2	+2
Recovery	%	Variable, see Table 15-2	N/A	N/A

Note: Exchange rate CAD$1 = US$0.95

Table 16-3: Gold Equivalent Recovery by Ore Type for Pit Optimization

Ore Type	Recovery (%)
Sulphide	85
Transition	85
Oxide	88

Table 16-4: Sample Calculation of Net Gold Price

Item	Unit	Value
Gold Price – US	US$/oz	1,300.00
Conversion oz to g	-	31.1048
Gold Price – US	US$/g	41.80
Exchange rate (US/CAD)	-	0.95
Gold Price – Canadian	$/g	44.00
Royalty	1.50%	0.66
Refining	0.10%	0.04
Net Gold Price	$/g	43.30

Sensitivity analysis was performed to assess the impact of varying operating costs and pit wall angles. Based on the 85% pit shell, the processing cost and mining cost were both varied by ±20%, in 10% increments. Similarly, the pit wall angles were varied by ±2° from the base case. As expected, the pit shells show the greatest sensitivity to the assumed metal prices, with lesser sensitivity to mining cost.

16.3.3 Roads

Mine service roads will be pioneered by dozers as single-lane, 15 m wide, balanced cut/fill accesses with a collector ditch and soil salvage windrowed up-slope locally.

The existing exploration road network can be incorporated into the access development for the pit. The access roads will be expanded into mine haul roads and subsequently widened as required by means of end-dumping with suitable NAG4 or NAG5 waste rock from the pit.

The haul road widths are based on a 290 tonne (930E-4SE) size haul truck with a width of 8.7 m. Road widths are 37 m to 40 m depending on ditching and berm requirements. Road grades are limited to 10% in-pit and 8% ex-pit. British Columbia mine safety regulations require runaway lanes spaced as required for all roads over 5% grade.

Haul cycles were calculated for each design period—initially by month and later by year—over the life of the mine. Haul simulations were carried out based on the designated Komatsu 930E-4SE 290 tonne haul trucks. For trucks hauling overburden, the capacity was reduced by 5% to allow for carryback of cohesive soil materials. Rolling resistance was estimated to be 6% within 50 m of the loading face and 50 m of the dump, and 4% on all established roads. A maximum truck speed limit of 40 km/h was set for flat or inclined roads, reducing to 25 km/h near shovel and dump points and 15 km/h around switchback corners. On the downhill segments, speeds were limited by braking data provided by Komatsu.

16.4 Consideration of Marginal Cut-Off Grades

Two distinct cut-off grades were considered for the Blackwater Project based on the destination of the ore, specifically whether the ore is being sent directly to the plant or to the ore stockpile. A higher cut-off grade was used for the stockpiled material to account for the cost of rehandling. For the purposes of cut-off grade calculation only, a lower gold recovery of 79% was assumed for stockpiled material. These cut-off grades are used to determine the destination of blasted material and therefore do not include mining costs except for ore rehandled from the stockpile.

Cut-off grade values are based on a gold price of $1,300/oz. The plant direct feed cutoff grade is AuEq 0.26 g/t, and the ore stockpile cutoff grade is AuEq 0.32 g/t. The cut-off grade calculation includes the following costs:

- Minimum profit
- Operating cost (ore mining, hauling cost, processing, G&A)
- Sustaining capital cost for mining, TSF, and the mill
- Royalty and refining cost
- Reduced recovery for stockpiled ore (79%).

The costs for determining cut-off grade were provided in Table 15-1. An elevated cut-off grade strategy has been selected to minimize the Project payback period and maximize the net present value (NPV). During the first 10 years of the Project, where a surplus of ore will be mined, the highest-grade ore will be sent to the mill and the rest stockpiled for processing at the end of the mine life.

The cut-off grade calculation was reviewed upon the finalization of the feasibility study costs, gold recoveries and royalty payments. The net impact of these changes is to increase the cut-off grade from 0.26 g/t to 0.28 g/t AuEq for direct feed ore and 0.32 g/t to 0.33 g/t AuEq for stockpiled ore. Due to the elevated cut-off grade strategy employed in the first 10 years of the Project, the impact of these increased cut-off grades would not be felt until late in the mine life and therefore is expected to be minimal in terms of overall Project economics.

16.5 Mine Schedule

The mine schedule is discussed in detail in Section 16.5.3 (Tables 16-5 and 16-6). The mine operations fleet is scheduled to start in Q4 Year -2 and end in Year 14. The stockpile is reclaimed from Year 15 to Year 17. Multiple constraints such as ramp-up considerations, TSF dam construction requirements, and metal production were considered in developing the production schedule. These considerations are discussed in the following sub-sections.

16.5.1 Pit Phases

A total of four phases were developed and incorporated into the mine production schedule. Some of these phases exist only for a short time, usually to increase ore grade or to produce material suitable for the TSF dam construction.

16.5.2 Pit Development

The development and phasing of the Blackwater deposit are described below (note: Q = quarter, M = month). Discussion of the waste types in the mine plan is included in Section 20; information on the tailings storage and waste rock facilities is included in Section 18.

Year -2 Q4 (Pre-production)

Initial pit development starts at the southern extent of the pit and is mined to the final pit wall. The pit is mined in a stepped fashion from west to east with four 12 m benches from elevation 1,620 m to 1,656 m. The west side of the pit is mined ahead of the east side to balance the rock versus overburden release for the TSF construction. Pit and waste rock storage facility areas are cleared. NAG5 and overburden waste is placed in both the West and East waste rock storage facilities, NAG4 waste is placed only in the West facility, and NAG3 and PAG waste is placed in the Site C TSF. The PAG waste is placed such that it will be flooded in the year following placement. Approximately 500,000 t of NAG5 material is used to construct initial roads.

Year -1 Q1-Q2 (Pre-production)

Mining continues to the southern extent of the final pit wall from the 1,608 m bench to the 1,632 m bench. PAG waste continues to be placed in the Site C TSF. Two separate ore stockpiles are established immediately west of the plant site for both high- and low grade ore. Approximately two-thirds of the final ore stockpile footprint will be cleared and prepared for future stockpile placement. NAG material is hauled down to the Site C main dam for construction as required, which the mine fleet starts placing in Q1 Year -1. Of the remaining overburden and NAG material, most is placed in the West facility and a smaller percentage in the East facility to manage haul times.

Year -1 Q3 (Pre-production)

Overburden mining is advanced at the eastern extent of the pit to the 1,620 m bench while the southwestern half of the pit is accelerated along the final pit wall to increase NAG5 release. PAG waste is placed in the Site C TSF. The access road to the bottom of the Site D dam is established in preparation for construction of the dam starting in Q4 Year -1.

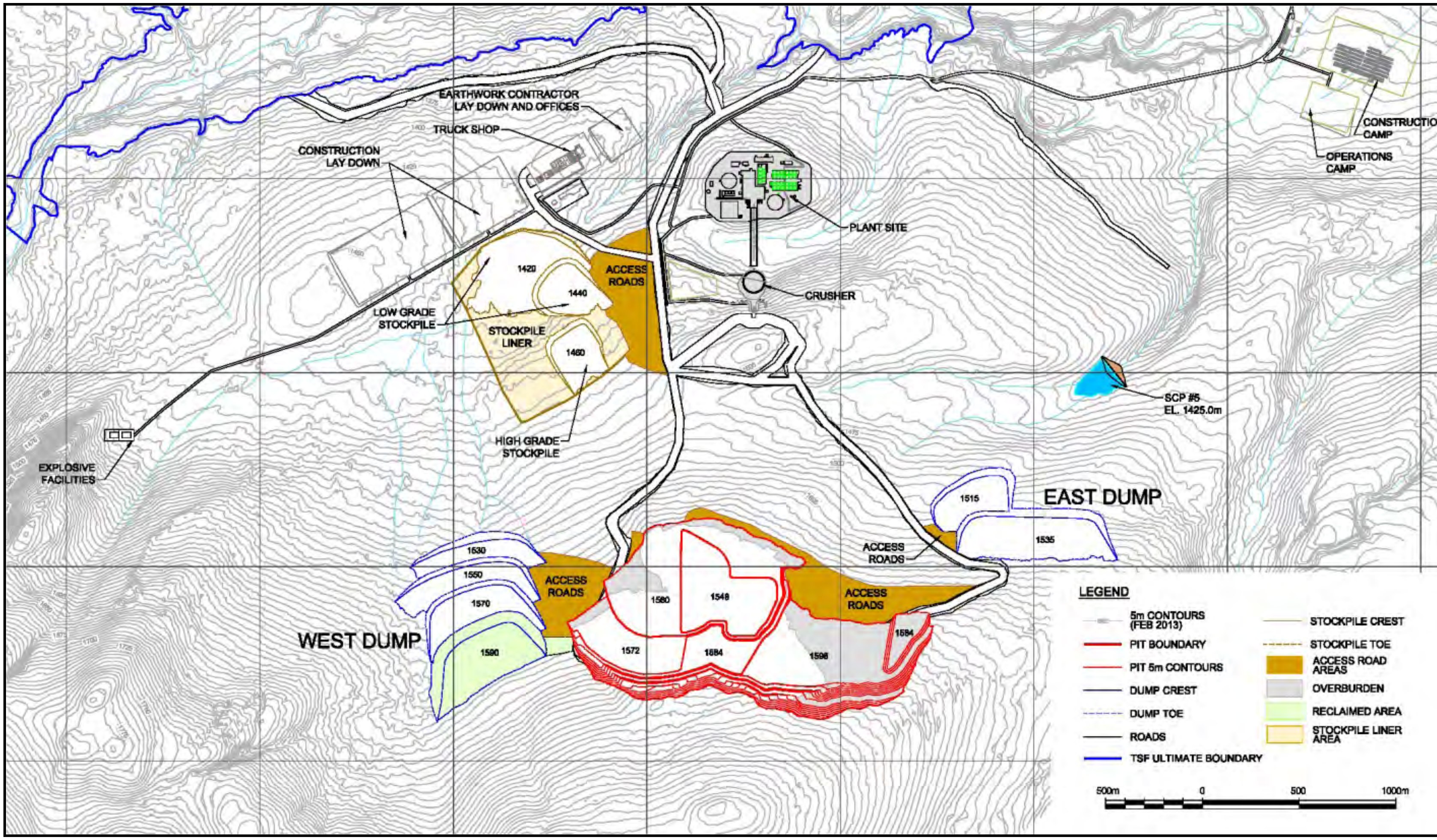
Year -1 Q1 – Year 1 M03 (Pre-production)

Mining continues in a similar pattern, with the west half of the pit being advanced one to two benches ahead of the east half while targeting the NAG5 zone along the southwestern pit wall. External facility waste continues to be split between the East and West facilities. The Site C saddle dam access road is upgraded to approximately the midway point to accommodate PAG waste placement in the upper part of the Site C TSF. NAG material is hauled down to the Site D TSF dam as required, which starts construction in Q4 Year -1. PAG waste is placed in the Site C TSF. All ore continues to be stockpiled, with high-grade ore stockpiling ending in March Year 1 as the mill starts up in April.

Year 1

The pit layout at the end of Year 1 is provided as Figure 16-1.

Figure 16-1: Phased Mine Plan – End of Year 1



Note: Figure prepared by Norwest, 2013

A higher-grade zone in the centre of the pit is mined down to the 1,548 m bench to maximize grades. The south extent of the pit continues to be mined in a staggered pattern from west to east to meet the various material release requirements and maintain the minimum rock/overburden split. External waste is predominantly placed in the West facility and a smaller amount in the East facility. Reclamation of the top lift (1,590 m) of the West facility starts in April Year 1. PAG waste continues to be placed in the Site C TSF. The access road for PAG placement in the Site D TSF is established in Q3 in preparation for PAG placement at the start of Year 2. The mill starts production in April Year 1. Low-grade ore continues to be stockpiled while the high grade stockpile starts to be reclaimed to maintain plant feed.

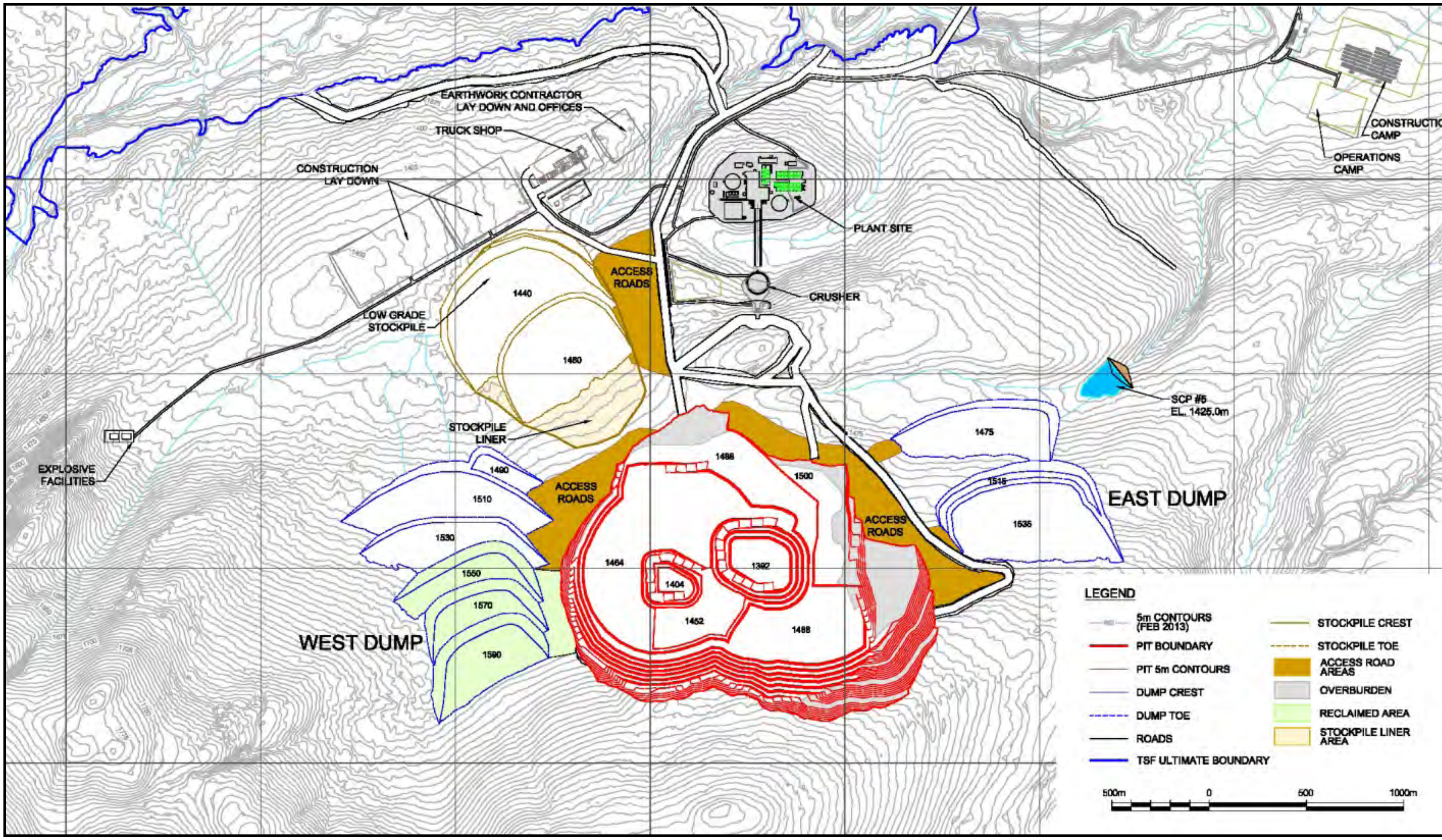
Year 2 to Year 3

The pit is mined down to the 1,512 m bench elevation in the main part of the pit, with the higher-grade core zone mined to 1,404 m elevation. The southeast side of the pit, which consists mostly of overburden, is mined down to an elevation of 1,536 m to 1,548 m. All PAG waste is now placed in the Site D TSF. Material from the high-grade ore stockpile is reclaimed during the first four months of Year 2, after which plant feed is maintained by direct ore release from the pit. Low grade ore continues to be stockpiled. External dump waste is split between the West and East facilities. The 1,570 m and 1,550 m lifts on the West facility are reclaimed in Q1 of Year 2 and Year 3, respectively. All PAG material is now placed in the Site D TSF.

Year 4 to Year 5

The pit layout at the end of Year 5 is included as Figure 16-2. The pit is mined down to the 1,452 m to 1,464 m bench and the higher-grade ore zone down to 1,392 m elevation. The southern part of the pit is mined down to the 1,488 m bench, leaving a final major package of overburden along the east wall, which is mined down to the 1,524 m bench. External waste is split between the West and East facilities. A lower lift is placed on the East facility ahead of the continuation of the upper lifts to manage the maximum lift height. An overburden stockpile is established in Q4 Year 4 at the south end of the Site D dam to provide an overburden supply for the final years of dam construction. Overburden is also sent to the Site C TSF to complete reclamation.

Figure 16-2: Phased Mine Plan – End of Year 5



Note: Figure prepared by Norwest, 2013

Year 6 to Year 7

The southwest part of the pit is mined down to the 1,344 m bench and the southern part to the 1,428 m bench. The overburden package on the east side of the pit is mined down to the 1,464 m bench and is nearly mined out. The excess overburden is placed in the overburden stockpile near the TSF to meet the remaining overburden requirements for the final years of dam construction. More than 50% of the waste rock released during this period is classified as PAG and as such is placed in the Site D TSF. Waste to the East facility is placed in the valley directly to the northeast of the final pit ramp entrance and ties into the existing bottom lift of the East facility. The West facility will have been advanced to the bottom lift to accommodate further waste placement on the two lifts above it while maintaining the maximum lift height of 40 m. Progressive reclamation of the West facility 1,530 m lift is completed in Year 6. Stockpiling of low grade ore continues and reaches the final elevation of 1,480 m.

Year 8 to Year 9

The pit is mined to the final wall along the entire perimeter, and all overburden is now mined out. The southwest part of the pit is mined down to the 1,296 m bench and the rest of the pit to the 1,380 m to 1,392 m bench. The final ramp on the west side of the pit, as well as access to the top lifts of the Site D dam, is also established.

Year 11

Mining in the southwest part of the pit is postponed while the rest of the pit is mined down to the 1,332 m bench. Once stockpiling of low-grade ore is completed, all low-grade ore will be direct fed to the plant in subsequent years. External waste is split between the East and West facilities, and waste going to the East facility is placed directly off the pit ramp entrance, tying-in to the crusher access.

Year 11 to Year 13

The southwest part of the pit is completed as the rest of the pit is mined down to the 1,224 m to 1,236 m bench. Waste release becomes minimal by the end of Year 13 as most material mined is now ore and mine production starts to ramp down.

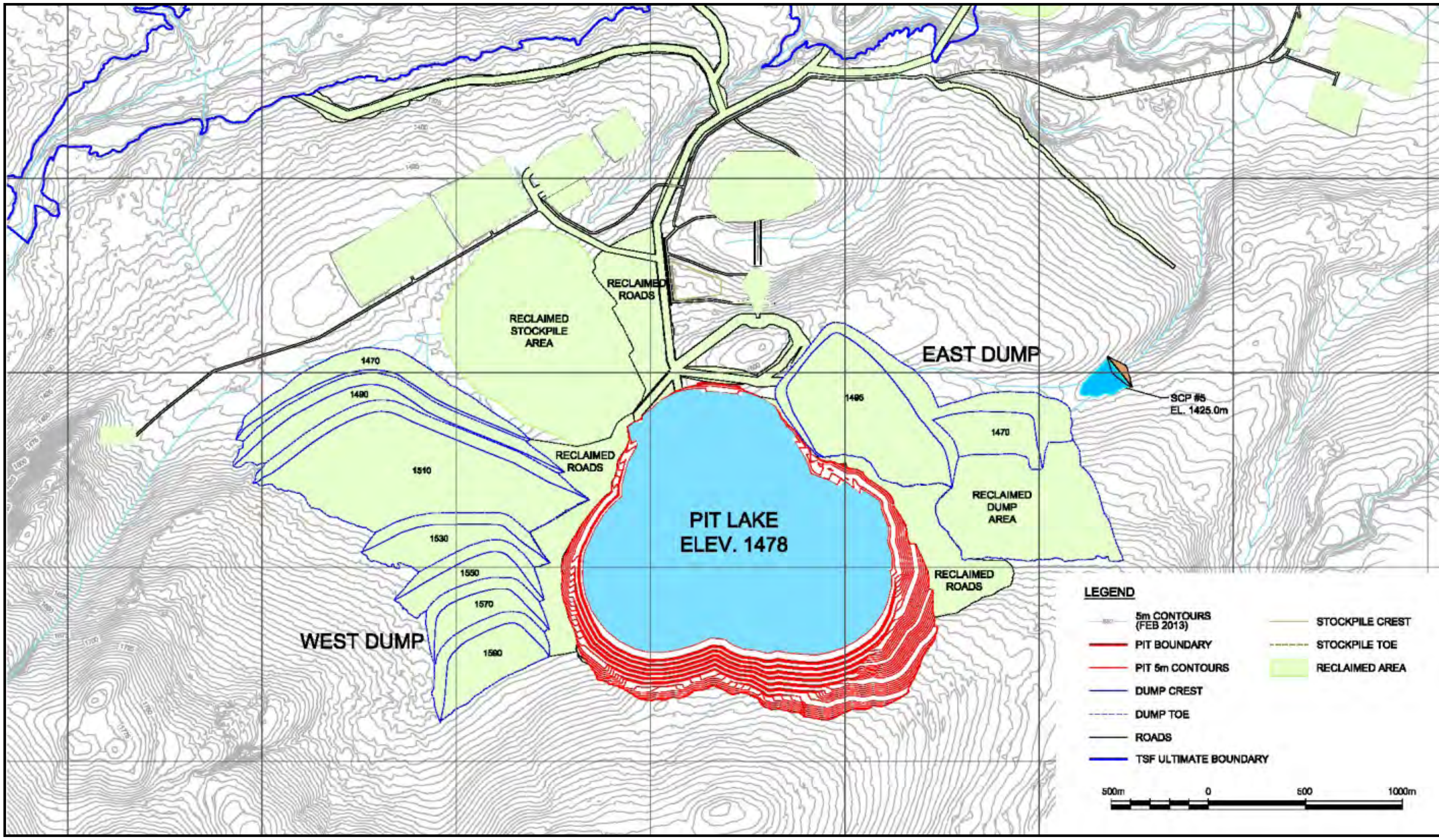
Year 14

All mining is completed this year. PAG waste is backfilled into the Phase 2 west pit.

Post-Year 14 Activities

Figure 16-3 shows the layout post the completion of open pit mining activities.

Figure 16-3: Phased Mine Plan – End of Year 17



Note: Figure prepared by Norwest, 2013

The low-grade stockpile is mined out and processed through the end of Year 17. During this period the bottom three lifts of the West dump are reclaimed. Once all ore is processed, the east half of the East facility is mined out to provide overburden for reclamation of the Site D TSF and the waste rock facilities. Reclamation is also carried out in all other previously unreclaimed areas.

16.5.3 Production/Throughput Rates

The design plant feed rate is 21.9 million tonnes per year (Mt/a) with an overall mining capacity of approximately 90 Mt/a. Several analyses were conducted to determine the optimum mine production level and ramp-up schedule. The main drivers for the mine schedule are:

- Overall Project timing – The initial mining fleet is assembled and operational by the start of Q4 Year -2. The mill is commissioned in April Year 1.
- Tailings storage facility construction – Construction of the TSF requires specific material types to be produced from the mine on a defined schedule. Much of the final schedule is driven by the need to release the better-quality (NAG5) waste rock.
- Elevated grade/stockpile – Ore grade is maximized in the early years of the Project to reduce the payback period. Lower-grade ore is stockpiled and processed in later years.
- Equipment erection and ramp-up – The rate at which equipment can be commissioned and ramped up to full productivity is a key input.

Plant feed increases from approximately 10.8 Mt during Year 1 to approximately 21.9 Mt once full production is reached. Excess ore above this level, ranging from 2 Mt to 7 Mt annually, will be stockpiled until Year 11 and fed to the plant from Year 15 to Year 17.

Gold production peaks in Year 3, at approximately 618,000 oz of gold, and again in Year 8, at approximately 581,000 oz of gold. These peaks illustrate the effect of the heterogeneous nature of the ore deposit geology on mine production, where large tonnages of ore are encountered in certain specific zones of the pit rather than being dispersed evenly throughout. This effect is also increased by the requirement to provide certain waste types to the TSF.

Table 16-5 summarizes annual plant feed schedule, including the tonnages sent directly to the mill, ore reclaimed from the stockpile, and the ore types, and also shows the annual cutoff grade for ore and for plant feed. The schedule is based on Measured and Indicated Mineral Resources; Inferred Mineral Resources have been set to waste. Table 16-6 provides the waste schedule.

The production schedule is based on 24-hour, year-round mining operations with limited weather delays and shutdowns. The total annual schedule is 355 operating days, or 8,520 hours. The mine will operate on two 12-hour shifts per day.

16.6 Mining Equipment

Loading will be carried out by a combination of hydraulic and cable shovels backed up by a large front-end loader. Equipment is sized as follows:

- 56 m^3 Cable Shovel (P&H 4100XPC class)
- 40 m^3 Hydraulic Shovel (PC8000 class)
- 28 m^3 Loader (L-1850 class).

To meet the targeted production levels, the mine plan relies on sufficient truck capacity to maintain the productivity of the loading units. The changing configuration of the mine over the Project life means that haul distances for both ore and waste will vary significantly, and consequently the truck fleet requirements will also vary. Trucks of 290 tonne capacity have been selected as a good match for the range of loading equipment, permitting three- to five-pass loading for the shovels.

Drill operations will be carried out continuously as part of the normal mining operation. Once full mine production is reached, drilling and blasting of approximately 7 Mt (dry) per month will be required to maintain production levels. The selected drills are sized for 200 mm and 250 mm drill holes and are capable of both rotary and hammer drilling. The drills will be required to drill to a depth of approximately 13.4 m in a single pass to allow for blasting benches up to 12 m high. Both diesel and electric blast hole drills are planned. Two electric drills are paired with the large electric cable shovel, primarily for use in waste mining. The remaining drills—five at peak production—are diesel-powered and will support both ore and waste mining.

Table 16-5: Annual Plant Feed and Ore Tonnages

	Year -2	Year -1	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
Plant Feed (t)	-	-	10,804,000	21,899,000	21,899,000	21,899,000	21,899,000	21,903,000	21,903,000	21,903,000
Au (g/t)	-	-	0.900	0.906	1.000	0.710	0.750	0.827	0.803	0.936
Ag (g/t)	-	-	5.546	5.384	5.842	5.416	4.958	6.155	4.255	6.706
Ore to Stockpile (t)	-	1,611,000	6,249,000	4,386,000	3,633,000	3,144,000	5,677,000	2,987,000	7,384,000	6,801,000
Ore to Mill (t)	-	-	10,284,000	21,159,000	21,899,000	21,899,000	21,899,000	21,903,000	21,903,000	21,903,000
Stockpile to Plant (t)	-	-	520,000	740,000	-	-	-	-	-	-
Cutoff grade for ore (AuEq g/t)	-	0.320	0.320	0.320	0.320	0.320	0.320	0.320	0.320	0.320
Approximate Annual Cutoff grade for plant feed (AuEq g/t)	N/A	N/A	0.37	0.35	0.35	0.37	0.38	0.34	0.38	0.38
Ore Oxide (%)	-	17%	11	7	11	8	5	6	2	-
Ore Transition (%)	-	83%	65	37	19	18	11	13	2	3
Ore Sulphide (%)	-	-	24	57	69	75	84	81	96	97

	Year 9	Year 10	Year 11	Year 12	Year 13	Year 14	Year 15	Year 16	Year 17	Total
Plant Feed (t)	21,903,000	21,903,000	21,902,000	21,902,000	21,902,000	21,902,000	21,900,000	21,900,000	5,121,000	344,445,000
Au (g/t)	0.839	0.688	0.609	0.634	0.687	0.879	0.399	0.399	0.399	0.738
Ag (g/t)	6.123	3.835	3.400	2.982	3.272	2.351	10.282	10.282	10.282	5.493
Ore to Stockpile (t)	3,126,000	5,183,000	-	-	-	-	-	-	-	50,181,000
Ore to Mill (t)	21,903,000	21,903,000	21,902,000	21,902,000	21,902,000	21,902,000	-	-	-	294,264,000
Stockpile to Plant (t)	-	-	-	-	-	-	21,900,000	21,900,000	5,121,000	50,181,000
Cutoff grade for ore (AuEq g/t)	0.320	0.320	0.260	0.260	0.260	0.260	-	-	-	-
Approximate Annual Cutoff grade for plant feed (AuEq g/t)	0.34	0.38	0.27	0.27	0.27	0.27	-	-	-	-
Ore Oxide (%)	-	-	-	-	-	-	7	7	7	4
Ore Transition (%)	2	-	-	-	-	-	22	22	22	11
Ore Sulphide (%)	98	100	100	100	100	100	71	71	71	85

Table 16-6: Waste Production by Material Type (tonnes)

Year	PAG1 and PAG2	NAG3	NAG4	NAG5	OVB	Total
Year -2	144,000	-	45,000	1,534,000	2,267,000	3,990,000
Year -1	5,488,000	1,787,000	2,015,000	8,985,000	12,373,000	30,648,000
Year 1	13,813,000	4,590,000	2,646,000	6,274,000	13,749,000	41,072,000
Year 2	26,936,000	3,548,000	2,583,000	8,488,000	13,697,000	55,252,000
Year 3	27,497,000	3,255,000	3,682,000	10,393,000	15,443,000	60,270,000
Year 4	37,004,000	5,400,000	3,664,000	11,425,000	5,896,000	63,389,000
Year 5	31,017,000	7,600,000	3,578,000	6,647,000	11,891,000	60,733,000
Year 6	39,175,000	7,875,000	3,804,000	8,031,000	7,882,000	66,767,000
Year 7	44,082,000	3,157,000	3,108,000	5,340,000	5,836,000	61,523,000
Year 8	37,925,000	10,078,000	3,789,000	7,440,000	2,558,000	61,790,000
Year 9	35,793,000	14,844,000	6,036,000	8,208,000	-	64,881,000
Year 10	26,581,000	9,895,000	5,865,000	14,556,000	-	56,897,000
Year 11	14,515,000	3,380,000	1,224,000	8,332,000	-	27,451,000
Year 12	10,741,000	1,400,000	1,205,000	1,550,000	-	14,896,000
Year 13	9,027,000	1,223,000	651,000	1,029,000	-	11,930,000
Year 14	6,961,000	1,303,000	278,000	167,000	-	8,709,000
Total	**366,699,000**	**79,335,000**	**44,173,000**	**108,399,000**	**91,592,000**	**690,198,000**

The exact size and configuration of the support equipment fleet will be finalized at the detailed engineering stage, but is expected to include units of the following type:

- Track-type dozers (650 kW, 450 kW, 153 kW)
- Wheel-type dozers (370 kW)
- Motor grader (7 m blade, 5 m blade)
- Hydraulic backhoes (300 kW, 150 kW)
- Articulated haul truck (36 tonnes)
- Water trucks for road dust control
- Various maintenance vehicles: fuel/lube trucks, heavy forklifts, tow truck + flat deck trailer, mobile cranes, tire handler.

Table 16-7 summarizes the primary and support equipment requirements over the LOM (including pre-production).

16.7 Blasting and Explosives

Blast patterns and loading parameters are based on 12 m bench heights. The selected blast hole diameters are 200 mm in ore and strong rock, and 250 mm in weak to medium strength rock.

Table 16-7: Primary and Support Mining Equipment

Equipment Number - Annual	-2	-1	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17
Loading																			
$56m^3$ Cable Shovel			1	1	1	1	1	1	1	1	1	1	1						
$40m^3$ Hydraulic Shovel (Diesel+Electric)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
$28m^3$ Front End Loader	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Drills																			
250mm Drill (Diesel)	1	2	4	4	4	4	4	4	4	4	4	4	3	3	3	3	0	0	0
250 mm Drill (Electric)			1	2	2	2	2	2	2	2	2	2	1						
120 mm Drill	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1			
Trucks																			
290t Haul Truck	8	14	19	22	23	23	23	27	27	27	27	27	27	24	24	24	8	8	8
36t Haul Truck	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2
Excavator																			
300kW - Utility	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
150kW - Utility	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Dozers																			
650kW			2	2	2	2	2	2	2	2	2	2	2	1	1	1			
450kW	3	5	5	5	5	6	6	6	6	6	7	5	5	3	3	3	2	1	1
153kW, 21t - Utility	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
370kW Wheeled	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Grader																			
397kW, 66t, 7.3m blade	1	2	3	3	3	3	3	4	4	4	4	4	3	2	2	2	1	1	1
200kW, 4.9m blade	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Other																			
$7m^3$ buck FEL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Water Truck (150t truck)	1	2	2	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1
Fuel/Lube Truck	1	2	2	2	2	2	2	2	2	2	2	2	2	2	1	1	1	1	1
Lowboy	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Light Plant	6	7	9	9	10	10	9	9	9	9	9	8	6	5	5	5	4	4	2
crew bus	1	2	3	4	4	4	4	4	4	4	4	4	3	2	2	2	1	1	1
Pickup trucks	20	20	21	20	20	20	20	20	20	20	20	20	20	20	20	20	10	10	3
Maintenance																			
Mechanics truck	2	3	4	4	4	4	4	4	4	4	4	4	3	2	2	2	1	1	1
Welding truck	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Hiab (9 tonnes)	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Mobile Crane - 55t	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Integrated tool handler	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1
Tire handler FEL	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1	1

Where blast holes are wet, a mix of 70% emulsion / 30% ANFO will be used for blasting. Dry blast holes in weak to medium strength rock will use 100% ANFO, whereas dry blast holes in ore and strong rock will use a 25% emulsion/ 75% ANFO blend for improved fragmentation. Where necessary, dewatering and hole liners will be employed to prevent the loss of explosives and to improve blasting performance. On average an estimated 25% of blast holes are expected to be wet.

