



NEW GOLD INC.

**TECHNICAL REPORT ON THE
RAINY RIVER MINE, ONTARIO, CANADA**

NI 43-101 Report

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July 25, 2018

CAUTIONARY NOTE WITH RESPECT TO FORWARD LOOKING INFORMATION

Certain information and statements contained in this report are “forward looking” in nature. All information and statements in this report, other than statements of historical fact, that address events, results, outcomes, or developments that New Gold and/or the Qualified Persons who authored this report expect to occur are “forward-looking statements”. Forward-looking statements are statements that are not historical facts and are generally, but not always, identified by the use of forward-looking terminology such as “plans”, “expects”, “is expected”, “budget”, “scheduled”, “estimates”, “forecasts”, “intends”, “anticipates”, “projects”, “potential”, “believes” or variations of such words and phrases or statements that certain actions, events or results “may”, “could”, “would”, “should”, “might” or “will be taken”, “occur” or “be achieved” or variations, including the negative connotation, of such terms. Forward-looking statements include, but are not limited to, statements with respect to anticipated production rates; grades; projected metallurgical recovery rates; infrastructure, capital, operating and sustaining costs; the projected life of mine; the proposed pit design phase development and potential impact on cash flow; estimates of Mineral Reserves and Mineral Resources and realization of such Mineral Reserves and Mineral Resources; the future price of gold; government regulations; the maintenance or renewal of any permits or mineral tenures; 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 significant disruptions affecting the operation of the mine; (2) the availability of certain consumables and services and the prices for diesel, natural gas, cyanide, fuel oil, electricity and other key supplies being approximately consistent with current levels; (3) labour and materials costs increasing on a basis consistent with current expectations; (4) that all environmental approvals, required permits, licenses and authorizations will continue to be held on the same or similar terms and obtained from the relevant governments and other relevant stakeholders within the expected timelines; (5) certain tax rates; (6) the timelines for exploration activities; and (7) assumptions made in Mineral Resource and 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, decrease of future gold prices; cost of labour, supplies, fuel and equipment rising; adverse changes in anticipated production, including discrepancies between actual and estimated production, Mineral Reserves, Mineral Resources and recoveries; exchange rate fluctuations; title risks; regulatory risks, and political or economic developments in the United States; changes to tax rates; risks and uncertainties with respect to obtaining necessary permits, land use rights and other tenure from the State and private landowners 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, in New Gold’s latest Annual Information Form, dated March 29, 2018, Management’s Discussion and Analysis for the three and six months ended June 30, 2018 and its other SEDAR filings from time to time. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. All forward-looking statements herein are qualified by this cautionary statement. Accordingly, readers should not place undue reliance on forward-looking statements. 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.

CAUTIONARY NOTE TO U.S. READERS CONCERNING ESTIMATES OF MEASURED, INDICATED AND INFERRED MINERAL RESOURCES

Information concerning the Rainy River Mine 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 accordance with National Instrument 43-101 (“NI 43-101”) under guidelines set out in the Canadian Institute of Mining, Metallurgy and Petroleum (“CIM”) Standards on Mineral Resources and Mineral Reserves adopted by the CIM Council on November 27, 2010. 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 great 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 category. Readers are cautioned not to assume that all or any part of an “Inferred Mineral Resource” exists, or is economically or legally mineable.

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1 SUMMARY

EXECUTIVE SUMMARY

The purpose of this report is to provide an update, for public disclosure, of the Rainy River Mine (Rainy River or the Mine) operations, near Fort Frances, Ontario, Canada, and to support the Mineral Resource and Mineral Reserve estimates for the Mine as of June 30, 2018. This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

New Gold Inc. (New Gold) is an intermediate gold mining company with four producing assets. The New Afton and Rainy River Mines in Canada, the Mesquite Mine in the United States and the Cerro San Pedro Mine in Mexico (which transitioned to residual leaching in 2016) provide the company with its current production base. In addition, New Gold owns 100% of the development stage Blackwater project located in British Columbia, Canada.

New Gold completed a takeover of Rainy River Resources Ltd. (RRR) on October 15, 2013. The Rainy River Mine is 100% owned and operated by New Gold. The Mine commenced processing ore on September 14, 2017 and subsequently announced its first gold pour on October 6, 2017. Commercial production was achieved in mid-October. From an accounting perspective, New Gold recognizes commercial production effective November 1, 2017. In 2017, the Mine produced 37,047 ounces of gold.

Mineral Resource and Mineral Reserve estimates have been created utilizing acceptable methodologies and the classification of Measured, Indicated, and Inferred Mineral Resources, stated in Table 1-1, and Proven and Probable Mineral Reserves, stated in Table 1-2, meet the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves adopted by the CIM Council on May 10, 2014 (CIM (2014) definitions).

Open pit Mineral Resources are reported at cut-off grades of 0.3 g/t to 0.5 g/t gold equivalent (AuEq) for Low Grade and 0.5 g/t AuEq for Direct Processing material. The underground Mineral Resources are reported at a cut-off grade of 2.0 g/t AuEq. Measured and Indicated Mineral Resources exclusive of reserves are estimated to total 63.109 million tonnes at grades

of 1.06 g/t Au and 3.8 g/t Ag, containing 2,142,000 ounces of gold and 7,651,000 ounces of silver. Inferred Mineral Resources are estimated to total 8.871 million tonnes at grades of 1.10 g/t Au and 2.4 g/t Ag, containing 313,000 ounces of gold and 672,000 ounces of silver. All Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

TABLE 1-1 MINERAL RESOURCES – EFFECTIVE JUNE 30, 2018

	Tonnes (000s)	Grades, g/t		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Direct Processing					
Open Pit (OP)					
Measured	2,996	1.14	5.7	109	550
Indicated	26,541	1.12	3.4	957	2,921
OP Measured and Indicated	29,537	1.12	3.7	1,066	3,471
Inferred	3,697	1.06	3.2	126	385
Underground (UG)					
Measured	-	-	-	-	-
Indicated	7,934	3.06	8.6	780	2,206
UG Measured and Indicated	7,934	3.06	8.6	780	2,206
Inferred	1,215	3.59	2.7	140	107
Low Grade (LG)					
Measured	2,462	0.35	3.3	28	261
Indicated	23,175	0.36	2.3	268	1,713
LG Measured and Indicated	25,637	0.36	2.4	296	1,974
Inferred	3,959	0.37	1.4	47	180
Combined Mineral Resources					
Measured	5,458	0.78	4.6	137	811
Indicated	57,651	1.08	3.7	2,005	6,840
Total Measured and Indicated	63,109	1.06	3.8	2,142	7,651
Total Inferred	8,871	1.10	2.4	313	672

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated using long-term metal prices of US\$1,375 per ounce gold, US\$19.00 per ounce silver and a C\$/US\$ exchange rate of 0.77. Metal recoveries of 95% for Au and 70% for Ag were used.
3. Mineral Resources are reported at cut-off grades for direct processing material of 0.50 g/t AuEq for open pit and 2.0 g/t AuEq for underground and between 0.30 g/t and 0.5 g/t AuEq for low grade resources. The gold equivalency formula is as follows: $AuEq\ g/t = Au\ g/t + ((Ag\ g/t * 19 * 70) / (1375 * 95))$.
4. Bulk density ranges from 2.70 t/m³ to 3.08 t/m³.
5. Mineral Resources are exclusive of Mineral Reserves.
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability.
7. Open pit Mineral Resources are constrained by a conceptual pit shell.
8. Totals may not add due to rounding.

The Qualified Person (QP) is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

Open pit Mineral Reserves are reported at cut-off grades of 0.3 g/t AuEq and 0.5 g/t AuEq for Low Grade and Direct Processing ores, respectively. The underground Mineral Reserves are reported at a cut-off grade of 2.2 g/t AuEq. Proven and Probable Mineral Reserves are estimated to total 120.451 million tonnes at grades of 1.09 g/t Au and 3.2 g/t Ag, containing 4,220,000 ounces of gold and 12,398,000 ounces of silver.

TABLE 1-2 MINERAL RESERVES – EFFECTIVE JUNE 30, 2018

	Tonnes (000s)	Grades, g/t		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Direct Processing Reserves					
Open Pit (OP)					
Proven	21,468	1.22	2.5	842	1,739
Probable	50,409	1.15	3.1	1,860	5,069
Total OP P&P (direct processing)	71,878	1.17	2.9	2,701	6,807
Underground (UG)					
Proven	-	-	-	-	-
Probable	8,954	3.55	9.5	1,021	2,728
Total UG P&P (direct processing)	8,954	3.55	9.5	1,021	2,728
Low Grade Reserves					
Proven	7,407	0.38	2.0	90	478
Probable	26,900	0.36	2.4	308	2,086
Total OP LG P&P	34,307	0.36	2.3	398	2,564
Stockpiles					
Proven	5,313	0.58	1.8	99	299
Probable	-	-	-	-	-
Total Stockpile P&P	5,313	0.58	1.8	99	299
Combined OP & UG					
Proven	34,189	0.94	2.3	1,031	2,516
Probable	86,263	1.15	3.6	3,189	9,882
Total OP & UG P&P	120,451	1.09	3.2	4,220	12,398

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.

2. Effective date of Mineral Reserves is June 30, 2018; estimated by Nicholas Kwong, P. Eng., an employee of New Gold Inc.
3. The open pit Mineral Reserves were estimated using an operating pit design determined from Measured and Indicated Mineral Resources and supporting a mine plan of 27 ktpd. The design is based on a gold price of US\$1,275 per troy ounce and a silver price of US\$17 per troy ounce.
4. Open pit Mineral Reserves were estimated at a cut-off grade of 0.5 g/t AuEq for direct processing and 0.3 g/t AuEq for low grade material for stockpiles (see note 6 below for basis of AuEq).
5. Underground Mineral Reserves were estimated at a cut-off grade of 2.2 g/t AuEq for stoping and 0.53 g/t AuEq for development (see note 6 below for basis of AuEq).
UG cut-off grade assumptions:
 - Au price US\$1,275 per troy ounce, Ag price US\$17 per troy ounce
 - Exchange rate 1.3 C\$: 1 US\$ (1 C\$ = 0.77 US\$)
 - Planned hangingwall and footwall dilution of 0.5 m and 0.25 m respectively; unplanned dilution 4.3%
 - Average mining recovery: 92%
 - Mill Au and Ag recovery: 95% and 60% respectively
 - Breakeven cost of \$107/t of mill feed, inclusive of costs for mining, processing, G&A, refining & transport, royalties and sustaining capital allowance.
6. AuEq is equal to Au g/t + 0.0084*Ag f/t.
7. Totals may not add due to rounding.
8. Tonnes and grades are in metric units.

The QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimate.

CONCLUSIONS

The QPs offer the following interpretations and conclusions:

GEOLOGY

The Rainy River deposit is an auriferous volcanogenic massive sulphide (VMS) system with a primary syn-volcanic source and possibly a secondary syn-tectonic mineralization event.

MINERAL RESOURCES

With regard to the quality assurance/quality control (QA/QC) monitoring programs in place at Rainy River:

- Results of gold and silver certified reference materials (CRMs) indicate that the primary laboratory measures gold and silver in samples accurately.
- Results of the blank sample program indicate that there is little to no contamination of samples during sample preparation.
- Results of the duplicate sample program indicate no significant bias present in the preparation or analysis of the samples.

The sample preparation, analysis, and security procedures at Rainy River are adequate for use in the estimation of Mineral Resources.

The Mineral Resource database is sufficiently reliable for grade modelling and Mineral Resource estimation.

Open pit Mineral Resources are reported at cut-off grades of 0.3 g/t AuEq to 0.5 g/t AuEq for Low Grade and 0.5 g/t AuEq for Direct Processing material. The underground Mineral Resources are reported at a cut-off grade of 2.0 g/t AuEq.

Measured and Indicated Mineral Resources are estimated to total 63.109 million tonnes at grades of 1.06 g/t Au and 3.8 g/t Ag, containing 2,142,000 ounces of gold and 7,651,000 ounces of silver. Inferred Mineral Resources are estimated to total 8.871 million tonnes at grades of 1.10 g/t Au and 2.4 g/t Ag, containing 313,000 ounces of gold and 672,000 ounces of silver. The Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

MINERAL RESERVES

Open Pit Mineral Reserves

The open pit Mineral Reserves are estimated to be 71.88 million tonnes of direct processing ore grading 1.17 g/t Au and 2.9 g/t Ag, 34.3 million tonnes of low grade ore grading 0.36 g/t Au and 2.3 g/t Ag, and 5.3 million tonnes of above-ground stockpiled ore grading 0.58 g/t Au and 1.8 g/t Ag. The Mineral Reserves include factors for the cut-off grade, dilution, and minimum mining width.