Cushion blasting will be used for any blast patterns adjacent to an interim or final pit wall to prevent overbreakage of the wall and to maintain its overall stability and integrity. This will also reduce the surface area of the ultimate walls and limit acid production and metal leaching. All cushion blasts will use a 200 mm hole diameter. It is estimated that wall control blast patterns will be used for 14% of the total tonnage blasted. This percentage was derived from the total area of interim pit walls. Revised geotechnical requirements developed during detailed design and operations for some shorter, interim walls should allow this proportion to be reduced, leading to lower drill-and-blast requirements and therefore reduced costs.

Frost blasts may be required to break up the overburden sufficiently when daytime high temperatures do not exceed freezing. Typical frost blasts would be designed to a powder factor of 0.16 kg/t to break the frozen overburden layer sufficiently to limit the size of the frozen pieces. This will allow the excavator to load the frozen overburden efficiently and thus increase its productivity when digging this type of material.

The full-service vendor will provide the main explosives plant and storage, mobile manufacturing unit (MMU), loading personnel, and support vehicles. The explosives plant and storage facility will be located to the northwest of the pit.

17.0 RECOVERY METHODS

17.1 Process Flow Sheet

The overall process flow sheet is included as Figure 17-1.

17.2 Plant Design

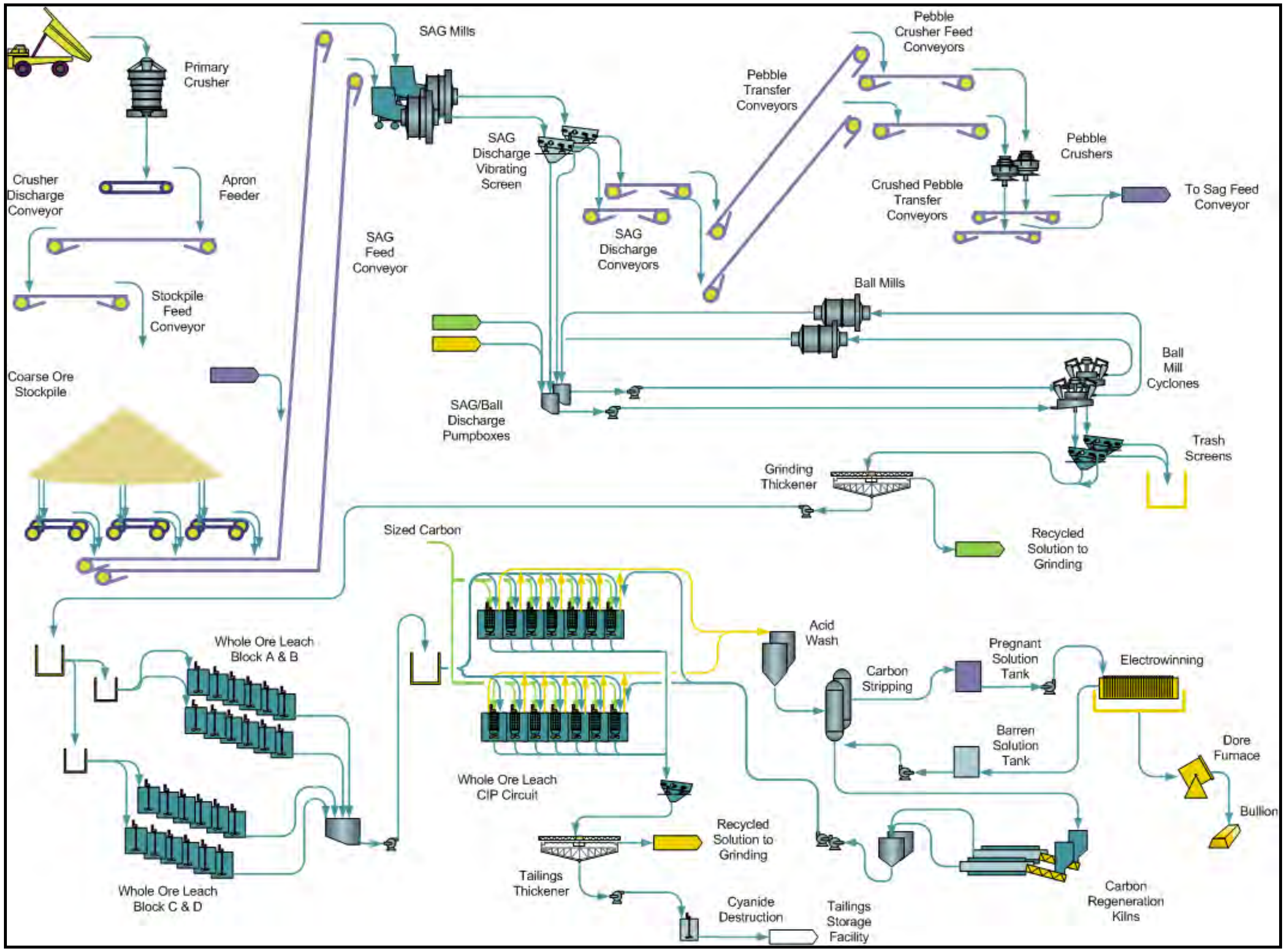
The process plant facility will consist of a primary crushing plant, a coarse ore stockpile (COS), a SAG/ball mill/crusher (SABC) grinding circuit, pre-leach thickening, whole ore cyanide leaching , carbon-in-pulp (CIP) recovery of precious metals from solution, elution of precious metals from carbon, and recovery of precious metals by electrowinning followed by smelting to doré. The plant will also have facilities for carbon regeneration, tailings thickening, and cyanide destruction.

Run-of-mine ore will be transported in 290 tonne mine haul trucks to a gyratory crusher, where each truck will dump from one of two dumping positions into the crusher feed pocket. The crushed ore will be transferred to an apron feeder and fed to a conveyor for delivery to a coarse ore stockpile (COS).

Reclaimed ore from the COS will be conveyed to feed two primary grinding SAG mills, each with its own separate reclaim and lime addition systems. Each SAG mill will be in closed circuit with a vibrating screen and a pebble crusher and will feed an associated ball mill. Each ball mill will operate in closed circuit with a dedicated cyclone cluster to produce a product size of 80% passing 150 µm.

The ore will be ground with cyanide solution recycled from the pre-leach thickener to the cyclone feed pumpbox; this will initiate the gold leaching process. After classification by the cyclone system, the thickened cyclone underflow at 50% solids, will report to the leach circuit, where lime will be added to maintain the pH to the range of 10.5 to 11 to prepare for cyanide leaching. The slurry will be split between four trains of leach tanks to achieve a leach residence time of 30 hours. Discharge from the leach trains will be recombined and sent to two lines of 400 m^3 CIP tanks for gold and silver adsorption.

Figure 17-1: Process Flowsheet



Note: Figure prepared by AMEC, 2013

The loaded carbon generated from the CIP tanks will be pumped to a pressure elution circuit with two 20 tonne capacity stripping vessels. Once gold has been desorbed, the carbon will be sent for regeneration in two electric kilns, each operating at 1.25 t/h. Quenching and screening will prepare the reactivated carbon for reintroduction into the CIP circuit.

Gold eluate, meanwhile, will be sent to electrowinning cells to produce a gold–silver precipitate sludge. Loaded cathodes will be pressure-washed in place to produce a sludge containing the precious metals. The sludge will be filtered and dried, then mixed with fluxes and smelted on site to produce gold–silver bars.

Slurry discharged from the CIP tanks will report to a CIP tails thickener before being sent to cyanide destruction. Cyanide-bearing liquid overflow recovered from the thickener will be recycled back to the process to lower the cyanide consumption as well as cyanide destruction costs. Slurry will be thickened to 60% solids in a tailings thickener before being introduced into the cyanide destruction circuit.

Reclaimed water will be used to dilute the slurry fed into the cyanide destruction process to 45%. A one-stage SO2/air process will be employed, via three parallel reactors. The system will include SO2 generation and storage as well as feed systems for SO2, lime, and copper sulphate.

17.3 Product/Materials Handling

The major process design criteria are summarized in Table 17-1.

17.3.1 Primary Crushing and Stockpiling

The process plant begins at the primary crusher, where the elevation of the dump pad is set at 1,485 m to match the exit ramp from the pit and avoid any additional uphill haulage once the haul trucks have left the pit. The orientation of the crusher makes use of the northern slope of the hillside, thereby minimizing the amount of excavation for the crusher tower and the MSE wing walls required to create the lower pad for the crusher discharge and feed conveyors to the coarse ore stockpile (COS). Based on the initial geotechnical assessment, the primary crusher will to be founded on bedrock, minimizing foundation requirements.

Table 17-1: Summary of Major Process Design Criteria

	Unit	Design	Design Input*
Ore			
Ore specific gravity	-	2.74	T
Au – Average head grade	g/t	0.74	N
Ag – Average head grade	g/t	5.5	N
S – Average head grade	%	1.8	N
Bond ball mill work index (Wi) 50th %	kWh/t	18.3	T
Bond ball mill work index (Wi) 75th %	kWh/t	19.7	T
Average abrasion Index	-	0.19	T
Axb	-	27.5	T
Grinding Circuit			
Milling rate	t/d	60,000	N
ROM F_{80}	mm	256	N/T
Primary crusher availability	%	75	I
Stockpile live capacity	tonnes	60,000	C
SAG mill feed, F_{80}	mm	133	C
SAG mill product, P_{80}	µm	1,460	C
Ball mill grind, P_{80}	µm	150	N
Ball Mill circulating load	%	315	C
Pebble crusher circulating load	%	30	
Grinding circuit availability	%	92	I
Leach Circuit			
Leach feed thickener unit area	m^2/t/d	0.075	T
Type of circuit	-	CIP	A
Residence time, leach tanks	h	30	A
Residence time, CIP tanks	h	1.5	A
Cyanide consumption	kg/t	0.40	C
Carbon concentration	g/L	50	A
Leach tails thickener unit area	m^2/t/d	0.075	T
Elution Circuit			
Stripping method	-	Pressure Zadra	N
Number of circuits	-	2	A
Carbon batch size per circuit	tonnes	20	A
Pressure	kPa	500	A
Temperature	°C	150	A
Carbon Regeneration			
Type	-	Indirect	N
Method of heating	-	Electric	N
No. of Kilns	-	2	A
Rate	t/d	80	A
Cyanide Destruction Circuit			
Number of stages	-	1	
Residence time	min	60	E
Oxidant	-	SO_2/air	A
SO_2 addition	g/g CN	6	I
Residual cyanide, WAD	mg/L	TBC	N

From the discharge of the crusher, a stacking conveyor feeds the 42 m high COS further down the hillside; the arrangement minimizes the length of the conveyor and permits it to operate at a relatively flat grade of 6.5°. The base of the COS is set at elevation 1,420 m and the reclaim tunnels beneath at elevation 1,410 m. The reclaim tunnels will be constructed within a cut zone of overburden material and are not designed to be founded on rock, but rather to rest on a layer of structural fill over undisturbed native material.

The reclaim conveyors exiting the tunnels will cross the valley bottom horizontally, supported on a rockfill embankment, before rising at an 11° incline up the opposite hill slope and into the mill building at a elevation of about 1,450 m.

The primary crusher is a gyratory-type crusher designed to crush at an average rate of 3,333 t/h to a P_{80} of 133 mm.

The dump pocket capacity is 650 tonnes (live capacity of 406 m^3), twice the size of an average truckload, to reduce the waiting time for trucks. Normal practice is for trucks to dump only when ore levels in the pocket are low. The surge pocket under the crusher will also have the capacity of two truckloads. The dump pocket will be equipped with a water spray, an agglomerative dust suppression (ADS), or a “fogging” system. The apron feeder discharge chute will be equipped with a baghouse-type dust collector.

Crushed ore will be transferred from the surge pocket to the crusher discharge conveyor at an average rate of 3,333 t/h. The apron feeder under the crusher will be equipped with a variable-speed drive (VFD), allowing it to control the loading on the crusher discharge conveyor. The feeder will be set at a withdrawal angle of 60°.The crusher discharge conveyor will feed the COS feed conveyor, which will be an uncovered conveyor structure constructed with a standard walkway on both sides. The belt will be equipped with a rip detection system that will automatically stop the conveyor in the event any tearing is detected.

The COS will have a live capacity of 60,000 tonnes, which translates to 24 hours of nominal process plant operation. The total live and dead storage capacity will be 300,000 tonnes, equivalent to about five days of normal operation. With the use of a bulldozer, this allows the process plant to continue operating for the duration of a complete crusher concave/mantle relining, when necessary.

The COS will be equipped with two lines of three reclaim apron feeders sized such that two feeders per line can deliver the design rate. The apron feeder discharge chutes will be equipped with a baghouse-type dust collector to control dust in the tunnel.

17.3.2 Grinding

The simulation results established that the base case flowsheet to achieve 60 kt/d at P_{80} 150 µm and 75th percentile hardness is two parallel lines, each with one semi-autogenous grinding (SAG) mill, pebble crusher, and ball mill.

Each SAG mill feed conveyor will be equipped with a rip detection device to alert personnel of belt problems.

The SAG mill itself will be trunnion-supported and driven by two variable-speed motors. The SAG mill will discharge to a dedicated vibrating scalping screen with 13 mm square apertures to achieve a SAG transfer P_{80} of 2.1 mm. Oversized material will be diverted to pebble crushers, whereas the transfer undersize will be sent to the SAG / ball mill discharge pumpbox for classification via the ball mill cyclones.

Lime (1 kg/t) will be added to the SAG mill, along with 140 mm (5.5") size steel ball grinding media.

The ball mill coupled with each SAG mill will operate in closed circuit configuration with cyclones to produce a P_{80} of 150 µm. The ball mill will generally use standard 2.5" balls, but these can be increased to 4" as required. Each ball mill will rotate unidirectionally in the opposite direction to the other. Trommel magnets will be installed on the ball mills to remove balls and ball chips from the ball mill discharge slurry. After discharge from the ball mill, the slurry will pass through a grizzly to pick out any remaining large steel before entering the cyclone feed pumpbox.

The strategy for the design of the SAG mill liners for the 2013 Feasibility Study was primarily to ensure that the grate size and open area enabled consistent production in the comminution circuit. After a JKSimMet simulation of the grinding circuit, the SAG mill grates were adjusted to provide 12% open area and all 3.5" pebble ports for the mass transfer of critical size material out of the SAG mill to the pebble cone crusher; this pebble port size is at the maximum for industry-accepted design parameters and is required to ensure the SAG mill meets the production target, particularly considering the hardness of the ore.

The design of the SAG mill liners will be developed further in the detailed engineering phase. The ball mill liner design will also be carried out in detailed engineering.

The cyclones used to classify ball mill feed versus leach feed will operate at a feed density of 55% solids by weight to produce a circuit product size P_{80} of 150 µm.

The grinding area will be equipped with two overhead cranes for mill relining, SAG mill screen replacement, and other maintenance. In addition, a sump pump will be installed for cleanup of the basement floor in case of spills.

Oversized SAG discharge will be sent to a pebble crusher to manage a circulating load as high as 30%. The pebble crusher will have a retractable VFD belt feeder to allow for changes in feed rate as well as to provide easy access for relining and maintenance.

A bypass system on the pebble crushing facility will automatically divert uncrushed pebbles to an emergency stockpile or to a conveyor for return to the SAG mill feed conveyor. The pebble crusher will also be protected from tramp metal by a metal detector on the pebble conveyor that will automatically activate the diverter gate to the bypass position when metal is detected on belt. A surge bin ahead of the crusher will ensure that the feed rate can be controlled and to enable choke-feeding.

A dust collection system will be provided at the crusher discharge chute to manage dust generated by the crusher.

This area will be equipped with an overhead crane for crusher relining and other maintenance, and a sump pump will be installed for cleanup of the basement floor.

17.3.3 Leach Feed Thickening

Ground ore from the SAG / ball mill circuit will flow from the cyclone clusters by gravity to two trash screens for the removal of organic materials, metal, and other miscellaneous tramp materials. The undersize from the two trash screens will flow by gravity to the leach feed thickener, whereas the oversize will be diverted to a trash screen bin, which will be emptied periodically.

The leach feed thickener preliminary sizing indicates an 80 m ø high-rate type with an auto-diluting feed well. The feed slurry density of 30% solids will be increased to a target of about 55% solids in the underflow after thickening. The speed of the underflow pumps underneath the thickener will be varied to control the solids density of the feed to the leach circuit.

Thickener overflow water will be sent to the cyclone feed pump boxes.

17.3.4 Whole Ore Leach and CIP

The grinding thickener underflow stream will be diverted to a leach feed tank where it will be mixed with cyanide to achieve a concentration of 500 g/L.

The slurry will be diverted between two distribution tanks, each of which will provide feed to two whole ore leach (WOL) lines.

The WOL line will consist of four blocks of six tanks in series, each 18 m ø x 20 m high. Slurry will overflow from one tank to the next as it makes its way through the line. Each tank will be equipped with manual bypass valves and a bypass line traversing the tank to allow for isolation of a specific tank as required.

Mechanical agitators in the tanks will maintain the slurry in suspension. Air will be injected into the tank from a gas sparger system at the base to maintain the solution oxygen level needed for process kinetics. To prevent the generation of hydrogen cyanide, the slurry pH will be maintained at a minimum of 10.5 by feeding milk-of-lime to the first five tanks.

Once leaching is completed, the slurry from all four leach lines will be recombined in the CIP tank pumpbox and pumped to the CIP tanks.

The pump-cell CIP circuit arrangement will consist of two lines of seven 400 m^3 tanks operating in carousel mode. In this mode of operation each tank has its own discrete batch of carbon, which spends a definite period of time in the circuit before the entire batch is removed to elution. Gold loadings are expected to be more uniform, with gold-in-carbon and gold-in-solution profiles easier to manage and predict, thus improving the accuracy of gold accounting.

Since the activated carbon is not transferred from contactor to contactor, no transfer pumps are required, and the interstage screens are sized to suit the flow. This reduction in carbon pumping is expected to reduce carbon consumption.

The circuit will operate at a carbon concentration of 50 g/L of slurry and will need a total of approximately 280 tonnes of carbon. It is anticipated that 40 t/d of loaded carbon will be generated.

Two 30 tonne overhead cranes will be located in the area for maintenance access as needed.

17.3.5 Stripping Circuit

The slurry containing loaded carbon from the CIP circuit will be pumped to a vibrating carbon recovery screen. The carbon washed from the screen will fall through a chute into two acid wash vessels. The remaining slurry on the recovery screen will flow through the screen deck to be collected in a screen undersize launder and pumped back to the CIP feed.

Pressure Zadra technology will be applied for the carbon stripping process. In addition to the two acid wash vessels, the circuit will consist of two carbon stripping vessels, one pregnant solution tank and one barren solution tank. Carbon will be processed in batches of 20 tonnes, such that two elution cycles can be achieved per day. Thus, with each strip cycle having a duration of 11 hours, it will be possible to strip 80 t/d of carbon during high silver feed grade periods.

Each strip cycle consists of three distinct phases. The first is acid washing, where a 4% w/w concentration of hydrochloric acid (HCl) is used to fill the vessel containing the loaded carbon from the bottom up. The acid is recirculated for 1.5 hours at a maximum solution contact rate of two bed volumes per hour.

This is followed by a neutralization rinse in the same vessel. A 1% w/w solution of sodium hydroxide (NaOH) is circulated through the carbon over a period of 1.5 hours; the maximum solution contact rate for this phase is two bed volumes per hour.

The last phase is elution, where the carbon is transferred to two carbon stripping / elution vessels. Here, the carbon is subjected to an upflow of strip solution, consisting of fresh water, 1% sodium hydroxide, and 0.1% sodium cyanide (NaCN). This phase is carried out at a pressure of 500 kPa and a temperature of 150°C. It lasts 8 hours and produces a solution strip volume of 640 m^3.

Eluate is bled out of the stripping vessel at a constant rate to a pregnant solution tank capable of holding all 640 m^3 of strip solution per cycle. This pregnant solution is cooled by inline heat exchangers installed in downstream eluate piping. After the heat exchangers, the solution will flow directly to electrowinning cells in the gold room.

After stripping, the barren carbon is cooled by a water rinse step lasting an hour in the strip vessel. This carbon is then pumped from the strip vessel to two carbon regeneration circuits, each consisting of a 1.5 m x 3.6 m vibrating carbon dewatering screen and a regeneration kiln. The screened carbon is sent to the carbon regeneration kiln while the undersize is sent to a fines tank. The material from the fines tank is pumped through a carbon fines filter press, and captured carbon is stored in bags. Periodically, the carbon fines will be treated in an off-site smelter to recover credit for residual gold and silver values contained therein.

Both carbon regeneration kilns will be indirectly heated electric kilns, each rated at a nominal production of 30 t/d at an availability of 75%. The normal operating temperature for these kilns is 675°C. After heating, the carbon drops into two carbon quench tanks where it is cooled with water and from where it is sent to the carbon sizing screen. Oversized carbon from this screen will be stored in a dedicated storage tank before being pumped back into the CIP circuit as needed. Undersize will join the other carbon fines in the carbon fines tank prior to filtration.

17.3.6 Electrowinning and Refining

Solution will be pumped directly from the 9.1 m ø x 9.1 m high pregnant solution tank to ten 150 ft^3 electrowinning (EW) sludging cells, arranged in five lines of two, at an overall feed solution flow rate of 96 m^3/h. The cells will have 304 stainless steel perforated plate cathodes and punch plate anodes. After electrowinning, the eluate will flow to the barren solution tank and be recycled to elution as part of the carbon stripping process. The target metal concentration for the barren solution is ≤1.8 g/t gold and 4.9 g/t silver.

The air in the electrowinning room will be treated by an overhead scrubber and fan assembly before being released to the atmosphere. The fan will be capable of moving 15,000 m^3/h of air.

Three times per week, a set of three or four electrowinning cells will be cleaned to recover gold into sludge dislocated from the cathodes. This will be performed by washing the cathodes in place with a high-pressure washer to dislodge as much sludge as possible. The sludge will flow to the electrowinning sludge pumpbox and be pumped to two sludge filter presses for dewatering. Cake produced by these presses will be transferred to pans and placed in an electric drying oven at 600°C before being mixed with a prescribed flux to prepare for charging the refining furnace.

The nominal flux formula will be an industry-typical mixture of borax, potassium nitrate, and silica, at the respective ratios of 0.4, 0.3, and 0.3 in g/g of gold.

The furnace will be an induction-style unit, 750 kW, supplied complete with a power pack, power ramping control, and fume collection. A high-temperature baghouse will collect fumes from the furnace. The gas will be cleaned further through a high-temperature dust collector carbon filter.

The melted metal will be poured into a series of moulds to produce doré, while the slag produced will be poured into slag moulds. After cooling, the slag will be broken up, with the high-grade slag material being re-poured to increase recovery and low-grade slag recycled to the grinding circuit.

17.3.7 Cyanide Destruction and Tailings

The cyanide destruction process will take place in three parallel tanks, each 14 m ø x 14 m high. Each tank will have a high-power agitator to assist with gas dispersion in the slurry. The gas will be a mixture of SO_2 and air supplied through a sparging system. The slurry will be fed into the system at a density of 45% solids diluted from 60% solids with TSF reclaim water. The circuit will have a residence time of 60 minutes at the rated throughput.

The anticipated cyanide feed concentration of 100 mg/L in the feed will be reduced to less than 1 mg/L in the TSF. SO_2 will be fed at a rate of 5 g/g of cyanide in solution, or an estimated 1,661 kg/h. To maintain the slurry at the required pH level of 8.5, milk-of-lime will be added at a level of 1.4 kg per kg of SO_2. It will also be necessary to add copper sulphate ($CuSO_4$) at a concentration of 15 mg/L, or 33 kg/h, to maintain the reaction kinetics.

After cyanide destruction, all three tanks will discharge into a tailings tank, from where the slurry will be routed to the TSF.

17.3.8 Tailings Thickening

Tailings generated in the CIP circuit will initially be screened through the carbon safety screens to capture any attritioned carbon particles that remain in the discharge slurry. The undersize from the screens will be sent to the tailings thickener, while the oversize will be stored for later processing through the carbon sizing screen.

The tailings thickener preliminary sizing indicates an 80 m ø high-rate type with an auto-diluting feed well. The feed slurry density of 50% solids will be increased to a target of about 60% solids in the underflow after thickening. The speed of the underflow pumps beneath the thickener will be varied to control the solids density of the thickened tailings. Flocculant will be added at a nominal rate of 0.02 kg/t of feed solids to assist with the tailings thickening.

Overflow water from the tailings thickener will be recycled back to the grinding cyclone feed pumpboxes so that the cyanide recovered with this solution can be reused. Underflow solids will be sent to cyanide destruction.

17.4 Energy, Water, and Process Materials Requirements

17.4.1 Water

The bulk of the water requirements for the process plant will be met with reclaim water recovered from air compressors, column heat exchangers, the thickeners, and the TSF. The reclaim water tank at the plant site will be 18 m ø x 20 m high, and will use a combination of low-pressure process water pumps and booster pumps to distribute the water to the points required.

Fresh water obtained from Tatelkuz Lake will be stored for use in both a gland seal water tank, 4.5 m ø x 5.8 m high, and as fire water, stored in the fresh water / fire water tank, 12 m ø x 14 m high. Three firewater pumps will be installed on the firewater section of the tank: one diesel, one electric, and one jockey. Pumps will also be installed to forward water as needed to the process building, and the truck shop. A portion of the water from this tank will be treated in a potable water treatment plant and stored in a small, 5 m ø x 6 m high potable water tank.

17.4.2 Reagents

The reagent preparation area includes receiving systems and mixing / preparation / metering systems for flocculant, caustic, cyanide, copper sulphate, sodium metabisulphate, antiscalant, and lime. Hydrochloric acid will be handled in the strip area. These systems will all be located in a separate reagent building designed for easy access by transport trucks delivering bulk and packaged reagents.

Reagent requirements are as follows:

- Flocculant will be delivered to the plant in super sacs. The bags will be stored indoors, and a 7- to 10-days' supply will be kept in the mixing area
- Caustic soda will be delivered in liquid form in bulk transport trucks carrying approximately 35 tonnes
- Sodium cyanide will be delivered in solid bulk form in ISOtainers by trucks carrying approximately 20 tonnes. Smaller bag/boxes of sodium cyanide will also be available at all times as a contingency. The transport, management and storage of cyanide will be consistent with the International Cyanide Management Code
- Lime will be delivered in bulk by trucks containing approximately 40 tonnes each and equipped with a pneumatic unloading system
- Copper sulphate will be delivered in powdered form in 1,000 kg super sacs. The sacs will be emptied onto the feed bin of an agitated tank where the powder will be mixed with water to the required concentration. The solution mixture will then be pumped to a holding tank and pumped to the cyanide destruction trains as needed
- Hydrochloric acid will be delivered in bulk liquid form by truck, pumped to a dilute acid tank, and distributed as needed
- Sulphur will be delivered in bulk molten form by tanker trucks and processed in a sulphur burner package provided by a vendor to produce sulphur dioxide (SO_2) for the cyanide destruction circuit

- Sodium metabisulphite will be delivered in super sacs, which will be unloaded into a dedicated mix tank complete with an agitator, a dust collector, and an exhaust fan. The sodium metabisulphite system is a backup system to the generation of SO_2 from molten sulphur
- Antiscalant will be delivered to site in liquid totes, transferred to a mix tank, and pumped to use areas.

17.4.3 Air

Three main air compressors fitted with intake filters and silencers will feed plant air into a receiver for subsequent distribution to different parts of the plant such as the refinery, carbon regeneration, sulphur burner, sludge slurry pump, and filter press areas. Some of this air will be fed to a dedicated receiver in the grinding plant, and some will be fed to two instrument air dryers and a receiver that will provide instrument-quality air for distribution throughout the plant.

A dedicated, self-contained air service system will be provided at the gyratory crusher. This will consist of an air compressor with its own service air receiver, air dryer, and instrument air receiver. Another independent air system will be provided in the reagents area.

A further four dedicated process air compressors, each equipped with an intake filter and a silencer, will be provided for the whole ore leaching lines and cyanide destruction.

17.5 Power

Power requirements are discussed in Section 18.

17.6 Comments on Section 17

The QP notes:

- The concentrator will be a 60,000 t/d facility employing the conventional whole ore cyanide leach process to produce doré.
- Overall concentrator availability is estimated at 92%. This is a typical value seen for properly maintained concentrators of this size where SABC circuits are used.
- Design criteria are based on JKTech testwork, JKSimMet grinding simulations, and laboratory leaching, settling, and carbon absorption tests carried out by various laboratories, consultants, and vendors. Where no data are available, process design criteria were developed in consultation with New Gold based on industry averages and modelling by equipment suppliers and independent consultants

- The overall design utilizes large equipment while maintaining a simple and conventional flowsheet

18.0 PROJECT INFRASTRUCTURE

18.1 Project Layout

The overall Project facilities and major infrastructure cover the mine site area, tailings storage facility (TSF), camp site, airstrip, main access road, and water supply system from Tatelkuz Lake. A layout plan is presented in Figure 18-1.

18.2 Road and Logistics

18.2.1 Access Roads

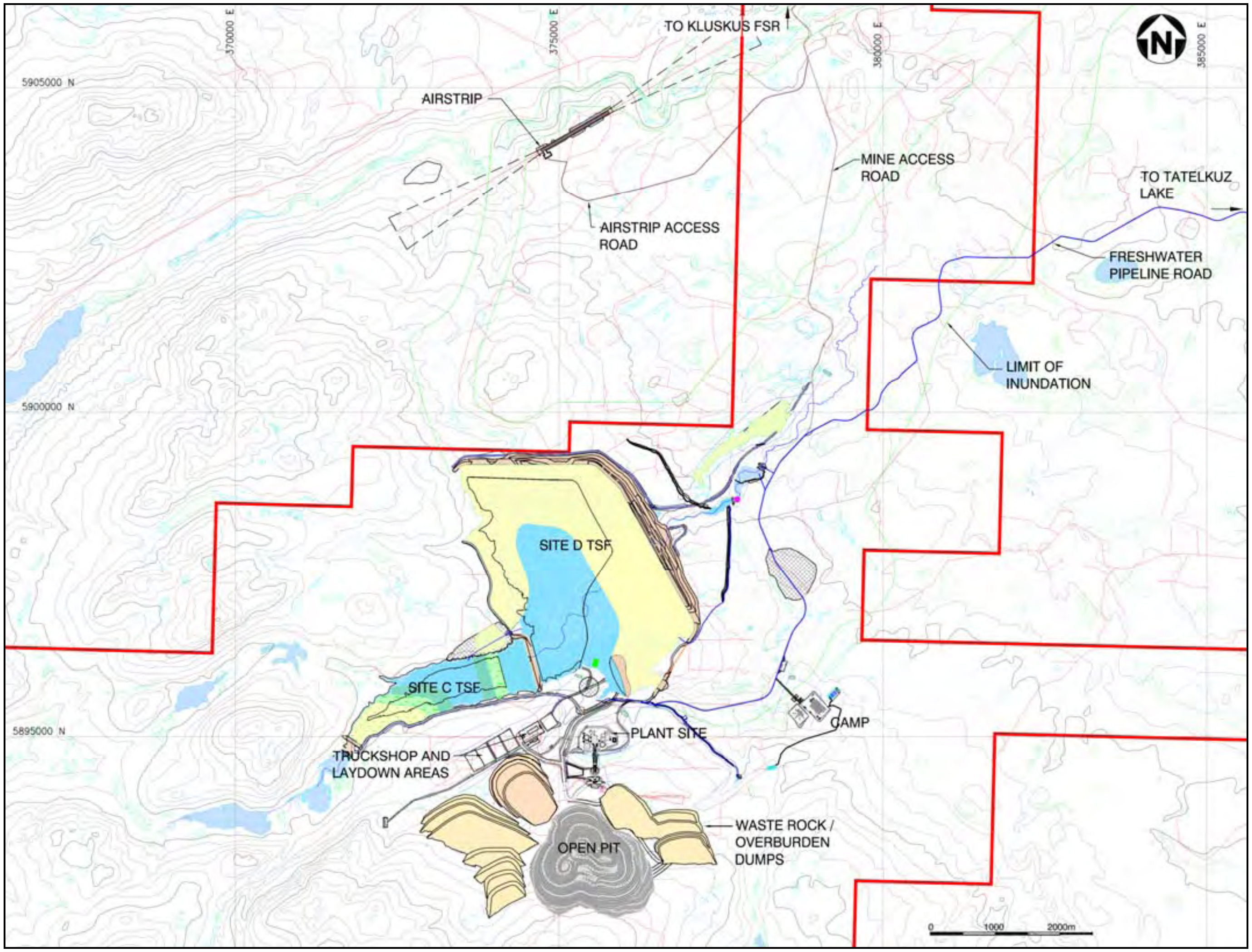
Ground access to the site will be off Highway 16, about 120 km west of Prince George, following the existing Kluskus Forest Service Road (FSR) southwesterly for about 125 km before turning east onto a 15 km long dedicated access road to the mine site. An airstrip for transportation of personnel will be constructed 10 km due north of the mine site. The road to the airstrip will branch off the mine access road about 1 km from the FSR junction and 14 km from the plant site (refer to Figure 18-1).

The Kluskus-Ootsa Forest Service Road is the primary means of access to the Blackwater site. The FSR was built in the early 1970s and is currently maintained by Canfor Forest Products. During Project operations, the Blackwater mine will likely become the main user of the FSR and will be responsible for primary maintenance.

As with most FSRs in Northern British Columbia, heavy traffic is restricted during the spring breakup, which is typically mid-March to the end of May; load limitations are usually in place for most of April and May. New Gold will upgrade part of the FSR to meet the future year-round operational needs of the Blackwater Project.

A study was conducted to identify the upgrades required, current and future road use, interactions with other road users, and the capital and operating costs associated with the upgrades. Engineering drawings were also prepared to support cost estimating and permitting applications.

Figure 18-1: Project Site General Arrangement



Note: Figure prepared by AMEC, 2013

A new 16 km long mine access road (MAR) will replace the existing exploration access road to the site. The MAR will originate at 123+973 km on the Kluskus FSR and extend south to the mine site. Some sections of the water supply pipeline and the power transmission line will parallel the MAR. The road right-of-way will therefore be wide enough to accommodate these structures. The water supply line joins the road right-of-way approximately 7.2 km from the mine site, and the transmission line parallels the road right-of-way all the way from the Kluskus FSR.

The road will be used for heavy traffic during mine construction and has been designed for year-round all-weather access. The road will be 10 m wide, two-lane, have a design speed of 60 km/h, and incorporate five bridges of varying lengths and types. The road design includes ditching to control erosion as well as culverts and cross drains as required.

Geotechnical investigations have identified potential borrow pits for road surfacing aggregates within the road right-of-way.

18.2.2 Airstrip

An aerodrome will be constructed approximately 16 km from the construction camp for the transportation of personnel to and from the mine site during the construction phase of the Project. The airstrip will be sized for Dash 8 class aircraft. The facilities will include an access road, airstrip / taxiway, and small terminal building.

The airstrip has been designed to meet a target year-round availability of 90%. On days when the airstrip is not accessible due to weather restrictions, aircraft will be diverted to the Prince George airport. Alternate bus transportation will be in place to provide transportation to site under those situations.

18.3 Geotechnical Investigations

Knight Piésold completed an evaluation of the geotechnical and hydrogeological conditions of the Blackwater area through extensive site investigation programs in 2012 and 2013. These programs support engineering studies for the tailings and water management systems, plant site, and other mine site infrastructure proposed in the Davidson Creek watershed and for the open pit design on the slopes of Mt. Davidson. Drill hole, ground geophysics, and test pit locations were adjusted as the program progressed and site conditions became better understood.

18.3.1 Site Investigations

The 2012 and 2013 site investigation programs included:

- Excavating 305 test pits to investigate the near-surface material characteristics and foundation conditions
- Drilling 28 geotechnical drill holes utilizing ODEX drilling techniques with standard penetration tests (SPTs) in the surficial materials and diamond drill coring (HQ3) with packer permeability tests in bedrock
- Drilling 66 geotechnical drill holes utilizing sonic drilling techniques
- Drilling 16 geotechnical drill holes in the open pit utilizing geomechanical logging techniques
- In-situ packer hydraulic conductivity testing (Lugeon single packer) permeability tests during rock mass drilling in ODEX drill holes and geomechanical open pit investigations
- Installing 35 standpipe piezometers and five vibrating-wire piezometers in select geotechnical drill holes to investigate static groundwater levels and evaluate the rock mass permeability in the TSF area
- Installing multiple vibrating wire piezometers in 12 observation holes and 12 geomechanical drill holes in the open pit
- Installing 28 monitoring wells developed for long-term groundwater quality monitoring
- Conducting 34 response tests in the screened completion zone of standpipe piezometers and monitoring wells
- Conducting down-hole seismic surveys at five plant site drill holes
- Laboratory testing of surficial materials and rock core samples to determine geotechnical material parameters for the different types of materials encountered
- Rock strength laboratory testing on selected representative core samples to evaluate strength properties and to verify rock mass classification
- Completing 35.3 km of seismic refraction surveys to develop profiles of subsurface bedrock and the saturated water table elevations
- Completing 5.2 km of high-resolution resistivity and induced polarization profiling to develop bedrock and the saturated water table profiles
- Two pumping wells to assess hydrogeology in the open pit.

18.3.2 Site Stratigraphy

The stratigraphy of the surficial materials and bedrock from surface downward is as follows:

- Holocene deposits
- Fraser glaciation deposits
  - glaciofluvial deposits
  - glacial till deposits
  - glaciolacustrine deposits
- Interglacial fluvial deposits
- Older glacial deposits (predominantly glacial till from an earlier period of glaciation)
- Reworked regolith (reworked completely weathered bedrock)
- In-situ regolith (completely weathered bedrock)
- Intact bedrock.

18.3.3 Tailings Storage Facility

The Site D Main Dam alignment was established to optimize use of the natural topography of the Davidson Creek watershed, allowing for efficient and long-term storage of mine waste. The foundation conditions are characterized by a surficial glacial sequence ranging from 4 m to 96 m thick overlying bedrock. Surficial materials at the north abutment are particularly thick, ranging from 14 m to 96 m. The completely weathered bedrock horizon ranges in thickness from approximately 2 m to 30 m. Static groundwater levels range from 7 m to 32 below ground surface, mirroring the surface topography. The groundwater depth is consistent with the saturated zone identified by seismic refraction survey lines.

Drilling and test pits were completed in the TSF Site D basin to evaluate the near-surface conditions for potential seepage pathways. The conditions are characterized by a surficial glacial sequence ranging from 3 m to 94 m thick overlying bedrock; this sequence is deepest in the centre of the Davidson Creek valley. Completely weathered bedrock ranges from approximately 3 m to 30 m thick. Static groundwater levels range from 4 m to 25 m depth, mirroring the surface topography.

The dominant surficial material type at the Site C Main Dam site is low-permeability glacial till ranging in thickness from 27 m to 89 m. Glaciofluvial sand and gravel deposits are also prevalent in this area and overlie the glacial till to form meltwater channel terraces between approximately 10 m and 35 m thick on either side of

Davidson Creek. A completely weathered bedrock horizon was identified on the south side of the valley, ranging from approximately 12 m to 35 m in thickness. The completely weathered bedrock horizon was not encountered on the north side of the valley. Static groundwater levels range from 3 m to 31 m, mirroring topography.

Drilling and test pits were completed within the TSF Site C basin to evaluate the near-surface conditions for potential seepage pathways. The conditions are characterized by glaciofluvial sand and gravel deposits ranging from 1 m to 7 m thick overlying the bedrock. Static groundwater levels range from 1 m to 4 m and mirror the surface topography. The bedrock geology consists of slightly weathered andesitic volcanic rocks. In-situ hydraulic conductivity testing indicates the rock mass has low permeability.

The Site C West Dam is located at the west end of TSF Site C. Drill holes and seismic lines have been completed in this area to investigate the subsurface conditions and optimize the dam alignment. To avoid slope instability and environmental concerns (small lake to west of dam) identified farther upstream at the drainage divide, the dam location was shifted downstream to its current position approximately 100 m to the northeast; seismic refraction surveys indicated this was a more favourable site. The surficial materials are approximately 5 m at this location and are composed of dense fluvial sand and gravel deposits with more than 15% fines content. This composition and the bedrock properties are thought to consistent with those at the previously proposed dam site. Bedrock is fragmentals (felsic tuff and felsic lapilli tuff). The completely weathered bedrock horizon was not encountered. Hydraulic conductivity in the bedrock is low, with values ranging from $4x10^{-7}$ m/s to $3x10^{-8}$ m/s. The static groundwater level is shallow, at 3 m.

The environmental control dam (ECD) and interception trenches will be located approximately 1 km downstream of the Site D Main Dam. Conditions in this area are characterized by a surficial material sequence ranging from 24 m to 108 m thick that is deepest in the centre of the Davidson Creek valley. The completely weathered bedrock horizon in this area ranges from approximately 2 m to 30 m thick. The static groundwater level is a reflection of the surficial topography and ranges from 21 m to 53 m.

Evaluation of potential seepage pathways during 2013 identified that:

- Drilling and mapping of the surficial glaciofluvial channel deposits within the Site C and Site D basins found they were not continuous or hydraulically connected to inter-glacial fluvial deposits.

- Interglacial fluvial deposits were found to be localized, discontinuous, and absent or thin in drill hole core.
- Assessment of the hydraulic conductivities of the highly weathered bedrock yielded values of $3x10^{-5}$ to $7x10^{-8}$ m/s, indicating that the highly weathered bedrock has the potential to be a possible seepage pathway beneath the TSF and would daylight downstream of the Site D Main dam. Subsequent seepage control measures have been incorporated into the design to mitigate this using the ECD and seepage interception trenches.
- The primary cutoff trench for seepage control will be constructed beneath the Site D Main Dam. Seepage from the Site C Main Dam will be captured and contained in TSF Site D. The Site C West Dam will require a 5 m cut-off trench to key the dam into bedrock.

18.3.4 Waste Rock Facilities

Geotechnical conditions in the area proposed for the West waste facility are characterized by surficial materials ranging from 18 m to 75 m in thickness, being thickest at lower elevations. Several eskers and localized kames and ablation till were identified in the footprint area of the West facility. Static groundwater levels range from 3 m to 4 m below surface and mirror the subsurface surficial topography.

Geotechnical conditions in the area proposed for the East waste facility are characterized by a surficial material sequence ranging from 24 m to 108 m thick, with the thickest deposits at the lower elevations. A sub-glacial meltwater corridor was identified in the footprint area of the East facility. Static groundwater levels range between artesian conditions to a depth of 12 m. The bedrock geology consists of andesitic volcanic rocks that are highly weathered at less than 5 m depth.

18.3.5 Low-Grade Stockpile

Sand and gravel deposits comprising a meltwater corridor and kame complex were identified in the footprint of the stockpile. The foundation materials are characterized by glaciofluvial surficial deposits up to 18 m thick overlying the highly weathered andesitic bedrock. The static groundwater level is at approximately 13 m depth below ground surface.

18.3.6 Plant Site

The plant site will be located on a topographic high between the open pit and the TSF at elevation 1,433 m.

The surficial materials at the plant site are primarily glaciofluvial deposits composed of sand and gravel and interbedded silty sand and silty gravel materials. The deposits range in thickness from 1 m to 44 m, being thinnest across the topographic high of the plant site and increasing on the flanks of the hill. The material is dense to very dense below 5 m depth. The static groundwater table was encountered at approximately 7 m below ground surface in the sand and gravel unit.

Andesite bedrock is highly weathered for approximately the first 5 m above moderately to slightly weathered bedrock. The RQD is greater than 60% and RMR is classified as FAIR to GOOD. Rock compressive strength testing ranged from 70 to 275 MPa indicating very strong rock. Hydraulic conductivity of the rock mass was low and ranges from $1x10^{-7}$ m/s to $3x10^{-8}$ m/s.

18.4 Borrow Sources

Potential borrow source locations were identified as follows:

- Approximately 500,000 m^3 of material would be generated from excavations at the planned plant site. The excavated material would be suitable for use as backfill but may require some processing to be used as structural fill or in MSE wall construction
- Some 400,000 m^3 of suitable borrow materials may be available at a location between the plant site and Site C Main dam, within 2 km of the plant site area
- In excess of 3 Mm^3 of materials suitable for use as sub-base material or for processing to produce other well-graded materials could be sourced from a site approximately 5 km down the proposed mine access road from the plant site, just beyond the construction camp location
- In excess of 3 Mm^3 of sand and gravel materials are available from an esker deposit located approximately 10 km from the plant site area. These materials will need to be crushed and/or screened to produce the desired quantities and grain size distributions.

Concrete aggregate suitability testing was performed on two samples, one from the plant site, and the second from the esker deposit. The materials were found to be in compliance, with the exception of the testing for alkali–silica reactivity. The addition of supplementary cementing materials such as fly-ash will mitigate against the potential for alkali-silica reactivity expansion by neutralizing the excessive alkalinity of the cement with silicic acid at the early stage of the cement setting.

18.5 Waste Characterisation

18.5.1 Classification

Mine waste was classified based on its predicted acid generation potential into potentially acid generating (PAG) or non-acid generating (NAG) as shown by the calculated neutralization potential ratio (NPR). NAG waste rock was further classified as to its metal leaching potential based on zinc content. Classification criteria are as follows:

- Overburden (NAG)
- Waste Rock
  - PAG1 – NPR ≤ 1.0 (PAG)
  - PAG2 - 1.0 < NPR ≤ 2.0 (PAG)
  - NAG3 – NPR > 2.0 and Zn ≥ 1,000 ppm (NAG-ML)
  - NAG4 – NPR > 2.0 and 600 ≤ Zn <1,000 ppm (NAG)
  - NAG5 – NPR > 2.0 and Zn < 600 ppm (NAG)
- Ore and Tailings (PAG)
- Low Grade Ore (PAG)

Median NPR values for PAG waste rock (particularly PAG1) are relatively low, indicating a relatively short lag time to acid production; this was verified in laboratory kinetic (humidity cells) and field tests (field bins). PAG waste rock requires subaqueous disposal to prevent ARD formation.

In contrast, median NPR values for NAG waste rock are relatively high, indicating very little potential for acid production. Segregation of waste rock with zinc greater than 1,000 ppm (NAG3) results in high NPR and relatively low zinc and cadmium concentrations in the remaining NAG4 and NAG5 waste rock. NAG4 and NAG5 waste rock can be stored in out-of-pit waste dumps, while NAG3 will be stored subaqueously in the TSF to minimize metal leaching.

Low-grade ore is classified as PAG with a relatively short lag time, and the low-grade ore stockpile is expected to generate acidic drainage with elevated metals until the ore is processed. The low-grade ore will be placed on a compacted till liner with a drainage collection system. The drainage will be neutralized with lime prior to discharge to the TSF.

Sulphide and transition ore tailings are classified as potentially acid generating and will be kept saturated or submerged within the TSF during operations to prevent ARD.

Oxide ore contains very low sulphur, significantly less than transitional and sulphide ore types, and most of the sulphur is expected to be NAG sulphate sulphur Zinc, cadmium, and arsenic are also lower in the oxide ore compared to transitional and sulphide ore types. Lead is higher in oxide ore but has low mobility at the neutral pH expected in the TSF. Oxide ore tailings will be placed as the final lift over transitional and sulphide ore tailings in the TSF Site C to reduce the potential for ARD and metal leaching.

18.5.2 Waste and Ore Classification Block Model

A block model was developed based on the ABA test results and exploration geological metal dataset to classify waste rock and ore blocks. The same criteria for waste rock were used for ore blocks. The acid generation potential of the waste and ore was estimated using the calculated neutralization potential ratio (NPR, where NPR = NP/AP) values derived from sulphur and neutralization potential (NP) assays and estimates.

Concentrations of sulphur were converted to acid potential (AP) values for use in the NPR calculation. Values for NP were determined directly in ABA or estimated from the calcium assay from the larger geological metal dataset; the latter assumed calcium was only present as calcium carbonate and was the sole source of acid neutralization capacity. Separate sulphur and NP block models were prepared and then the estimates combined to determine the NPR for ore and waste rock blocks.

Overburden was classified as NAG based on ABA testing. Overburden blocks were identified from the overburden/rock interface mapped during the exploration program.

ABA data were available for 805 samples of drill core, mainly representing 10 m intervals. These data were composited into 12 m (bench height) down-hole intervals and the resulting Ca and S composite concentrations used to calculate NP and AP values.

In holes with continuous ABA data, those results were used in place of the calculated NPR values (representing 386 samples). In holes with isolated ABA analyses, NPR data from 434 of the samples were merged with 15,636 calculated NPR composites. Duplicates of the merged ABA data were inserted to double the weight of these samples.

AP and NP values were estimated using the ID^2 method in three passes. The first pass used a 50 m isotropic search with no hard boundaries. A minimum of three and maximum of eight composites from at least two drill holes were required to estimate a block. The second pass treated the main north–south fault structure as a hard boundary and used a 200 m isotropic search for blocks not estimated in the first pass. The minimum and maximum number of composites required was the same as used in the first pass. A third pass also treated the fault as a hard boundary and used a 500 m isotropic search for blocks not estimated in the first two passes. A minimum of two and maximum of eight composites were required to estimate a block, with no restriction on the number of holes required.

NPR values were calculated from the estimated AP and NP values. In three areas of very low Ca and S (< 0.02%) a value of three was assigned for NPR.

18.5.3 Zinc Block Model

Samples from core drilling from 2010 to January 2013 were analyzed by ICP methods for 36 elements, including zinc, to characterize a total of 291,437 m of drill core.

Zinc concentrations were estimated using ID^2 in three passes. Search parameters were identical to the NPR block model runs.

The results of the waste and ore classification block model were used to develop the mine waste production schedule and the waste management plan.

18.6 Low-Grade Stockpile Design

The low-grade stockpile will be placed directly to the northwest of the pit (refer to Figure 18-1). It will be constructed from the bottom up in 20 m lifts. Lower lifts will be resloped to an overall slope of 3:1 as they are completed to maintain stability until the stockpiled material is processed.

In the pre-production period the stockpile area will be split into higher- and lower-grade areas to allow higher-grade ore to be fed to the process plant in the initial years of operations.

18.7 Waste Storage Facilities

18.7.1 Waste Handling

Knight Piésold provided a staged construction plan for the tailings storage facility (TSF), which included a listing of minimum annual material requirements. This plan is the key driver of the waste handling strategy, considering that a large proportion of the non-PAG waste is required to construct the TSF.

In addition to the TSF construction, several other considerations affected the waste mining schedule:

- Minimizing overburden mining in wet periods (nominally April, May, and September)
- Maximizing overburden mining in frozen periods (nominally December to February)
- Placing overburden on the outer layers of dump lifts to simplify reclamation
- Providing adequate low-reactivity rock for use as plating material for trafficability while working with the overburden and other weaker materials
- Maintaining an overburden dump as a source of reclamation material
- Creating an overburden stockpile for TSF construction in the later years.

The general design goals and constraints for the waste management plan include:

- Maximize backfill in mined-out areas of the open pits
- Minimize the footprint of the waste rock piles outside the pit limits
- Minimize the number of affected drainage basins
- Leave final slopes composed of overburden to facilitate reclamation
- Segregate overburden in the east dump to preserve overburden for future potential use as reclamation material
- Minimize haul distances / times for waste haulage to manage trucking costs
- Provide progressive reclamation opportunities where practical.

18.7.2 Waste Facility Design Criteria

NAG waste (NAG4 and 5) and overburden not used for construction of the TSF will be placed in either the East or West waste facility locations. All NAG4 material will be placed in the West waste facility in order to capture the runoff from the facility in the TSF due to the somewhat higher metal leaching potential of this material. During the final year of mining (Year 14) approximately 8 Mt of PAG material will be backfilled in the southwestern part of the pit.

The mined overburden will be distributed between the two waste facilities based on the ratio of rock to overburden placed in a given period. A minimum of 10% of waste rock will be placed with the overburden to provide plating and maintain the overall integrity of the waste facilities.

The East waste facility, to be sited east of the pit (refer to Figure 18-1), is the primary storage location for overburden and NAG5 waste not required for TSF construction. Some areas of this dump will consist primarily of overburden (80%) that can be used to reclaim the TSF and other mine facilities at closure. The dump configuration and sequencing are designed to accommodate the use of overburden for reclamation. The overall dump has a capacity of approximately 50 Mt and will be constructed in a series of lifts to enhance stability and minimize resloping requirements at closure.

The West waste facility, to be located west of the pit (refer to Figure 18-1), is the primary storage location for overburden and NAG4 waste not required for TSF construction. The waste facility has a capacity of approximately 87 Mt and, similar to the East facility, will be constructed in a series of lifts to enhance stability and minimize resloping requirements at closure. Some of the West dump lifts will be progressively reclaimed during operations.

The design criteria for the waste rock facilities are shown in Table 18-1.

The stability ratings for the East dump, West dump, and low-grade stockpile were assessed using the rating system described in the interim guidelines provided by the BC Mine Waste Rock Pile Research Committee (BC MWRPRC, 1991). All three areas have been classified as “Class II” low failure hazard dumps (rating of 500).

Stability analyses have been carried out for each of the facilities in accordance with the recommended level of effort indicated in the guidelines. The analyses were conducted using the limit equilibrium computer program SLOPE/W and a similar methodology as was used for the TSF embankment assessment.

The stability of each dump was assessed for a deep-seated failure surface propagating a minimum of 100 m from the crest of the pile, a mid-sized failure surface propagating a minimum of 50 m from the crest, and a smaller-scale failure surface allowed to form just at the crest of the pile. The results of the analyses satisfy the requirements for factors of safety indicated in the guidelines. The seismic analyses showed that any embankment deformations during earthquake loading from the operating basis earthquake (OBE) and maximum design earthquake (MDE) would be minor and would not result in significant displacement in the piles.

Table 18-1: Waste Dump Design Criteria

Item	Unit	Value
Maximum Slope Angle	degrees	37 (1.3H:1:V)
Maximum Reclaimed Slope Angle	degrees	18 (3H:1V)
Maximum Dump Lift Height*	m	40
Maximum Foundation Grade	degrees	15

* Maximum dump lift height is the maximum vertical distance from the dump crest to the toe of the lift.

18.8 Tailings Storage Facilities

18.8.1 Site Selection

Five potential locations for storage of 300 Mm^3 of tailings plus PAG waste rock were identified within 10 km of the deposit area. The alternatives assessment considered economic, operational, and environmental factors and concluded that Sites C and D were the best options. TSF Site C and Site D were compared further, and the results of the comparative economic analysis indicated that the preferred TSF, from an economic perspective, was TSF Site C.

Subsequent alternatives analysis identified that the most effective management of PAG was to have it subaqueously disposed in the TSF in a hybrid Site C/Site D case. Subaqueous disposal was selected as the preferred method of waste rock disposal due to reduced environmental risk, reduced engineering complexity, reduced corporate liability, and lower overall costs.

18.8.2 Hazard Assessment

The Canadian Dam Association Dam Safety Guidelines (CDA, 2007) were used to determine the dam classification and suggested minimum inflow design flood (IDF) and earthquake design ground motion (EDGM) for the Project tailings dams. The tailings dams were classified by considering the potential incremental consequences of a failure.

The following suggested design flood and earthquake levels were adopted from the CDA guidelines for the construction and operational phases of the Project:

- IDF – 2/3 between 1-in-1,000 year and probable maximum flood (PMF)
- EDGM – 1-in-5,000 year return period.

The following design flood and earthquake levels were adopted for the closure phase of the Project:

- IDF – PMF
- EDGM – MCE (1-in-10,000-year event).

18.8.3 Tailings Characteristics

A laboratory testing program was conducted to determine the geotechnical and physical characteristics of the Blackwater tailings. The specific gravity of the tailings solids was determined to be 2.79, and the material can be described as non-plastic sandy-silt with trace clay. The particle size distribution of the tailings sample was approximately 44% fine sand, 46% silt, and 10% clay.

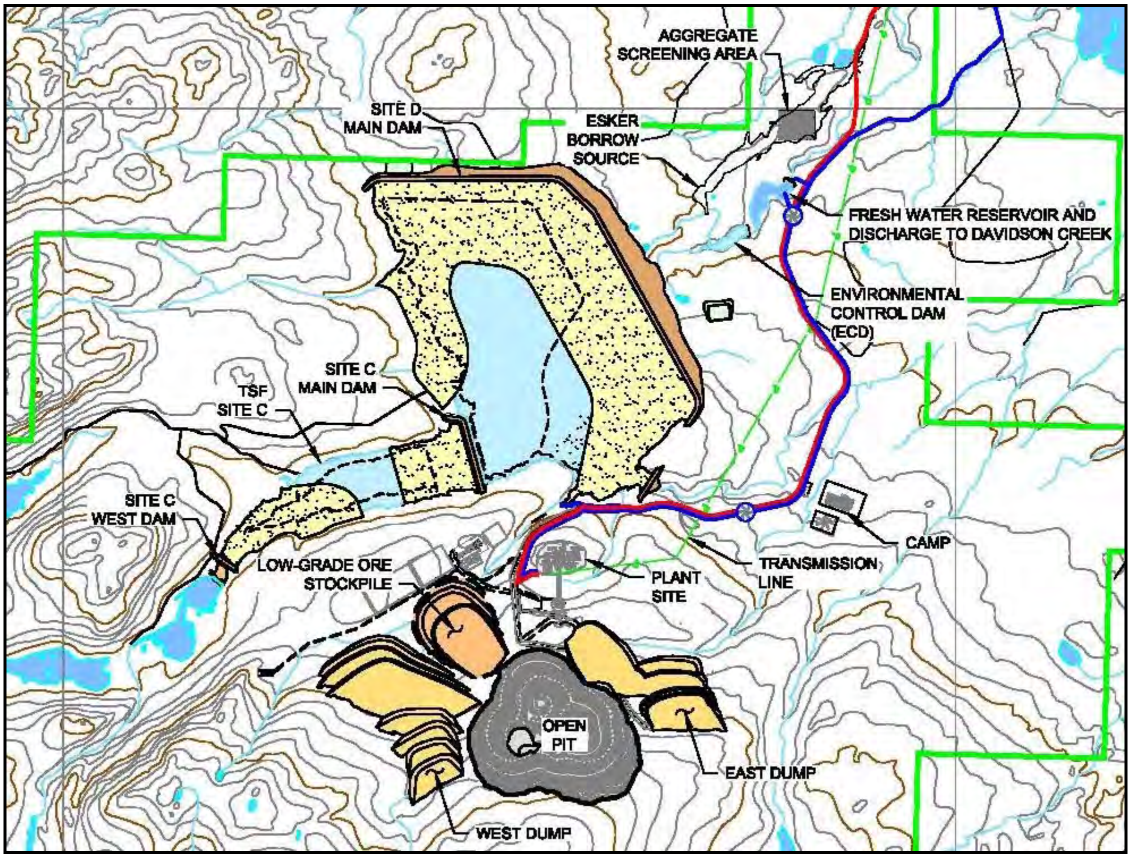
Undrained settling, drained settling, and air-drying tests were carried out to provide information on the effect of initial slurry solids content on the settling and permeability characteristics of the material and the effect on water recovery and achieved density. Tests were performed for a target solids content equal to 50%.

Laboratory tests conducted to determine the consolidation and permeability characteristics of the tailings included slurry consolidometer testing, a low-stress slurry consolidation test, and a falling head permeability test (conducted on settled tailings after completion of drained settling). The tailings rheology characterized the slurry as a non-Newtonian, Bingham plastic slurry typical of tailings slurries. It is fairly coarse and clean with good packing density and low rheology. The rheology results were used to establish depositional velocities and frictional pressure losses in the tailings pipeline design.

18.8.4 Facility Design

The facility has been designed to contain 453 Mm^3 of tailings and waste rock material and will require 73.9 Mm^3 of construction material, about 65.8 Mm^3 or 89% of which will be waste rock and overburden from the open pit. Use of open pit materials is fully integrated into the mine plan to minimize the need for borrow material. The general TSF arrangement is included as Figure 18-2.

Figure 18-2: TSF Simplified General Arrangement



Note: Figure prepared by Knight Piésold, 2013. Map north is to top of the figure. The distance from the open pit to the main dam at Site C is approximately 3 km

The TSF Site C Main Dam will be constructed to elevation 1,353 m and the TSF Site D Main Dam to elevation 1,340 m. Specific overall features of the TSF are:

- Cofferdams and sediment ponds to manage water during construction by either routing water around the TSF or directing water to the TSF for collection
- Three zoned water-retaining earth-rockfill dams referred to as the Site C Main Dam, Site C West Dam, and Site D Main Dam
- Environmental control dam (ECD) and interception trenches to capture seepage downstream of the Site D Main Dam and direct water to the TSF
- Collection channels that route water to the TSF
- Tailings distribution system
- Reclaim water system

- Tailings beaches
- Supernatant water pond
- Designated PAG/NAG3 waste storage areas within the TSF.

TSF Site C

TSF Site C will be constructed first to provide storage capacity for start-up of the process plant. The facility was designed to contain the first two years of tailings and PAG waste rock. The embankment foundations will be cleared and stripped in preparation for fill placement for each stage. The facility will be expanded using downstream construction methods.

Initially, two cofferdams and a downstream sediment pond will be constructed. The first cofferdam for the Site C Main Dam will be built into the dam footprint and sized to contain flows from a 1 in 10-year wet August through October period and a 1 in 10-year, 24-hour storm, with 2 m of additional freeboard. A sediment pond will be constructed downstream of the Site C Main Dam footprint to allow for collection and pumpback of sediment-laden construction runoff. The second cofferdam will be constructed upstream of the Site C West Dam and will create and isolate the TSF from the fish compensation pond and channel to be constructed in the upstream part of the catchment.

Construction of the Stage 1A dam will commence following completion of the sediment and erosion control features. The dam will be built to elevation 1,300 m by the end of Year -2. Construction of the Stage 1B dam will begin in early Year -1, immediately following completion of Stage 1A. The dam will be raised to elevation 1,325 m to provide sufficient capacity to impound PAG waste rock generated during preproduction pit stripping and a start-up pond up to 10 Mm^3, with additional capacity to contain the IDF. Construction of Stage 2 will commence thereafter and will be completed by the end of Year -1. The dam will be raised to elevation 1,340 m, providing the additional capacity required to contain the first year of tailings and PAG waste rock.

In the first year of operations, the final lift (Stage 3) on the Site C Main Dam will be completed to elevation 1,353 m. Construction of the final lift will commence in early 2017 with a downstream shell zone step-out of approximately 50 m. The step-out will be constructed by the mine fleet during the first three quarters of the year, with the actual dam raise taking place late in the year.

In addition, the Site C West Dam will be constructed to elevation 1,353 m.

TSF Site D

TSF Site D will be constructed to provide storage capacity for the rest of the operational life of the Project (Years 3 to 17). The facility is designed to contain tailings, PAG waste rock, and an operational supernatant pond of at least 10 Mm^3, with additional capacity to contain the IDF. The facility will be expanded using a centreline construction method.

Initially, a cofferdam and a downstream sediment pond will be constructed for water management purposes. The cofferdam will be built into the ultimate dam footprint and sized to contain flows from a 1 in 10-year wet October through December period and a 1 in 10-year 24 hour storm, with 1 m of additional freeboard. A spillway to bypass the construction area has been sized to manage a 1 in 200-year storm event. The cofferdam and spillway will be constructed in late Year -2 to allow for water collection from the TSF Site D drainage area. A temporary pump system and pipeline sized to move 1,500 m^3/h of water has been designed for this location to route the spring and summer high flows up to TSF Site C to build the start-up water pond.

Construction of Stage 1A for TSF Site D will commence in Year -1. The COT will be excavated on the abutments first during the earlier months, with in-creek works taking place during the summer low flow period beginning in August. The dam will be built to elevation 1,240 m by the end of Year -1.

Construction of the Stage 1B dam will commence beginning in early Year 1, immediately following completion of Stage 1A. The dam expansion will consist of upstream and downstream step-outs of the shell zone (Zone C) at elevation 1,240 m. The step-outs will be constructed in approximately 65 m wide sections and be completed by the mine fleet using select material from pit stripping.

Construction of Stage 2 will commence in Year 2 and the tailings dam will be raised to elevation 1,272 m, providing enough capacity to impound PAG waste rock generated during Year 2, tailings and PAG waste rock generated during Year 3, and a pond of up to 10 Mm^3 with additional capacity to contain the IDF.

The tailings dam will be raised annually thereafter to maintain the storage capacity of the TSF. Each raise is designed to provide enough storage for the following year of tailings and PAG waste rock and to maintain a pond allowance of at least 10 Mm^3 with additional capacity for storage of the IDF.

18.8.5 Seepage

Seepage will be controlled primarily by the low-permeability core zone constructed prior to the development of the tailings beach, the cutoff trenches, and the low-permeability foundation materials. Seepage from the TSF will result from infiltration of ponded water directly through the embankment fill and the natural ground, and from expulsion of pore water as the tailings mass consolidates.

Special design provisions incorporated into the tailings dam design to minimize seepage losses include the development of extensive tailings beaches to isolate the supernatant pond from the dam, hydraulic barriers, embankment drainage collection systems, and toe drains at the downstream toe of the dams to reduce seepage gradients. Additional seepage collection ditches constructed along the toe of the embankments will collect seepage and surface runoff and direct the flow to the pumpback systems.

Secondary seepage collection at the ECD will be achieved by constructing a collection dam approximately 1 km downstream at a topographic low point in Davidson Creek. A pumpback system will manage seepage and storm water inflows. Recovered water will be pumped to TSF Site D, and the collection pond will be kept dewatered to the maximum extent practical. Seepage through this dam will be captured in an embankment drain system and sump and be pumped back to the ECD pond.

Two seepage interception trenches, one on each side of Davidson Creek, will be excavated through the surficial sand and gravel terraces downstream of the Site D Main Dam and will report to the ECD pond. The locations of the seepage interception trenches are based on the results of geotechnical drilling along the proposed alignments. The trenches will be excavated and keyed into the low-permeability overburden horizon and will be approximately 3.3 km long and typically 5 m to 15 m deep.

Groundwater monitoring wells have been installed in the downstream areas below Site D and could be converted to recovery wells, if required, to recover any foundation seepage. Additional monitoring wells will be installed as required before Site D is commissioned.

The Site C West Dam seepage will be controlled in the long term by constructing a pond upstream of the TSF at an elevation higher than the TSF supernatant pond, and by selectively discharging the benign oxide tailings to hydraulically separate the supernatant pond from the West Dam.