The open pit Mineral Reserves are stated at a cut-off grade of 0.5 g/t Au for Direct Processing and 0.3 g/t Au for the Low Grade stock pile.

Open pit mining utilizes conventional truck and hydraulic shovel with 10 m high benches in the open pit.

The design parameters, development and mining method are considered to be appropriate for the deposit.

Underground Mineral Reserves

The underground Probable Mineral Reserves are estimated to be 8.95 million tonnes grading 3.55 g/t Au and 9.5 g/t Ag. The Mineral Reserves include factors for cut-off grade, dilution, minimum mining width, and mining extraction.

The underground Mineral Reserves are stated at a cut-off grade of 2.2 g/t Au, which is higher than the breakeven cut-off grade of 1.7 g/t Au. The higher cut-off grade was used to generate an underground production schedule that terminates when the open pit processing is complete.

The underground mining will utilize long hole open stoping for areas less than ten metres wide and cut and fill stoping with cemented aggregate fill for areas wider than ten metres.

The mine will be a decline access mechanized mine using medium scale mechanized equipment.

The design parameters, development and mining method are considered to be appropriate for the deposit.

MINING

Open Pit Mining

The life of mine (LOM) plan was developed after evaluating multiple iterations of mining sequence including combinations of mining sequencing with different mining rates, mine sequencing alternatives and other economic parameters.

Mining at Rainy River is conducted using conventional open pit mining methods. The drill, blast, load and haul cycle uses large open pit equipment matched to the hydraulic shovel and diesel electric haul truck fleet. Waste material is categorized and stored in specific waste dump locations. The typical waste designations are potential acid generating (PAG), non-potentially acid generating (NPAG), and overburden. Ore is designated in direct feed to the processing plant and low grade material is stored in a stock pile for end of mine life processing.

The planned equipment fleet is appropriate for the production rates, schedule, operating conditions and LOM plan.

Underground Mining

The planned single heading advance rate and the rapid production build up to the design production rate pose a moderate risk to the project schedule. There are multiple stoping areas that will be operated concurrently, which will provide significant flexibility in the event of delays in any one area.

Additional time may be required in the production schedule to allow for infill drilling and analysis prior to the commencement of production in an area.

Areas where the dip is less than 55° may suffer some additional ore loss and/or dilution, or higher costs to recover all the ore in the stope designs.

Ground conditions are generally expected to be “Fair” to predominantly “Good” based on Feasibility Study level geotechnical investigations. Geotechnical analyses and modelling support the planned open stope designs. In some non-entry areas, large hangingwall exposures will exist, supported by relatively small pillars (as designed). Controlled and uncontrolled failures of the hangingwall or pillars in these locations may generate additional dilution or loss of ore.

Where cemented backfill is used, an aggregate fill with an average 4% cement content has been assumed. Waste rock fill with no cement may also be used in stopes where there will be no subsequent mining against the fill.

The planned equipment fleet is appropriate for the mine as planned; ventilation benefits may be realized through the use of electric (battery and/or trolley) units in place of diesel units.

METALLURGICAL TESTING AND DEVELOPMENT OF DESIGN CRITERIA

The metallurgical testing programs supporting the engineering of the Rainy River process facilities were performed in support of the Feasibility Study prepared for New Gold in 2014, which is the primary source for this report. The testing programs were extensive and provided the design information required.

The design criteria developed for the process design engineering were sourced from the metallurgical testwork and from information supplied by equipment suppliers.

PROCESS

The Rainy River process plant was commissioned in September 2017 and has been working through initial operating and maintenance issues and plant optimization. The production rates are approaching the targeted 21,000 tonnes per day (tpd) and have been achieved on a daily basis, however, plant availability has been lower than projected and so the average production rates on a monthly basis have been below target.

Gold grades have been slightly lower than projected and both gold and silver recoveries have been lower than projected. Operations indicate that the primary focus through the start-up period has been on gold recovery with silver being of secondary economic concern. Key parameters affecting gold and silver recovery are grind size, 80% passing 75 microns, cyanide concentration, retention time in the leaching circuit, and carbon adsorption and elution in the Carbon-in-Pulp (CIP) and elution circuits. Consistent operation is also a factor as stabilizing of the circuit is required following start-ups and shut-downs resulting in reduced performance. Operating personnel are focusing on these areas in order to improve production rates and recoveries.

Some of the key issues encountered during plant commissioning and start-up included:

- Mill motor and variable frequency drive (VFD) programming issues, which caused frequent shut-down of the mill.
- Material discharge from the coarse ore stockpile to the mill feed conveyor, including plugged apron feeder discharge chutes and some conveyor damage resulting in down-time.
- Carbon containment and carbon transfer from the CIP tanks to the carbon elution circuit.
- Carbon elution circuit control and performance, including eluent heating and temperature control.
- Cyanide destruction, SO₂-air, circuit operation and performance.
- Tailings pumping
- Reagent mixing and distribution

These types of challenges are common during plant start-ups and most will be remedied through adjustments during continuous operation of the plant and correction of issues identified during scheduled maintenance periods. Several unit operations have been targeted for review and modification and are the focus of a plant optimization and expansion study.

PLANT EXPANSION STUDY

New Gold has identified sections of the plant, including the carbon elution and cyanide destruction circuits that require modification to achieve required performance. Taking this into consideration, Ausenco Limited (Ausenco) was retained to perform a review of the elution circuit and to prepare the New Gold Rainy River 24 ktpd Debottlenecking Feasibility Study report, which was issued on March 15, 2018.

EQUIPMENT SIZING REVIEW

Ausenco performed a review of the major process plant equipment to determine whether they have sufficient capacity for operating at 24,000 tpd on a steady state basis. Ausenco contacted the vendors for the major process equipment and report that all of the equipment reviewed has the capacity for steady state 24,000 tpd operation.

Some of the equipment will be very close to maximum capacity at 24,000 tpd and so the capability to handle significant surges in production rates or flow will be limited. It will be important to maintain the 92% equipment availability in order to consistently reach production targets. Equipment that is close to capacity at 24,000 tpd and should be reviewed in more detail include:

- The semi-autogenous grinding (SAG) mill will be at 96% of motor capacity assuming an operating work index of 13.2 kWh/t and the ball mill will be at 94% capacity assuming a work index of 13.0 kWh/t. Both mills are close to maximum capacity and may be sensitive to variations in ore hardness. FL Smidth suggested a site audit and baseline mill performance review.
- The final tailings pumps will be reviewed due to issues with pumpbox overflow during surges in flow. This is on the current Ausenco project list for review discussed in this document. A systematic review of the tailings pumping system will be performed in conjunction with the tailing dam construction schedule.
- The water management pumps, Pinewood River water return pumps, the mine rock pond water pumps and fresh water pumps should be reviewed with respect to the water management plan and tailings dam construction schedule. Ausenco recommends that the site water balance be reviewed to confirm the projected pumping requirements.

REVIEW OF COMMINUTION CIRCUIT DESIGN AND GOLD PLANT ASSESSMENT

Following the plant expansion program, which is designed to increase the mill production rate from 21,000 tpd to 24,000 tpd, New Gold commissioned Ausenco to perform a more extensive review of the processes and facilities to determine the maximum operating rate that the plant can achieve without the addition of new grinding equipment. Adding grinding equipment is not

being considered at this time, as it would significantly increase capital costs and the associated down time would decrease production.

Moderate capital cost additions such as the replacement of pump motors or installing additional screens is being considered. These types of modifications can be installed without significantly affecting production.

The study is being performed in three phases: a Concept Study phase, a Feasibility Study Phase and an Execution Phase. The schedule for the project will begin with the current Conceptual Study reports and the final implementation phase is to be completed and commissioned by 2019. The production forecast projects total production of 7.5 million tonnes milled in 2018, 8.5 million tonnes milled in 2019 and 2020, and 9.9 million tonnes milled in 2021, which equates to 27,000 tpd. The two study reports include:

- Ausenco, 2018, New Gold Rainy River Review of the Comminution Circuit Design July 2018.
- Ausenco, 2018, New Gold Rainy River Gold Plant Assessment, July 10, 2018.

The main observations from the review of the historical information and control philosophy are that:

- The circuit was designed for a harder ore than is currently being processed.
- The primary crusher is producing a very fine SAG mill feed.
- Under current operating conditions the grinding mill's installed power is underutilized (66% for the SAG mill and 74% for the ball mill). Operating data indicate that the ball mill is consistently operating in the 11 kWh/t range and the grind size is coarser than design at 89 um.
- The fine SAG mill feed results in a coarse transfer size to the ball mill.
- The grinding load is being pushed to the ball mill, which is producing a product particle size with a P_{80} of approximately 90 um rather than the design product size of P_{80} 75 um.
- An analysis of the ball mill cyclones indicated that the grinding circuit is operating with a circulating load between 500% and 600% instead of the projected 300% with a 400% maximum given in the process design criteria.
- The ball charge could be increased to improve the grind size and the circulating load. A decrease in the circulating load would have a positive impact on the capacities of the downstream equipment.

MINE ROCK AND OVERBURDEN STOCKPILES – STAGE 1 STUDY

A Stage 1 preliminary scoping level evaluation was performed to determine the most effective method for improvement of the foundation materials beneath the proposed mine rock stockpile areas. There are many areas of instability due to soil types and moisture. The study consisted of identifying methods of foundation improvement, preparing preliminary designs for each alternative, preparing cost estimates for each alternative, and then evaluating the results.

The alternatives that were considered applicable for the clay soils at Rainy River included:

- Wick drains
- Stone columns
- Removal and replacement (shear keys)
- Deep soil mixing
- Electro-osmosis
- Vacuum consolidation
- Flattening the slope

FURTHER DEVELOPMENT WORK - STAGE 2 STUDY

Based on the capital cost estimates and schedules for installation, the following alternatives were selected for further investigation in a Stage 2 study.

- The Base Case – Wick drains.
- Alternative 1 – Removal and replacement or shear key, only in areas with shallow clay deposits. Wick drains or slope flattening will still be required for zones with deeper clay.
- Alternative 2 – Flattening the slope by expanding the east mine rock stockpile.

Some additional optimization work was performed on the base case wick drain alternative. The ground improvement widths in Section A areas were reduced from 180 m to 125 m by including a 50 m width without wick drains and a 50 m to 75 m width with wick drains. This configuration provides significant cost improvement in the wick drain option. This optimized base case will be investigated in the Stage 2 study.

ENVIRONMENTAL, SOCIAL, COMMUNITY AND RECLAMATION/CLOSURE

New Gold has made a significant commitment to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against their Health, Safety, Environment and Corporate Social Responsibility

Policy last reviewed and approved by their Board of Directors on July 25, 2018. Key elements include:

Health and Safety – This includes promoting and protecting the well-being of employees through safety-first work practices and a culture of safety excellence. Commitment to leading industry practices and systems in health and safety that focus on prevention of accidents and incidents are a priority

Environment – This includes a commitment to preserving the long-term health and viability of the natural environments affected by the project and operation.

Labour Practices – This includes a commitment to upholding fair employment practices and encouraging a diverse workforce, where people are treated with respect and are supported to realize their full potential.

Community Engagement and Development – This requires a commitment to establishing relationships based on mutual benefit and active engagement with host communities to contribute to healthy communities and development.

At the time of this review, the Mine Environmental Department was adequately staffed, but resources appeared to be stretched. Subsequent to the site visit, the environmental lead for the Mine retired. This role and responsibilities have been assumed by internal staff. New Gold contracts AMEC Foster Wheeler (AMEC) to carry out inspections and third-party groundwater monitoring.

At the time of the site visit, the Mine reported two non-compliance related issues associated with an effluent discharge in excess of permit limits and failing to report an effluent discharge above permit limits. New Gold pleaded guilty to the charges and was fined \$187,500.

The Mine environmental budget for 2018 was reviewed and appears adequate/reasonable for the size of operation, issued permits, and conditions imposed.

New Gold has developed a reclamation and closure plan that satisfies corporate and regulatory requirements. Reclamation and closure planning requires regular review and update to accommodate change(s) in planned and constructed facilities/operations.

RECOMMENDATIONS

The QPs offer the following recommendations:

MINERAL RESOURCES

- Where possible, identify mappable geologic features that act as controls to the distribution of gold and silver mineralization in the deposit to further refine the geologic constraints to future mineral resource estimates.
- As mining progresses, continually assess reconciliation performance of the mineral resource block model against mined production, and where appropriate, refine the geostatistical parameters and block model grade interpolation methods to further improve Mineral Resource estimation accuracy and precision.

MINERAL RESERVES

- Refine procedures for accurate grade control sampling, pit bench estimation and mine planning to ensure reconciliations between mine-to-mill production and reserves-to-mine production remain within acceptable levels of confidence.

MINING

Open Pit

- Further development of wall control and slope monitoring techniques and procedures in both rock and overburden as the pit progresses.
- Completion of the investigation of geotechnical waste dump foundation stabilization of overburden.
- Further operator and maintenance training, coaching, and upskilling to increase the production rate and reduce dilution to achieve the schedule defined in this report.
- The overall project economics are sensitive to total mining costs. A strategic review of efficiency, productivity, and costs is recommended in the near term.

Underground

- Close monitoring of the mine development and the rapid implementation of remedial actions in the event of development advance shortfalls.
- Review of the footwall geometry in areas where the dip is less than 55° as part of the stope planning.

- Cavity monitoring surveys as part of the production records and reconciliation of production to the Mineral Reserve estimates.
- Ground control monitoring systems for analysis of the hangingwall and pillar stability in the open stope areas.
- Further consideration and review of the stability of the of the overall stope hangingwall for the open stopes, including review of rib pillar dimensions as local mining knowledge increases.
- Introduction of a microseismic monitoring system for deeper regions of the mine and for areas below the pit.
- Continued testing to verify field performance of backfill binder content.
- Review of the schedule to ensure that there is time allotted for infill drilling and data analysis between the initial development and the commencement of stoping.

PROCESS

The process facilities are still in the start-up and initial operation stage. It is recommended that the operations and maintenance personnel continue to develop their procedures and to identify opportunities that will improve plant availability. This process is well advanced and the studies and projects that Ausenco are currently pursuing will result in improvements.

Ausenco performed a review of the major process plant equipment to determine whether they have sufficient capacity for operating at 24,000 tpd on a steady state basis. Ausenco contacted the vendors for the major process equipment and report that all of the equipment reviewed has the capacity for steady state 24,000 tpd operation.

It appears that some of the equipment will be very close to maximum capacity at 24,000 tpd and so the capability to handle significant surges in production rates or flow will be limited. It will be important to maintain the 92% equipment availability in order to consistently reach production targets. There were suggestions by some of the equipment suppliers for additional design review as part of the expansion program. It is recommended that ore hardness be reviewed with respect to the mine plan to provide a consistent feed to the mills at the higher production rate. In addition, the water balance and pump capacities in the water management systems should be reviewed.

The question of converting the elution heating from indirect heating to direct should be reviewed with respect to scale formation and its effect on the in-line elution heaters. This

conversion is not recommended due to the severe scale reported in the existing heat exchanger. A different type of heat exchanger such as a shell and tube may be more appropriate as it would be easier to clean by hydro-blasting than the plate and frame.

ENVIRONMENTAL CONSIDERATIONS

- Promote cross training of employees to reduce reliance on key personnel (Environmental, Indigenous Relations, etc.).
- Set up/implement Environmental Management System (EMS). Perform regular audits (at least annual). This will allow for tracking and assessment of success/failures and help with Cross Training recommendation above.
- Continue consultation and coordination with First Nations and public.
- Perform potentially acid generating (PAG) rock confirmation testwork.
- Start interim reclamation to reduce post mining obligations
- Related to above, start revegetation test plots to determine what species and composition works best for the Mine area and environment.

CAPITAL AND OPERATING COSTS

The cost estimate for implementing project closure is estimated to be C\$97 million. Closure costs must be reassessed on a regular basis during LOM to determine present-time rates within the terms of the Mining Act Regulations.

While the mine and mill production rates and operating costs are quite variable as the plant has only been operating a few months, the base operating costs appears reasonable and in-line with expectations for the LOM at C\$2.7/t in mining and C\$8/t to C\$10/t (excluding on site refining costs) range in processing. However, it is recommended watching these costs going forward to confirm the budgeted values are appropriate.

There is a discrepancy between the LOM average of C\$1.54/t compared to actuals of C\$24.65/t for general and administration (G&A) costs due to the low tonnage processed in 2017 versus a normal year of 8.6 million tonnes per annum (Mtpa) and higher overhead costs during start-up. It is recommended watching these costs going forward to confirm the budgeted values are appropriate.

ECONOMIC ANALYSIS

Under NI 43-101 rules, producing issuers may exclude the information required for Item 22 – Economic Analysis, on properties currently in production, unless the Technical Report includes a material expansion of current production. New Gold is a producing issuer, and the Rainy River Mine is currently in production, and the minor expansion from 21,000 tpd to 24,000 tpd is not considered a material expansion. New Gold has performed an economic analysis of the Rainy River Mine using the estimates presented in this report and confirms that the outcome is a positive cash flow that supports the statement of Mineral Reserves.

TECHNICAL SUMMARY

PROPERTY DESCRIPTION AND LOCATION

The Rainy River Mine is located in Ontario, Canada at approximately Latitude 48° 50 North and Longitude 94° 01' West. The property is located in the Township of Chapple, District of Rainy River, in western Ontario, approximately 35 km northwest of Emo, and 405 km west of Thunder Bay.

LAND TENURE

The Rainy River property comprises a portfolio of 210 patented mining rights and surface rights claims (which includes 18 Crown Leases over 28 leasehold interests) and 81 unpatented claims, which rights and claims are owned by New Gold covering an aggregate area of approximately 23,122 ha. The Mine is located in the townships of Fleming, Mather, Menary, Patullo, Potts, Richardson, Senn, Sifton, and Tait. The area of the Mine is approximately 6,053 ha. All unpatented claims are in good standing.

EXISTING INFRASTRUCTURE

The Mine is located approximately 50 km to the northwest of Fort Frances, the nearest large town in western Ontario. The property is centred in Richardson Township (part of Chapple Township) in northwestern Ontario, approximately 162 km along Highway 17/Highway 71/Regional Road 600 south of Kenora, and 418 km along Highway 11/Highway 71/Regional Road 600 west of Thunder Bay. These access roads are sealed allowing year-round access.

HISTORY

Exploration in the Rainy River area began in 1967. Various companies and government organizations were active on and around the project area from 1967 to 1989. Nuinsco held the claims to the property from 1990 to 2004, with RRR continuing from 2005 to 2013 when New Gold completed a takeover of RRR on October 15, 2013.

The project was acquired by Nuinsco in 1993. Nuinsco exploration activities from 1993 to 2004 included geophysical surveys and diamond drilling, which led to the discovery of the 17 Zone in 1994, 34 Zone in 1995, and 433 Zone in 1997.

Upon acquisition of the property from Nuinsco in June 2005, RRR relogged key sections of the drill core, and completed several exploration and infill drilling campaigns.

Ten Mineral Resource estimates were prepared for Rainy River from 2003 to 2017. Authors of these reports include Mackie et al. in 2003, Caracle Creek International Consulting Inc. (CCIC) in 2008, SRK Consulting (Canada) Inc. (SRK) in 2009, 2010, 2011, and 2012, BBA Inc. (BBA) and collaborators in 2014 Feasibility Study, and New Gold and SRK in 2015. A total of seven of these Mineral Resource estimates are documented in previous technical reports prepared for the Mine which are available on SEDAR. The current Mineral Resource estimate reported in Table 1-1 supersedes all previous estimates.

GEOLOGY AND MINERALIZATION

The Mine is located within the Late Archean Rainy River Greenstone Belt (RRGB) which formed approximately 2.7 billion years ago (Ga). The RRGB forms part of the western Wabigoon Subprovince, located in the Superior Province of the Canadian Shield. The Wabigoon Subprovince is a 900 km long, east-west trending composite volcanic and plutonic terrane comprising distinct eastern and western domains separated by rocks of Mesoproterozoic age.

The western Wabigoon domain is predominantly composed of mafic volcanic rocks intruded by tonalite-granodiorite intrusions. The volcanic rocks which were largely deposited between approximately 2.74 Ga and 2.72 Ga, range from tholeiitic to calc-alkaline in composition, and are interpreted to represent oceanic crust and volcanic arcs, respectively. These are

succeeded by approximately 2.71 Ga to 2.70 Ga volcano-sedimentary sequences and by locally deposited, unconformable, immature clastic sedimentary sequences.

The volcanic rocks have been intruded by a wide variety of plutonic rocks including synvolcanic tonalite-diorite-granodiorite batholiths, younger granodiorite batholiths, monzodiorite intrusions and monzogranite batholiths and plutons. The intrusions were emplaced over a large time span between approximately 2.74 Ga to 2.66 Ga.

Rainy River is centred on Richardson Township. To the north of Richardson Township lies the Sabaskong granitoid batholith. The Black Hawk Stock lies to the east of Richardson Township. A package of metasedimentary rock is found south of Richardson Township. Wedged in between these lithologies are a series of tholeiitic mafic and structurally overlying calc-alkalic intermediate to felsic metavolcanic rocks, striking almost east-west and dipping to the south. Intermediate dacitic rocks host most of the Rainy River gold mineralization.

Structural analysis suggests that the current geometry and southwesterly plunge of the gold mineralization at Rainy River is the result of high strain deforming features associated with gold mineralization and rotating the ore plunge parallel to the stretching direction

A total of four main styles of mineralization have been identified on the Rainy River Mine:

- Moderately to strongly deformed, auriferous sulphide and quartz-sulphide stringers and veins in felsic quartz-phyric rocks (ODM/17, Beaver Pond, 433 and HS Zones);
- Deformed quartz-ankerite-pyrite shear veins in mafic volcanic rocks (CAP/South Zone);
- Deformed sulphide-bearing quartz veinlets in dacitic tuffs/breccias hosting enriched silver grades (Intrepid Zone); and
- Copper-nickel-platinum group metals mineralization hosted in a younger mafic-ultramafic intrusion (34 Zone).

The formation of the Rainy River deposit has been attributed to known auriferous VMS systems with a primary synvolcanic source and possibly a secondary syntectonic mineralization event.

EXPLORATION STATUS

In July 2013, New Gold began the process of acquiring RRR and ultimately completed the acquisition of the company in October 2013.

In November 2013, SRK completed an updated Mineral Resource estimate for the Mine which served as the basis for an updated Feasibility Study, which was subsequently completed in February 2014. The November 2013 resource estimate was the ninth estimate completed for Rainy River.

From 2014 to 2015, New Gold completed a Mobile Metal Ion (MMI) geochemical survey and a hyperspectral alteration study to determine potential vectors to gold mineralization. The results of this work were used to define prospective satellite targets to the known Mineral Resources. Total drilling during this period comprises of 163 core holes totalling 55,044 m, and eight reverse circulation holes totalling 1,485 m designed to test for potential extensions to the known open pit resources and infill drilling. Additionally, 115 condemnation holes were completed totalling 23,390 m.

In January 2015, New Gold acquired a 100% interest in three mineral properties located within the Rainy River area through the acquisition of Bayfield Ventures Corp. (Bayfield). These properties included the Burns Block claim located immediately east of the current open pit. The company subsequently re-logged 317 core holes totalling 102,380 m from the Burns Block claim and integrated this information into the drill hold database and mineral resource and reserve estimates for the Mine.

Between 2016 and 2017 New Gold conducted two infill drilling campaigns to upgrade estimation classification for open pit and underground mineral resources in the ODM zone. Fifty-three core holes totalling 8,397 m were completed in the upper levels of the planned open pit and 15 core holes totalling 8,021 m were completed in the deeper levels of the planned underground mining area.

Additional exploration advances during 2016 included the completion of a detailed study of the geology of the Rainy River gold deposit that was part of a M.Sc. thesis project by Ms. Mireille Pelletier (Pelletier, 2016).

MINERAL RESOURCES

The Rainy River database comprises data from 2,185 core holes (921,688 m), drilled by Nuinsco, Bayfield, RRR, and New Gold up to December 31, 2017. The updated 3D geological

model is comprised of wireframes for eight major lithological units and fourteen distinct mineralized zones with further domains.

In order to carry out the geostatistical analysis and grade estimation, the raw assay data was composited to 1.5 m lengths. Capping of very high gold grades was applied to the composited data levels and adjusted for each resource domain and each metal. Two unrotated, sub-blocked models, representing the open pit and underground volumes at the Main Zone of the Rainy River deposit, were created in Vulcan. As no new drilling or geologic modelling was done on the Intrepid Zone, the Intrepid block model was not changed since the 2015 update generated by SRK.