The elevation of the supernatant pond in Site C will be controlled passively by the closure spillway for the facility. The fresh water pond upstream of the TSF will create a hydraulic barrier, forcing fresh water seepage toward the TSF. The area between the two dams will be backfilled and graded toward the fresh water pond. A seepage sump and pumpback system will be constructed between this pond and the TSF, which will be used for monitoring water quality.

18.8.6 Static Stability and Seismic Deformation

Analyses were carried out to investigate the stability of the embankment under both static and seismic loading conditions. The stability analyses were carried out using the limit equilibrium computer program SLOPE/W. In this program a systematic search is performed to obtain the minimum factor of safety from a number of potential slip surfaces. Factors of safety have been computed using the Morgenstern-Price method.

In accordance with international recommendations (ICOLD, 1995) and standard industry practice, the minimum acceptable factor of safety for the tailings embankment under static conditions is 1.3 for short-term operating conditions and 1.5 for long-term (steady-state and post-closure) of the TSF. A factor of safety of less than 1.0 is acceptable under earthquake loading conditions, provided that calculated embankment deformations resulting from the seismic loading are not significant and that the post-earthquake stability of the embankment maintains a factor of safety greater than 1.2, implying there is no flow slide potential.

The seismic stability assessment of the TSF included an estimation of seismically induced deformations of the dam from the operating basis earthquake (OBE) and maximum design earthquake (MDE) events. The OBE has been defined as the 1-in-500 year earthquake with a mean peak ground acceleration (PGA) of 0.04 *g* and design earthquake magnitude of 8.5. The MDE corresponds to the 1-in-5,000 year earthquake with a mean PGA of 0.08 *g* and design earthquake magnitude of 8.5. To demonstrate the robustness of the embankment design to seismic loading, the 1 in 10,000-year earthquake was also considered. The PGA for the 1-in-10,000 year event is 0.11 *g*.

The results of the stability analyses satisfy the requirements for factors of safety and indicate that the proposed design is adequate to maintain both short-term (operational) and long-term (post-closure) stability. The seismic analyses indicate that any embankment deformations during earthquake loading from the OBE, MDE and 1-in-10,000 year event would be minor and would not have a significant impact of the available embankment freeboard or result in any loss of embankment integrity. The results also show that the embankments are not dependent on tailings strength to maintain overall stability and integrity.

18.8.7 Deposition Plan

Tailings from the process plant will be thickened and delivered by gravity through a pipeline from the thickener to either the TSF Site C or the TSF Site D TSF. The pipeline will be laid parallel to a service road from the process plant to an energy-dissipating drop box where the tailings distribution pipelines to TSF Site C and Site D will separate. The tailings drop box is designed to reduce the extra head for early stages or for discharge at closer locations along the dams. The design includes a bypass pipeline around the tailings drop box to ensure sufficient head is available for delivery of tailings to the far ends of Site C and Site D.

The tailings pipelines are sized to ensure gravity flow for the entire duration of the mine life while dissipating as much energy head through friction as possible.

Additional pipelines extending from the drop box to either TSF Site C or Site D will be constructed to allow for emergency discharge to each TSF area. The discharge locations will be far enough away from the reclaim barges to ensure that the reclaim water intakes are free of sediment.

18.8.8 PAG and NAG 3 Disposal Area

A filling curve was developed for each site that takes into account the storage characteristics of the facility and includes the approximate rate of rise of the tailings and waste rock horizon, supernatant pond allowance, and IDF freeboard.

The PAG disposal area will be developed at the same or similar rate of rise as TSF filling level but will be several metres higher to provide a dry, stable placement surface for truck traffic. The design objective for the PAG area is to flood the waste rock within one year of placement. The maximum elevation of the waste storage area will remain at an elevation where it can be flooded by the supernatant pond in the case of premature closure. At closure, this waste storage area will be covered by 0.3 m of overburden and submerged below the final closure tailings elevation.

The PAG disposal area during preproduction and Year 1 of operations will be within and near the centre of TSF Site C. The disposal area will expand as a fill platform with overall slopes at angle of repose. The tailings beaches will provide a low-permeability barrier between the coarse, permeable waste rock and the tailings embankments.

The PAG disposal area beginning in Year 2 will be within TSF Site D. Waste placement will commence near the upstream zone of the TSF Site D Main Dam below an elevation of approximately 1,240 m and will extend west up the valley as a lobe. Beginning in Year 3, the PAG fill platform will be located in the east and north areas of the facility. The fill platform will rise slightly above and with the TSF filling level from Year 3 until Year 14 with the when mining ceases.

The PAG disposal fill platform will typically rise at a rate that stays at least 5 m below the current dam crest elevation. The fill platform will typically meet the advancing tailings beach approximately 500 m to 1,000 m away from the dam.

The surface of the PAG disposal area will be progressively covered by tailings and the supernatant pond between Year 15 and 17 during low-grade ore processing when waste rock production has ceased.

18.8.9 Monitoring

Geotechnical instrumentation will be installed along one plane through the Site C West Dam, Site C Main Dam, and fresh water reservoir, and in five planes through the Site D Main Dam. The instrumentation will be installed during construction and over the life of the Project. The geotechnical instrumentation will consist of vibrating wire piezometers, slope inclinometers, and movement monuments, and will be installed in the foundations, embankment fill, and embankment crests.

Instrumentation monitoring will be carried out routinely during construction and operations. Daily measurements will be taken and analyzed during construction to monitor the response of the embankment fill and the foundation from the loading of the embankment fill. The frequency of monitoring for the piezometers and inclinometers may be decreased to bi-monthly readings once the effects of initial construction have dissipated. Surface monuments will be surveyed at least twice per year during operations.

18.9 Water Management

18.9.1 Objective

Water within the Project area will be recycled and used to the maximum practical extent by collecting runoff from the mine site area. Site runoff water will be collected and stored within the TSF and used to inundate the PAG waste rock and tailings solids to prevent acid rock drainage (ARD) and minimize metal leaching (ML). Excess water will be stored in the supernatant ponds within the TSF and recycled to the mill for use in the process. The water supply sources for the Project are as follows:

- Runoff from the catchment above the Project site
- Direct precipitation onto the TSF and runoff from the mine site facilities
- Water recycle from the TSF supernatant ponds

- Groundwater from open pit dewatering and depressurization
- Water extracted from two wells east of the camp area for potable and firewater use
- Fresh water pumped from Tatelkuz Lake for plant fresh water needs and to mitigate flow reductions in lower Davidson Creek for downstream fisheries
- Periodic water supply from Tatelkuz Lake to supplement requirements for processing or to saturate PAG waste rock within the TSF, if required.

18.9.2 Sediment and Erosion Control

A construction phase sediment and erosion control plan (SECP) was prepared for the Project as part of the study. Best management practices (BMPs) will be implemented before and during construction.

A sediment control pond (SCP) has been designed for each major area of disturbance:

- Construction laydown areas (SCP #1)
- Plant site and crusher area (SCP #2)
- Construction camp area (SCP #3)
- Aggregate screening area (SCP #4).

The sediment control ponds were designed to accommodate a live storage equal to the 1-in-10 year, 24-hour storm event with 0.5 m of freeboard and to settle out sediment particles sized 0.01 mm (and larger), while providing a retention time of at least 20 hours. Each pond and pond outlet spillway was designed to withstand a 1-in-200 year, 24-hour storm event. The collection and diversion ditches were designed for the 1-in-10 year, 24-hour storm event. All ditches will be accompanied by a silt fence installed downslope of the ditch to prevent potential residual sediment movement along undisturbed slopes.

Pre-stripping of the open pit will commence in late Year -2. The pre-stripped material will be used in dam construction or disposed of in the East and West dumps. SCP #5 will be constructed before any disturbance in the open pit area. The pond is designed to manage runoff from the east side of the open pit and the ultimate extents of the East dump. SCP #5 will accommodate a live storage equal to the 1-in-10 year, 24-hour storm and will provide a retention time of over 20 hours for settling suspended sediments. The spillway is designed to manage the 1-in-200 year, 24-hour storm. SCP #5 will discharge to a diversion channel and the water will subsequently flow to collection at the TSF Site D cofferdam for storage and re-use in processing.

Construction of the TSF Site C cofferdam will commence as soon as permits are received. The cofferdam is expected be in service until October of Year -2, at which point the construction of Stage 1A of the Site C Main Dam elevation will rise above the cofferdam, rendering the cofferdam obsolete. The cofferdam is designed to store the 1 in 10-year, 24-hour storm event, as well as the 1 in 10-year wet August, September, and October water volumes. Downstream of the dam, SCP #6 will be constructed before Site C cofferdam construction begins.

Construction of the TSF Site D cofferdam will commence during construction of the Site C Main Dam in October of Year -2. It is designed to store the 1 in 10-year, 24-hour storm event, as well as the 1 in 10-year wet October, November, and December water volumes. The cofferdam includes a spillway, which has been designed to safely pass the 1 in 200-year, 24-hour storm event peak flow. The sediment control pond (SCP #7) downstream of the cofferdam will be constructed prior to any disturbance in the vicinity of the Site D cofferdam.

18.9.3 Operational and Closure Water Management

The construction, operations, and closure water management strategies for the Project have been developed by identifying the size and position of the planned mine site facilities and establishing estimated catchment area boundaries based on the mine site development concept. All site drainage during operations and closure will drain by gravity to the TSF. Virtually all seepage from the TSF and waste rock dumps will also be collected and directed to the TSF. This simplifies water management, spill control, and closure in addition to providing water for the process.

The water stored in the TSF Site C start-up pond will serve as the primary process water source at the start of mill operations until the end of Year 2, with additional water being drawn from the TSF Site D pond (via the pump system at the cofferdam), as necessary. Once tailings deposition in TSF Site D commences in Year 3, and until the end of mining operations in Year 17, the TSF Site D pond will be the primary source of process water. Additional makeup water, if required during that time, will be provided by the TSF Site C pond. The pond in TSF Site C, as of Year 3, will be allowed to accumulate naturally to the closure spillway elevation at or below 1,343 m and then overflow into the pond of TSF Site D in approximately Year 27. Fresh water required for the mill (e.g., gland and reagent mixing water) and any additional water to ensure PAG waste rock and tailings in the TSF remains inundated will be supplied by the fresh water supply pipeline from Tatelkuz Lake.

Groundwater inflow and surface runoff to the open pit, including water from the vertical depressurization wells, will be collected and discharged to TSF Site D and will be recycled for use in the milling process until the cessation of open pit mining in Year 15.

The pit dewatering system will be decommissioned in Year 15 and the pit will begin to fill with water while the low-grade ore is processed through the mill from Year 15 to 17. Once mill operations cease in Year 17, the surplus inflow to TSF Site D (inflow minus losses) will be pumped to the open pit to accelerate pit filling and associated flooding of PAG rock exposed in the ultimate pit walls. Once the open pit is full, predicted in Year 35, it will overflow via a spillway to the TSF Site D pond. Subsequently, the TSF Site D pond will overflow via the closure spillway and discharge channel to a plunge pool in Davidson Creek downstream of the ECD.

During closure, passive treatment wetlands will be constructed in the sediment pond, ECD, and water reservoir downstream of TSF Dam D to polish TSF seepage. Additional passive treatment wetlands will also be constructed on the surface of TSF Site C in Years 4 and 5 and on the surface of TSF Site D in Years 18 and 19.

18.9.4 Water Management Systems

Environmental Control Dam

The primary seepage collection point downstream of the TSF will be located approximately 1 km downstream at a topographic low point in Davidson Creek. The collection pond will be created by constructing a 12 m high dam (ECD) across Davidson Creek. The dam is designed to contain continuous seepage and runoff from events up to the 1 in 100-year, 24-hour storm. A spillway is designed to pass the 1-in-200 year, 24-hour storm. The pond will be fed by two interception trenches. The primary pumpback system at the ECD is designed to maintain the pond at a minimum water level. The ECD will have an embankment drain system, seepage collection sump and monitoring device, and secondary pumpback system to collect and recycle seepage.

Water Reclaim System

Water reclaimed from the tailings ponds at TSF Sites C and D will be delivered to the reclaim water tank at the mill. The water will consist of supernatant from the settled tailings and runoff from precipitation and snowmelt within the reporting catchment areas. The reclaim pipeline will consist of 750 mm diameter steel pipe for the initial high-pressure sections closest to the reclaim pumps and 800 mm diameter HDPE DR 13.5 and DR17 pipe for the rest of the line.

The reclaim water system will initially utilize a barge-mounted pump station equipped with four 750 kW vertical turbine pumps sized to deliver 3,200 m^3/h of reclaim water. The reclaim barge will be anchored in TSF Site C during Years 1 and 2 of operations and then moved to TSF Site D late in Year 2 for the remainder of mine operations and closure. A reclaim structure will be constructed in TSF Site C during Year 2. The reclaim structure design is similar to the ECD pump station design, consisting of a concrete intake structure with stop-logs and trash racks. The structure will be set at elevation 1,337 m and requires 3 m of submergence to operate. The pumping system will include four vertical turbine pumps of the same type and motors as the barge-mounted system for consistency of spare parts and ease of maintenance.

The reclaim structure has been designed to operate within a 3.5 m fluctuation in TSF pond level, which equates to a 5 Mm^3 fluctuation in pond volume. The TSF pond level will be controlled passively with a spillway to the north of the Site C Main Dam. The spillway will have an invert at elevation 1,343 m, which will allow for long-term availability of the Site C reclaim structure as a backup system.

Low-Grade Stockpile

The low-grade ore (LG) stockpile will be developed over a prepared low-permeability foundation with surface water and seepage collection and monitoring systems. Drainage from the LG stockpile is expected to be acidic and contain elevated metals; therefore the drainage will be collected, neutralized, and discharged to the TSF.

Runoff from the upslope undisturbed catchment will be diverted around the stockpile to the TSF by a 3 m wide diversion channel excavated adjacent to a 6 m wide access road. Foundation drains will be installed in areas of existing drainage lines or in any areas of seepage encountered during topsoil stripping.

NAG Disposal Areas

The East and West dump layouts have been refined to minimize surface water control requirements. Foundation drains will be installed in areas of existing drainage lines or when excessive seeps or springs are encountered during clearing and grubbing. Non-contact surface water will be diverted around the dumps during operations and closure and will be field-fit with the advancing fill platforms. Water that infiltrates through the dump will be collected in ditches near the toe of the dumps and routed to a sediment basin before discharge to the TSF.

18.9.5 Water Balance

A monthly operational and closure water balance was developed for the Blackwater Project using the GoldSim© software package. The model estimates the magnitude and extent of any water surplus and/or deficit conditions in the TSF based on a range of possible climatic conditions. The model period included one year of preproduction (Year -1) and 17 years of operations, at a nominal milling rate of 60,000 t/d dry, and

16 years of closure until the TSF discharges to Davidson Creek. The model incorporates the following major Project components:

- Open pit
- Mill
- LG stockpile
- TSF Site D
- TSF Site C
- East and West waste rock and overburden dumps.

The water balance model assumed that a start-up pond of at least 6 Mm^3 (under average conditions) will accumulate in TSF Site C in the year before mill start-up. Of this amount, 1.0 Mm^3 will accumulate behind the TSF Site C Main Dam based on runoff from its contributing upslope catchment (7.1 km^2), and the other 5.0 Mm^3 will be from undisturbed contributing catchments (25 km^2) at TSF Site D. This latter runoff will be collected behind a cofferdam at the TSF Site D Main Dam location and be pumped to the TSF Site C pond until the start of operations. The minimum operating pond volumes for TSF Sites C and D were assumed to be 3.0 Mm^3 and 7.5 Mm^3, respectively.

The facilities are in a balance or surplus condition throughout operations, given that the water accumulated within the supernatant ponds in TSF Sites C and D, including from open pit dewatering, will satisfy the mill process requirements under average precipitation conditions. Fresh water required for the mill (1.05 Mm^3/a) will be provided by pumping from Tatelkuz Lake. From Years 5 to 10, when tailings are being deposited in TSF Site D, reclaim water will be withdrawn from both the supernatant ponds of both TSF Site D and Site C to meet the process water requirements. The additional process water requirement from TSF Site C is largely needed during the winter months, when precipitation falls as snow.

Excess water becomes available with snowmelt during the spring freshet period, and the system then operates in a surplus condition. The amount of surplus increases over time due to increasing runoff from the expanding disturbed areas of the mine facilities, and also from decreases in waste rock production and corresponding decreases of water storage in waste rock voids for the mine rock stored within TSF Site D.

The maximum operating pond volume for TSF Site D was based on a spillway elevation of 1,336 m. However, TSF Site D is predicted to exceed its maximum operating pond volume in the last year of operations; therefore, a portion of the upstream contributing catchment might be diverted around the facility to Davidson Creek in the later years of mine life.

The open pit begins filling in Year 15 when the dewatering system is decommissioned after open pit mining has ceased. At the end of mill operations in Year 17 (after processing the low-grade ore), the surplus inflow to TSF Site D (inflow minus losses) is pumped to the open pit to aid in pit filling.

TSF Site C continues to overflow naturally into TSF Site D via a closure spillway in Year 27. The open pit takes approximately 21 years (Year 35) from the end of mining to fill based on average precipitation conditions. TSF Site D is predicted to begin discharging via the closure spillway in that same year. Therefore it will take a total of 18 years after the end of mill operations before the system discharges to Davidson Creek.

A stochastic analysis was completed to analyze the range of possible cumulative pond volumes available in TSF Site C and TSF Site D over the mine life, as defined by the 95th percentile wet and dry values (5% and 95% chance of being equalled or exceeded in any month, respectively). This range of volumes also indicates possible active or live storage capacity in the TSF ponds for a reasonably large range of anticipated climatic conditions. The stochastic water balance highlights the sensitivity of the TSF pond volumes to the assumed climatic inputs and pond minimum/maximum operating capacities. The stochastic results indicate that for extreme conditions (fifth percentile), the accumulated TSF ponds, open pit, and associated contributing catchments are not able to supply enough water to meet the process water requirements until Year 11, and the system will operate in a deficit condition until that time.

The stochastic analysis indicates that under extreme dry conditions the supplemental makeup water requirement to support operation of the mine is less than 3 Mm^3/a. The fresh water supply system has enough design capacity to augment this deficit.

The TSF operational pond capacity is sufficient to manage a reasonable range of surplus water conditions. Water levels will be monitored to ensure that the water balance assumptions are accurate and that an operational surface water discharge from the TSF does not occur until closure. As a contingency, non-contact surface water diversions can be created to divert runoff around the TSF, and the TSF dams can be raised more frequently, if required.

18.10 Onsite Built Infrastructure

The process plant site will be on a hilltop between the orebody to the south and the tailings storage facility (TSF) to the north. The plant site buildings and most of the site facilities lie within the Davidson Creek valley and upstream of the TSF catchment area, thus minimizing any direct impact on other watersheds in the area. Surface drainage from the min site will flow to the TSF. The elevation of the plant site platform allows for a gravity tailings system for the mine life.

The primary crusher will be constructed on the hill slope just to the north of the orebody and south of the plant site, across a shallow valley. The crusher and coarse ore reclaim system will be oriented perpendicular to the main grinding area in the plant to ensure equal feed size distribution to each of the two SAG mills. The crushed ore will be discharged onto a 140 m long coarse ore conveyor running to the coarse ore stockpile (COS), downslope of the crusher.

Two identical reclaim conveyors will transport the crushed ore directly to the SAG mills on 450 m long conveyors to be mounted on an earth embankment that crosses the valley floor and continues up to the plant site, where they will be elevated on bents to enter into the mill building.

The mill building will be centrally located within the main process area. The elution and refinery building will be attached directly on the north side and all other process infrastructure distributed strategically around it. The intent of the compact arrangement of facilities is to minimize piping runs and interconnections between process areas. The whole ore leach tanks will be oriented at 90° from the process building to take advantage of the sloping terrain and permit gravity flow from tank to tank. The site electrical substation will be directly beside the plant to minimize the length of large power cables running to the mill motors.

An emergency spill containment pond will be constructed at the east end of the plant site to collect any major spills from the process areas, such as the grinding mills, the CIP tanks, and the WOL tanks. In each area, a sump will be connected to a 600 mm buried collection drain line that discharges directly into the pond.

The main truck shop will be on a ridge west of the plant site to provide easy access for maintenance of the haul trucks. The truck shop will house heavy vehicle repair bays, various specialty service bays, a warehouse, and maintenance offices. The layout has been designed to allow for expansion by up to two bays in the future. The administration and emergency services buildings will be constructed on a separate platform just downslope of the truck shop. The administration building will provide offices for the administrative staff and change facilities for administration and mining personnel. The emergency services building will house an ambulance and fire truck, as well as the first aid attendants and equipment. Having the truck shop and administration buildings close to each other will facilitate coordination between the administration staff and the mining operations.

Because the western ridge top is large and fairly flat, it is well suited for construction laydown use. Minimal earthworks will be required, and laydown areas will be prepared as needed through the construction phase of work.

An explosives storage facility about 2 km farther to the west will store ammonia nitrate prill and emulsion for use in the pit. The facility will be operated by a third-party explosives contractor, with all but proprietary equipment owned by New Gold. The explosives contractor will be responsible for setting up and maintaining the site, ensuring that an adequate supply of explosives is available at all times to meet mining demands, and delivering the explosives to the holes in the pit as dictated by the mine plan.

A fuel farm will be constructed alongside the haul route from the pit to the TSF, between the plant site and the truck shop. Haul trucks will refuel on one side (the mine side) and smaller service vehicles on the opposite side to minimize safety issues with the different sized vehicles. In the same vein, separate service roads for the smaller site vehicles will access all facilities, minimizing interference with the haul trucks. Where the haul road cuts between the truck shop / administration area and the process area, it will incorporate an overpass for the haul trucks before continuing out in the direction of the TSF. Service vehicles will be able to pass safely under the overpass between the two main areas and to travel on a separate road parallel to the haul road toward the TSF until reaching a juncture with the main site access road. After this point, both types of vehicle will need to share the rest of the road to the TSF and related facilities. In this case, mine operations will need to implement strict controls to ensure the safety of all personnel using the common roads.

The initial site construction works will make use of the existing 250-person exploration camp, which will be expanded rapidly to accommodate up to 426 persons within the first few weeks of construction. This will minimize any initial delays in site mobilization. The main camp site for the rest of the construction workforce, and eventually for permanent staff, will be about 3 km east of the plant site, adjacent to the main access road. This camp will accommodate up to 880 construction workers and management personnel. Upon completion of the construction works, the management facility will be refurbished for use by the operations staff; the temporary sections will be dismantled and removed from site. A security office and visitor parking area just ahead of the turnoff to the camp area will regulate all traffic and visitors on the mine site.

Wells will be developed near the new camp area to supply water for the temporary and operations camps. The water will be treated and distributed around the camp site for domestic use. Waste water will be collected and treated and the effluent released in a Rapid Infiltration Basin (RIB) in accordance with regulatory requirements. Given the separation of the camp site and mine/plant development area, potable water and sewage treatment systems will be required at the plant site. It is planned to relocate the treatment plants from the expanded exploration camp to the plant site during the latter stages of construction when the exploration camp is decommissioned.

A refuelling service dispensing both diesel and gasoline fuel will be established at the camp site.

In consideration of safety clearances, the entire plant site area—including the fuel tank farm, primary crusher, and truck shop—falls outside the 300 m flyrock clearance limits around the ultimate open pit boundary.

Figure 18-3 is an overview of the mine layout plan. Figure 18-4 shows the proposed plant site layout. Figure 18-5 shows the plant facilities layout.

18.11 Power and Electrical

The Blackwater Project will require up to 120 MW of power. A 139.5 km, 230 kV overland transmission line will be constructed to connect to the BC Hydro grid at the Glenannan substation located near the existing Endako mine, 65 km west of Vanderhoof, B.C., as recommended in a Systems Impact Study commissioned by BC Hydro. The study identified a number of upgrades to the substation and requirements for system reinforcement, which have been incorporated into the Project costs.

The transmission line has been routed to make use of existing access and to cross recently logged areas as much as practical along its alignment (Figure 18-6). The alignment was scrutinized to minimize impact on the environment and local stakeholders, and the design was optimized for reliability and constructability to reduce the effects of terrain on cost and construction. Alternative line alignments and BC Hydro interconnection points were contemplated throughout the design process before settling on the current alignment.

Figure 18-3: Mine Site Layout

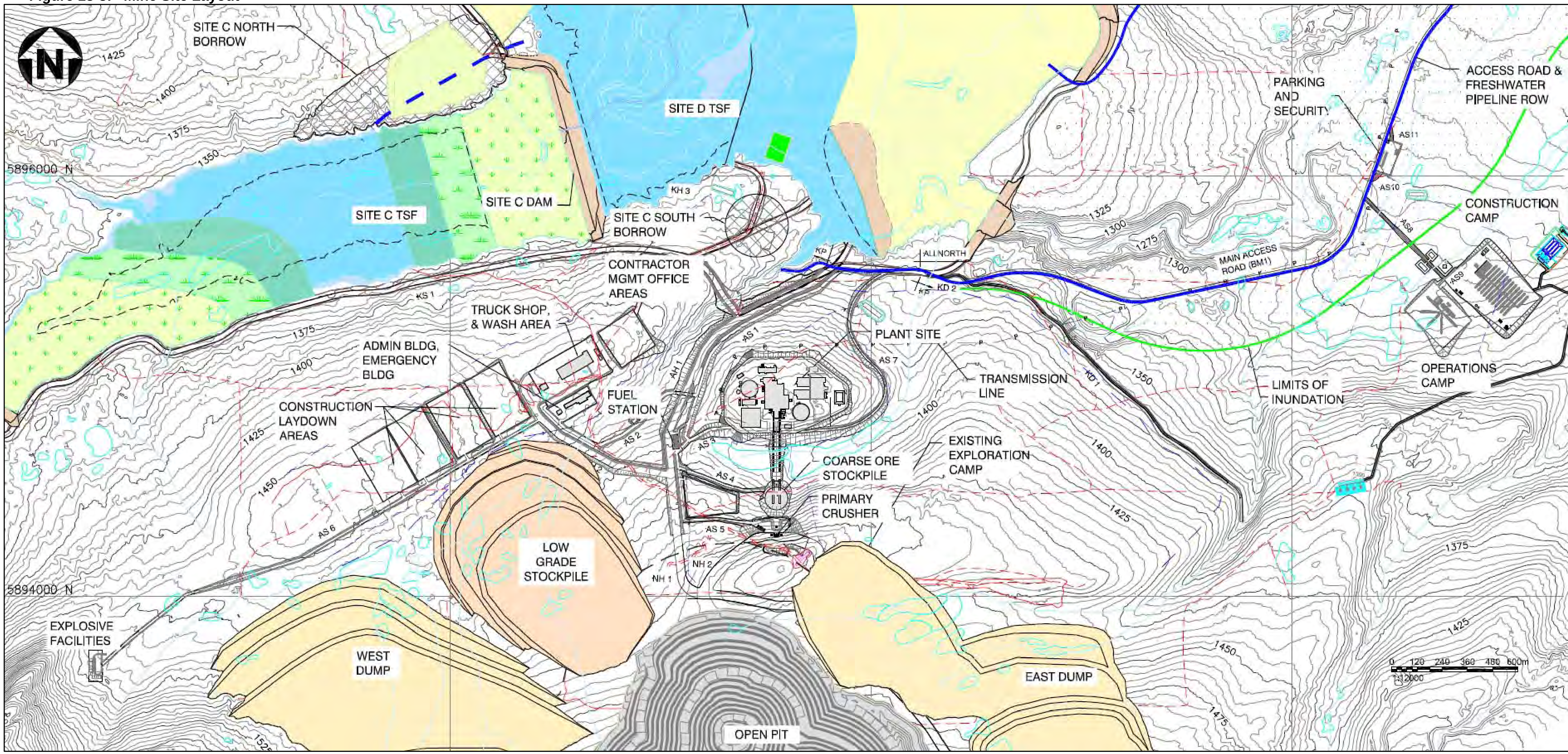
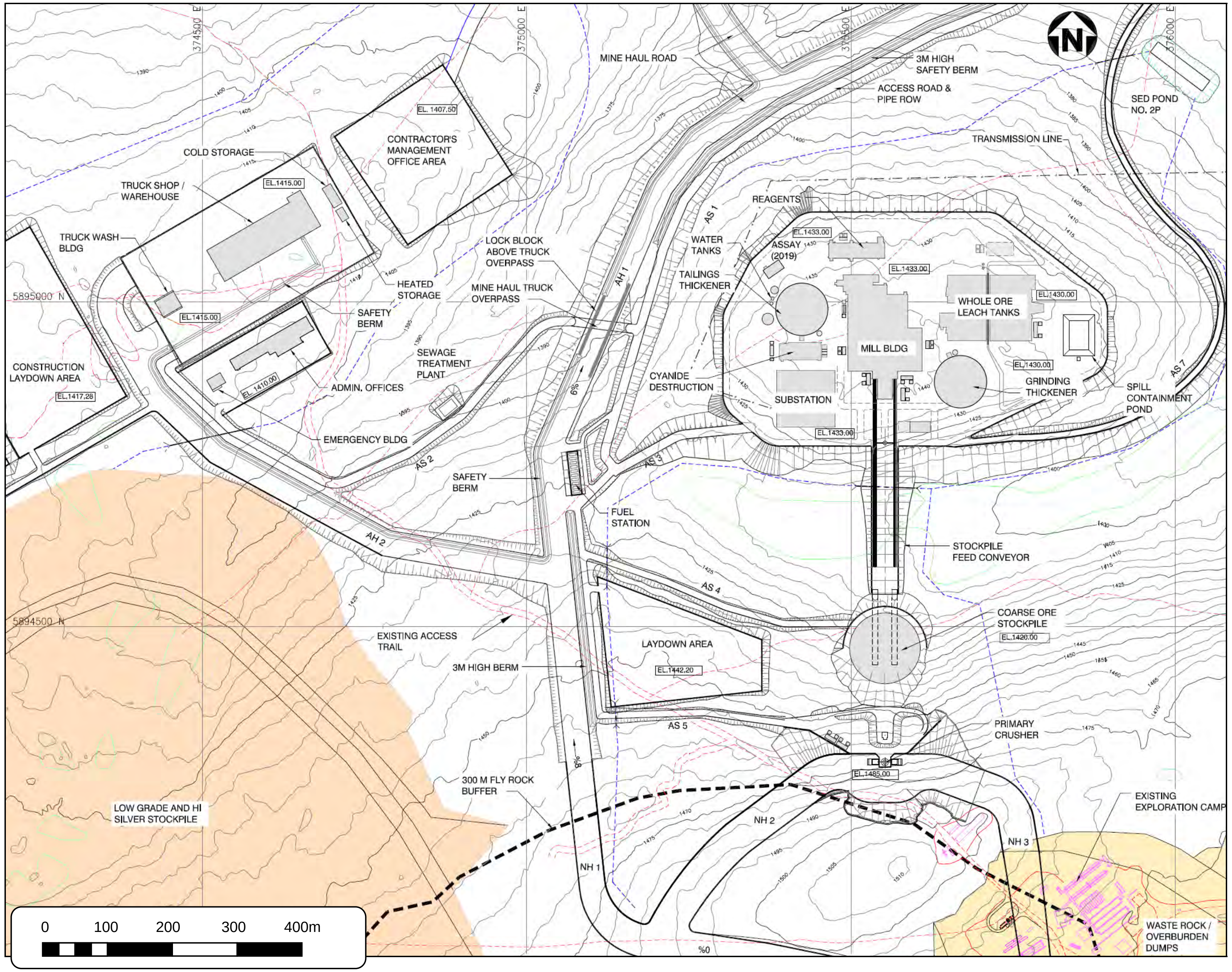
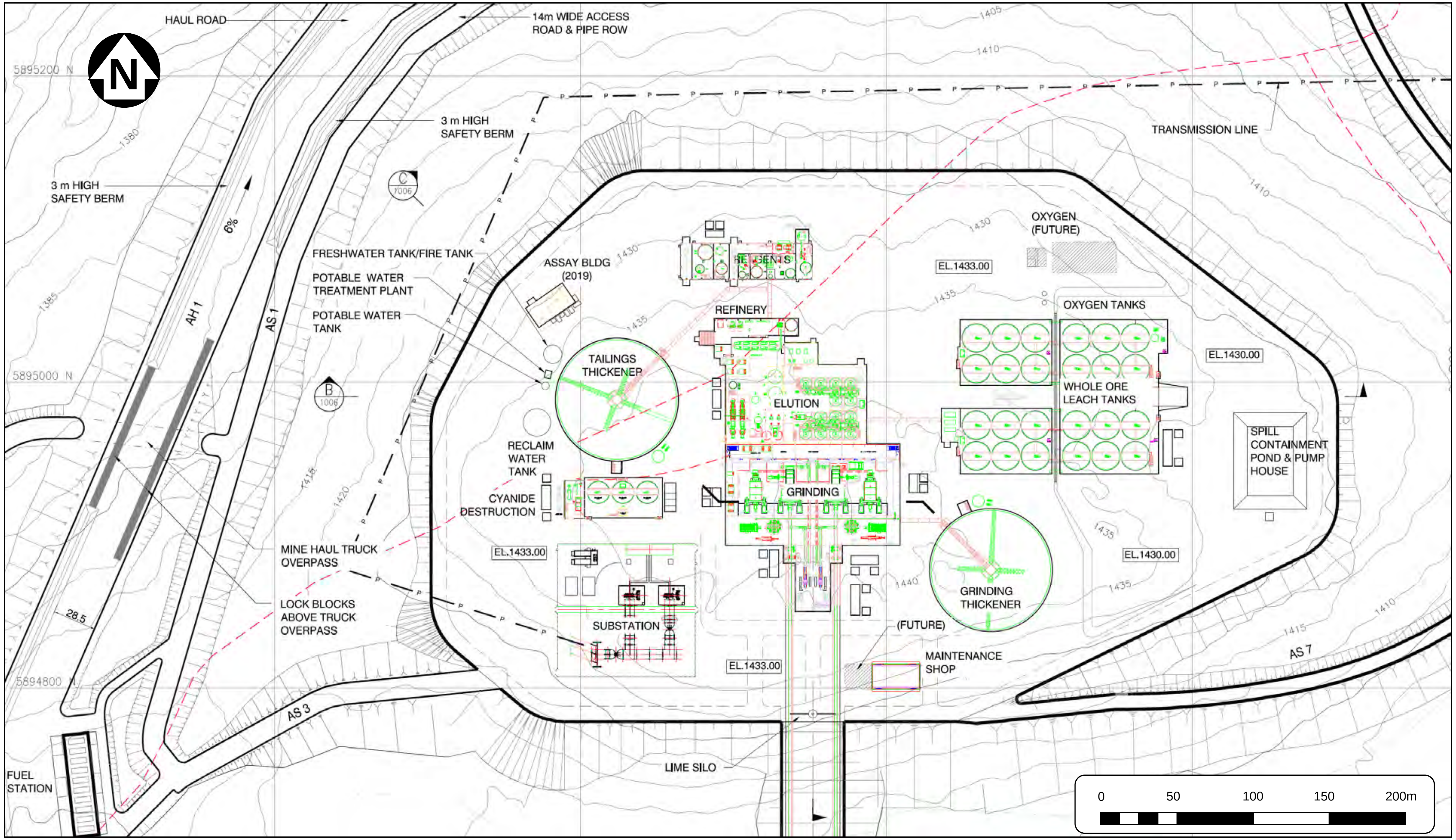