Variogram model parameters are effectively unchanged from the 2015 resource estimate completed by SRK. Metal grades were estimated separately using ordinary kriging estimator (OK) from capped composite data within respective domains. Grade interpolation was completed in two or three estimation passes using search ellipse ranges derived from variography, and semi-soft boundaries. Usually, the first pass ranges corresponded to 90% of the variogram sill. Specific gravity was estimated using inverse distance squared (ID^2), where supported by sufficient data density, and assignment based on lithology and mineralization domain, where not.

Mineral Resources are classified primarily on the basis of a block's distance from the nearest informing composites and on variography results. Classification is based on gold data alone.

Open pit Mineral Resources are reported at cut-off grades of 0.3 g/t AuEq and 0.5 g/t AuEq for Low Grade and Direct Processing material, respectively. The underground Mineral Resources are reported at a cut-off grade of 2.0 g/t AuEq. Measured and Indicated Mineral Resources are estimated to total 63.109 million tonnes at grades of 1.06 g/t Au and 3.8 g/t Ag, containing 2,142,000 ounces of gold and 7,651,000 ounces of silver. Inferred Mineral Resources are estimated to total 8.871 million tonnes at grades of 1.10 g/t Au and 2.4 g/t Ag, containing 313,000 ounces of gold and 672,000 ounces of silver. The Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

MINERAL RESERVES

The pit optimization analyses for the mineral reserve and LOM plan were run on the Measured and Indicated Mineral Resources to determine the economics of extraction by open pit methods using US\$1,275/oz Au and US\$17.00/oz Ag price. Only blocks classified as Measured or Indicated Resources were included in the pit optimization process.

The final pit design was developed from the pit optimization results and included 35 m wide ramps at a 10% grade. Open pit Mineral Reserves are reported within the final pit design at cut-off grades of 0.3 g/t and 0.5 g/t AuEq for Low Grade and Direct Processing ores, respectively.

Underground Mineral Reserves were estimated by the application of mine development and stoping plans to convert the Indicated Mineral Resources to Probable Mineral Reserves. The underground Mineral Reserve estimates are based upon the use of mechanized long hole stoping with a combination of open stopes and backfilled stopes.

The underground Mineral Reserves are reported at a cut-off grade of 2.2 g/t AuEq. Total Open Pit and Underground Proven and Probable Mineral Reserves are estimated to be 120.451 million tonnes at grades of 1.09 g/t Au and 3.2 g/t Ag, containing 4,220,000 ounces of gold and 12,398,000 ounces of silver.

MINING METHOD

The Rainy River Mine is an open pit and underground gold-silver mining project. The open pit mine is a conventional truck and shovel open pit mining operation, which utilizes hydraulic shovels and 220-tonne trucks as the primary mining equipment. The open pit mine operates at a rate of 180,000 tpd of ore and waste and has an overall strip ratio of 3.7:1.0 (W:O). The underground mine is a mechanized ramp access mine that will use long hole stoping to exploit the underground Mineral Reserves at a rate of approximately 2,300 tpd after 2021.

The open pit is anticipated to be mined out by 2026. It is then planned that mill production continues with underground and stockpile feed through 2032.

Underground Mineral Reserves have been identified in ten zones to date. Several of the zones are extensions of the open pit ore bodies while others are up to one kilometre away from the open pit. Current underground Mineral Reserves have been defined at depths ranging from

near surface (e.g., Intrepid) to approximately 600 m below surface within an area approximately 2.8 km long and 0.9 km in width. Gold mineralization at Rainy River remains open at depth beyond the limits of exploration drilling completed to date.

Ore grade gold mineralization occurs in sub-vertical horizons from three metres to 20 m thick. Widths over 15 m are rare and the weighted average thickness is eight metres. A three metre minimum width is used for defining Mineral Reserves. The ore zones generally dip at 60° or more but can flatten locally to 45°. The planned mining method relies upon gravity ore flow along the footwall.

Underground Mineral Reserves are planned to be mined using a decline access mechanized long hole stoping method with open stoping, and stopes with waste rock fill and cemented aggregate fill. Stopes less than ten metres wide will be mined by open stoping with eight-metre thick rib pillars generally spaced 40 m along strike. Stopes greater than 10 m wide will be backfilled.

MINERAL PROCESSING

PROCESS PLANT

The Rainy River process facilities were originally designed to process 21,000 tpd, or 7.67 Mtpa, from the open pit and underground mines. The target production was to be 19,500 tpd from the open pit mine and 1,500 tpd from the underground mine. The operating availability of the primary crusher is 65% and the operating availability of process plant is 92%.

The process flowsheet consists of the following unit processes:

- Primary gyratory crushing;
- Coarse ore stockpile, discharged through draw pockets by apron feeders;
- SAG mill feed conveyor;
- Primary SAG mill grinding;
- Pebble crushing;
- Secondary ball mill grinding;
- Gravity concentration of primary cyclone feed slurry using centrifugal concentrators;
- Intensive cyanide leaching of the gravity concentrate using an Acacia reactor;
- Screening of cyclone overflow prior to thickening to remove trash;
- Grinding circuit thickening;

- Cyanide leaching using eight tanks in series;
- Carbon in pulp gold recovery using seven tanks in series;
- Cyanide destruction using the SO₂ (sulphur dioxide) air process in two tanks in series;
- Carbon stripping using the Zadra process;
- Acid washing of the carbon using hydrochloric acid;
- Electrowinning of the carbon eluent and gravity concentrate leach solution; and
- Casting of gold and silver bars in an induction furnace.
- Cyanide destruction using the INCO SO₂-air process to reduce the cyanide to less than 5ppm CN_{TOTAL}

In early 2018, a plan to expand mill throughput to 24,000 tpd was completed. The mine plan and mill production in this Technical Report is based on the 24,000 tpd milling rate.

TAILINGS MANAGEMENT AREA

The detoxified slurry flows from cyanide destruction to the tailings pumpbox. The tailings slurry is then pumped by two 356 mm x 304 mm, 550 kW centrifugal pumps in series to the tailings management area.

Reclaim water is pumped from the tailings management area to the process water tanks. The reclaim water demand for the process facilities is 1,080 m³/h.

WATER MANAGEMENT POND

Fresh water is currently supplied to the water management pond from the Pinewood River with two, 225 kW vertical turbine water intake pumps. After sufficient reclaim water is available from the tailings management pond, the pumps will be moved from the Pinewood River to the water management pond pump station and will pump through the same pipelines back to the Pinewood River. Fresh water is pumped to the firewater tank, the truck filling station, the truck shop, and the truck wash facility.

WATER DISCHARGE POND

The water discharge pond receives water by gravity from the water management pond. Water then flows by gravity from the water discharge pond to the wetlands.

MINE ROCK POND

The mine rock pond has a capacity of approximately 2.93 Mm³ and receives water from the open pit and eventually from underground mine dewatering. Water from the mine rock pond is

pumped to the reclaim water tank, the tailings pumpbox, the process water tank and the cyclone feed pumpbox. The mine rock pond is not available during the winter due to freezing. Water will be supplied from tailings management area reclaim water only during the winter months.

PROJECT INFRASTRUCTURE

PRIMARY SITE ACCESS ROADS

The mine site access roads and onsite roads make use of existing road and easements, upgrading and extending them as required. The main entrance to the site is the east access road, which connects the Korpi Road from Finland (Highway 71) with the Roen Road. Branches of the Roen Road connect the main access road to the Plant Site to the south and the tailings management area (TMA) via the Haul Road 13. A branch to the north provides access to the Explosive Magazine and the emulsion plant.

The access roads to the tailings facilities are wide enough to accommodate tailings and reclaim water pipelines and light vehicle and truck traffic. The roads are surfaced with crushed stone.

Plant site roads connect the process plant area to the coarse ore stockpile at the primary crusher, the low-grade stockpile, the underground portal, and the open pit.

MINE HAUL ROADS

Mine haul roads connect the open pit to the overburden and waste rock stockpiles, crusher pad, mine facilities, including the truck shop and truck wash buildings, and tailings dam.

MINE SERVICES FACILITIES

The mine services facilities include the truck shop and truck wash facilities. The facility dimensions were based on Komatsu 830E (220-tonne) class haul truck.

GENERAL OFFICES AND ASSAY LABORATORY

The office facilities are prefabricated type buildings made up of 12' x 60' modules. There are three main buildings; the administration, the mine office and dry, and the plant office. The office requirements for each building were designed to accommodate the projected staffing requirements.

MAIN ADMINISTRATION BUILDING

The main administration is located by the mine entrance and houses administration and safety personnel and consists of 10 offices and space for open plan work stations. There is a small kitchen and three meeting rooms.

MINE OFFICE AND DRY

The mine office is located next to the truck shop and houses the mine, maintenance and engineering office staff. The building has dry facilities with lockers for men and women.

PLANT OFFICE

The plant office is located on the west side of the process building between the leach tanks and the pre-leach thickener and connected to the main mill building via a corridor. The building houses process operations and maintenance staff and has a dry facility as well.

PARKING AREA

Parking is located adjacent to mill building room for 150 vehicles and two buses. During the site visit it was noted that all personnel were being transported to the mine site by bus. Personal vehicles were to be allowed in the weeks following the visit.

ASSAY LABORATORY

The assay laboratory is located adjacent to the administration building and was commissioned in December 2016 through February 2017. The lab is designed to process 200 mine blast hole and mill solids samples per day. The assay lab has facilities for:

- Sample preparation including weighing, drying, crushing and splitting;
- Fire assaying, including a balance room for weighing final gold and silver buttons;
- Atomic absorption (AA) spectrophotometers for analysis of the gold and silver following fire assay;
- LECO Analyzers for carbon and sulphur analyses;
- Wet chemical lab for solution samples;
- Environmental lab;
- Two offices, a lunch room, and two washrooms.

FUEL STORAGE AND DISPENSING

The fuel island is located near the crusher on the main haul road to the plant. The tank farm is located outside the blast radius of the pit and consists of seven 80,000 L double walled storage

tanks for a total capacity of 560,000 L. This volume provides sufficient fuel for six days of operation based on daily fuel consumption of 95,000 L.

The light vehicle fuel station consists of horizontal double walled tanks with the appropriate distribution equipment for light vehicles. Diesel and gasoline are available for light vehicles, which are primarily diesel pick-up trucks.

EXPLOSIVE MAGAZINE STORAGE AND EMULSION PLANT

The explosive magazine storage area and emulsion plant are located on a dedicated road to the north of the east access road. The facilities were constructed and are being operated by the explosive supplier. The explosive magazine is located midway up the road and the emulsion plant is located at the end of the road in an isolated area.

ELECTRICAL POWER AND COMMUNICATIONS

The total power connected for the project is estimated to be 57 MW. Electricity is supplied by a 16.7 km long 230 kV power line, and connected to the regions existing 230 kV Hydro One power line currently connecting Fort Frances and Kenora.

The main 230 kV to 13.8 kV substation is located to the northeast of the concentrator building. Two main 230 kV to 13.8 kV, 42/56/70 MVA transformers are used for combined power of 100 MVA. This provides capacity for future expansion and mitigates the risk of downtime due to transformer failure. A 15 kV Gas Insulated Switchgear, complete with electrical protection devices is included.

Electricity for the underground mine is to be provided by a 13.8 kV line routed down the decline ramps in the mine. Power will be delivered from the main substation by an overhead power line to the mine portal.

EMERGENCY POWER

There are two emergency generators, one supplying 4.16 kV and a second supplying 600 V power. During a power outage, a programmable logic controller (PLC) will manage the critical loads. The loads are grouped into fixed loads such as lighting, heating, sequential loads such as leach tank agitators, cyanide destruction tank, and manually operated loads, such as sump pumps, rake mechanisms, and reactive heating.

COMMUNICATION

A fibre optic loop connects all areas of the operation. The fibre optic lines are run on the 27.6 kV overhead distribution lines and transmit voice, video, and data on the following systems:

- Telemetry, data acquisition and control between the process plant and exterior process equipment;
- Computer network between all departments;
- Local telephone lines;
- Computer network for maintenance on all electrical equipment;
- Fire detection;
- Video surveillance and access control systems;
- Electrical tele-protection equipment.

TAILINGS MANAGEMENT AREA

The TMA covers an area of approximately 765 ha and design is based on an ultimate settled density of 1.4 t/m³, providing storage capacity for the estimated 74 Mm³ of tailings to be produced over the LOM. The facility is bounded by natural topography in the northeast and by impoundment dams along the remaining perimeter and has the capacity for further expansion.

TAILINGS DEPOSITION PLAN

Tailings will be deposited from spigots located inside the perimeter of the TMA dams in order to develop beaches to protect each of the dams. This will separate the ponds from the dams and allow reclaim of water using barge pumps. After 2024, the spigots will require extension inwards to displace the water in the ponds and minimize the volume of water required to cover the tailings for mine closure.

The starter dam has a crest elevation of 371.5 metres above sea level (MASL). The dams will be raised sequentially over the life of mine to the maximum dam height of 23.5 m to contain the design total tonnage of 94.6 Mm³.

TAILINGS DAM DESIGN

Tailings management dams are designed for the most severe flood and earthquake criteria due to their potential for failure. The site has low to moderate seismic risk with a 0.096 g horizontal peak ground acceleration for a 10,000-year return period earthquake.

The design criteria factors of safety used in the construction of the dams include:

- Short term, end of construction with induced pore pressures: 1.3
- Long term, when excess pore pressures have fully dissipated: 1.5
- Rapid draw down of the Water Management Pond slope: 1.2
- Worst case, with potentially slicken-sided upper varved clay: 1.0
- Pseudo-static loading with a seismic coefficient of 50% of the peak ground acceleration, PGA, 1.0

The design slopes for the impoundment dams were initially 4H:1V for heights up to 15 m for approximately 70% of the dam length. After a review of the design by the Ministry of Natural Resources and Forestry (MNR) the design has been changed to require 11H:1V slopes for all of the dams.

The primary dam construction materials are mine waste rock and selected clay from the open pit development, which are available during the preproduction and early periods of operation.

TAILINGS MANAGEMENT AREA DAM ENGINEERING AND CONSTRUCTION SCHEDULE

TMA dam engineering and construction is a continuous process that is scheduled to provide capacity for tailings placement through the life of the mine. The TMA is divided into a water management pond, which was constructed first to be able to manage water containment and distribution. The water required for start-up of the process facilities was provided by pumping water from the water management pond, which was filled by pumping water from the Pine Wood River. Tailings containment is divided into three cells. Cell 1 is the start-up cell, which will provide capacity for mill tailings through March 2018, followed by Cells 2 and 3, the first lifts of which will be constructed by October 2019. Completion of the construction of the first lift of the three cells is considered Stage 1. Subsequent stages will consist of increasing the heights of the dams in all three cells. General constraints on the dam construction schedule are the availability of clay material for the core and placement time required for that material.

MARKET STUDIES

Gold and silver markets are mature global markets with reputable and refiners located throughout the world. Gold is a principal metal traded at spot price for immediate delivery. The average gold price for 2017 was US\$1,257 per troy ounce. The three-year and five-year rolling average prices through the end of December 2017 are US\$1,222 and US\$1,269 per

troy ounce, respectively. This Technical Report uses the current bank consensus price for gold of US\$1,300.00 per troy ounce for mineral reserve reporting. The Mine produces a gold-silver doré product that will be shipped for refining by one of the internationally-established refiners.

ENVIRONMENTAL, PERMITTING AND SOCIAL CONSIDERATIONS

New Gold has made a significant commitment to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against their Health, Safety, Environment and Corporate Social Responsibility Policy last reviewed and approved by their Board of Directors on February 19, 2015.

New Gold has collected and documented pre-project baseline environmental conditions to better design the Mine and obtain the various permits/authorizations required to avoid and/or mitigate environmental impacts. This information will be used to assess compliance with issued permits and to determine reclamation and closure success.

Feasibility and permitting did not reveal any environmental aspects that are considered limiting to permitted development. The Mine has all of the permits and authorizations to construct and operate.

The Mine tracks and reports good standing with the local community, including local First Nations bands and the Métis Nation of Ontario. They have signed Participation Agreement(s) with the impacted local Indigenous groups

CAPITAL AND OPERATING COST ESTIMATES

Total LOM sustaining capital costs are estimated to total C\$1,344 million as summarized in Table 1-3.

TABLE 1-3 CAPITAL COSTS SUMMARY

Description	Unit	2018	2019	2020	2021	2022	2023	24-31	LOM Total
Open Pit	C\$(000)	76,734	133,329	71,510	42,442	41,670	31,903	67,888	465,476
Underground	C\$(000)	24,516	58,380	95,454	100,472	20,597	16,397	109,508	425,324
Process/Tailings	C\$(000)	129,145	54,436	7,240	20,341	26,850	1,576	108,277	347,864
Infrastructure	C\$(000)	3,197	680	1,290	150	440	190	2,040	7,987
Reclamation/Closure	C\$(000)	126	126	-	-	-	-	73,521	96,956 ¹
Grand Total	C\$(000)	233,717	246,950	175,494	163,404	89,557	50,067	361,234	1,343,607
UG Project Capital	C\$(000)	24,516	48,833	-	-	-	-	-	73,350
Sustaining	C\$(000)	209,201	198,117	175,494	163,404	89,557	50,067	361,234	1,270,258
Grand Total	C\$(000)	233,717	246,950	175,494	163,404	89,557	50,067	361,234	1,343,607

Note:

¹Contains \$23 million of post mining final closure costs in 2032.

The LOM unit operating costs are summarized in Table 1-4.

TABLE 1-4 UNIT OPERATING COST SUMMARY

Description	Unit	Value
Open Pit	C\$/t mined	2.7
Underground	C\$/t mined	72.5
Process	C\$/t milled	9.5
G&A	C\$/t milled	3.1
Royalties	C\$/t milled	0.5

2 INTRODUCTION

The purpose of this report is to provide an update, for public disclosure, of the Rainy River Mine (Rainy River or the Mine) operations, near Fort Frances, Ontario, Canada, and to support the Mineral Resource and Mineral Reserve estimates for the Mine as of June 30, 2018. This Technical Report conforms to NI 43-101 Standards of Disclosure for Mineral Projects (NI 43-101).

This Technical Report has been prepared by New Gold Inc. (New Gold), an intermediate gold mining company with four producing assets. The New Afton and Rainy River Mines in Canada, the Mesquite Mine in the United States, and the Cerro San Pedro Mine in Mexico (which transitioned to residual leaching in 2016) provide the company with its current production base. In addition, New Gold owns 100% of the Blackwater project located in British Columbia, Canada.

New Gold completed a takeover of Rainy River Resources Ltd. (RRR) on October 15, 2013. RRR, a wholly-owned subsidiary of New Gold, is the operator of the Rainy River Mine. Rainy River Mine commenced processing ore on September 14, 2017 and subsequently announced its first gold pour on October 6, 2017. Commercial production was achieved in mid-October. From an accounting perspective, New Gold recognizes commercial production effective November 1, 2017. In 2017, the mine produced 37,047 ounces of gold.

SOURCES OF INFORMATION

For this Technical Report, a site visit was carried out by Nicholas Kwong, P.Eng., Director Business Improvement, Andrew Paul Hampton, P. Eng., Principal, KSN Mineral Process Associates (KSN), and Lee Patrick Gochnour, QP, MMSA, Gochnour & Associates, Inc., on October 2-4, 2017. Mr. Kwong's most recent site visit was on July 9, 2018. Michele Della Libera, P.Geo. (MDL), Director Exploration for New Gold, has visited the site multiple times since July 2013, most recently on July 9-13, 2018. Mr. Binsar Sirait, Director of Mining Engineering for New Gold, visited the site multiple times since January 2016, most recently on June 13, 2018. Ms. Nussipakynova visited the site on April 11, 2018. During the site visits and subsequent meetings, discussions were held with:

New Gold Corporate

- Mark Petersen, Vice President Exploration, New Gold Inc.
- Binsar Sirait, Director of Mining Engineering, New Gold Inc.
- Mauro Bassotti, Director Operations Geology, New Gold Inc. (former)
- John Bligh, Manager Technical Data Services, New Gold Inc.

Rainy River Mine

- Hubert Schimann, Mine Manager
- Peter Caldwell, Underground Technical Superintendent
- Alfri Hamdani, Mine Planner
- Tim Schwartz, Chief Geologist
- Darrol VanDeventer, Chief Engineer
- Tony Lord, Maintenance Manager
- David Hall, Mill Manager (former)
- Temesghen Teshale, Metallurgist
- Tom Galesic, Senior Business Analyst (former)
- Sylvia Smeeth, Land and Property Administrator
- Darrell Martindale, Environmental Manager (former)
- Stacy Jack, Manager Community Relations (former)

Mr. Kwong is responsible for Sections 15, 16, 19, 21, 22 and 24 of this report and shares responsibility for Sections 1, 2, 3, 18, 20, 23, 25, 26, and 27. Mr. Della Libera is responsible for Sections 4, 5, 6, 7, 8, 9, 10, 11, and 12, and shares responsibility for Sections 1, 2, 25, and 26. Ms. Nussipakynova is responsible for Section 14 and shares responsibility for Sections 1, 2, 12, and 25. Mr. Hampton is responsible for Sections 13, 17, and 18 and shares responsibility for Sections 1, 2, 3, 4, 5, 6, 21, 25, 26, and 27. Mr. Sirait shares responsibility for the portions of Sections 1, 2, 15, and 16 that relate to open pit Mineral Reserves and mining methods. Mr. Smith shares responsibility for the portions of Sections 1, 15, 16, 25, and 26 that relate to underground Mineral Reserves and mining methods. Mr. Gochnour is responsible for Section 20.

The documentation reviewed, and other sources of information, are listed at the end of this report in Section 27 References.

LIST OF ABBREVIATIONS

Units of measurement used in this report conform to the metric system. All currency in this report is US dollars (US\$) unless otherwise noted.

a	annum	kWh	kilowatt-hour
A	ampere	L	litre
bbl	barrels	lb	pound
btu	British thermal units	L/s	litres per second
°C	degree Celsius	m	metre
C\$	Canadian dollars	M	mega (million); molar
cal	calorie	m ²	square metre
cfm	cubic feet per minute	m ³	cubic metre
cm	centimetre	μ	micron
cm ²	square centimetre	MASL	metres above sea level
d	day	μg	microgram
dia	diameter	m ³ /h	cubic metres per hour
dmt	dry metric tonne	mi	mile
dwt	dead-weight ton	min	minute
°F	degree Fahrenheit	μm	micrometre
ft	foot	mm	millimetre
ft ²	square foot	mph	miles per hour
ft ³	cubic foot	MVA	megavolt-amperes
ft/s	foot per second	MW	megawatt
g	gram	MWh	megawatt-hour
G	giga (billion)	oz	Troy ounce (31.1035g)
Gal	Imperial gallon	oz/st, opt	ounce per short ton
g/L	gram per litre	ppb	part per billion
Gpm	Imperial gallons per minute	ppm	part per million
g/t	gram per tonne	psia	pound per square inch absolute
gr/ft ³	grain per cubic foot	psig	pound per square inch gauge
gr/m ³	grain per cubic metre	RL	relative elevation
ha	hectare	s	second
hp	horsepower	st	short ton
hr	hour	stpa	short ton per year
Hz	hertz	stpd	short ton per day
in.	inch	t	metric tonne
in ²	square inch	tpa	metric tonne per year
J	joule	tpd	metric tonne per day
k	kilo (thousand)	US\$	United States dollar
kcal	kilocalorie	USg	United States gallon
kg	kilogram	USgpm	US gallon per minute
km	kilometre	V	volt
km ²	square kilometre	W	watt
km/h	kilometre per hour	wmt	wet metric tonne
kPa	kilopascal	wt%	weight percent
kVA	kilovolt-amperes	yd ³	cubic yard
kW	kilowatt	yr	year

3 RELIANCE ON OTHER EXPERTS

This report has been prepared by New Gold. The information, conclusions, opinions, and estimates contained herein are based on:

- Information available at the time of preparation of this report,
- Assumptions, conditions, and qualifications as set forth in this report, and
- Data, reports, and other internal information, and other third-party sources, which are referenced as appropriate throughout this report.

New Gold has relied on an opinion on ownership by DLA Piper dated January 11, 2018 entitled “Confirmation of title re Project and Infrastructure Lands comprising the Rainy River Gold Mine.” and this opinion is relied on in Section 4 and the Summary of this report.

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

4 PROPERTY DESCRIPTION AND LOCATION

The Rainy River Mine is located at Latitude 48° 50' North and Longitude 94° 01' West in Ontario, Canada. The property is located in the Township of Chapple, District of Rainy River, in western Ontario, approximately 35 km northwest of Emo, and 405 km west of Thunder Bay. A location map for the Mine is presented in Figure 4-1.