Figure 18-4: Overall Plant Site Layout



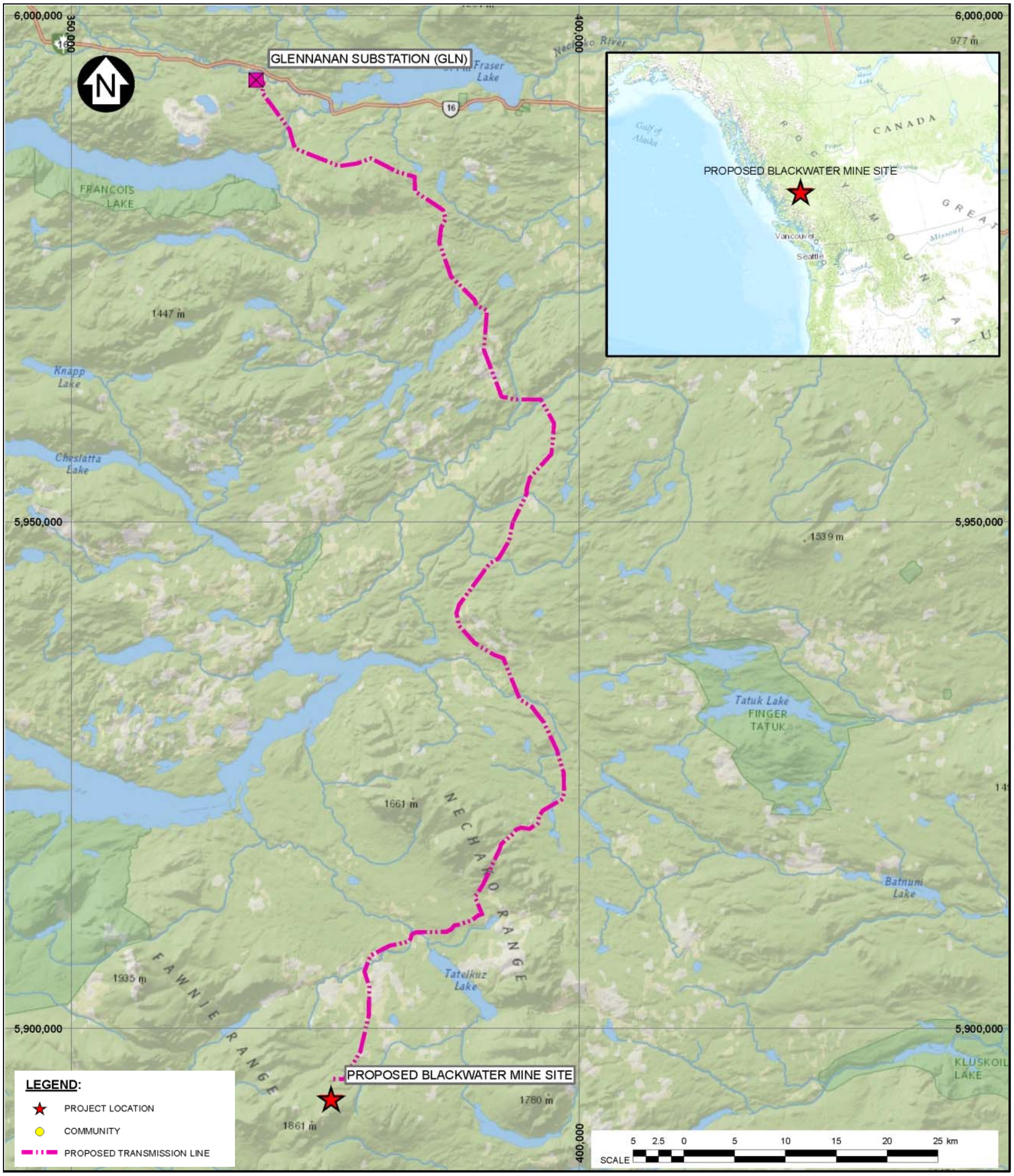
Note: Figure prepared by AMEC, 2013

Figure 18-5: Plant Facilities Layout



Note: Figure prepared by AMEC, 2013

Figure 18-6: Proposed Power Transmission Line Routing



Note: Figure prepared by AMEC, 2013

LiDAR terrain mapping, aerial photography, and site geotechnical assessments of the highest-risk terrain areas were used to complete the transmission line engineering along the preferred alignment. This information allowed the line designers to determine optimal structure locations that avoid areas of problematic soil conditions and other terrain hazards as much as possible. Geotechnical conditions were generally found to be favourable, and no significant problems are anticipated for the transmission line tower foundations.

Many sections of the proposed transmission line will be accessible by existing road infrastructure. A study of road access identified locations where new road access will be required for construction and maintenance during operations.

The incoming transmission line will terminate at the site main substation adjacent to the main process facilities. The substation will have incoming circuit breakers, motorized isolating disconnect switches, power transformers, switchgear, and protective equipment for the transformation of power from the transmission voltage level of 230 kV to the site distribution / utilization level of 25 kV. The site protection scheme will interface with BC Hydro using “Point of Wave” control and load shedding as required and as identified in the BC Hydro System Impact Study.

The anticipated electrical load for the Blackwater site is as follows:

- Peak load 107 MW
- Average load 98.4 MW
- Power factor 98%

The main substation will be adjacent to the mill grinding building, where the largest electrical loads are located, to minimize cabling costs and losses. The main substation will consist of two power transformers. The transformer secondaries will be connected to the primary distribution centre (PDC) for power distribution around the site. Power will be distributed to the mine facilities at 25 kV, three-phase, 60 Hz through radial feeders originating at the PDC and routed around the site in cable and tray and on overhead power lines.

The primary power supply to the open pit will be a single 25 kV feed pole line running from the PDC at the main substation. Portable substations will transform the power to 7.2 kV for the mine shovels and drills. The system will initially consist of two portable substations for two electric shovels and two electric drills, and has been sized to allow for the installation of a third portable substation in the future. Additional portable substations, also powered from the mine 25 kV pole line, will be used for mine dewatering operations.

Emergency power will be available from a standby power station sized to provide power to the process and ancillary electrical equipment in the event of a utility power failure. The plant will consist of a minimum of two modular gensets rated at a nominal 3.0 MW. The temporary construction power generation equipment will be used as the source of backup power supply for the permanent camp.

18.12 Fuel

Fuel will be delivered by tanker truck to the main storage tanks at the mine site. The tanks are designed to hold a total volume of approximately 1 million litres (ML) of fuel to meet surface and mining requirements for a period of seven days in the event of weather-related delays in fuel delivery.

At the main fuel storage area, fuel will be dispensed directly from the tanks to haul trucks and fuel transport trucks by means of high-flow (Wiggins-type) nozzles and receivers. The fuel transfer trucks will deliver fuel to mobile equipment and to various other smaller tanks at the mill site. This truck delivery system was selected over hard piping between tanks.

The ten diesel tanks will be contained within a bunded area lined with HDPE to provide secondary containment.

Low sulphur diesel will be used as long as it is available. Standard temperature diesel fuel will be used at the mine site during the summer months, and standard winter grade diesel will be used during the winter months because the winter design low temperatures are above the clouding temperature of the winter grade fuel.

The selection of diesel-burning equipment will depend on actual product specifications from the fuel supplier/refinery.

18.13 Water Supply

18.13.1 Fresh Water Supply

Knight Piésold conducted a trade-off study to identify potential long-term freshwater water sources for the Project. The study found that Tatelkuz Lake was the most practical option, had the lowest environmental impact, and also had the lowest risk with regard to security of water supply. Tatelkuz Lake is northeast of the mine site area and is fed by a watershed of approximately 395 km^2.

The selected water intake location on Tatelkuz Lake was based primarily on preference by local stakeholders because of the presence of a pre-existing forestry clear-cut that approaches the lake. The planned pipeline route from Tatelkuz Lake to the fresh water reservoir follows existing forestry roads to the maximum practical extent; the route from the fresh water reservoir to the plant site and TSF Site D will be along a pipeline service corridor adjacent to the mine access road.

The fresh water supply system is designed to provide:

- Continuous fresh water supply from Tatelkuz Lake for plant fresh water needs, and to mitigate flow reductions in lower Davidson Creek
- Occasional water supply from Tatelkuz Lake to supplement requirements for processing or to saturate PAG waste rock within the TSF, if required.

Five design flow rate options were examined for the fresh water supply system. The assessment considered storage requirements for normal operation of the system, surplus pumping capacity, and pipe size requirements. A design flow rate of 2,400 m^3/h was selected to minimize the required storage capacity for normal operation of the system. Standard pipe sizes of 610 mm steel or 750 mm HDPE can accommodate this flow rate.

Storage is needed as a contingency in the event of system repair or maintenance. Minimum surplus capacity has been sized to ensure the system has additional pumping capacity and availability to provide an additional 3 Mm^3 of water per year as a contingency against a series of extremely dry years.

The design flow rate of the system is driven largely driven by the in-stream flow needs (IFN) for fish in lower Davidson Creek. IFN will be supplied to Davidson Creek from the preproduction period, when Davidson Creek is cut off at TSF Site D, through the operational life of the Project and closure period until post-closure, when the open pit and TSF will be discharging water of suitable quality for release to Davidson Creek.

Water intake will be via a wet-well, which will be a permanent two-level concrete structure constructed on the shore line of Tatelkuz Lake. The structure will have a steel superstructure, an installed crane, and removable roof hatches for pump installation, removal, and maintenance. The pipeline to the mine site will be a combination of 610 mm diameter standard steel pipe, to be used for the initial high-pressure sections closest to each booster pump station, and 710 mm diameter HDPE DR17 pipe for the rest. The section of pipeline from the lake to the fresh water reservoir will be approximately 13.6 km long with a total static head of 243 m. One booster pump station (booster station 1) will be required to reach the reservoir.

18.13.2 Fresh Water Reservoir

The fresh water reservoir will be an in-creek water body downstream of the final seepage collection point at the ECD. The reservoir will be created by constructing a dam approximately 14 m high across Davidson Creek. The embankment is designed as a water-retaining structure and will have a low-permeability core with appropriate filter zones and random fill shell zones. The reservoir will have a storage capacity of about 400,000 m^3 of fresh water. The size of the reservoir was based on an estimate of contingency storage required for a reasonable range of possible malfunctions in the fresh water supply system.

A spillway will route storm flows through the reservoir and around the dam. The spillway is designed to manage the 1-in-200 year, 24 hour storm in accordance with Dam Safety Guidelines (CDA, 2007) for a significant-consequence dam.

The reservoir slopes will be covered with riprap or otherwise protected to prevent sediment-laden water from being released downstream. Appropriate sediment control BMPs will be implemented on natural drainages entering the reservoir to ensure the success of the plan.

Water will be discharged from the reservoir to Davidson Creek through two pipelines near the downstream toe of the dam: a 150 mm diameter pipe for the IFN flows and a concrete-enclosed 610 mm diameter steel pipe for flushing flows (channel maintenance). The discharge outlet will be equipped with a fish screen and will discharge to a riprap-lined outfall channel for energy dissipation and to assist with aeration to raise dissolved oxygen levels in the discharged water.

The water release conditions will be controlled by a temperature and flow control system (TFCS) consisting of temperature and flow measurement devices and associated control logic feedback loops on the discharge pipeline. A reservoir bypass line will connect directly to the water supply pipeline to allow for direct discharge of the required IFN during reservoir maintenance. It can also be used to provide cooler water as required for fisheries in Davidson Creek.

18.13.3 Fire and Potable Water Supply and Distribution

Two wells approximately 1 km east of the camp area will be the source of water supply for the temporary and operations camps. A test well in the area indicated that a single well would have the potential for a flow rate of more than 16 m^3/h, which meets the estimated average demand for the camps at their peak occupancy during construction. With both wells operating, the system will be able to refill the fire water tank within eight hours, in accordance with fire water requirements.

Water from the wells will be treated in modular portable water treatment plants (PWTPs) then pumped to a 560 m^3 fire / fresh water storage tank at the camp site. This tank will provide more than 24 hours of storage during construction and more than four days of storage during operations. Potable water requirements for the existing and expanded exploration camp will increase as the camp is expanded from 250 to 426 beds. A 150-person PWTP unit will therefore be procured to augment the existing systems. At the end of construction, the PWTP will be relocated to the process plant area to provide potable water for the mine dry and domestic use.

The fire / fresh water storage tank for the temporary and operations camps will have a reserve supply of 170 m^3 of fire protection water. This capacity will be capable of providing 114 m^3/h (500 US gpm) for 1.5 hours, in accordance with NFPA requirements for ordinary hazard occupancies.

18.14 Domestic Waste Disposal

18.14.1 Sewage

The sewage treatment system for the construction and operations camps will consist of an aerated sewage lagoon and two temporary package plants, each sized for use by 500 persons. During operations, the package plants will be taken off line and all flow will be to the aerated sewage lagoon. The lagoon will provide a long-term treatment option that is robust and simple to operate. Sludge from the package plants will also be disposed of the lagoon, eliminating the need for a separate process to manage the sludge. The treated effluent will be pumped to a Rapid Infiltration Basin for final disposal to ground as per B.C. regulatory requirements.

The expanded exploration camp will make use of the existing sewage treatment system, consisting of two RBC packaged treatment plants and an infiltration field. The system will require only slight modification to accommodate the increase from 250 to 426 beds at this camp. Once the exploration camp is decommissioned, the two RBC treatment plants will be relocated to the mill area to service the plant, administration, and truck shop buildings.

Sewage from the various sources will be collected in sewage lift stations and pumped to the treatment plants via buried force mains covered with insulation boards for freeze protection, and insulated and heat-traced as necessary. The treated effluent will be discharged to the tailings pond for final disposal.

18.14.2 Recyclable Materials

Recyclable materials such as scrap metal, unusable machinery, aluminum cans, drained oil filters, and antifreeze will be segregated at the point of generation to avoid becoming mixed with other wastes. These materials will be collected and stored in a centralized location where they will be compacted, packaged, and palletized for shipping to recycling facilities in B.C. or further afield if necessary.

18.14.3 General Refuse

Depending on the type of material, general refuse (i.e., pallets, cardboard, non-recyclable containers, construction waste, putrescible and non-putrescible refuse) will either be placed directly in the solid waste facility (SWF) or incinerated and then placed in the SWF. The SWF would be co-located with the overburden portion of the West waste rock dump for ultimate burial.

18.14.4 Hazardous Waste Management

Hazardous waste will be safely stored in a curbed and secured area until it can be transported off site for disposal to a permitted hazardous waste facility.

18.14.5 Universal Waste

Universal waste is a subset of hazardous waste subject to a less-stringent regulation. Universal wastes include batteries (Ni-Cad, mercury, lithium), thermostats, discarded mercury-containing equipment (instruments, barometers, manometers, temperature and pressure gauges, mercury switches), and lamps (fluorescent tubes, incandescent light bulbs, mercury vapour lamps). Universal waste will be collected and managed in designated areas. The materials will be shipped off site to permitted recycling facilities in B.C. or elsewhere as necessary. Universal waste will be accumulated on site for up to a year.

18.15 Security

The proposed Blackwater mine site will employ a number of security systems to ensure the security of personnel, materials, and product. Wherever possible security systems will be automated or computerized to reduce requirements for security staff. Primary access to site will be regulated by swipe cards, which will be issued to all employees. This system will control access to highly sensitive areas on site, such as the process control server room. Visual security will be provided by a network of IP-enabled closed-circuit television (CCTV) cameras installed throughout the site. Camera imagery will be fed to the central security control room and the mill process plant control room. Security staff will be provided with radios for wireless communication and may be provided with tablets for access to the CCTV camera system.

18.16 Communications

Telus currently provides telecommunications service from an existing microwave tower at the Blackwater site. The bandwidth is 150 Mbps, which will be expanded to 250 Mbps for Project construction and normal operations. At present, a 3G (third generation) wireless communication system with full cellular or data service is available on site.

Data service will be brought to site at a later date via a permanent fibre optic cable running along the power line. The in-plant communications system will then transition to the fibre optic cable as the primary communication connection; the microwave system will be retained for secondary backup.

The site-wide communications system will be based on a fibre optic backbone connected to the following systems:

- IT network (VoIP telephone and local area network)
- Control system network
- Security network
- CCTV network
- Fire alarm network.

18.17 Comments on Section 18

The QPs note:

- The overall Project facilities and major infrastructure cover the mine site area, TSF, camp site, airstrip, main access road, and water supply system from Tatelkuz Lake
- Project infrastructure has been designed to have a minimal footprint. The TSF and waste rock dumps are located near the open pits and within the same or adjacent sub-drainages
- The Blackwater site will be accessed via the Kluskus Forest Service Road, which connects to Provincial Highway 16 near Vanderhoof. In addition, a new 16 km access road will be constructed from 124 km of the FSR to the plant site
- An airstrip will be built for use during the construction phase of the Project to increase accessibility and reduce travel time to the Project site

- Power will be supplied to the Blackwater site by connection to the BC Hydro grid. The line follows existing resource roads and other previously disturbed areas as much as practical
- A fibre optic cable will be installed along with the main transmission line to provide high bandwidth telecommunications access to the site
- Fresh water for the Project will be sourced from Tatelkuz Lake. The fresh water will be used to mitigate flow reductions in Davidson Creek downstream of the TSF and for Project operations as required
- All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure
- Based on an extensive geochemical evaluation, some of the waste rock and the tailings are classified as PAG and/or ML. A schedule has been developed to place the different classes of material either within the TSF or in the waste dumps, depending on the characterisation of the material. Some material can be used for construction purposes
- The two waste dumps have a combined capacity of 137 Mt. The TSF has a storage capacity of 453 Mm^3 of tailings and waste rock material

19.0 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

No formal marketing studies have been completed.

New Gold currently has operative refining agreements with two firms for refining of gold and silver doré produced from New Gold's Mesquite Mine in Nevada. For this operation, New Gold's gold production is sold on the spot market, by marketing experts retained by or on behalf of New Gold. The terms contained within the sales contracts are typical of, and consistent with, standard industry practice, and are similar to contracts for the supply of doré elsewhere in the world.

New Gold is likely to market any future production from Blackwater in the same manner as that currently employed at Mesquite.

Gold can be readily sold on numerous markets throughout the world and it is not difficult to ascertain its market price at any particular time. Since there are a large number of available gold purchasers, New Gold would not be dependent upon the sale of gold to any one customer. Gold could be sold to various gold bullion dealers or smelters on a competitive basis at spot prices.

19.2 Commodity Price Projections

Commodity pricing is based on base case metal prices and exchange rates consistent with current consensus estimates.

19.3 Contracts

New Gold expects that terms contained within any sales contract that could be entered into would be typical of, and consistent with, standard industry practices, and be similar to contracts for the supply of gold elsewhere in the world.

19.4 Comments on Section 19

In the opinion of the QPs, New Gold will be able to market gold produced from the Project. Sales contracts that could be negotiated would be expected to be within industry norms. However, the majority of production would be expected to be spot marketed.

20.0 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

20.1 Baseline Studies

Environmental and social baseline study areas were defined to characterize the current conditions in the areas potentially affected by Project components or activities. Results of work completed to date are included as summaries in the following sub-sections.

20.1.1 Atmospheric Environment

The Project is located in a remote area with minimal existing anthropogenic emission sources. As such, there are no significant sources of air and noise emissions in the Project area. Air quality background concentrations are low compared to ambient air quality objectives.

The estimated and measured baseline noise levels or equivalent sound pressure levels (SPL) are low, at about 30 decibels (dBA), with almost no difference between daytime (30 dBA) and night-time (29 dBA). The average equivalent SPL at the proposed pit area is also low, at 31 dBA.

20.1.2 Terrestrial Environment

The Project area is characterized by mountainous topography at the mine site and gentle rolling terrain along the linear components such as the transmission line.

Morainal parent materials are the most commonly mapped sediments within the Project study area. This material is variable in thickness, ranging from a few centimetres to over several metres, as determined from on-site investigations. The dominant soil type in terms of reclamation suitability throughout the area is rated as Fair. This rating applies to soils derived from both morainal and glaciofluvial sediments. Terrain stability ratings indicate that the most of the area to be used for the Project is stable in terms of slope stability and accelerated erosion, while potentially unstable or unstable slopes occur locally without compromising Project facilities.

Whitebark pine (*Pinus albicaulis*), a species listed under the *Species at Risk Act* (*SARA),* is found at higher elevation at the proposed mine site. This species will be affected by the development of the open pit and the waste rock dumps. A management plan will be developed to mitigate the Project effects, and the closure and reclamation plan will incorporate measures to restore whitebark pine in the mine site area.

Wildlife species such as caribou, moose, and grizzly bears known to be present in the Project area are representative of the boreal forest. An ungulate winter range (U-7-012) designated for caribou lies immediately west of the study area. The surveys conducted around the mine site found that wildlife and winter wildlife habitat values are moderate and that the area does not include any high-value or important wintering habitat for ungulates.

Nineteen wildlife species of conservation concern were observed in the study area. A management plan will be implemented to mitigate the potential effects on these species, and the closure and reclamation plan will include measures to rehabilitate wildlife habitat.

The transmission line crosses the Stellako River Wildlife Management Area. The wildlife management plan will address potential effects in this area, and the transmission line will be aligned as close to the existing BC Hydro right-of-way as possible.

20.1.3 Aquatic Environment

The surface water quality baseline program included 25 sites, although some were discontinued after changes to the Project design and consultation with agencies. The program started in March 2011. Streams were sampled monthly and lakes quarterly; during the spring freshet, sampling was conducted weekly.

Surface water quality is typical of interior streams in B.C. The water is low in nutrients with generally low or undetectable trace metals. At some of the sampling sites, naturally occurring metal concentrations of such parameters as arsenic, cadmium, copper, iron, mercury, and zinc exceeded water quality objectives, as often seen in B.C. streams.

Sediment samples were collected during summer low flows at the water quality sampling sites. Arsenic, iron, and manganese exceeded B.C. sediment guidelines at some of the sites, but the results were not atypical for streams in mineralized areas of B.C., where sediment guidelines are often exceeded naturally. Healthy aquatic populations were found in all area streams, and thus exceedances of guidelines do not indicate natural impairment of aquatic ecosystems.

Groundwater quality was monitored at 13 well sites. With the exception of naturally occurring aluminum, lead, arsenic, iron, and manganese, the results were typical of groundwater quality meeting applicable guidelines.

As defined by the provincial conservation data center, there are no Red-listed (endangered or threatened) wetlands that are affected by the mine. Provincially Blue-listed (special concern) wetland ecosystems were found to occupy approximately 39 ha (0.9%). Wetland management and compensation plans have been developed to manage the potential effects of the Project on these Blue-listed wetlands ecosystems.

No anadromous fish are present in the area of the mine site, and the Project will not place mine waste in any lakes.

Rainbow trout and kokanee are the most common fish species found in streams potentially affected by the Project. Rainbow trout spawn in most of the streams in the spring, while kokanee spawn in Davidson Creek, Creek 661, and other streams in the surrounding area from mid-July to early September. Eggs incubate over winter. Juvenile kokanee migrate downstream to Tatelkuz Lake immediately after they emerge from incubation gravel in the spring to feed, grow, and mature. Other species identified (in much lower numbers) in the area include mountain whitefish, northern pike minnow, lake chub, longnose sucker, brassy minnow, burbot, white sucker, and slimy sculpin.

Streams within the mine site area are used seasonally, mainly by juvenile rainbow trout. A lack of overwintering habitat in the streams limits seasonal fish distribution, although most adult fish reside in lakes year-round because the lakes provide suitable winter habitat.

Because the mine site will affect habitat currently being used by fish, a fish habitat mitigation and compensation plan has been developed for the Project.

20.1.4 Socio-Economic

The Project and its components extend through areas accessed by various land users, communities, and Aboriginal groups. The Project has been designed to avoid private land and, through consultation with stakeholders, the Project footprint has been modified to minimize effects on lands covered by range, guide outfitter, and trapping tenures. The Project area does not cross or overlap any federal or provincial parks or any designated Agricultural Land Reserve (ALR).

The transmission line and the Kluskus FSR extend from the mine site to the B.C. Highway 16 (Yellowhead Highway) corridor that connects more urbanized areas, including Vanderhoof and Fraser Lake. Vanderhoof is a district municipality with a population of approximately 4,500 residents. The major industries are lumber and agriculture. Community facilities include schools, a health centre, a Royal Canadian

Mounted Police (RCMP) detachment, and a fire hall. Vanderhoof also provides regional services for smaller rural communities nearby.

The village of Fraser Lake between Burns Lake and Vanderhoof along the Yellowhead Highway has approximately 1,300 residents. Fraser Lake is close to the operating Endako mine, which is major source of employment. The other main industry is lumber. Community services include a school, health centre, RCMP detachment, and fire hall.

Like much of British Columbia, the Project site and surrounds include a number of Aboriginal communities on what are deemed to be "unsettled areas"—meaning treaties were not signed historically between the Crown (federal and/or provincial governments) and First Nations. In the absence of treaties, Aboriginal groups have asserted Aboriginal rights and title over "Traditional Territories," which are often large tracts of Crown land. In the Project area, many of these asserted traditional territories overlap with other claims. For instance:

- The mine site is located within the Traditional Territories of Ulkatcho First Nation and Lhoosk'uz Dene Nation.
- Off-site infrastructure such as the Kluskus FSR and the transmission line crosses the Traditional Territory of Nadleh Whut'en First Nation, Nazko First Nation, Stellat'en First Nation, and Saik'uz First Nation.

The nature of Aboriginal rights and precise locations of Aboriginal titles in the region remain undefined.

20.1.5 Heritage

Most of the lands within the Project development area have low to moderate potential for protected archaeological resources. The exception is an area in proximity to the Stellako River and terraces at lower elevation bordering Davidson Creek, where archaeological sites such as lithic scatters and cultural depressions have been identified. Other local heritage features include two historic trails, the Messue Wagon Trail and the Cheslatta Trail, and cultural heritage sites such as cambium-stripped trees, shaped trees, blazed trees, and box traps. Historical remains from 19th and 20th century mineral exploration and timber harvesting activities also have been observed.

A paleontological resource study, including a 2013 field assessment, confirmed the presence of Ashman Formation bedrock containing fossils within and immediately adjacent to the proposed transmission line right-of-way, where it traverses the hills north of Tatelkuz Lake. Most of the fossils observed in this rock unit are fragmental and/or indeterminate, due in part to their preservation in thinly bedded shale.

Potential Project effects on archaeological, historical, cultural heritage, and paleontological sites have been mitigated by minimizing or adjusting the Project footprint and implementing procedures for site protection, systematic data recovery, surveillance, and monitoring.

20.2 Potential Environmental Issues

Potential effects of the Project and their planned mitigation include:

- Early identification and avoidance of key sensitive areas in the region of the Project:
  - the Blackwater River drainage, which the mine site will avoid
  - ungulate winter range (habitat) for the Tweedsmuir-Entiako Caribou herd, which the mine site will avoid
  - growth areas for Whitebark pine (*Pinus albicaulis*), which is a Blue-listed species in B.C. and was recently added to Schedule 1 of the federal *Species at Risk Act* (*SARA*) as endangered.
- Minimizing the overall footprint of the Project and ensuring that the tailings storage facilities (TSF) and waste rock dumps are located near the open pits and within the same or adjacent sub-drainages. Relocating the existing exploration access road and maximizing use of the new mine access road to the site will also minimize the overall footprint.
- Chance-find protocols will be implemented, and employees will be trained on archaeological, paleontological, and historic site preservation.
- Wildlife management, vegetation, and invasive species best management practices (BMPs) will be adopted for the construction and operation phases of the Project.
- Salvaged soils will be used at closure to rehabilitate disturbed habitat for use by wildlife and vegetation, focusing on endangered species such as the whitebark pine
- A key environmental consideration is the presence of fish-bearing water bodies in the Project area, particularly in the lower reaches of Davidson Creek, where kokanee spawning habitat has been identified. The Project footprint avoids those lower reaches and proposes to mitigate for flow reductions in Davidson Creek immediately downstream of the mine site by pumping water from Tatelkuz Lake in

sufficient volumes to avoid a harmful alteration destruction or disruption (HADD) of fish habitat.

- Flow reductions will be mitigated in downstream Davidson Creek for fish habitat, and a comprehensive fish habitat compensation plan has been developed for lost habitat within the Project footprint. The cost of fish habitat compensation has been estimated and is included in the capital cost estimate.
- Flow mitigation measures involve the pumping of fresh water from Tatelkuz Lake to Davidson Creek to satisfy in-stream flow requirements to sustain the different life functions of rainbow trout and Kokanee. A combination of fish habitat compensation projects is currently being proposed, including the enlargement of existing lakes, creating ponds, and rehabilitating disturbed fish habitat.
- Potential effects on air quality will be mitigated mainly by applying standard dust-suppression water along haulage and other roads, establishing speed limits, and providing diesel equipment with low-sulphur fuel and appropriate maintenance. Noise will be mitigated by placing stationary equipment in sheltered and enclosed locations.
- The transmission line alignment will avoid or mitigate effects on existing private property, range, guide outfitters, and trapping tenures.
- Policies will be implemented to prevent hunting and fishing by Project employees while working for the Project.
- The construction and operations camps will reduce pressures on local community infrastructure and services.
- Road traffic controls will be used to manage traffic and minimize the risk of accidents along the transportation corridor.
- The Project will support local business and contractors in benefiting from economic opportunities.
- The Project will monitor community well-being and assist employees with counselling services to prevent negative effects related to increased income and changes from traditional land use activities.
- A health and safety plan will be developed to protect the health of mine site workers, and a monitoring plan will be implemented to measure potential contamination of country foods by exposure of vegetation, wildlife, and fish to heavy metals.

20.3 Closure Plan

The mine design includes a robust closure plan with simplified water management requirements, reflecting the compact Project layout and integrated waste management strategy. The plan is based on proven practices and does not depend on active long-term treatment. All Project components will be decommissioned and reclaimed according to best industry practices and provincial and federal regulations. Proposed end land use objectives for mine closure are wildlife, recreation, and return of the land to traditional Aboriginal use.

PAG waste rock and the TSF beaches will be covered with 30 cm of overburden to isolate the tailings porewater from the pond supernatant and to facilitate reclamation. On closure, the upland TSF beaches will be vegetated. Emergent and bog wetlands will be established in submerged and saturated areas of the tailings. The TSF Site C will be partially reclaimed in Years 4 and 5, and Site D and the rest of Site C will be reclaimed after the completion of mill operations in Year 17.

Once mill operations cease, TSF supernatant from Cell D will be pumped to the mined-out pit until the pit is full while maintaining adequate water cover over the tailings. Ultimately the pit lake will overflow to the TSF and thence to Davidson Creek. Discharged water is predicted to meet effluent quality criteria and, after mixing, receiving water quality criteria. Contingency plans for post-closure water management include wetland and pit lake treatment and continuing to pump TSF seepage to the TSF or pit lake. Monitoring will continue post-closure.

About 63 ha of the West dump will be progressively covered with a minimum of 30 cm of overburden and vegetated. The remaining 105 ha of the West dump and the 72 ha East dump will be reclaimed at closure. Drainage from the reclaimed dumps will continue to flow by gravity to the TSF post-closure. If the water quality is acceptable, drainage from the East dump may be discharged to Creek 661 to re-establish pre-mining drainage patterns.

The compacted till pad for the low-grade stockpile may become contaminated with acidic drainage and metals during operations. This material will be tested and excavated as required and placed into the TSF or open pit.

All buildings and facilities will be removed except concrete foundations, which will be broken up and covered with overburden and vegetated. Other than those required for long-term monitoring of the site, roads will be ripped and reclaimed.

Knight Piésold prepared the closure cost estimate using a template provided by New Gold and based on the closure plan concepts developed by Knight Piésold, New Gold, and AMEC. The basis of estimate is largely founded on neat line quantities for the proposed closure earthworks and water management activities, considering mine construction and reclamation costs for other, similar projects in Northern B.C. and Knight Piésold's experience in the design and construction of waste and water management facilities.