The project survey control is based on the Universal Transverse Mercator (UTM) coordinate system. It is based on the Zone 15 North projection, using the North American Datum 1983 (NAD 83). The UTM coordinates place the Rainy River Mine at 5,409,500N and 425,500E at a nominal elevation of 360 MASL.

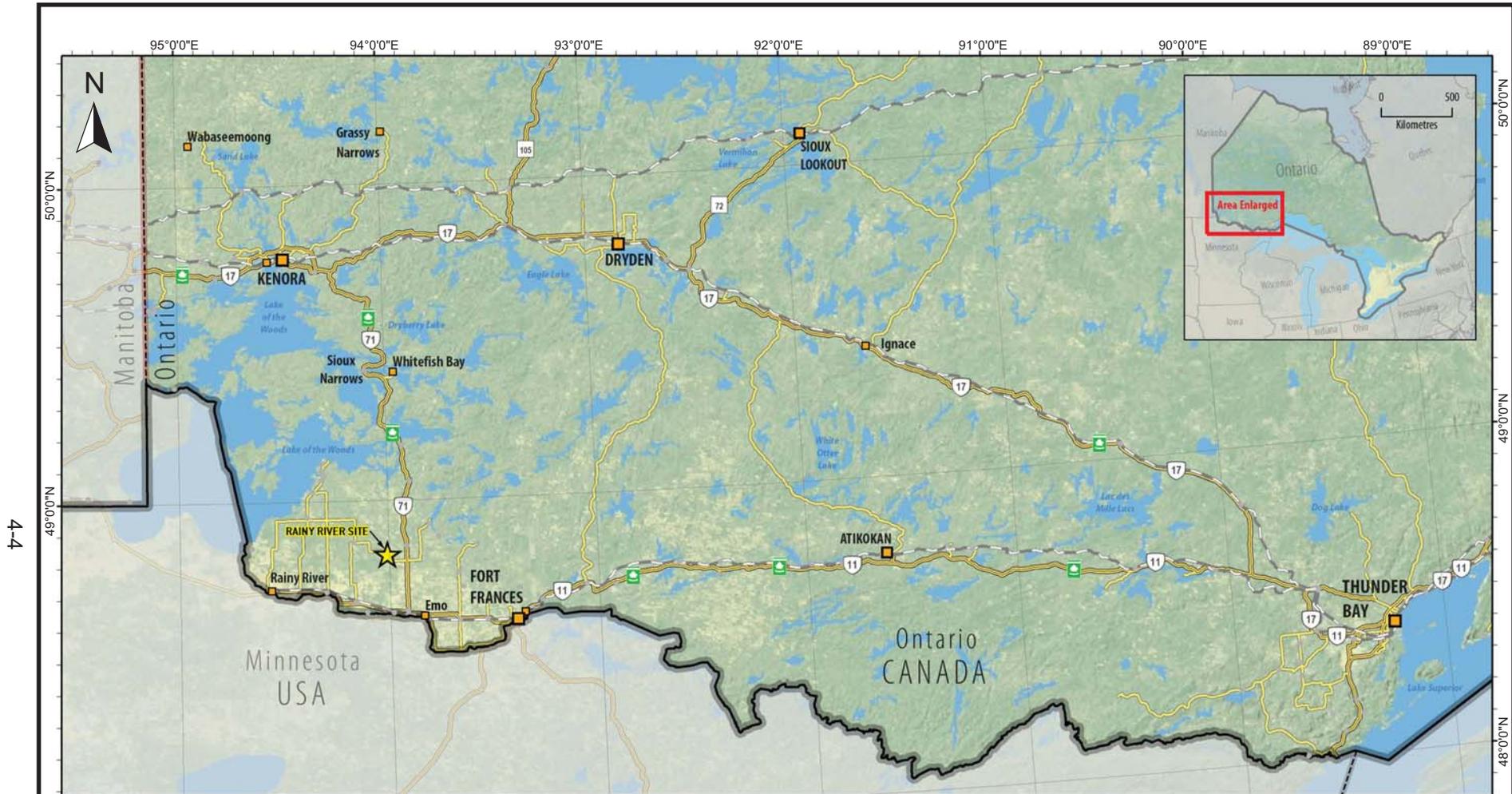
LAND TENURE

The Rainy River property comprises a portfolio of 210 patented mining rights and surface rights claims (which includes 18 Crown Leases over 28 leasehold interests) and 81 unpatented claims, which rights and claims are owned by New Gold covering an aggregate area of approximately 23,122 ha. The Rainy River Property is located in the townships of Fleming, Mather, Menary, Patullo, Potts, Richardson, Senn, Sifton, and Tait. The Mine area is approximately 6,053 ha. All unpatented claims are in good standing. A land tenure map is shown in Figure 4-2, and a list of the unpatented claims and their expiry dates is presented in Table 4-1.

TABLE 4-1 SUMMARY OF LAND CLAIMS

Township/ Area	Claim Number	Recording Date	Claim Due Date	Status	Percent Option
FLEMING	3019809	2004-May-17	2019-May-17	A	100%
FLEMING	4211671	2006-Jun-26	2019-Jun-26	A	100%
FLEMING	4244241	2009-Jan-28	2019-Jan-28	A	100%
FLEMING	4244243	2009-Jan-28	2019-Jan-28	A	100%
FLEMING	4245258	2009-Jan-28	2019-Jan-28	A	100%
FLEMING	4245259	2009-Jan-28	2019-Jan-28	A	100%
FLEMING	4245260	2009-Jan-28	2019-Jan-28	A	100%
MENARY	4208866	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208867	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208868	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208869	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208870	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208871	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208872	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208873	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208874	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208875	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4208876	2005-Oct-26	2019-Oct-26	A	100%
MENARY	4244244	2009-Jan-28	2019-Jan-28	A	100%
NELLES	4205809	2006-Nov-22	2018-Nov-22	A	100%
NELLES	4250319	2011-Jun-13	2019-Jun-13	A	100%
NELLES	4254475	2011-Jan-26	2019-Jan-26	A	100%
NELLES	4254476	2011-Jan-26	2019-Jan-26	A	100%
NELLES	4254477	2011-Jan-26	2019-Jan-26	A	100%
NELLES	4254478	2011-Jan-26	2019-Jan-26	A	100%
NELLES	4254479	2011-Jan-26	2019-Jan-26	A	100%
NELLES	4260559	2010-Dec-02	2018-Dec-02	A	100%
NELLES	4260560	2010-Dec-02	2018-Dec-02	A	100%
NELLES	4260561	2010-Dec-02	2018-Dec-02	A	100%
NELLES	4260562	2010-Dec-02	2018-Dec-02	A	100%
NELLES	4260563	2010-Dec-02	2018-Dec-02	A	100%
PATTULLO	4205805	2006-Nov-27	2019-Nov-27	A	100%
PATTULLO	4205814	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4205815	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4205816	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4205817	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4205818	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4205820	2006-Nov-22	2018-Nov-22	A	100%
PATTULLO	4214438	2010-Mar-03	2019-Mar-03	A	100%
PATTULLO	4214439	2010-Mar-03	2019-Mar-03	A	100%

Township/ Area	Claim Number	Recording Date	Claim Due Date	Status	Percent Option
PATTULLO	4214440	2010-Mar-03	2019-Mar-03	A	100%
PATTULLO	4214442	2010-Mar-03	2019-Mar-03	A	100%
POTTS	3012554	2007-Mar-13	2019-Mar-13	A	100%
POTTS	4207826	2006-Feb-20	2019-Feb-20	A	100%
POTTS	4211670	2006-Jun-26	2019-Jun-26	A	100%
POTTS	4211672	2006-Jun-26	2019-Jun-26	A	100%
POTTS	4218605	2007-Apr-19	2019-Apr-19	A	100%
POTTS	4224811	2008-May-06	2019-May-06	A	100%
POTTS	4224812	2008-May-06	2019-May-06	A	100%
POTTS	4224813	2008-May-15	2019-May-15	A	100%
POTTS	4244242	2009-Jan-28	2019-Jan-28	A	100%
POTTS	4245254	2009-Jan-28	2019-Jan-28	A	100%
POTTS	4245255	2009-Jan-28	2019-Jan-28	A	100%
POTTS	4249688	2010-Mar-01	2019-Mar-01	A	100%
RICHARDSON	1105427	1992-Oct-15	2019-Oct-15	A	100%
RICHARDSON	1105428	1992-Oct-15	2019-Oct-15	A	100%
RICHARDSON	1105430	1992-Oct-15	2019-Oct-15	A	100%
RICHARDSON	1161080	1991-Dec-19	2018-Dec-19	A	100%
RICHARDSON	1161081	1991-Dec-19	2018-Dec-19	A	100%
RICHARDSON	1161592	1994-Mar-01	2019-Mar-01	A	100%
RICHARDSON	1161604	1994-Mar-01	2019-Mar-01	A	100%
SENN	3008455	2004-Jun-21	2019-Jun-21	A	100%
SENN	3008456	2004-Jun-21	2019-Jun-21	A	100%
SENN	3012530	2006-Feb-13	2019-Feb-13	A	100%
SENN	3016066	2006-Feb-13	2019-Feb-13	A	100%
SENN	3016067	2006-Feb-13	2019-Feb-13	A	100%
SENN	3016068	2006-Feb-13	2019-Feb-13	A	100%
SENN	3016069	2006-Feb-13	2019-Feb-13	A	100%
SENN	3016070	2006-Feb-13	2019-Feb-13	A	100%
SENN	4244249	2009-Jan-28	2019-Jan-28	A	100%
SIFTON	1218904	2012-Jan-09	2019-Jan-09	A	100%
SIFTON	3016793	2009-Sep-25	2019-Sep-25	A	100%
SIFTON	4205802	2006-Nov-27	2019-Nov-27	A	100%
SIFTON	4205803	2006-Nov-27	2018-Nov-27	A	100%
SIFTON	4205804	2006-Nov-27	2018-Nov-27	A	100%
SIFTON	4205819	2006-Nov-22	2019-Nov-22	A	100%
SIFTON	4214441	2010-Mar-03	2019-Mar-03	A	100%
SIFTON	4276421	2015-Jul-16	2019-Jul-16	A	100%
SIFTON	4276487	2017-Oct-13	2019-Oct-13	A	100%
TAIT	4253992	2011-Jan-11	2019-Jan-11	A	100%
TAIT	4253993	2011-Jan-11	2019-Jan-11	A	100%



4-4

Legend:

Boundaries	Populated Places
Province	0 - 500
State	501 - 501
US/Canada	501 - 2000
Transportation	2001 - 5000
Railway	5001 - 100000
Highway	
Major Road	



July 2018

Source: New Gold, 2018.

Figure 4-1

New Gold
Rainy River Mine,
Rainy River, ON, Canada

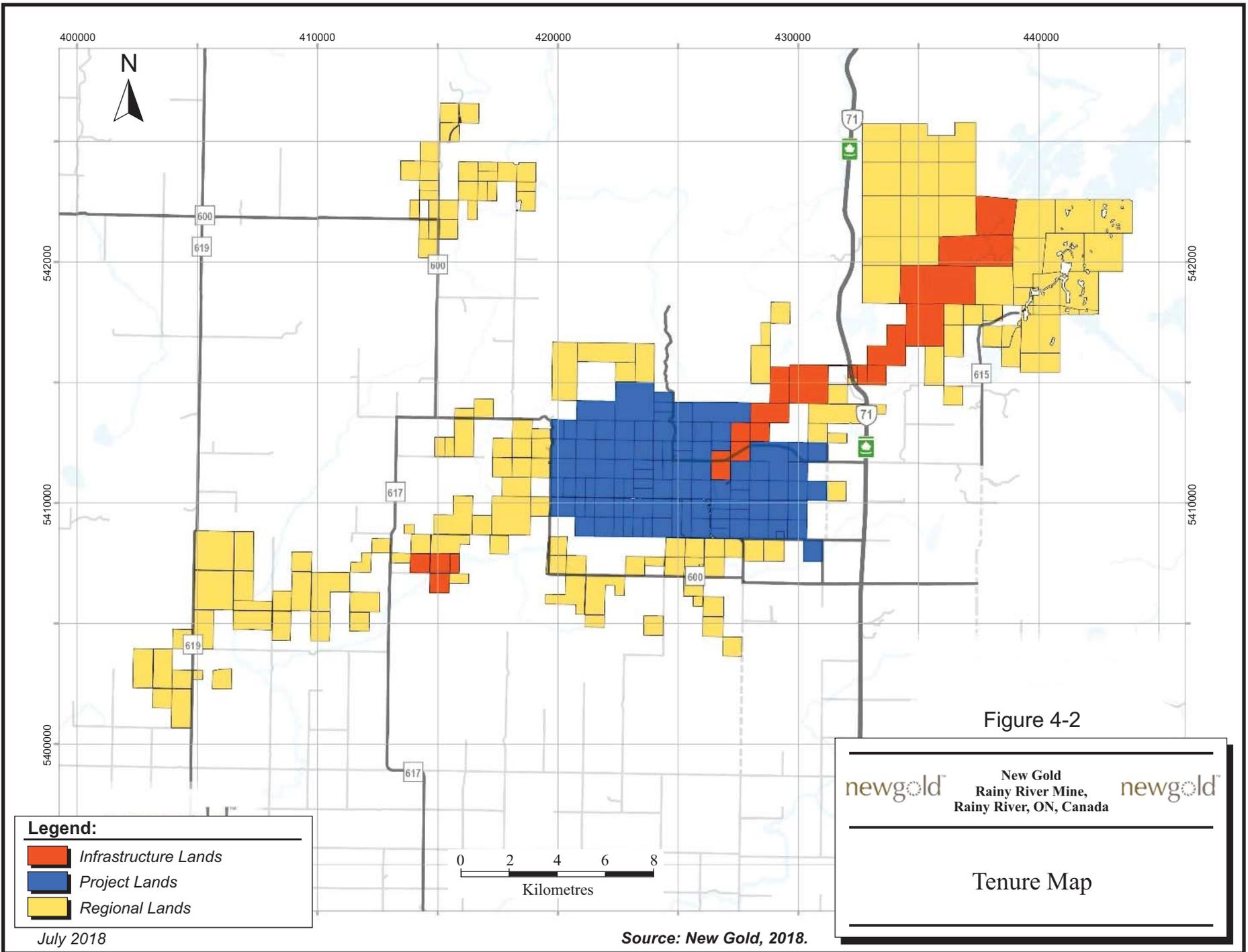
Site Location

ROYALTY AGREEMENTS

Royal Gold, Inc. through its wholly owned subsidiary RGLD Gold AG (Royal) entered into a \$175 million Purchase and Sale Agreement with New Gold in July 2015. The agreement provides Royal with a percentage of the gold and silver production from the Rainy River Mine. New Gold will deliver to Royal:

- 6.5% of the gold produced at Rainy River until 230,000 ounces have been delivered, and 3.25% thereafter.
- 60% of the silver produced at Rainy River until 3.1 million ounces have been delivered, and 30% thereafter.
- Royal will pay New Gold 25% of the spot price per ounce of gold or silver.

New Gold is not aware of any environmental liabilities on the property and has obtained all required permits to conduct the proposed work on the property. New Gold is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work program on the property.



5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

The following section has been summarized from the Feasibility Study of the Rainy River Mine, prepared in 2014 for New Gold by BBA Inc. (BBA) and collaborators (BBA, 2014).

ACCESSIBILITY

The Rainy River Mine is located approximately 50 km to the northwest of Fort Frances, the nearest large town in western Ontario.

The property is centred in Richardson Township (part of Chapple Township) in northwestern Ontario, approximately 152 km south of Kenora, following east along Highway 17 and south on Highway 71 to Korpi Road. Alternative access from Thunder Bay is approximately 418 km along Highway 11 to Emo, Ontario then north on Highway 71 to Korpi Road. These access roads are sealed allowing year-round access.

The Canadian National Railway is located 21 km to the south and runs east-west, immediately north of the Minnesota border. The nearby towns and villages of Fort Frances, Emo, and Rainy River are located along this railway line.

CLIMATE

The climate is typically continental, with extremes in temperatures ranging from +35°C to -40°C, from summer to winter. Annual rainfall in the region averages approximately 60 cm, with heaviest rains expected from June to August, when an average of approximately 30 cm of rain is recorded. An average of 150 cm snowfall is recorded annually in the region.

LOCAL RESOURCES

There are three small towns within immediate driving distance of the Rainy River Mine: Emo (pop. 1,305, 34 km by road), Rainy River (pop. 909, 63 km by road), and Fort Frances (pop. 8,103, 63 km by road).

INFRASTRUCTURE

Hydroelectricity is produced north of Kenora at various locations, as well as west and east of Thunder Bay. A medium-sized coal-powered thermal power station is located east of Fort Frances and another is located near Thunder Bay.

There is a ready supply of water in the area from lakes and rivers. Ground water is also likely to be in plentiful supply, given the abundance of standing water and rivers within the region. The major primary drainage system in the area includes Rainy Lake, which lies to the southeast and is drained by the Rainy River which flows west along the Minnesota border to Lake of the Woods, which in turn feeds into the Lake Winnipeg watershed.

PHYSIOGRAPHY

The Rainy River Mine is divided into two physiographical regions. These regions are separated by a distinct northwest to southeast divider, locally termed the Rainy Lake/Lake of the Woods Moraine, which traverses the countryside immediately to the north of Richardson Township. To the north and east of this moraine, there is a substantial amount of bedrock exposure and topographic relief can be up to 90 m. This relief contrast is controlled by the geology of the granitic batholiths, which have eroded more deeply than the adjacent supracrustal rocks of the Canadian Shield. The area has been subjected to the Whiteshell glacial event from the Labradorian ice centre to the northeast.

The region to the south and west of the moraine is comprised of lowlands. Topographic relief in this region is minimal, glacial overburden is typically twenty to forty metres thick, drainage is poor, and outcrop is limited to less than one percent of the surface area. This area has been exposed to successive glaciations from the northeast and west.

Where covered, the bedrock is immediately overlain by Labradorian till, which in turn is overlain by thick, glaciolacustrine silts and clays of Glacial Lake Agassiz and easterly transported clay and carbonate-rich Keewatin till. Some poorly drained areas are also covered by a thick peat layer.

Vegetation in the area is categorized within the northeastern hardwood region immediately adjacent to the southern margin of the boreal forest.

6 HISTORY

The following section has been summarized from a previous Technical Report prepared for New Gold by BBA dated February 14, 2014, which in turn references documentation of exploration in northwestern Ontario that is archived in the Ministry of Northern Development and Mines (MNDM) offices at Kenora.

PRIOR OWNERSHIP

Exploration in the Rainy River Mine area began in 1967. Various companies and government organizations were active on and around the Rainy River area from 1967 to 1989. These included Noranda, Ontario Division of Mines, Ministry of Natural Resources, International Nickel Corporation of Canada (INCO), Hudson's Bay Exploration and Development (Hudbay), the Ontario Geological Survey (OGS), and Mingold Resources Inc. (Mingold Resources).

Nuinsco held the claims to the Rainy River area from 1990 to 2004, with RRR continuing from 2005 to 2013 when New Gold completed a takeover of RRR on October 15, 2013.

EXPLORATION AND DEVELOPMENT HISTORY

Following the noting of anomalous copper in the region, Noranda registered claims in 1967 and performed geophysics. In 1971, the Ontario Division of Mines, Ministry of Natural Resources continued exploration works through the mapping of the north-central part of the Rainy River Greenstone Belt (Blackburn, 1976). This was followed up by INCO, who undertook ground geophysics, and drilled two holes (results unknown). In 1972, Hudbay undertook airborne and ground geophysics, which was followed up in 1973 with 54 drill holes in the Rainy River Mine region. There was insufficient encouragement to continue and exploration was curtailed.

In 1988, the OGS produced a regional geological map (Map P.3140) of the area based on the interpretation of aeromagnetic data and geological mapping carried out by Johns (1988). This mapping was supported by an OGS rota-sonic drilling program on a 3 km drill grid completed between 1987 and 1988.

The OGS program resulted in the discovery of a “gold grains-in-till” anomaly in Richardson Township.

Mingold Resources followed up on this anomaly in 1988 and staked 85 claims and optioned patented lands in Richardson and some neighbouring townships. Mingold Resources’ use of various sampling methodologies on the till, including reverse circulation (RC) drilling, gave inconclusive results.

Rainy River was acquired by Nuinsco in 1993 and Nuinsco’s exploration activities from 1993 to 2004 are summarized in Table 6-1. Exploration successes of note include the discovery of 17 Zone in 1994, 34 Zone in 1995, and 433 Zone in 1997.

TABLE 6-1 SUMMARY OF NUINSCO EXPLORATION ACTIVITIES

Year	Activity	Company
1993	Rota-sonic drilling	Midwest Drilling
1993	IP survey	Val d'Or Géophysique
1993	Magnetometer survey	Val d'Or Géophysique
1993	Landsat linear study	DOZ Consulting Group
1993	Reconnaissance mapping and sampling	Nuinsco Resources
1994	Rota-sonic drilling	Midwest Drilling
1994	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
1994	Diamond drilling	Ultra Mobile Diamond Drilling
1994	Grid mapping and sampling	Nuinsco Resources
1994	Soil Sampling/Enzyme Leach	Nuinsco Resources
1995	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
1995	Diamond drilling	Ultra Mobile Diamond Drilling
1995	IP survey	JVX Geophysics
1995	Trenching and stripping, mapping	Nuinsco Resources
1995	Soil Sampling/Enzyme Leach	Nuinsco Resources
1996	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
1996	Diamond drilling	Ultra Mobile Diamond Drilling
1996	Diamond drilling	Bradley Brothers Diamond Drilling
1996	UTEM survey	Lamontagne Geophysics
1996	Surface pulse EM	Crone Geophysics
1996	Surface pulse EM	JVX Geophysics
1996	Borehole pulse EM	Crone Geophysics
1996	Borehole pulse EM	Crone Geophysics
1996	Borehole pulse EM	JVX Geophysics
1996	IP survey	JVX Geophysics

Year	Activity	Company
1996	Magnetometer survey	JVX Geophysics
1996	Outcrop stripping	Nuinsco Resources
1997	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
1997	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
1997	Diamond drilling	Ultra Mobile Diamond Drilling
1997	Diamond drilling	Bradley Brothers Diamond Drilling
1997	Airborne EM and Magnetic survey	Geoterrex-Dighem
1997	Surface pulse EM	Crone Geophysics
1997	Borehole pulse EM	Crone Geophysics
1997	Borehole pulse EM	Crone Geophysics
1997	IP survey	Quantec IP
1997	Local detailed mapping	Nuinsco Resources
1997	Outcrop stripping	Nuinsco Resources
1998	Surface pulse EM survey	Crone Geophysics
1998	Diamond drilling	Ultra Mobile Diamond Drilling.
1998	Reverse circulation drilling	Bradley Bros.- Overburden Drilling
1998	Line cutting/Magnetometer survey	Mtec Geophysics Inc.
1998	Diamond Drilling	Ultra Mobile Diamond Drilling
1999	Diamond Drilling	Ultra Mobile Diamond Drilling
1999	Diamond Drilling	Bradley Brothers Diamond Drilling
2000	Airborne EM and Magnetic Survey	Aeroquest Limited
2000/2001	Geochemical Compilation	Franklin Geoscience and Nuinsco Personnel
2001/2002	Magnetotelluric Geophysical Survey	Phoenix Geophysics
2001	Mapping/Prospecting	Nuinsco Resources
2001/2002	Diamond Drilling	Diamond Drilling, Bradley Brothers
2004	Diamond Drilling	

Source: Mackie et al. 2003

Note. IP – induced polarization; EM –electromagnetics, UTEM - University of Toronto electromagnetic system

Upon acquisition of the property from Nuinsco in June 2005, RRR relogged key sections of the drill core and input available data into a GIS database. In excess of 100 RC holes were completed to better define the gold-in-till and gold-in-bedrock anomalies.

Several exploration and infill drilling campaigns were undertaken from 2005 to 2013 by RRR, the details of which are included in Section 10. The Intrepid Zone was covered by a mobile metal ion (MMI) soil survey in 2013 which identified anomalous gold over a prominent magnetic low trend similar to the magnetic trend that hosts the majority of the Rainy River Mine deposits.

A summary of exploration activities by RRR, including commissioned studies and excluding drilling, is provided in Table 6-2.