The closure and reclamation costs are annual costs incurred during the mine life for progressive reclamation, and continue out to Year 50. These costs, discounted to the last year of operations, are approximately $86 million.

A salvage value of $78.4 million has been estimated for Project assets upon closure. Mining equipment, major process equipment, and capital spares are assumed to be sold on an "as-is where-is" basis. The value for scrap metal was based on the estimated reclaimed quantities to be sold to the scrap metals market.

20.4 Permitting

A large number of federal or provincial permits are required for mine construction and subsequent operations. The process of obtaining the provincial permits is partially coordinated by the Ministry of Forests, Lands and Natural Resource Operations through a Mine Development Review Committee. Some of the provincial permits have legislated timelines, while others do not. The federal permits and authorizations do not have legislated timelines, and some permits originating from federal agencies may not be issued within the review period set out in the B.C. legislation.

20.4.1 Exploration Activities

New Gold holds permits for the exploration activities currently being conducted on the Project.

20.4.2 Environmental Assessment Process

The Blackwater Project will be subject to an environmental assessment under the British Columbia *Environmental Assessment Act* (BCEAA) and the *Canadian Environmental Assessment Act (2012)* (2012 CEAA). The assessment process will be coordinated through the BC Environmental Assessment Office (BC EAO) with input and direction on federal matters by the Canadian Environmental Assessment Agency (CEAA).

Environmental and social baseline studies started in 2011 and continued through 2012 and 2013 to provide a comprehensive understanding of the current conditions in the Project area.

The environmental assessment process for the Project officially started in October 2012 with the acceptance of the Project Description by both the BC EAO (*BCEAA*) and the CEAA. The sequence of decisions describing the current status of the Project environmental assessment process is as follows:

- In November 2012, the CEAA issued a letter wherein New Gold was notified that the Project meets the definition of a “designated project” under the 2012 CEAA, and the submitted Project Description meets the requirements of the *Prescribed Information for a Description of a Designated Project Regulations* under the 2012 CEAA.
- In November 2012, BC EAO issued an Order under Section 10 of the BCEAA indicating that the Project requires an Environmental Assessment Certificate. Following the issuance of this order, preliminary versions of the draft Application Information Requirements (AIR) were presented to BC EAO for review.
- On February 19, 2013, the CEAA released Final EIS Guidelines for the Project and in its letter of transmittal confirmed that the Project would not be referred to a review panel and that the CEAA would conduct the assessment.
- On February 20, 2013, the CEAA announced that participant funding was available for the review of the Project. On May 16, 2013, the CEAA announced the allocation of $198,637 in federal participant funding to eight Aboriginal groups.
- On April 30, 2013, the Working Group, composed of provincial and federal agencies, local government representatives, and Aboriginal groups, met to introduce the Project and review the Version D of the draft AIR.
- On July 9, 2013, the BC EAO issued the Section 11 Order. Schedule A to the Order under Section 11 of the BCEAA provides New Gold guidance and direction with respect to the scope, procedures, and methods for the environmental assessment of the Project.
- On August 28, 2013, a draft version E of the AIR addressing comments received from the Working Group was sent to BC EAO and the CEAA for review.
- On October 9, 2013 a 30-day Public Comment Period commenced on draft version F of the AIR. BC EAO approval of the AIR is expected during Q4 2013/Q1 2014.

20.4.3 Key Provincial Permits

The provincial government will follow the “One P” process for the Project involving combined applications and reviews for all required provincial permits. Two key provincial permits are required to commence construction:

- *Environmental Management Act* (*EMA*) permit for any discharges from the Project to surface waters, including seepage from the TSF during operations and sediment control pond discharges during construction
- *Mines Act* permit to allow construction and operation of the Project.

The One P process envisages that many of the permits required to construct and operate the Project mine, transmission line, and other associated infrastructure will be applied for at approximately the same time, either in a "bundle" or separately at the discretion of the responsible agency.

The referral process will be managed by the British Columbia Ministry of Forests, Lands and Natural Resource Operations (BC MFLNRO) (the coordinating provincial ministry), which will refer all provincial permit applications to other provincial and local governments, Aboriginal groups, and interested stakeholders.

New Gold will ensure all required permitting level details are included with applications.

20.5 Key Federal Permits

Federal authorizations will be required under the federal *Fisheries Act*:

- Schedule 2 amendment under the *Fisheries Act* requires an Order in Council and applies to placing mine waste into a natural water body that is frequented by fish.
- Section 35(2) authorization under the *Fisheries Act* for a Harmful Alteration, Disruption, or Destruction (HADD) of fish habitat. A Schedule 2 amendment is required before a Section 35(2) authorization is issued for a tailings impoundment; both require fish habitat compensation plans accepted by Fisheries and Oceans Canada. A new Fisheries Act is expected in 2014.

A Navigable Waters Protection Act exemption under Section 23 may be required pending the new *Navigation Protection Act* coming into force in late 2014. An exemption under the *Navigable Waters Protection Act* is not required to commence construction in navigable waters but is required for deposition of tailings in such waters.

The federal Major Projects Management Office (MPMO) coordinates government reviews of project applications for federal approval under the *CEAA 2012* and for federal permits (key federal permits listed above), and recommends applications for federal permits and authorizations be submitted with the EIS. Once the federal Minister of the Environment approves the project in principle, the Agency will transfer coordination of federal permitting to ministries responsible for administering federal acts and regulations. For the Project, federal ministries will include Fisheries and

Oceans Canada (DFO), Environment Canada, Transport Canada, and Natural Resources Canada. The MPMO will continue to oversee the process and ensure federal responsibilities for timely review and Aboriginal consultation are fulfilled.

20.6 Additional Major Permits for the Project

Additional major permits for the Project not listed above include:

- Provincial permits normally bundled:
  - Provincial licences and authorizations under the *Water Act*, e.g., withdrawal of water from Tatelkuz Lake and "use" (as defined in the Act) of Davidson Creek water, including the TSF
  - Air discharge permit(s) for any point source emissions (under EMA)
  - Refuse permit (under EMA)
  - Municipal sewage registration for an approximate 1,500-person construction camp
  - Licence of Occupation or other Crown tenure (for lands outside the mining lease)
  - Occupant Licence to Cut/Special Use Permit(s) to clear merchantable timber
  - Free Use Permit to clear non-merchantable timber;
- Mining lease
- *Mining Right of Way Act* Permit
- *Heritage Conservation Act* permit(s)
- Explosives licence(s) from Natural Resources Canada (obtained by the explosives contractor).

Key agencies include:

- Ministry of Forests, Lands and Natural Resource Operations
- Ministry of Energy and Mines
- Ministry of Environment
- Fisheries and Oceans Canada
- Environment Canada
- Natural Resources Canada
- Transport Canada.

20.6.1 Permitting Schedule

The preparation of the permitting applications will be completed in parallel with the Application/EIS review process such that the critical path provincial bundled permit applications, DFO authorizations, and *Navigable Waters* exemption (if required) will be filed immediately following provincial and federal ministerial decisions.

New Gold expects that key permits required for initial construction of the proposed Project will be obtained by early 2015.

20.7 Considerations of Social and Community Impacts

New Gold is committed to building strong and responsive relationships that support the company's commitment to leaving a positive legacy in the communities and regions in which it operates. New Gold began engagement and consultation with the public and Aboriginal groups on acquisition of the property in 2011. Since that time, New Gold continues to work closely with the public and Aboriginal groups to identify issues, perspectives, interests, and concerns relative to the Project.

New Gold is an active member of the local community and has a Project office in Vanderhoof that provides residents with access to current information about the Project. Through discussion with BC EAO, public and stakeholder consultation work is primarily focused on the Vanderhoof and Fraser Lake communities.

Key land users (e.g., ranchers, trappers, and guide outfitters) in closest proximity to the Project are also being consulted. The Project footprint has been modified to minimize effects on lands covered by range, guide outfitter, and trapping tenures. The Project has also been designed to avoid private land.

Consultation activities have included open houses, meetings, site tours, workshops, and focus groups. Consultations to date indicate there is widespread community support for the development of the Project.

The Project will create approximately 595 permanent jobs. The construction work force will be roughly 1,200 on average, peaking at 1,500. New Gold is committed to maximizing local employment and contracting opportunities. New Gold will work collaboratively with community partners, use existing training programs to prepare local workers, and establish new programs for specific training where necessary. It is expected that local communities will benefit directly and indirectly from the Project.

20.7.1 First Nations

Although the duty to consult lies with Crown agencies, the BC EAO (pursuant to the Section 11 Order) assigns certain responsibilities for undertaking procedural aspects of the Crown's duty to consult with potentially affected Aboriginal groups. The BC EAO formally delegated aspects of its consultation responsibilities to New Gold through a Section 11 Order distributed on July 9, 2013. The Section 11 Order directs New Gold to consult the following five Aboriginal groups (Schedule B Aboriginal Groups):

- Lhoosk'uz Dene Nation
- Nadleh Whut'en First Nation
- Saik'uz First Nation
- Stellat'en First Nation
- Ulkatcho First Nation.

In addition, BC EAO identified the following three Aboriginal groups as having interests in the regional area of the Project (Schedule C Aboriginal Groups):

- Nazko First Nation
- Skin Tyee Nation
- Tsilhqot'in National Government.

New Gold has initiated communications focused on early engagement and information-sharing with all the Aboriginal groups listed above. Since 2011, New Gold has held more than 100 meetings with Aboriginal groups in addition to extensive written communication via e-mails and letters. Consultation activities have included numerous presentations to Aboriginal leaders, community meetings, site tours, and one-on-one meetings.

New Gold is committed to meaningful consideration of the comments, questions, and concerns of public and Aboriginal groups. Information gathered through consultation activities is captured in a formal information management system that supports tracking and monitoring of comments and issues raised, as well as the context in which they were raised. Primary issues and comments raised by Aboriginal groups during consultation relate to potential effects on water quality, fish and fish habitat, caribou herds, and other wildlife and habitat, including the potential effects from increased traffic. Socioeconomic interests raised relate primarily to potential employment and economic opportunities.

Traditional Knowledge (TK) and Traditional Land Use (TLU) provide information about the value of environmental and cultural components to Aboriginal groups. New Gold engaged representatives of Aboriginal groups that could potentially be affected by the Project to conduct TK/TLU studies during 2012 and 2013.

New Gold has entered into Exploration Cooperation Agreements with the Ulkatcho First Nation, Lhoosk'uz Dené Nation, and Skin Tyee Nation and is actively pursuing Agreements for construction and operation of the Project with Aboriginal groups.

20.8 Comments on Section 20

The QPs note:

- New Gold conducted extensive environmental baseline studies. Two key areas requiring careful environmental management were noted:
  - Potential effects on fish due to a flow reduction in Davidson Creek downstream of the TSF; the Project footprint avoids those lower reaches and proposes to mitigate for flow reductions in Davidson Creek immediately downstream of the mine site by pumping water from Tatelkuz Lake in sufficient volumes to avoid a harmful alteration destruction or disruption (HADD) of fish habitat
  - Effects on whitebark pine (*Pinus albicaulis*); a management plan will be developed to mitigate the Project effects, and the closure and reclamation plan will incorporate measures to restore whitebark pine in the mine site area.
- The closure plan has simplified water management requirements resulting from the compact Project layout and integrated waste management strategy.
- The closure plan employs proven practices and is not dependent on long-term active treatment and monitoring.
- Proposed end land use objectives for mine closure are wildlife habitat and return of the land for traditional use by Aboriginal groups
- The environmental assessment process for the Project officially started in October 2012 with the acceptance of the Project Description by both the BC EAO (*BCEAA*) and the CEAA.
- A large number of federal or provincial permits are required for mine construction and subsequent operations. Some of the provincial permits have legislated timelines, while others do not. The federal permits and authorizations do not have legislated timelines, and some permits originating from federal agencies may not be issued within the review period set out in the B.C. legislation

- New Gold is actively consulting with First Nations, government, and other stakeholders that could potentially be affected by the Project. The intent of the consultation is to increase the mutual awareness and understanding of the Project and its potential effects, and to explore potential strategies to mitigate negative effects and enhance positive ones.

21.0 CAPITAL AND OPERATING COSTS

21.1 Capital Cost Estimates

Capital cost estimates were prepared by the following parties as part of the 2013 Feasibility Study:

- New Gold: Cost estimates for EPCM, Owner's costs, environmental costs, involvement with indirect costs and contingency
- AMEC: Design and estimates for process and site infrastructure, overall compilation of direct, indirect, contingency, escalation costs. Estimate of Knight Piésold engineering scope. Estimating and consolidating sustaining capital cost
- AMEC E&I: Fish Habitat and Compensation Plan and cost
- Valard: Material take-offs and cost estimates for the 230 kV off-site transmission line
- Knight Piésold: Design and quantities for mine stockpiles and waste for LGS, East dump, and West Dump; tailings facilities design for TSF Site C, Site D, and the Environmental Control Dam; design and prepare quantities for tailings distribution and water reclaim; design and quantities of off-site facilities including Tatelkuz lake intake and pump station; estimates for mechanical equipment; closure costs
- Norwest: Cost estimates for mine preproduction, mine equipment, and mine operating costs, excluding pit infrastructure
- Allnorth: Design and cost estimates for main access road, Kluskus FSR, airstrip, road along off-site transmission line, and road along the fresh water supply line.

21.1.1 Basis of Estimate

The capital cost estimate for the Blackwater Project was developed to provide an estimate suitable for the 2013 Feasibility Study phase, including costs to design, procure, construct, and commission the facilities. In consideration of the current state of design and procurement, this estimate falls under the AACE Class 3 Estimate classification and its accuracy is expected to be within +15%/-10% of final Project cost. Capital costs will be capitalized until commercial production, which is defined as 30 days at an average of 60% of production capacity.

The cost estimate is based on a combination of material take-off (MTO) data, design drawings, vendor quotes, manufacturers' information, and industry standards and rates. Key assumptions used in generating the estimates were:

- Diesel: CAD$1.042/L

- Electrical power: CAD$0.051/kWh
- United States dollar: USD$0.95 to CAD$1
- Euro: EUR0.73 to CAD$1
- Australian dollar: AUD$0.98 to CAD$1.

The estimate was developed in accordance with the Work Breakdown Structure (WBS) for the Project and was based on the following:

- Process flow diagrams
- Project scope of facilities
- Equipment list
- Design criteria
- General arrangement drawings and site layout plans
- Electrical single-line diagram
- Supplemental sketches where required
- Equipment specifications for major equipment
- Budget quotations from vendors for major equipment
- Budget pricing for bulk materials
- Geotechnical and hydrogeological reports
- Regional climactic data
- Engineering discipline material take-offs
- Project development schedule
- Project execution plan
- Project work breakdown structure (WBS)
- AMEC's In-house data.

In addition to quantified items, various estimate allowances are included in the capital cost estimate to take into account the state of engineering design at the time the estimate was prepared. As engineering design definition increases, more items become quantified and design allowances decrease.

Estimates for contractors' construction equipment are included in the direct costs of the Project, on each activity line item as appropriate per discipline. Allowances to cover wastage and overbuy of bulk materials during the course of construction are included within the material costs only. Estimate factors are applied to the estimate to cover quantities that are not generally quantified by engineering. They are intended to cover additional quantities purchased, moved, installed, or erected during the course of construction.

The construction indirects were estimated jointly by AMEC and New Gold based on the proposed construction execution plan and using AMEC's Indirects Model as a tool.

New Gold provided the engineering and procurement (EP) costs after an assessment of proposals for EPCM services. The estimate includes basic and detailed engineering for the overall Project, including the TSF. Engineering support post-mechanical completion is also included. New Gold also provided construction management (CM) costs after an assessment of proposals for EPCM services. The CM estimate covers the field- or site-based services required to construct the facilities within the scope described.

21.1.2 Cost Assumptions

Labour

Wage rates for construction crews were established based on data collected from ten contractors and executed contracts from a recently completed project in British Columbia. Construction labour rates developed for the early phase of construction, before the airstrip is in operation, include travel costs. Once the airstrip is completed, charter flight costs for all construction and management personnel are included in the Owner's costs and not in labour rates.

To account for less-than-ideal conditions at the Blackwater site, productivity factors were incorporated into the construction labour unit work-hours as multipliers on the base manhours. Productivity factors are not applied to earthmoving activities because any site-specific or equipment-based productivity loss is included in the unit rate.

Camp Costs

The estimate was based on using a combination of rented camp accommodation and the existing 250-bed camp on site. The estimated on-site manhours were incorporated into the construction schedule to determine monthly camp size requirements.

The costs of the construction camp were based on a quotation, adjusted to suit the final determined camp requirements.

Catering costs were based on an average of $55/person-day on site, as per an assessment of four quotations, during the main construction period from Year -2 to Q2 Year 1.

All camp operating costs in the Year -3 construction period are included in Owner's costs.

Temporary Facilities

The provision for temporary facilities includes all temporary buildings required during construction. Estimates were based on purchasing the office and building facilities. Electrical and communications services were estimated for each facility as required. Facilities were assumed to have water and sewage holding tanks. An allowance is included for CM office supplies and equipment.

Construction Support

Construction support includes materials, equipment, and personnel to maintain the site during the construction period. Estimates were based on durations from the construction schedule

Construction Utilities

Construction utilities were based on the construction schedule and temporary building load list.

Construction Services

This account includes all temporary construction support services required during construction and commissioning.

Health, Safety, Security, and Environment (HSSE)

HSSE costs were developed based on the construction schedule and AMEC in-house data.

Freight, Logistics, Taxes, and Duties

Freight costs were estimated using information provided by vendor quotations, freight forwarder quotations and weights of materials and equipment. Freight for mining equipment, pre-engineered buildings, modular buildings, and field-erected tanks were included in the direct costs.

Provincial sales tax (PST) of 7% was included in the estimate based on an assessment of the August 2013 Provincial Government Bulletin PST 111. In general, equipment used for transporting or processing ore is PST exempt. PST is generally applicable on procured bulk materials. PST was assumed to be not applicable to labour or contractor construction equipment. GST was not included in the estimate.

Vendor Representatives

A list of mechanical and electrical process equipment that requires vendor services was prepared to ensure validation of manufacturer warranties and on-site provision of expertise and supervision in construction erection, pre-commissioning testing, or plant commissioning. Travel time, airfares, lodging, and other out-of-pocket expenses were accounted for as a rate per round trip.

Start-up and Commissioning

An allowance is included for contractor support during start-up and commissioning.

First Fills

New Gold provided the budgetary cost to supply plant first fills, which includes such items as grinding media, lubricants, and reagents. First fills do not include general warehouse inventory.

Spare Parts

Capital spare parts were based on 3.5% of the mechanical and electrical equipment costs. An allowance of 1% of mechanical and electrical equipment cost was included for start-up spares and another 1% for operational spares.

21.1.3 Owner's Costs

New Gold provided Owner's costs for inclusion in the capital cost estimate. These include personnel-related costs (salaries, travelling, offices) for the Owner's team, permitting, insurances, camp costs (catering and camp maintenance) during Year -3, and safety and training for Operations personnel. Owner costs are estimated at $77.94 M (Table 21-1).

21.1.4 Contingency

AMEC performed an initial contingency analysis using a risk analysis program (@RISK) to generate a range of probable costs. Of the costs included in the analysis, the contingency was determined to be 14.8% based on the 85th percentile of the risk simulation results. The contingency was estimated to be $200.4 million, which is 11.4% of the total direct and indirects costs of the Project (Table 21-2).

21.1.5 Escalation

The capital cost estimate is based on Q3 2013 dollars and excludes escalation factors such as inflation and changes in commodity prices. For the purposes of estimation of Mineral Reserves, these costs have been brought forward to Q4 2013, with no escalation applied.

To obtain an indication of the final escalated cost of the Project from 2013 to 2017, anticipated escalation rates over this time were sourced from vendors and literature, and were applied to various cost categories within the estimate. Overall escalation was determined to be approximately $150 million, or 7.6% of the base estimate (including contingency), equivalent to 2.1% compounded annually.

21.1.6 Sustaining Capital

Sustaining capital costs are the ongoing capital expenditures required to sustain operations. The equipment replacement schedule was based on average life span, considering regular maintenance, practical experience standards for the mining industry, and vendor recommendations. Costs are expressed in Q3 2013 Canadian dollars with no allowance for escalation or interest.

The total sustaining capital cost for the Blackwater Project is estimated to be $680.71 million (Table 21-3). This cost represents approximately $1.99/t over the life of the mine.

Table 21-1: Owner's Costs

Item	Project Costs ($M)
Project Team	29.38
Environmental, Community, and Public Affairs	20.35
Safety and Security	4.73
Human Resources	4.85
Mill Operations	4.27
Information Technology (IT)	4.73
Finance and Administration	9.69
Camp Costs (Year -3)	1.15
Start–up (until Commercial Production) *	(1.21)
Total Owner Cost	**77.94**

Note: * Owner's costs also include the start-up period, including mining, process, and G&A costs as well as TSF construction and revenue during ramp-up until achieving 30 days at 60% of production capacity (Commercial Production).

Table 21-2: Contingency

Contingency Results at P_{85}	Costs Evaluated for Contingency	Total Project Costs
Estimate before Contingency ($M)	1,353.33	1,762.42
Contingency @ P_{85} ($M)	200.42	200.42
Contingency (%)	14.8	11.4
Estimate Including Contingency ($M)	-	1,962.83

Table 21-3: Summary of Sustaining Capital

		Year of Project ($M)									
WBS		**01**	**02**	**03**	**04**	**05**	**06**	**07**	**08**	**09**	**10**
1000	Mining*	66.73	33.66	16.20	12.98	10.37	30.73	6.01	4.86	5.32	2.62
2000	On-Site Infrastructure	-	34.12	-	-	-	-	-	-	-	-
3000	Processing Plant	0.37	2.27	2.08	3.33	2.92	3.22	4.82	3.28	3.37	4.03
4000	Tailings Facilities*	32.86	66.54	37.61	35.06	19.29	17.94	20.48	28.04	16.77	6.77
6000	Off-Site Facilities	-	6.84	-	-	-	-	-	-	-	-
9000	Indirects	0.86	13.15	0.82	0.78	0.40	0.38	0.46	0.64	0.38	0.15
P000	Provisions	4.19	17.36	5.42	5.49	2.59	2.84	3.37	5.10	2.57	1.39
	Total	**105.00**	**173.94**	**62.13**	**57.64**	**35.57**	**55.12**	**35.14**	**41.93**	**28.41**	**14.96**

		Year of Project ($M)								
WBS		**11**	**12**	**13**	**14**	**15**	**16**	**17**	**18**	**Total**
1000	Mining	3.90	0.99	3.21	1.92	1.37	0.84	-	-	201.70
2000	On-Site Infrastructure	-	-	-	-	-	-	-	-	34.12
3000	Processing Plant	3.89	3.67	3.02	3.97	1.67	2.17	0.26	0.25	48.58
4000	Tailings Facilities	6.24	6.95	7.91	5.62	5.49	-	-	-	313.57
6000	Off-Site Facilities	-	-	-	-	-	-	-	-	6.84
9000	Indirects	0.14	0.17	0.18	0.14	0.13	-	-	-	18.78
P000	Provisions	1.30	1.44	1.43	1.26	0.97	0.33	0.04	0.04	57.13
	Total	**15.47**	**13.22**	**15.74**	**12.91**	**9.64**	**3.34**	**0.29**	**0.29**	**680.71**

Note: * Mining sustaining capital includes haulage of pit run waste materials to the tailings facilities for construction.

21.1.7 Capital Cost Summary

The initial capital cost estimate for the facilities described in this Report is approximately $1,963 million (US$1,865 million). Total LOM sustaining capital is estimated to be $681 million (US$647 million). Reclamation and closure costs net of salvage value of $7.4 million have been applied to Year 17 and are included in the financial model.

The capital cost estimate and financial model exclude allowances for cost escalation over the Project duration.

A summary of the capital cost estimate is included as Table 21-4.

21.2 Operating Cost Estimates

The operating cost estimates were prepared by the following parties:

- New Gold: Inputs to labour costs, vendor preferences, G&A components and costs, including insurance and environmental costs
- AMEC: Estimates for reagent and steel consumables consumption and costs, spare parts utilization, labour costs (with input by New Gold), freight, and electrical power; mill staffing plans
- Norwest: Estimates for mine operating costs, excluding environmental and closure costs; mine schedule detailing the delivery of feed to the plant; mine staffing plans
- Knight Piésold: Tailings and waste management area operation including tailings pipelines, power line maintenance, and fresh water systems
- Allnorth: On-site and access road maintenance

21.2.1 Basis of Estimate

Labour costs were developed by New Gold for all direct Blackwater employees and are based on yearly compensation surveys published by PricewaterhouseCoopers. Salaries were referenced against other New Gold operations and adjusted accordingly. The final cost to the company for each position was built up from base salary plus incentives, benefits and insurance. Labour costs are presented as a yearly salary for all employees.

Consistent energy cost assumptions have been applied to mining, processing, and G&A costs.

Table 21-4: Capital Cost Estimate Summary – by Major Area

Area	Description	Cost (CAD$M)
1000	Mining*	285.93
2000	On-Site Infrastructure	166.31
3000	Processing Plant	632.03
4000	Tailings Facilities	90.33
5000	Project Access Corridor	12.36
6000	Off-Site Facilities	126.74
	Total Direct Cost	*1,313.70*
8000	Owner's Costs	77.94
9000	Indirect Costs	370.78
	Total Indirect Cost	*448.72*
	Total Direct + Indirect Cost	*1,762.42*
	Contingency	200.42
	Total Capital Cost	**1,962.83**

Note: * Mining sustaining capital includes haulage of pit run waste materials to the tailings facilities for construction.

New Gold provided a diesel fuel cost of CDN$1.042/L based on an average of summer and winter rates. New Gold benchmarked the diesel rate with nearby facilities and applied discounts to reflect the road improvement and Patronage allocation.

An average electricity tariff rate of $0.046/kWh was assumed for Blackwater based on the BC Hydro tariff rate calculator Version October 18, 2012, Schedule 1823. In addition to the electricity tariff, 3% is included for line losses and 7% for PST, for an overall rate of $0.051/kWh.

Consumables include the following materials:

- Mill reagents
- Assay reagents
- Grinding media
- Steel liners.

The reagent quantities were developed from the process design criteria, taking into account testwork, the impact of the mass balance, and scaling factors from laboratory to industrial use.

Grinding media consumptions in the SAG and ball mills were estimated from the Bond Abrasion Index formula with input by AMEC. Wear rates for crusher liners are based on published Metso Linear Wear Rate charts.

Unit prices for consumables were obtained from current New Gold supply contracts or from budgetary RFQs received by AMEC. Reagent consumption estimates are based on extensive metallurgical testing and on general experience in the industry for similar ore types. Estimates include freight to site.

The costs of assay reagents are based on the estimated number of samples required for mining and processing and an estimated cost per sample. Assay costs are carried as a fixed annual cost.

Media supply unit costs are based on quotations from Moly-Cop for steel balls, ME-Elecmetal for SAG mill liners, Metso for ball mill liners, and AMEC in-house data for primary and pebble crusher liners.

The operating cost estimate is inclusive of costs from the beginning of Commercial Production to the end of production, including waste management facilities. Cost allocation boundaries are as follows:

- For mining, all costs incurred before April Year 1 are considered capital costs.
- Costs from April Year 1 to August Year 1 are considered Owner's capital start-up costs.
- All costs after August Year 1 are considered to be either operating costs or sustaining capital.

Goods and Services Tax (GST) is not included in the operating cost estimate. Provisions are made for British Columbia Provincial Sales Tax (PST). Exchange rate assumptions were the same as those used in the capital cost estimate.

21.2.2 Mine Operating Costs

The mine operating costs consist of the following components:

- Equipment operating cost – the activities of drilling, blasting, loading, hauling, mining support, and equipment maintenance. Equipment operating costs are calculated from the total annual operating hours based on the equipment productivities and the cost per SMU hour to operate the equipment.
- Salary and hourly personnel – mine department salary staff and general mining labour; maintenance and operator labour included as part of the equipment operating cost. During full production, the mine is expected to employ 44 salaried personnel and 345 hourly personnel.

- Miscellaneous costs – miscellaneous tools and equipment necessary to support mine operation, such as surveying, software (MineSight©, AutoCAD, SlopeW), geotechnical instrumentation for pit and dumps, and annual air survey.

The total cost, including support, is approximately $2 billion to mine 1.1 Bt (billion tonnes) of material, including ore rehandle and the overburden stockpile. The total unit cost for the Project is approximately $1.80/t. The average unit mining cost is $1.42/t for ore (including rehandle from ore stockpile) and approximately $1.15/t for waste. The unit cost is higher in the early years because of the small amount of material being moved and the higher cost of pre-stripping. The costs near the end of mine life increase again due to the long uphill hauls to get out of the pit.

The total operating cost for all activities is approximately $1.88/t, or $5.33/t of ore mined (Table 21-5) during the operations period.

21.2.3 Process Operating Costs

The process operating costs are based on the equipment included in the process flow diagrams. Two types of costs are included in the cost model: fixed and variable. Fixed costs are constant each year and include labour and maintenance. Variable costs are based on the annual tonnage produced by the plant calculated in accordance with the mine production provided from Norwest. Steel consumables and reagents are variable costs. Power costs are generally fixed costs with crushing and grinding areas as variable costs for tonnage ramp up considerations.

The processing operating cost estimate is divided into the following categories:

- Labour: Process labour costs were estimated by assuming a work force of 105 personnel, comprising operations, day crew (including tailings management), reagent crew, maintenance, technical, and laboratory staff. Based on AMEC in-house data, this work force level is consistent with typical 60,000 t/d gold processing plants. The process labour costs were estimated at $12.8 million per year. Until Year 3, only 6 instead of 15 laboratory technicians are included because of the delayed start-up of the assay laboratory. Laboratory services will be contracted out before Year 3.

Table 21-5: Mining Unit Costs (operations period)

Item	Including Reclaimed tonnes*		Excluding Reclaimed tonnes**	
	$/total t	$/ore	$/total t	$/ore
Loading	0.22	0.59	0.23	0.66
Hauling	0.87	2.31	0.92	2.61
Drilling	0.10	0.27	0.11	0.31
Blasting	0.24	0.63	0.25	0.71
Support Equipment + Miscellaneous	0.27	0.70	0.28	0.79
Salary	0.08	0.22	0.09	0.25
Total	**1.79**	**4.72**	**1.88**	**5.33**

Note: *Total tonnes with stockpile reclaimed = 1,018 million tonnes; ** Total tonnes mined = 968 million tonnes

- Power: Process power costs were based on estimated operating power loads for the processing equipment, including water supply, tailings reclaim, and TSF water management. The load list includes operating time, mechanical loading, and efficiency factors. The plant power cost is estimated at $42 million per year. Until Year 3, only 40% of assay laboratory power is included due to the delayed start-up of the assay laboratory. In addition, power is not included for the mill maintenance shop until Year 3 due to its delayed start-up. Power costs for water supply include fresh and process water pumping and are estimated at $2.39 million per year.
- Steel consumables: Grinding media and steel liners costs are estimated at $41.5 million per year. Grinding media consumptions in the SAG and ball mills are based on the average Blackwater ore abrasion index (0.19 average) from Lakefield Research formulae with input by AMEC.
- Reagents: Total reagent costs were estimated at $2.13/t milled over the life of the mine.
- Spare parts and maintenance supplies: Maintenance and spare parts include the equipment and typical building maintenance of the process areas. Maintenance materials are estimated at 5% of direct mechanical, electrical, and HVAC plant equipment capital cost. Total process plant electrical equipment capital cost is $204 million and results in an annual maintenance materials cost of $10.2 million. Maintenance supplies for the assay laboratory and mill maintenance shop are excluded before Year 3 based on when the areas will be established. Water supply equipment capital cost is $5.4 million and results in an annual maintenance materials cost of $0.270 million. Mill relining contracts for the SAG and ball mills are included under maintenance and are estimated based on the relining frequency, the number of mills, and AMEC's in-house data for the contract cost. An estimated annual cost of $1 million is included for mill relining contracts. Maintenance costs are included for the mobile equipment related to the process plant. These costs are fuel and maintenance based on estimated hourly usage. The annual process plant mobile equipment cost is estimated to be at $0.697 million.

Life-of mine unit operating costs for processing are summarized in Table 21-6.

21.2.4 General and Administrative Operating Costs

G&A costs included senior management, administration, insurance, and other costs required to support the operation. The G&A operating costs are summarized in Table 21-7 and include the following considerations:

- G&A labour costs were estimated based on a schedule of employees and data on salaries for similar operations in B.C.
- Costs for annual insurance premiums and fees are estimated at $2.7 million per year based on quotes received.
- An average cost of $55 per person per day was assumed for the total camp and catering costs.
- AMEC estimates that 5 MW of power will be required to operate the camp and administration buildings on site. In addition, AMEC estimates that 2.6 MW of power will be required to operate mine-related areas including the truck shop, truck wash, pit dewatering, and explosives storage area. The total G&A power cost is estimated at $3 million per year.
- Power line maintenance costs include access road maintenance, vegetation control, climbing and inspection, ground line treatment, pole replacements, and allowances for unexpected failures. Total costs for power line maintenance were estimated at $0.7 million per year.
- Site services and management costs were based on an estimate for operating mobile equipment costs and 0.5% of the total installed cost for all the G&A buildings and utilities that require maintenance. The total estimated cost for site services and management is $1.9 million per year.

Table 21-6: Life-of-Mine Processing Costs

Area	($/t milled)
Labour	0.58
Reagents and Consumables	2.14
Steel Consumables	1.89
Power	2.03
Spare Parts and Maintenance Supplies	0.57
Total	**7.20**

Table 21-7: Summary of G&A Operating Costs

Area	Annual G&A ($M/a)	Unit Cost ($/t milled)
G&A Labour	4.84	0.22
Camp and Catering	6.72	0.31
Insurance	2.70	0.12
Camp and Administration Power	3.08	0.14
Power line Maintenance	0.71	0.03
Site Services Costs	1.90	0.09
Mining Lease and BC Assessment Tax	2.33	0.11
Access Road Maintenance	2.68	0.12
Bussing Costs	1.00	0.05
Vanderhoof Office	0.20	0.01
Bus Depot Security	0.30	0.01
Health, Safety, and Security	1.07	0.05
Environmental	0.90	0.04
Human Resources	1.61	0.07
IT and Communications	0.53	0.03
Other G&A	0.60	0.03
Total	**31.17**	**1.43**

- Mining lease fees are based on a fixed rate of $20 per hectare of mining lease area. The Blackwater property does not sit within a municipal boundary, and the Project is not expected to pay municipal property taxes. The property does sit within the Cariboo regional district and is expected to pay a BC Assessment Tax based on a fixed rate of $20.52 per $1,000 of capital improvements. The value of capital improvements is estimated as the total installed materials cost for all infrastructure on site, adjusted by a typical estimated factor used by the BC Assessment office.

- New Gold would become the primary user of the Kluskus Forest Service Road (FSR) during operations. The access road runs from Highway 16 to the plant site entrance. Access road operating costs cover annual road maintenance, sanding, gravel, grading, and snow removal. The road maintenance cost is estimated to be $2.7 million per year, based on consultation with the current primary road user/maintainer and internal estimates.
- Bus transport costs were estimated at $50 per person per rotation and to total $1.0 million per year. This cost assumes bus transportation from Vanderhoof, the designated hub city, to site. Employees being bussed to site will need to park their vehicles at the bus depot in Vanderhoof. Security for this parking lot will be required 24 h/d to discourage theft of employees' property. The estimated cost of a security contract for this service is $0.3 million per year.
- The Blackwater Project will maintain a small office in Vanderhoof to support local community relations activities and other off-site activities. Based on current office expenses, an annual cost of $0.2 million was estimated.
- New Gold developed an estimate based on HSE expenses at its other operations. HSE costs for Blackwater are expected to total $1.1 million per year.
- Based on expected requirements and comparisons to its other similar operations, New Gold developed a budget estimate for non-labour environmental department operating costs. The non-labour environmental operating costs are estimated to be $0.9 million per year.
- New Gold estimated the human resources operating costs by comparison to its other operations. Human resources costs for Blackwater are estimated to be $1.6 million per year.
- Based on current IT and communication expenses at Blackwater and assumptions to account for the increased number of personnel on site, IT and communications expenses were estimated to be $0.5 million per year.
- Other G&A costs represent ancillary miscellaneous costs related to office management, business travel, legal fees, and other assorted costs. Other G&A costs are estimated to be $0.6 million per year.

21.2.5 Operating Cost Summary

The average life-of-mine (LOM) operating cost estimate is summarized by major area in Table 21-8, and is based on a total of 341.8 Mt of ore milled over the mine life at an average rate of 21.9 Mt per year. All costs are expressed in Q3 2013 Canadian dollars with no allowance for contingency.

Table 21-8: Summary of Blackwater LOM Operating Costs

Area	LOM Total Cost ($M)	Unit Cost ($/t milled)
Mining	1,823	5.33
Process	2,462	7.20
G&A	488	1.43
Total	**4,773**	**13.96**

Costs exclude royalties, GST, depreciation, escalation, and product transport costs, which are handled separately in the Blackwater economic model.

21.3 Comments on Section 21

The QPs note:

- The initial capital cost estimate is $1,963 million
- Total LOM sustaining capital is estimated to be $681 million
- Reclamation and closure costs net of salvage value of $7.4 million have been applied to Year 17 and are included in the financial model in Section 22
- LOM operating costs are estimated at $14.49/t of ore milled, and $12.48/t of ore milled after accounting for silver credits
- Total LOM all-in sustaining cash costs are estimated at $14.47/t of ore milled
- The average LOM cost after silver credits and royalty is US$578/oz Au produced
- The average LOM all-in sustaining cash cost is US$670/oz Au produced

22.0 ECONOMIC ANALYSIS

The results of the economic analysis discussed in this section represent forward-looking information as defined under Canadian securities law. Actual results may differ materially from those expressed or implied by forward-looking information. The reader should refer to Section 1.3 of this Report for more information regarding forward-looking statements, including material assumptions (in addition to those discussed in this section and elsewhere in this Report) and risks, uncertainties and other factors that could cause actual results to differ materially from those expressed or implied in this section (and elsewhere in the Report).

22.1 Methodology Used

The financial analysis was carried out using a discounted cash flow (DCF) model. All cash inflows and outflows throughout the life of the Project are discounted at a given discount rate to a present value and are then added to yield a net present value (NPV).

The DCF model for the Blackwater Project assumes 100% equity financing (no debt) as a simplifying assumption, and all figures are expressed in Q3 2013 constant dollars or real terms. No inflation or real cost escalation was assumed. Cash flows are assumed to occur mid-period. All monetary figures are expressed in Canadian dollars unless otherwise specified.

Under the model, mining pre-production starts in Year -2, and Year 1 marks the first year of commercial production resulting in revenue. In Table 22-6, calendar years commencing with 2014 (Year -3) have been inserted. These references to specific dates and calendar years are for illustrative purposes only and would apply only if a formal construction decision in respect of the Project was made on December 31, 2014. The reader should note that no such construction decision has been made. The association of specific dates or years to quantities of material mined or produced, revenues, operating or capital costs incurred, taxes or royalties paid, depreciation, cash flows or economic analysis is for illustrative purposes only and does not reflect the expected timing of a construction decision or the development and operation of the Project.

22.2 Financial Model Parameters

The Project value is determined on a pre-tax and after-tax basis with discounting to the start of Year -2 (2015), which marks the first year of Project construction. Project expenses incurred in Year -3 are treated in the cash flow model as occurring at the start of Year -2.

An after-tax valuation was also ascertained using the taxation assumptions stated in Section 22.2.4.

Mining pre-production starts in Year -2, and Year 1 marks the first year of commercial production resulting in revenue.

The DCF model is based on the underlying economic assumptions listed in Table 22-1 to Table 22-3.

22.2.1 Mineral Resource, Mineral Reserve, and Mine Life

The analysis uses the Mineral Resources presented in Section 14 that were modified to produce the Mineral Reserves in Section 15. Total Proven and Probable Mineral Reserves are estimated at 344.4 Mt grading 0.74 g/t Au and 5.5 g/t Ag.

22.2.2 Operating Costs

Mining costs average $5.33 per tonne of ore milled, or $1.79 per tonne of material moved for the life of mine. Processing costs are estimated at $7.20 per tonne of ore milled. General and administration costs are estimated at $1.43 per tonne of ore milled.

22.2.3 Capital Costs

The construction period of the Project is estimated to be two years. The major capital outlays are scheduled during the two years before production, Year -2 and Year -1, with the first year of production being Year 1. The total initial capital for the Project is estimated to be $1,963 million (US$1,865 million). All capital related to exploration and study development is considered sunk since it is scheduled to be complete by the end of 2013.

Sustaining capital over the LOM is estimated at $681 million (US$647 million).

Table 22-1: Economic Inputs

Item	Unit	Value
Metal Prices		
Gold	US$/oz	1,300
Silver	US$/oz	22
Discount Rate	%	5.0
Exchange Rate	CAD/USD	0.95
Refinery Deductions		
Payable Factor, Au	%	99.9
Payable Factor, Ag	%	95.0
Bullion Refining Treatment Charge	$/oz Au + Ag	0.8
Transport and Insurance	$/oz Au	2.45

Table 22-2: Royalty Requirements Covering Blackwater Deposit

Claim Number	Claim Name	Remaining NSR Royalty (%)	Remaining Buyout Amount
509273	Davidson	0	Purchased
515809	Dave	1.5	Nil
515810	Jarrit	1.0	Nil

Table 22-3: Operating Cost Statistics (unit costs averaged over mine life)

Operating Cost Area	LOM Total ($M)	LOM $/Tonne Milled ($M)	LOM $/Ounce Produced ($M)
Mining	1,823	5.33	260
Processing	2,462	7.20	351
General and Administration	488	1.43	70
Refining	73	0.21	10
Transport and Insurance	17	0.05	2
Total Royalties	93	0.27	13
Subtotal	4,957	14.49	707
Silver Credit	(686)	(2.01)	(98)
Costs Net of Royalty and Ag Credit	**4,271**	**12.48**	**609**

22.2.4 Taxation Considerations

The tax rate calculations and taxation assumptions used in the after-tax NPV calculations are summarized in this sub-section.

The federal government imposes income tax on mining income at the same rate that applies to other types of income. The federal rate applicable to resource profits is 15%. British Columbia's taxation of the resource sector is generally harmonized to the federal system. The provincial corporate income tax rate applicable to mining income is 11%. A combined rate of 26% is used in the model to compute the federal and provincial tax liability in respect of the Blackwater Project. All deductions and rates are based on currently enacted legislation and are subject to change.

Under the British Columbia Mineral Tax Act mining taxes are imposed in two stages:

- 2% tax on net current proceeds
- 13% tax on net revenue.

The initial 2% tax is a form of minimum tax, which is deductible in full against the 13% tax. Both the net current proceeds tax and the net revenue tax have been factored in the Blackwater after-tax life-of-mine model.

The 13% net revenue tax is computed by deducting the capital costs, exploration costs, pre-production development costs and an annual investment allowance from the net current proceeds and therefore it only becomes payable once the capital costs incurred in developing the mine have been fully recovered. Both the 2% net current proceeds tax and the 13% net revenue tax have been factored in the after-tax NPV calculations for Blackwater.

Federal and provincial tax legislation provides a number of deductions, allowances, and credits that are specifically available to taxpayers engaged in qualifying mining activities. The most notable of these deductions are Canadian Exploration Expenditures (CEE), Canadian Development Expenses (CDE), and capital costs eligible for Class 41 of the capital cost allowance system. Because these deductions and allowances are only available when incurred, a high-level assumption was made with regard to the allocation of expenditures between the three categories in the life-of-mine model.

A further consideration when calculating the after-tax economics for Blackwater is that the project is held within a corporate entity that also includes New Gold's New Afton operation located in British Columbia and the company's two Canadian-based corporate offices, one of which is in Vancouver, British Columbia and the other in Toronto, Ontario. As such, New Gold should be able to realize tax synergies between different assets by utilizing tax attributes interchangeably amongst its portfolio of assets, all with the goal of maximizing New Gold's overall profitability rather than that of any one operation or project.

Among other things, Blackwater's after-tax economics will be dependent on the timing of its development, the related build-up of New Gold's tax attributes and the ultimate allocation and utilization of these tax attributes. For purposes of the base case after-tax economics, only the deductions, allowances and credits that are specifically related to the Blackwater project have been included. No allocation of potential attributes related to New Gold's current, or future, corporate administrative expenses, interest expenses or capital costs at the company's other Canadian-based assets has been made in the after-tax analysis.

At the end of 2012, on a combined basis, the New Gold corporate entity had over $900 million in CEE, CDE and Class 41 capital cost allowance, beyond those specifically related to Blackwater. Going forward, New Gold could use these attributes to shelter taxable profits from one or both of its New Afton and Blackwater projects and plans to do so in a manner that maximizes its overall profits. Further, certain attributes should continue to build over time as, on an annual basis New Gold's corporate administration expenses are estimated to approximately US$30 million and its annual total finance costs are approximately US$52 million. If all, or some, of the above noted CEE, CDE and/or Class 41 allowances as well as these annual expenses were to be allocated to Blackwater, the after-tax results would improve when compared to the Base Case.

In addition, the after-tax economics for Blackwater do not take into account any future corporate re-organizations or tax planning that New Gold may undertake to maximize its overall profitability.

22.2.5 Closure Costs and Salvage Value

Closure costs are estimated at approximately $86 million. A salvage value of $78.4 million has been estimated for Project assets upon closure.

22.3 Financial Results

Using a discount rate of 5%, the pre-tax and after-tax results are summarized in Table 22-4. Life-of-mine production is summarized in Table 22-5. Figure 22-1 shows annual gold production and cash costs (net of silver credits), and Figure 22-3 shows the Project value creation as depicted the annual pre-tax net cash flow and a cumulative discounted total. All figures are in CAD millions. The Blackwater Project cash flow schedule is summarized in Table 22-6.

Table 22-4: Valuation Metrics

5% Discount rate, Year -2 (2015)	Unit	Pre-Tax	After-Tax
Net Present Value	$M	1,044	616
IRR	%	11.3	9.3
Payback Period*	years	6.2	6.4

Note: * Payback period starts from commercial production using undiscounted cash flows

Table 22-5: Production Summary

Item	Unit	LOM Total	LOM Annual Average	Year 1–9 Annual Average*	Year 1–14 Annual Average*	Year 15-17 Annual Average
Life of Mine	years	17	NA	NA	NA	NA
Metal Price						
Gold	US$/oz	1,300	1,300	1,300	1,300	1,300
Silver	US$/oz	22	22	22	22	22
Production						
Ore Milled	kt	341,791	20,105	20,373	20,919	16,307
Waste Mined	kt	631,784	37,164	56,878	45,127	-
Strip Ratio W:O	-	1.88	1.88	2.28	2.07	-
Gold Grade	g/t	0.74	0.74	0.85	0.79	0.40
Silver Grade	g/t	5.50	5.50	5.61	4.70	10.28
Metal Feed						
Gold	koz	8,101	477	557	534	209
Silver	koz	60,398	3,553	3,675	3,159	5,391
Process Recoveries						
Gold	%	86.6	86.6	87.1	86.8	84.4
Silver	%	49.1	49.1	50.1	48.5	50.6
Metal Recovered						
Gold	koz	7,016	413	485	463	177
Silver	koz	29,606	1,742	1,842	1,531	2,726

Note: * Year 1 starts from Commercial Production (excludes pre-production)

Figure 22-1: Gold Production and Cash Costs

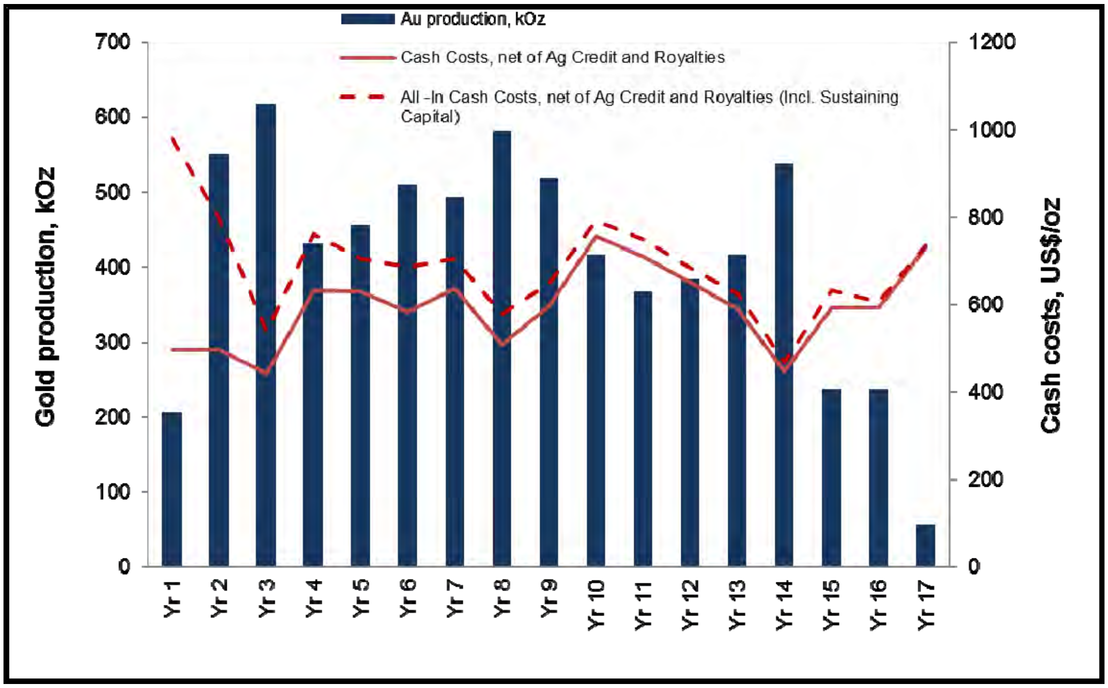


Figure 22-2: Life-of-Mine Cash Flow Projection (pre-tax and after-tax 5% discount rate)

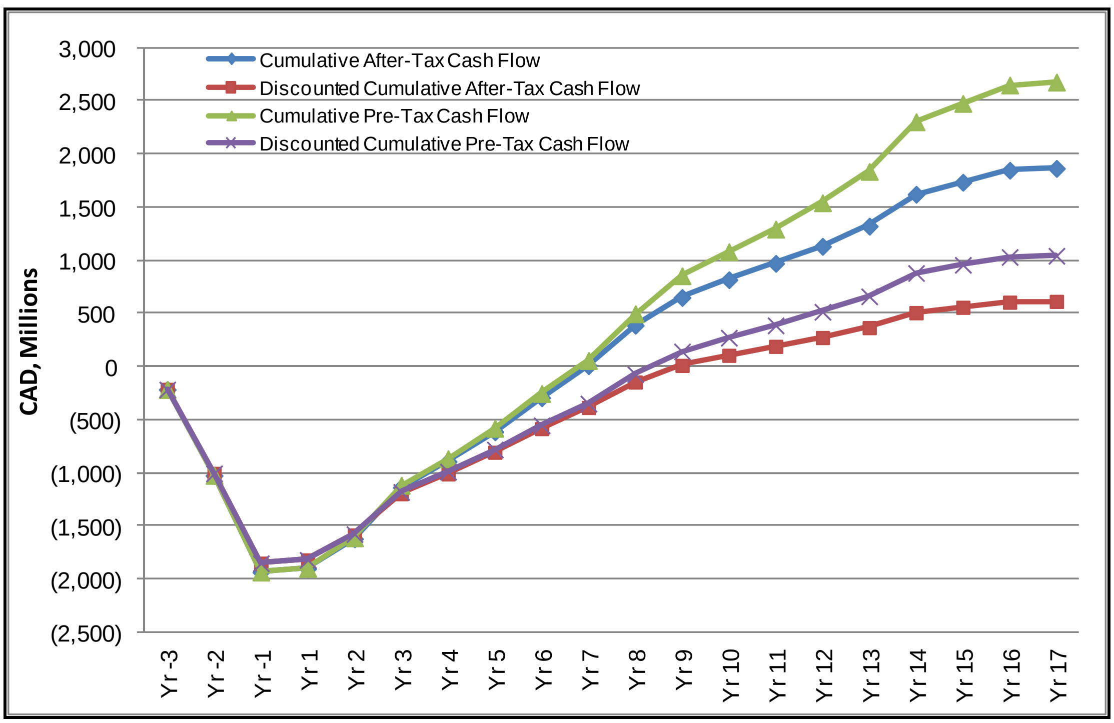


Table 22-6: Blackwater Project Cash Flows

Summary schedule	Date	Schedule	FS_Plan	Canadian dollars (2013 real terms)
Blackwater Project	11/25/2013	Inputs	5% discount rate, US$ 1,300/oz Au, US$ 22/oz Ag	Valuation date: January 1, 2015

NOTE: Excludes preproduction ore

Year ending 31 December		LOM	2014	2015	2016	2017	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	2033	2034	2035
Operating Metrics			Yr -3	Yr -2	Yr -1	Yr 1	Yr 2	Yr 3	Yr 4	Yr 5	Yr 6	Yr 7	Yr 8	Yr 9	Yr 10	Yr 11	Yr 12	Yr 13	Yr 14	Yr 15	Yr 16	Yr 17	Yr 18	
Mining																								
Ore mined (note 1)	ktonnes	336,152	-	-	-	9,852	25,545	25,532	25,042	27,576	24,890	29,287	20,704	25,029	27,088	21,902	21,902	21,902	21,902	-	-	-	-	-
Waste	ktonnes	631,784	-	-	-	17,295	55,251	60,271	63,389	60,732	66,767	61,523	61,790	64,881	56,897	27,451	14,886	11,931	8,709	-	-	-	-	-
Strip ratio (note 2)		1.88	-	-	-	1.8	2.2	2.4	2.5	2.2	2.7	2.1	2.2	2.6	2.1	1.3	0.7	0.5	0.4	-	-	-	-	-
Production																								
Ore to mill	ktonnes	341,791	-	-	-	8,150	21,899	21,899	21,899	21,899	21,903	21,903	21,903	21,903	21,903	21,902	21,902	21,902	21,902	21,900	21,900	5,121	-	-
Au grade	g/tonne	0.74	-	-	-	0.93	0.91	1.00	0.71	0.75	0.83	0.80	0.94	0.84	0.69	0.61	0.63	0.69	0.88	0.40	0.40	0.40	-	-
Au recovery	%		-	-	-	84.8	88.4	87.8	86.1	88.3	87.4	87.2	88.3	87.8	88.0	85.7	85.8	88.2	88.9	84.4	84.4	84.4	-	-
Gold	koz	7,016	-	-	-	206	551	618	431	456	509	493	581	519	417	387	383	417	538	237	237	55	-	-
Silver	koz	29,808	-	-	-	857	2,013	2,209	2,057	1,776	2,218	1,383	2,154	1,934	1,181	1,044	912	1,001	711	3,861	3,881	856	-	-
Commodity Prices																								
Gold	US$/oz	1,300	-	-	-	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	-	-
Silver	US$/oz	22.0	-	-	-	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	22.0	-	-
Operating Costs																								
Operating costs*	$M	(4,271)	(0.5)	(0.5)	(0.5)	(107.4)	(287.9)	(287.3)	(288.8)	(302.2)	(312.1)	(330.7)	(311.3)	(326.2)	(332.1)	(274.1)	(262.4)	(258.5)	(252.4)	(148.0)	(147.9)	(42.8)	-	-
Per unit costs	$/oz Au	609	-	-	-	520.7	522.6	465.0	665.7	663.1	613.5	670.1	535.4	628.4	796.6	746.1	685.2	620.5	468.4	624.0	623.5	768.3	-	-
* Mining, processing, G&A, refining, royalties and transport - net of Ag revenue																								
Capital Costs																								
Initial, construction capital																								
Mining Preproduction	$M	(100)	-	(24)	(75)	(1)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Mining Equipment	$M	(182)	(28)	(134)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Construction	$M	(1,700)	(185)	(651)	(833)	(31)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Subtotal - Initial/Construction capital	$M	*(1,963)*	*(213)*	*(809)*	*(909)*	*(32)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Sustaining capital																								
Mining	$M	(202)	-	-	-	(67)	(34)	(16)	(13)	(10)	(31)	(6)	(5)	(5)	(3)	(4)	(1)	(3)	(2)	(1)	(1)	-	-	-
On-site Infrastructure	$M	(34)	-	-	-	-	(34)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Processing Plant	$M	(49)	-	-	-	(0)	(2)	(2)	(3)	(3)	(3)	(5)	(3)	(3)	(4)	(4)	(4)	(3)	(4)	(2)	(2)	(0)	(0)	-
Tailings Facilities	$M	(314)	-	-	-	(33)	(67)	(38)	(35)	(19)	(18)	(20)	(26)	(17)	(7)	(6)	(7)	(8)	(6)	(5)	-	-	-	-
Off-site Facilities	$M	(7)	-	-	-	-	(7)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-
Indirects, contingency	$M	(76)	-	-	-	(5)	(31)	(6)	(6)	(3)	(3)	(4)	(6)	(3)	(2)	(1)	(2)	(2)	(1)	(1)	(0)	(0)	(0)	-
Subtotal - Sustaining capital	$M	*(681)*	-	-	-	*(105)*	*(174)*	*(62)*	*(58)*	*(36)*	*(55)*	*(35)*	*(42)*	*(28)*	*(15)*	*(15)*	*(13)*	*(16)*	*(13)*	*(10)*	*(3)*	*(0)*	*(0)*	-
Rehabilitation, closure costs (net of salvage)	$M	(7)	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	(7)	-	-
Subtotal - Rehabilitation, closure costs (net of salvage)	$M	*(7)*	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	-	*(7)*	-	-
Total capital expenses	$M	(2,651)	(213.5)	(809.1)	(908.6)	(138.7)	(173.9)	(62.1)	(57.6)	(35.6)	(55.1)	(35.1)	(41.9)	(28.4)	(15.0)	(15.5)	(13.2)	(15.7)	(12.9)	(9.6)	(3.3)	(7.7)	(0.3)	-
Financial Summary																								
Total revenue	$M	10,286	-	-	-	302	800	896	637	665	748	707	846	755	598	527	545	593	752	409	409	96	-	-
Net Operating Costs	$M	(4,863)	-	-	-	(123)	(326)	(332)	(331)	(338)	(355)	(357)	(353)	(362)	(355)	(284)	(280)	(278)	(266)	(228)	(228)	(59)	-	-
Royalties	$M	(93)	(0.5)	(0.5)	(0.5)	(3.9)	(8.9)	(6.7)	(3.3)	(4.9)	(8.7)	(5.6)	(8.4)	(9.3)	(4.6)	(3.9)	(3.7)	(3.5)	(3.0)	(5.0)	(5.0)	(3.5)	-	-
EBITDA	$M	5,329	(0.5)	(0.5)	(0.5)	174.8	465.8	558.0	302.7	321.5	384.1	344.6	484.4	384.1	238.4	228.6	261.6	311.6	483.5	176.6	176.7	33.3	-	-
Depreciation	$M	(2,651)	-	-	-	(216.3)	(234.7)	(241.3)	(247.4)	(251.3)	(257.3)	(261.2)	(177.3)	(173.3)	(172.6)	(68.8)	(60.9)	(58.6)	(57.4)	(57.1)	(54.8)	(60.1)	(0.3)	-
EBIT	$M	2,678	(0.5)	(0.5)	(0.5)	(41.6)	231.1	316.8	55.2	70.2	126.8	83.3	307.0	210.8	65.8	159.8	200.7	253.0	426.0	119.5	122.0	(26.8)	(0.3)	-
Change in NWC	$M	0	-	-	-	(2.7)	(4.5)	(1.7)	5.0	(0.3)	(1.2)	0.9	(2.8)	2.0	2.9	(0.2)	(0.7)	(1.0)	(3.4)	5.4	(0.0)	1.9	0.4	-
Free Cash Flow (Pre-Tax)	$M	2,678	(214.0)	(809.6)	(909.1)	35.4	287.4	494.2	250.0	285.8	327.8	310.3	439.8	357.7	226.3	212.9	247.7	294.9	467.2	172.4	173.4	27.5	0.1	-
Taxes	$M	(810)	-	-	-	(2.1)	(8.2)	(10.8)	(5.7)	(6.3)	(7.2)	(6.8)	(58.8)	(96.8)	(58.7)	(59.1)	(87.5)	(105.0)	(167.2)	(59.1)	(60.7)	(9.5)	-	-
Free Cash Flow (After-Tax)	$M	1,868	(214.0)	(809.6)	(909.1)	33.3	279.2	483.4	244.3	279.3	320.6	303.5	380.8	260.9	167.7	153.8	160.2	189.9	299.9	113.3	112.7	18.0	0.1	-
Discount factors (mid-period)			*1.000*	*0.976*	*0.929*	*0.885*	*0.843*	*0.803*	*0.765*	*0.728*	*0.694*	*0.661*	*0.629*	*0.599*	*0.571*	*0.543*	*0.518*	*0.493*	*0.469*	*0.447*	*0.426*	*0.406*	*0.386*	*0.368*

		Pre-Tax	After-Tax
Net present value, 2015	$M	**1,044**	**616**
Payback period	Years	**6.2**	**6.4**
IRR	%	**11.3**	**9.3**

Notes:
1. Ore mined excludes 5,639,000 tonnes mined in the pre-production period
2. Strip ratio excludes waste mined in the pre-production period

22.4 Sensitivity Analysis

Sensitivity analysis was performed on the Project to gauge its robustness against favourable and unfavourable changes to the following Project variables:

- Gold price
- Exchange rate
- Operating costs
- Capital costs.

Figure 22-3 indicates the Project is more sensitive to changes in the gold price and the USD:CAD exchange rate than to changes in capital or operating costs.

Four gold price, silver price, and exchange rate scenarios were used in the pre-tax model to evaluate the sensitivity on pre-tax NPV, IRR, and payback. The Base, Low, Moderate, and High gold price cases and results are shown in Table 22-7. This table demonstrates sensitivities to gold price, silver price, and exchange rate for the four selected scenarios. All other parameters, including capital costs and operating costs, are fixed.

The after-tax financial model was calculated at the same four gold price, silver price, and exchange rate scenarios that were used in the pre-tax model. The Base, Low, Moderate, and High gold price cases and after-tax results are shown in Table 22-8.

22.5 Comments on Section 22

The QP notes that an economic evaluation was prepared for the Project based on a pre-tax financial model. For the 17-year mine life and 341 Mt mill feed, the following Base Case financial parameters were calculated:

- Pre-Tax
  - $1,044 million NPV (pre-tax, Year -2 (2015)) at 5.0% discount rate.
  - 11.3% IRR
  - 6.2-year payback on $1,963 million capital cost.
- After-Tax
  - $616 million NPV (after-tax, Year -2 (2015)) at 5.0% discount rate.
  - 9.3% IRR
  - 6.4-year payback on $1,963 million capital cost.

Figure 22-3: Selected Sensitivities – Pre-Tax NPV at 5% Discount Rate

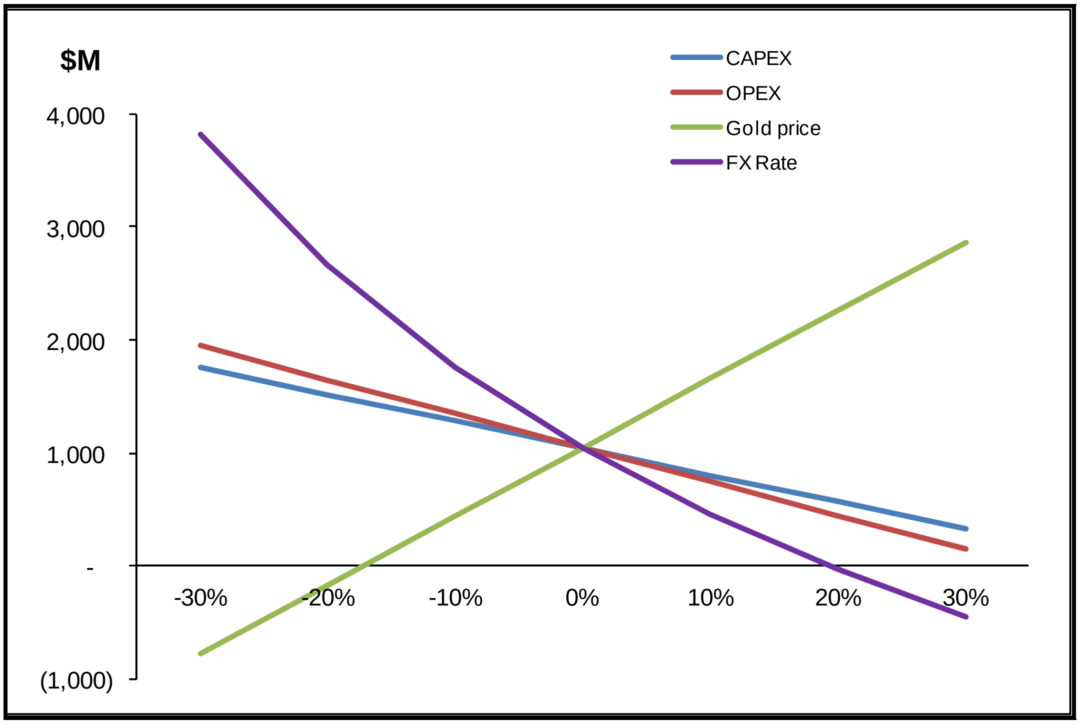


Table 22-7: Commodity Price and Exchange Rate Scenarios, Pre-Tax

Item	Unit	Case Base	Low	Moderate	High
Gold Price	US$/oz	1,300	1,150	1,450	1,600
Silver Price	US$/oz	22	20	24	26
Exchange Rate	US$/CAD$	0.95	0.93	0.97	1.00
NPV, CAD$	$M	1,044	432	1,631	2,120
IRR	%	11.3	7.8	14.4	16.8
Payback Period	years	6.2	7.5	5.1	4.5
NPV, US$	$M	991	402	1,582	2,120
IRR	%	11.3	7.8	14.4	16.8
Payback Period	years	6.2	7.5	5.1	4.5

Note: Pre-tax NPV at Year -2 (2015) and 5% discount rate.

Table 22-8: Commodity Price and Exchange Rate Scenarios, After-Tax

		Case			
Item	**Unit**	**Base**	**Low**	**Moderate**	**High**
Gold Price	US$/oz	1,300	$1,150	1,450	1,600
Silver Price	US$/oz	22	20	24	26
Exchange Rate	US$/CAD$	0.95	0.93	0.97	1.00
NPV, CAD$	$M	616	199	1,008	1,329
IRR	%	9.3	6.5	11.9	13.8
Payback Period	years	6.4	7.7	5.3	4.6
NPV, US$	$M	585	185	978	1,329
IRR	%	9.3	6.5	11.9	13.8
Payback Period	years	6.4	7.7	5.3	4.6

Note: After-tax NPV at Year -2 (2015) and 5% discount rate.

23.0 ADJACENT PROPERTIES

This section is not relevant to this Report.

24.0 OTHER RELEVANT DATA AND INFORMATION

24.1 Risks and Opportunities Assessment

New Gold developed and implemented a comprehensive risk and opportunities register during the Feasibility Study and tracked progress in addressing and advancing the items.

24.1.1 Risks

Project risks that were identified include:

- Economic risks: changes in metal prices and exchange rates; changes in input costs, primarily labour, fuel, and bulk materials for mining and processing; escalation in capital cost. Economic risks are almost completely out of the control of New Gold.
- Capital cost growth: costs for contractors, personnel, materials, and equipment are volatile; labour shortages may occur. Mitigation measures include provision for milestones in capital contracts let; provision of an attractive work site; employment of employ a cost engineer on the design team who will be responsible for tracking all major quantities and any design changes during detailed design and for providing insight into trends to minimize "cost creep"; employment of quantity surveyors during construction to check placed quantities against final design quantities; implementation of a rigorous Project controls system to provide progress, cost, and schedule monitoring and control
- Operating costs: operating costs are sensitive to changes in the price of labour, consumables such as diesel fuel, and contractor services. The drill-and-blast study assumed the mine rock could be blasted using a "low energy" blast pattern design more than 70% of the time. If the rock proves to be more competent than assumed, then more drilling and blasting could be required, leading to higher mining operating costs. Mitigation measures for labour costs include offering competitive employment contracts,
- Productivity: the assumed equipment and labour productivities are based on good regional practice and a new operation will need to invest in training and hiring to achieve the same levels. Equipment productivities depend strongly on operator experience. New Gold will employ similar strategies for employee hiring and training as successfully used at its New Afton mine.
- Dilution and ore loss: Geological and mining conditions may be more challenging and complex than has been assumed, possibly resulting in lower revenues than anticipated due to lower ore tonnages or feed grades. Implementing effective

grade-monitoring and grade-control procedures will be key in preventing higher dilution and/or ore losses than estimated in this study and in the ability to identify waste rock for TSF construction.

- High-grade silver in the mill feed: the presence of high-grade silver in the mill feed later in the mine life could increase the frequency of elution because of faster carbon loading and cause soluble losses of silver, resulting in lower overall silver recovery. Plant feed may have to be carefully blended, possible with material from the high silver grade stockpile, through close coordination of the mine and mill departments. Alternatives for dealing with the higher silver grades include:
  - adding two additional electrowinning cells in around Year 7 to increase the number of elution cycles
  - feeding high silver ore to the mill before a maintenance shutdown of the entire plant or of a single circuit to provide twice the leach and adsorption time for this material
  - adding a CIC circuit to scavenge soluble silver onto activated carbon, either in grinding or the tailings thickener, in later years. This could benefit strip frequency and thereby improve recovery and operating costs.
- TSF construction: higher costs may be encountered in integrating mining operations with TSF construction.
  - In the early mine life, the bulk of the waste material types with low potential for acid rock drainage (ARD) and metal leaching (ML) will be required for TSF dam construction. Waste capture rates of up to 80% will be needed in some years to meet the dam construction schedule. If less low ARD/ML potential waste (NAG5 and overburden) is available than anticipated, then the mine plan may need to target this better-quality material more specifically or to accept more NAG4 waste rock for dam construction.
  - Haulage costs and sustaining capital (e.g., more haul trucks) may be higher if more NAG waste rock is found to be ML and needs to be disposed of in the TSF.
  - When the detailed waste release schedule, the TSF dam construction schedule, and the material capture rates are developed, it may become apparent that additional trucks are required for longer hauls of waste materials.
  - Waste characterization for the Project is based on a comprehensive geochemical database and model and the testing to date, which found that NAG4 and NAG5 waste rock behave in a similar fashion. As a result, waste rock containing higher levels of zinc may be acceptable for TSF construction.
- Project delays: project delays could result from uncertainty in the federal and provincial environmental permitting process because of legislative and administrative changes.

- Lack of broad-based community support within First Nations communities could also affect Project timing, despite New Gold's considerable progress in consulting and reaching agreements with local Aboriginal groups on exploration and environmental assessment participation.
- Delays in mill start-up could occur if water supply is insufficient. This could happen if the two years of construction are very dry or if delays arise in commissioning the freshwater pumping system from Tatelkuz Lake. This could be mitigated by moving installation of booster station 3 and its pipeline from sustaining capital to Year -1.

24.1.2 Opportunities

Project opportunities that were identified include:

- Mineral resources
  - The existing pit limits include 2.7 Mt of inferred resource currently considered to be waste. Additional drilling and supporting studies may support upgrading of the classification category for some or all of this material.
  - Mineralization remains open at depth under the planned open pit and may represent an upside opportunity for future pit expansion with additional drilling
- Mining equipment
  - A high-level study in early 2013 assessed the potential for implementing a trolley assist system for the haul trucks. The assessment found that this option could possibly reduce costs in the later years of the mine life, depending upon diesel and electricity costs at the time.
  - Drilling is currently assumed to be conducted by PV271 class drills boring 200 to 250 mm holes. Once drilling experience is gained at site, it may be possible to use a larger class of machine capable of drilling larger holes, thus reducing drilling and blasting costs.
- Mine operations
  - Bench heights of 12 m were selected for mining both ore and waste, primarily because of dilution considerations. The large waste zones to be mined in some years could potentially be stripped more efficiently using higher benches without affecting selectivity, thus reducing waste mining costs.
  - Waste production could also be reduced and additional mineralization may be able to be recovered if pit slope angles could be increased.
- Process plant operation

- Oxygen can be used instead of compressed air for cyanide leaching and cyanide detoxification. Although high purity oxygen is not required for the Project, early-stage testing found it improved kinetics and potentially increased metal recovery; this could lead to a reduction in the number of leach tanks required and/or increased metal recoveries. Oxygen could be supplied by an on-site cryogenic plant (99% pure oxygen), a VSA oxygen system (90% pure oxygen), or by trucking to site (liquid oxygen). The VSA option is significantly less expensive than either of the other two alternatives. Further testing of oxygen sparging could be conducted in the operating plant or in a large-scale continuous pilot plant.
- Pre-crushing the SAG mill feed to a finer feed size could increase throughput by about 20% and reduce ball mill liner and media cost compared to a standard SABC circuit. A pre-crush circuit would have higher capital and power costs, SAG ball wear, layout space requirements, and SAG mill maintenance. This addition could be implemented if experience with the ore warranted it and the economic analysis showed it could be beneficial for production.

* Value engineering
  - Assess the potential for increased mill throughput due to softer oxide/transition ores in early years to maximize existing plant capacity
  - Re-evaluate the major mill and primary crusher foundation designs incorporating bedrock anchors to reduce concrete quantity
  - Optimize plant site layout to reduce earthworks
  - Evaluate additional on-site borrow sources to reduce borrow haul distances
  - Reassess single versus multiple process buildings as a potential cost reduction
  - Re-evaluate the water supply system to simplify design
  - Re-examine the water reclaim system design for TSF Site C
  - Assess potential integration of the construction and operations camps
  - Optimize the layout and construction of the low-grade stockpile to simplify low-grade ore placement and the drainage water collection system
  - Develop detailed tailings and PAG waste rock deposition plans to simplify closure plan
  - Consider delaying detailed design of the Project, so that all Project permits are obtained prior to development of the final construction schedule.

24.2 Project Execution Plan

A Project Execution Plan (PEP) has been prepared using feasibility-level engineering information, quantities, productivities, data, cost estimates, schedules, and other assumptions developed for the 2013 Feasibility Study.

New Gold will employ a Project Management Team (PMT) to manage and deliver the Project. The PMT will develop and implement Project Policies in the year before construction activities commence. The PMT, with input from the EPCM consultant, other consultants, and contractors, will develop and implement the necessary execution and management plans.

The PEP incorporates a Project Development Schedule that does not include any contingency for schedule delay. It is based on an assumption that New Gold Board Approval and Decision to Proceed will be obtained at the beginning of Year -3 to attain Commercial Production (defined as achieving 60% of nameplate throughput over a 30-day period) by mid-Year 1.

The Project is anticipated to require approximately 6,200,000 manhours, including New Gold Project and Operations personnel, the EPCM consultant, other engineering consultants, and construction and service contractors. The on-site workforce is projected to peak at 1,175 persons in the second construction year

The key operational staff for Blackwater will be hired 12 to 16 months ahead of the commissioning period, with the remaining crews hired three or more months before commissioning. The focus for the early hires, such as superintendents, supervisors, trainers, and planners, will be to set up processes, procedures, and training programs as well as to interview and hire crews. Hiring crews early for training, and having qualified trainers and supervision in place to train inexperienced staff and instil health and safety behaviours from Day 1, will be a key to a successful operation.

25.0 INTERPRETATION AND CONCLUSIONS

The QPs have reached the following conclusions and made the following interpretations as a result of the review of the 2013 Feasibility Study document:

25.1 Project Setting

- Mining activities should be capable of being conducted year-round
- There is sufficient suitable land area and surface rights available within the mineral claims for any future tailings disposal, mine waste disposal, and installations such as a processing plant, and related mine infrastructure.

25.2 Mineral Tenure, Surface Rights and Royalties

- New Gold holds 100% recorded interest in 227 mineral claims covering an area of 104,678 ha distributed among the Property and the Capoose, Auro, and Key claim blocks
- The Property claim block comprises 75 mineral cell claims totalling 30,578 ha. All Blackwater claims are 100% held in the name of New Gold. Sixty-nine claims expire in 2022. Two claims expire in October 2014 and four claims expire in January 2015. There are no other parties with beneficial interests in these mineral rights. None of the Blackwater cell claims are known to overlap any legacy or Crown granted mineral claims, or no-staking reserves
- The Blackwater deposit spans the Davidson claim (509273), the Dave claim (515809) and the Jarrit claim (515810)
- A review of surface rights in the vicinity of the Property was undertaken in December, 2013 and January, 2014. The majority of the Blackwater mineral claims comprising the Property are located on Crown lands. The review identified an overlapping private parcel, land reserves/notations, grazing tenures, forest recreation sites, forest tenures, trap lines, guide outfitter areas, and an ungulate winter range
- A review of surface rights in the vicinity of proposed electrical transmission line, water pipeline, airstrip and access roads (Linear Infrastructure) was undertaken in December, 2013. This review identified private parcels, a Land Act licence, rights of way, reserves/notations and a transfer of administration/control area; grazing tenures; forest tenures; a forest recreation site; traplines; guide outfitter areas; a wildlife management area; an agriculture land reserve; and third-party mineral tenures overlapping or in close proximity to proposed linear infrastructure routes

- New Gold's 100% interest in the Blackwater claim block is subject to four net smelter return (NSR) agreements:
  - Mineral claim 515809 (Dave Option): The optionors retained a 2.5% NSR. New Gold has purchased 40% of the Dave NSR royalty for $1,000,000, and a 1.5% NSR royalty remains. The claim covers a portion of the Blackwater deposit
  - Mineral claim 515810 (Jarrit Option): The optionors retained a 2% NSR royalty. New Gold has purchased half of the Jarrit NSR royalty for $1,200,000, and a 1% NSR royalty remains. The claim covers a portion of the Blackwater deposit.
  - Mineral claims 637203, 637205, and 637206 (JR Option): The optionors retained a 3% NSR royalty. New Gold may purchase two-thirds of the JR Claims NSR royalty for $1,000,000 at any time, such that a 1% NSR royalty would remain
  - Mineral claim 835014 (PS Option): The optionor retains a 2% NSR royalty, of which New Gold may purchase half for $1,000,000
- No other material encumbrances that are recorded against the Blackwater claims and are still active have been identified

25.3 Geology and Mineralization

- Knowledge of the deposit settings, lithologies, and structural and alteration controls on mineralization, and the mineralization style and setting is sufficient to support Mineral Resource estimation
- The deposit is considered to be an example of a low to intermediate epithermal system
- The deposit type used for exploration targeting is appropriate to the mineralization identified and the regional setting.

25.4 Exploration and Drilling

- The exploration programs completed to date are appropriate to the style of the known mineralization within the Project area
- Given the lack of bedrock exposure, no detailed surface geologic mapping has been carried out over the main deposit or surrounding area by New Gold, and geologic information has been obtained primarily by core drilling. Areas of shallow overburden near the centre of the deposit are potential targets for bulk sampling or trench mapping/sampling programs
- Geophysical surveys have proven useful to assist in interpreting deposit geology and identifying drill targets for future exploration

- The resolution and accuracy of the surface topography as interpreted from the 2011 LiDAR survey are considered sufficient to support detailed Project studies
- A total of 1,149 core drill holes (357,507 m) have been drilled in the Project area between 2009 and January 2013. Of this total, 134 were completed by Richfield, and 1,015 by New Gold. Drilling by parties other than Richfield and New Gold is referred to as legacy drilling; this drilling, completed between 1981 and the end of 2006 consists of 81 holes totalling 7,633 m. The legacy drilling is not used in resource estimation
- Gold and silver mineralization occurs within an irregularly-shaped system of stockwork and disseminated sulphides that strikes approximately east–west and dips moderately to the north. Depending on the inclination of an individual drill hole, and the local dip of mineralization, drill intercept widths are approximately equivalent to true widths.
- The quantity and quality of the lithological, geotechnical, collar, and down-hole survey data collected in the 2009–2013 exploration and infill drill programs are sufficient to support Mineral Resource estimation. There are no known sampling or recovery factors that could materially impact the accuracy and reliability of the results

25.5 Sample Preparation and Analysis

- Sampling methods are acceptable, meet industry-standard practice, and are acceptable for Mineral Resource and Mineral Reserve estimation and mine planning purposes.
- Bulk density determination procedures are consistent with industry-standard procedures, and there are sufficient bulk density determinations to support tonnage estimates
- Analysis is performed by accredited third-party laboratories
- Data from holes drilled between 1981 and 1994 have no documented QA/QC information, and they are not deemed acceptable for use in resource estimation.
- Later drill programs included insertion of blank, duplicate and CRM samples
- Quality control procedures implemented in 2012 for silver analysis shows acceptable levels of precision and accuracy for silver results. Previous concerns regarding the accuracy and precision of pre-2012 silver results due to lack of comprehensive silver QC is mitigated by the 2012 QC results.

25.6 Data Verification

- Verification is performed on all digitally-collected data on upload to the main database, and includes checks on surveys, collar co-ordinates, lithology data, and assay data. The checks are appropriate, and consistent with industry standards
- The process of data verification performed by the QPs indicates that the data collected from the Project during the 2009 to 2013 work programs adequately reflect deposit dimensions, true widths of mineralization, and the style of the deposits, and adequately support the geological interpretations, and the analytical and database quality
- QA/QC with respect to the results received to date for the 2009–2013 exploration programs is acceptable, and protocols have been well documented. A lower level of QA/QC exists for the seven drill holes completed between 2005 and 2006, but the data are considered to be sufficiently reliable to support a resource estimate.

25.7 Metallurgical Testwork

- An extensive metallurgical testwork program was carried out over the period 2008 to 2013 on samples that were composited to represent process plant feed in the mine development plan. These samples were obtained from two primary sources: a dedicated seven-hole metallurgical drilling program, and composites from 324 exploration drill holes
- Testwork was performed by industry recognized metallurgical laboratories
- A PEA was completed in 2012 that indicated that WOL was the most promising flowsheet. Mineralogical and diagnostic leach testing indicated that the primary areas of investigation required to optimize WOL processing were primary grind size, reagent addition, and leach retention time
- Information from further metallurgical process definition and variability testing was used to update the gold and silver recovery models for the 2013 Feasibility Study and to develop the recovery equations used within the mining model
- Estimated recoveries over the LOM are Au recovery of 86.6% and Ag recovery of 49.1%
- No elements that could be considered deleterious in the proposed process were identified from the testwork.

25.8 Mineral Resource Estimate

- Estimations of Mineral Resources for the Project conform to industry best practices, and meet the requirements of CIM (2010)

- The estimate is based upon a geologic block model that incorporates 286,966 individual assays from 309,516 m of core from 1,003 drill holes at a nominal drill hole spacing ranging from 25 m to 50 m. The drill hole database was supported by some 80,000 quality assurance/quality control (QA/QC) check assays
- Due to lack of QA/QC and accurate survey information, holes drilled before 2009 were not used for statistical analysis or grade estimation of the Mineral Resource, but were used in forming the lithological wire frame construction
- The Mineral Resource estimate is based on reasonable assumptions of economic extraction assuming open pit mining methods. AuEq cutoff values of 0.3 and 0.4 g/t and metal prices of US$1,400/oz Au and US$28/oz Ag were used in the pit shell. Additional assumptions included variable pit slope angles ranging from 23° to 43°; a mining cost of $1.64/t for mineralized material, waste mining cost of $1.94/t; ore processing cost of $6.85/t; sustaining capital for the mill of $0.18/t; G&A cost of $1.25/tonne; allocation for the tailings facility costs of $0.60/t; royalties at 1.5% of revenue; refining costs of 0.1% of Revenue, gold recovery of 88.0% (Oxide), 85.0% (Transition and Sulphide); gold recovery for stockpile material grading between of 0.3–0.4 g/t Au of 79%; silver recovery of 64.0% (Oxide), 58.0% (Transition), 44.0% (Sulphide). Silver recovery for stockpile material grading 0.3–0.4 g/t Au of 37%
- Measured and Indicated Mineral Resources total 397 Mt at 0.74 g/t Au and 5.5 g/t Ag. Inferred Mineral Resources are estimated at 17.6 Mt grading 0.66 g/t Au and 4 g/t Ag
- The following factors could affect the Mineral Resources: commodity price and exchange rate assumptions; pit slope angles and other geotechnical factors; assumptions used in generating the LG pit shell, including metal recoveries, and mining and process cost assumptions

25.9 Mineral Reserve Estimates

- Mineral Reserves have been developed from Mineral Resources by including geological, mining, processing and economic factors. The reserves are classified in accordance with the 2010 CIM Definition Standards for Mineral Resources and Reserves
- The Blackwater Mineral Reserve totals 344.4 Mt at 0.74 g/t Au and 5.5 g/t Ag.

25.10 Mine Plan

25.10.1 Geotechnical

- Open pit geotechnical conditions are based on a review of geomechanical information, stability analyses results, the mining equipment to be utilized, and experience from similar open pit operations in the Project region
- Surficial material – Bench face angles of 27° and height of 36 m with 20 m wide step-outs.
- Broken zone – Bench face angles of 60° and inter-ramp slope angles of 39° should be achievable. Maximum inter-ramp height should be limited to 120 m. Slope depressurization will be required in these areas.
- Competent rock – Bench face angles of 65° and inter-ramp slope angles of 45 to 46° should be achievable. Maximum inter-ramp heights should be limited to 200 m. The competent rock domain will be stable without groundwater depressurization.

25.10.2 Hydrology

- Water inflows to the Blackwater open pit will include both groundwater and surface water runoff. The contributions from groundwater will progressively increase as the pit extends below the groundwater table
- The 1-in-100 year return period storm has been used to size the pit surface water dewatering system and was estimated to be approximately 142,000 m^3
- A combination of in-pit and perimeter pumping wells will be used for slope depressurization and pit dewatering

25.10.3 Pit Optimisation and Production Schedule

- Large-scale open pit mining will provide process plant feed at a nominal rate of 60,000 t/d, or 21.9 million tonnes per year (Mt/a). Annual mine production of ore and waste will peak at 91.7 Mt/a. Ore grade is maximized in the early years of the Project to reduce the payback period. Lower-grade ore is stockpiled and processed in later years
- The average LOM stripping ratio is 2.00:1. The operational stripping ratio, excluding waste stripping during the development phase, is 1.88:1
- A total of four phases were developed and incorporated into the mine production schedule. Some of these phases exist only for a short time, usually to increase ore grade or to produce material suitable for the TSF dam construction

- The production schedule is based on Proven and Probable Mineral Reserves; Inferred Mineral Resources within the pit shell have been set to waste
- The production schedule is based on 24-hour, year-round mining operations with limited weather delays and shutdowns. The total annual schedule is 355 operating days, or 8,520 hours. The mine will operate on two 12-hour shifts per day
- The Blackwater pit is designed with 12 m bench heights for both ore and waste
- Loading will be carried out by a combination of hydraulic and cable shovels backed up by a large front-end loader. Haul distances for both ore and waste will vary significantly, and therefore truck fleet requirements vary; 290 t capacity trucks have been selected. Drilling and blasting of approximately 7 Mt (dry) per month will be required to maintain production levels. The exact size and configuration of the support equipment fleet will be finalized at the detailed engineering stage
- Mining operations will commence during the construction phase and by Year 1 the equipment fleet will comprise four 200 /250 mm diesel blast hole drills, two 40 m^3 hydraulic shovels, one 28 m^3 front-end loader, and fourteen 290 tonne trucks. These will be supplemented with backup graders, and track and rubber-tired dozers
- The total tonnage mined increases approximately 90 Mt/a in Years 3 through 10, and the mining fleet increases with the addition of four blast hole drills, one electric cable shovel, and 13 more haul trucks

25.11 Process

- The process plant facility will consist of a primary crushing plant, a coarse ore stockpile (COS), a SAG/ball mill/crusher (SABC) grinding circuit, pre-leach thickening, whole ore cyanide leaching , carbon-in-pulp (CIP) recovery of precious metals from solution, elution of precious metals from carbon, and recovery of precious metals by electrowinning followed by smelting to doré. The plant will also have facilities for carbon regeneration, tailings thickening, and cyanide destruction.
- The overall design utilizes large equipment while maintaining a simple and conventional flowsheet.
- The bulk of the water requirements for the process plant will be met with reclaim water recovered from the TSF and thickeners with water also re-used from air compressor cooling and elution column heat exchangers.
- Bulk and packaged reagents will be trucked to site and stored for use in the process plant
- Key process equipment will consist of the following:

- a primary crushing plant with a 1,520 x 2,870 mm (60" x 113") gyratory crusher
- a SAG/ball mill/crusher grinding circuit:
  - two 11.0 m diameter x 6.7 m (36' x 21.5') EGL, 17 MW SAG mills
  - two 8.2 m diameter x 12.8 m (27' x 42') EGL, 17 MW ball mills
  - two 1,000 kW pebble crushers
- whole ore leaching and carbon-in-pulp circuit:
  - 24 leach tanks of 18 m diameter
  - two trains of seven 400 m^3 capacity CIP tanks
  - two 80 m diameter thickeners
- three cyanide destruction vessels using the SO_2/air process on leach CIP residue before transfer in a single stream to the TSF.

25.12 Onsite Infrastructure

- The Blackwater site will be accessed via the Kluskus Forest Service Road. New Gold will undertake road improvements over a small section of the FSR. In addition, a new 16 km access road will be constructed from 124 km of the FSR to the plant site. New Gold will likely become the primary operator and user of the FSR by the time the Project is complete, considering that reduced logging operations are anticipated in the area at that time
- An airstrip will be built for use during the construction phase of the Project to increase accessibility and reduce travel time to the Project site
- An 880-person construction camp will be erected on site, which, together with the expansion of the existing camp from 250 to 426 persons, will provide accommodations for contractors and construction management staff. It will not be used in the operations phase. For operations a high-quality modular camp with a capacity of 420 persons will be constructed on site
- Onsite infrastructure to support mining and milling activities will include a primary crusher, reclaim conveyors, mill building, elution and refinery building, whole ore leach tanks, main truck shop, administration and emergency services buildings, explosives storage facility and fuel farm
- Power will be supplied to the Blackwater site by connection to the BC Hydro grid. A 139.5 km long 230 kV transmission line will be constructed from the BC Hydro Glenannan Substation to the Blackwater site
- Wells will be developed near the new camp area to supply water for the temporary and operations camps. The water will be treated and distributed around the camp site for domestic use.

- Fresh water for the Project will be sourced from Tatelkuz Lake, approximately 20 km northeast of the mine site. The fresh water will be used to mitigate flow reductions in Davidson Creek downstream of the TSF and for Project operations as required.

25.13 Waste Characterisation

- Some of the waste rock and the tailings are classified as potentially acid generating (PAG) and/or metal leaching (ML). Waste rock was classified based on its Neutralizing Potential Ratio (NPR) and ML potential as follows:
  - PAG1 – NPR ≤ 1.0 (PAG)
  - PAG2 – 1.0 < NPR ≤ 2.0 (PAG)
  - NAG3 – NPR > 2.0 and Zn ≥ 1,000 ppm (NAG-ML)
  - NAG4 – NPR > 2.0 and 600 ≤ Zn <1,000 ppm (NAG)
  - NAG5 – NPR > 2.0 and Zn < 600 ppm (NAG).
- PAG1 and PAG2 waste rock will be stored and submerged in the TSF within one year of mining to prevent the formation of acid rock drainage (ARD).
- NAG3 waste rock will be stored and submerged in the TSF within three to five years of mining to reduce metal leaching.
- NAG4 waste rock will be used for construction on site and in the downstream shell of TSF Dam D as required.
- NAG5 will be used for construction and in the downstream shell of TSF Dam D.
- Overburden is classed as non acid generating and will be used for construction on and off site
- Sulphide and transition ore tailings are classified as potentially acid generating and will be kept saturated or submerged within the TSF during operations to prevent ARD. Oxide tailings exhibit low ARD and ML potential and will be placed as the upper layer in TSF Site C.

25.14 Waste Rock Storage Facilities

- The East waste facility, to be sited east of the pit, is the primary storage location for overburden and NAG5 waste not required for TSF construction. The overall dump has a capacity of approximately 50 Mt and will be constructed in a series of lifts to enhance stability and minimize resloping requirements at closure.
- The West waste facility, to be located west of the pit, is the primary storage location for overburden and NAG4 waste not required for TSF construction. The West waste facility has a capacity of approximately 87 Mt

- The mined overburden will be distributed between the two waste facilities based on the ratio of rock to overburden placed in a given period. A minimum of 10% of waste rock will be placed with the overburden to provide plating and maintain the overall integrity of the waste facilities.

25.15 Tailings Storage Facility

- The TSF was designed to permanently store tailings, PAG1 and PAG2 waste rock, and NAG3 waste rock generated during the operation of the mine. The TSF has two adjacent sites, Site C and Site D
- The design of the TSF is supported by extensive geotechnical site investigations. The facility has been designed to contain 453 Mm^3 of tailings and waste rock material and will require 73.9 Mm^3 of construction material, about 65.8 Mm^3 or 89% of which will be waste rock and overburden from the open pit
- The TSF embankments will be engineered, water-retaining, zoned earthfill/rockfill dams with a compacted low-permeability core and appropriate filter/transition zones. A total of three embankments will be constructed across the two sites. The TSF Site C and Site D dams will be expanded using the downstream and centreline construction methods, respectively
- Site runoff water will be stored on site within the TSF, and the supernatant will be recycled back to the process plant. Additional water may be required to maintain a water cover over PAG waste rock and tailings in the TSF facility during some years of operation if conditions are drier than average. Fresh water for the mill or additional makeup water for the TSF to support operations will be pumped from Tatelkuz Lake.

25.16 Water Management

- All drainage from the mine will flow by gravity into the TSF to simplify water management, spill control, and mine closure
- During operations, drainage from the low-grade and coarse ore stockpiles may become acidic with elevated metals content; the drainage will be collected and neutralized with lime to increase the pH and precipitate metals before disposal in the TSF. Pit water is predicted to be of neutral pH with relatively low metals content during operations; it will be pumped to a small holding/monitoring pond, which will overflow to the TSF.
- Tailings slurry from the mill will be treated using the SO_2/air process. Laboratory testwork showed the process will be effective in lowering cyanide and metal concentrations.

25.17 Environmental Considerations

- New Gold conducted extensive environmental baseline studies and is preparing comprehensive environmental management plans for the Project. The environmental management plans have been integrated into the mining, processing, water, and waste management designs for the Project.
- Potential effects on fish due to a flow reduction in Davidson Creek downstream of the TSF will be mitigated by pumping water from Tatelkuz Lake and releasing it to the creek. A Fish Habitat Compensation Plan will be implemented such that no net loss of fish habitat due to direct mine footprint effects will occur.
- New Gold has developed a whitebark pine management plan to minimize negative effects on this species.
- The environmental assessment process for the Project officially started in October 2012 with the acceptance of the Project Description by both the BC EAO (*BCEAA*) and the CEAA.

25.18 Closure Plan

- The closure plan employs proven practices and is not dependent on long-term active treatment and monitoring. All Project components will be decommissioned and reclaimed according to best industry practices and provincial and federal regulations.
- Proposed end land use objectives for mine closure are wildlife habitat and return of the land for traditional use by Aboriginal groups
- The estimated closure and reclamation cost for the Project, discounted to the last year of operations, is approximately $86 million, including progressive reclamation conducted during the mine life. The estimated salvage value of the Project is about $78.4 million.

25.19 Permitting

- A large number of federal or provincial permits are required for mine construction and subsequent operations. The process of obtaining the provincial permits is partially coordinated by the Ministry of Forests, Lands and Natural Resource Operations through a Mine Development Review Committee. Some of the provincial permits have legislated timelines, while others do not. The federal permits and authorizations do not have legislated timelines, and some permits originating from federal agencies may not be issued within the review period set out in the B.C. legislation.
- Key permits, licences, and authorizations required to construct the mine include:

- BC Mines Act permit
- *BC Environmental Management Act* permits for discharges from the Project to surface waters and for air emissions
- Schedule 2 amendment under the federal *Fisheries Act,* which is required to place mine waste into a natural water body that is frequented by fish
- Section 35(2) authorization under the federal *Fisheries Act* for a harmful alteration, disruption, or destruction of fish habitat
- Section 23 application or exemption under the federal *Navigable Waters Protection Act* for effects on navigable waters
- Licences or approvals under the *BC Water Act* for the use of water
- Mining Lease.

25.20 Social Licence

- New Gold is consulting with First Nations, government, and other stakeholders that could potentially be affected by the Project.

25.21 Capital Costs

- The total estimated development capital cost for the Project is $1,963 million inclusive of a $200.4 million contingency. The development capital cost equates to $280 ($US266) per recoverable gold ounce over the life of the Project
- Total LOM sustaining capital is estimated to be $681 million, equivalent to an average of $97 ($US92) per recoverable gold ounce.

25.22 Operating Costs

- LOM operating costs are estimated at $14.49/t of ore milled, and $12.48/t of ore milled after accounting for silver credits. Total LOM all-in sustaining cash costs are estimated at $14.47/t of ore milled.
- The average LOM cost after silver credits and royalty is US$578/oz Au produced. The average LOM all-in sustaining cash cost is US$670/oz Au produced

25.23 Financial Analysis

- For the 17-year mine life and 341 Mt mill feed, the following pre-tax Base Case financial parameters were calculated:
  - $1,044 million NPV (pre-tax, Year -2) at 5.0% discount rate.
  - 11.3% IRR
  - 6.2-year payback on $1,963 million capital cost.

- The after-tax financial parameters are
  - $616 million NPV (pre-tax, Year -2) at 5.0% discount rate
  - 9.3% IRR
  - 6.4-year payback on $1,963 million capital cost.

- Sensitivity analysis was performed on the Project using metal price, exchange rate, operating costs and capital costs. The Project is more sensitive to changes in the gold price and the USD:CAD exchange rate than to changes in capital or operating costs.

25.24 Risks and Opportunities

- The major risks to the Project were identified as:
  - Changes to metal prices and exchange rate assumptions
  - Capital cost growth
  - Increases in operating costs
  - Productivity assumptions
  - Dilution control
  - Presence of high-grade silver in the mill feed
  - Integration of mining operations and the TSF construction
  - Permitting delays
  - Lack of social licence affecting permit grant

- Project opportunities included
  - Delineation of additional mineralization that could support higher-confidence resource categories through additional drilling
  - Use of a trolley assist system later in the mine life
  - Assessment of methods to reduce waste mining costs
  - Use of oxygen rather than compressed air for cyanide leaching and cyanide detoxification
  - Pre-crushing the SAG mill feed to a finer feed size
  - Value engineering initiatives.

25.25 Conclusions

AMEC considers that the scientific and technical information available on the Project can support proceeding with additional data collection, trade-off and engineering work and preparation of more detailed studies. However, the decision to proceed with a mining operation on the Project is at the discretion of New Gold.

26.0 RECOMMENDATIONS

There are no meaningful recommendations arising from the feasibility study. The decision to proceed with a mining operation on the Project is at the discretion of New Gold. New Gold is intending to continue with activities that will support the permitting process for the Project.

These activities form a two-phase work program, which will cost approximately $8–10 million in support of environmental studies and long-lead items required for Project permitting. The phases will be conducted concurrently, with the aim of achieving full Project permitting at the end of the program. Each phase is estimated to be $4–5 million to complete.

26.1 Phase 1, Environmental Impact Assessment

The aim of the Phase 1 work is completion and filing of the EIS including ongoing environmental baseline work.

Key aspects which will be included in these areas are:

- Completion of ongoing water quality, hydrology, hydrogeology, geochemical and metrological baseline work
- Completion of a Federal Environmental Impact Statement; and,
- Completion of a provincial environmental Assessment Certificate Application

26.2 Phase 2, Provincial and Federal Permitting

Phase 2 is designed around completion of engineering site investigations and design to enable completion of key Provincial (e.g. Mines Act) and Federal (e.g. Fisheries Act) permit applications.

Key aspects which will be included in these areas include

- Schedule 2 Amendment for TSF construction and operation
- Navigation Protection Act for filling of potentially navigable waters
- Fisheries Act Authorizations for impacts to fish habitat
- Provincial Mines Act to allow construction and operation of the proposed mine
- Timber cruise for transmission line, airstrip and mine site clearing
- Sewage treatment system for camp development.

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