TABLE 6-2 SUMMARY OF RRR EXPLORATION ACTIVITIES

Year	Activity	Company
2005	Re-Log 13 Diamond Drill Holes	L.D. Ayres
2005	Summary of Structural Observations	G. Zhang
2005	Re-Log 8 Nuinsco Diamond Drill Holes	L.D. Ayres
2005	Petrography and Mineralogy	R.P. Taylor
2005	Structure and Geology of Caldera Model	L.D. Ayres
2005	Structure and Geology of Richardson Twp	H. Paulsen
2006	Report of Re-Logging of Nuinsco DD Core	L.D. Ayres
2006	VTEM Airborne Geophysical Survey	Geotech Limited
2006	U-Pb Zircon Age Dating	Geospec Consultants Limited
2006	Petrographic and Mineralogical Report	E. Schandl
2006	Structure and Geology Review	K. H. Paulsen
2006	U-Pb Zircon Age Dating	Geospec Consultants Limited
2006	3D Borehole Pulse EM Survey	Crone Geophysics and Exploration
2007	IP Survey of 9 Holes, 3D Conductivity Inversion Models	JVX Limited
2007	Line Cutting	Archer Exploration Inc.
2007	Ground Gravity and EM Survey	Abitibi Geophysics
2008	Titan 24 Survey	Quantec Geoscience
2008	Airborne Magnetic Gradiometer Survey	Fugro Airborne Surveys, Corp.
2008	Regional Geophysical Interpretation	J. Siddorn – SRK Consulting (Canada) Inc.
2008	Socio-Economic Scoping Study Draft Report	Klohn, Crippen and Berger Ltd.
2008	Preliminary Pit Slope Design and Waste Management Assessment	Klohn, Crippen and Berger Ltd.
2009	Age Dating of Lithologies	University of Toronto Geochronology Lab
2009	Surficial Drainage Project	K. Smart Associates Limited
2009	Socio-Environmental Baseline Assessment	Klohn Crippen Berger Ltd.
2009	Acid Leach Test	Klohn Crippen Berger Ltd.
2009	Lidar Survey	Lidar Services International
2010	Preliminary Metallurgical Testing	SGS Canada Inc.
2010	Metallurgical Testwork	SGS Canada Inc.
2010	Environmental Baseline Studies	Klohn Crippen Berger
2010	DD 4 Geotechnical Drill Holes (1,405 m)	Klohn Crippen Berger Ltd.
2010	Review of Pit Slope Design	SRK Consulting (Canada) Inc.
2010	Memorandum of Understanding with Fort Francis Chiefs Secretariat	Rainy River Resources Ltd.
2010	Structural Study	SRK Consulting (Canada) Inc.
2010	M.Sc Thesis on Richardson Deposit	J. Wartman - University of Minnesota

Year	Activity	Company
2010	Pre-Feasibility Open Pit Slope Design	Klohn Crippen Berger
2010	New Core Logging Facility	C. Hercun, True-line Construction
2010	Line Cutting Geophysical Grid 33 km	Archer Exploration Inc.
2010	Titan Survey 33 km	Quantec Geoscience
2010	Application for Advanced Exploration Permit	G. Macdonald, K. Stanfield
2011	88 km High-Sensitivity Potassium Magnetometer Ground Survey	RDF Consulting
2011	Environmental Baseline Gap Analysis	AMEC Earth and Environmental
2011	First Quarter QA/QC Report	Analytical Solutions Ltd.
2011	Fugro AEM Survey	Fugro Airborne Surveys Corp.
2011	Report on Ground Gravity Surveys	Eastern Geophysics, Gerard Lambert
2011	Report on Borehole Surveys	Eastern Geophysics, Gerard Lambert
2012	Mobile metal ion soil surveys - various	Rainy River Resources Ltd.
2012	Report on 34 zone & Pinewood Ni, Cu & PGE mineralization	Revelation Geoscience Ltd.
2012	Intrepid specific gravity data	ALS Chemex Laboratory
2013	Soil gas hydrocarbon orientation survey	Rainy River Resources Ltd.

Note. VTEM – versatile time domain electromagnetic; LiDAR – light detection and ranging; AEM – airborne electromagnetics

HISTORICAL RESOURCE ESTIMATES

Ten Mineral Resource estimates were prepared for the Rainy River Mine from 2003 to 2017. Authors of these reports include Mackie et al. in 2003, Caracle Creek International Consulting Inc. (CCIC) in 2008, SRK Consulting (Canada) Inc. (SRK) in 2009, 2010, 2011, and 2012, BBA and collaborators in 2014 (Feasibility Study), and New Gold and SRK in 2015. A total of seven of these Mineral Resource estimates are documented in previous technical reports prepared for Rainy River which are available on SEDAR. The current Mineral Resource estimate contained in Section 14 of this report supersedes all previous estimates.

PAST PRODUCTION

There is no historical production from the Rainy River Mine.

7 GEOLOGICAL SETTING AND MINERALIZATION

REGIONAL GEOLOGY

The Mine is located within the Neoproterozoic Rainy River Greenstone Belt (RRGB) which formed approximately 2.7 billion years ago (Ga). The RRGB forms part of the western Wabigoon subprovince, located in the Superior Province of the Canadian Shield. The Wabigoon subprovince is located in the western portion of the Superior Province (Figure 7-1). It is a 900 km long, east-west trending composite volcanic and plutonic terrane comprising distinct eastern and western domains separated by rocks of Mesoproterozoic age (Percival et al., 2006).

The western Wabigoon domain is predominantly composed of mafic volcanic rocks intruded by tonalite-granodiorite intrusions. The volcanic rocks, which were largely deposited between approximately 2.74 Ga and 2.72 Ga, range from tholeiitic to calc-alkaline in composition, and are interpreted to represent oceanic crust and volcanic arcs, respectively (Percival et al. 2006). These are succeeded by approximately 2.71 Ga to 2.70 Ga volcano-sedimentary sequences and by locally deposited, unconformable, immature clastic sedimentary sequences.

The volcanic rocks have been intruded by a wide variety of plutonic rocks including synvolcanic tonalite-diorite-granodiorite batholiths, younger granodiorite batholiths, sanukitoid monzodiorite intrusions and monzogranite batholiths and plutons. The intrusions were emplaced over a large time span from approximately 2.74 Ga to 2.66 Ga (Percival et al., 2006).

In the region east of Fort Frances, the Wabigoon subprovince is bounded to the south by the late Archean, dextral Seine River–Rainy Lake and Quetico faults. The Quetico Fault splays off the subprovince boundary and strikes west through the western Wabigoon domain just south of the Mine. The RRGB is bounded to the north by the Sabaskong Batholith and to the east by the Rainy Lake Batholithic Complex and is contiguous with the Kakagi-Rowan Lakes Greenstone Belt to the north.

The regional metamorphic grade of the Archean rocks is greenschist to lower-middle amphibolite facies. Locally, adjacent to the intruding batholiths, upper amphibolite mineral assemblages are recognized.

Significant metallic mineral deposits hosted in the western Wabigoon domain include the Cameron Lake gold deposit hosted in the adjacent Kakagi–Rowan Lakes Greenstone Belt, the Hammond Reef gold deposit 190 km to the east of the Rainy River Gold Mine, and the Sturgeon Lake Volcanogenic Massive Sulphide (VMS) deposits 250 km to the northeast of the Rainy River Mine.

Three phases of the Quaternary Wisconsinan glaciation are recorded in the Rainy River area (Barnett, 1992). The Archean basement rocks and locally preserved Mesozoic sediments are overlain by till deposited from the Labrador Sector of the Laurentide Ice Sheet. Its provenance area is the Archean basement of the Canadian Shield to the northeast. In the Rainy River area, this till has been found to contain highly anomalous concentrations of gold grains, auriferous pyrite, and copper-zinc sulphides. As the Labradorean ice sheet retreated, a thick, electrically conductive, barren glaciolacustrine clay and silt horizon originating from glacial Lake Agassiz was deposited.

The Keewatin Sector of the Laurentide Ice Sheet then advanced over the area and deposited an argillaceous till of western provenance on top of the clay and silt horizon. The Mine area was therefore successively covered by the Labradorean and Keewatin ice sheets.



Figure 7-1

newgold[™] New Gold
 Rainy River Mine,
 Rainy River, ON, Canada

Superior Province Geological Map

LOCAL AND PROPERTY GEOLOGY

The geology of the Rainy River deposit is inferred from regional field mapping of limited rock exposures, extensive RC and diamond core drilling by Nuinsco, RRR and New Gold, OGS rota-sonic drilling, and airborne geophysics.

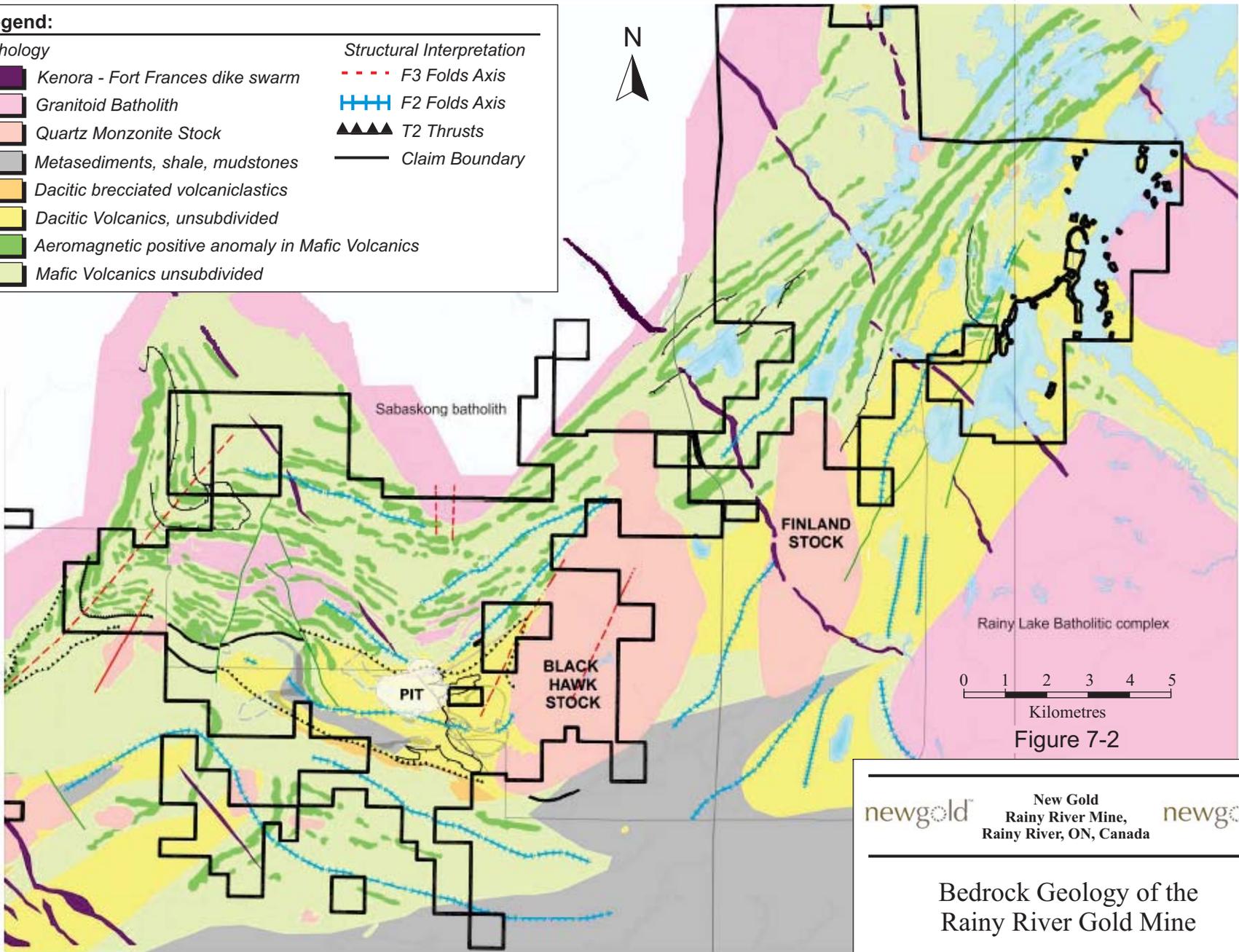
A bedrock geological interpretation produced by RRR for the area surrounding Rainy River is shown in Figure 7-2.

The Rainy River deposit is mainly hosted in intermediate to felsic, calc-alkaline metavolcanic rocks, bounded to the north and south by a series of tholeiitic mafic units. These strike almost east-west and dip to the south, subparallel to the main foliation recognized in the area. On the southern end of the property, metasedimentary rocks are found with a parallel orientation to the volcanic package. To the north of the property, the lower mafic volcanic sequence has been intruded by the trondhjemitic Sabaskong batholith. The Black Hawk monzonitic stock located to the east of the deposit truncates the felsic volcanic package of the volcanic sequence.

Intermediate dacitic rocks host most of the Rainy River gold mineralization.

Legend:

<i>Lithology</i>		<i>Structural Interpretation</i>	
	Kenora - Fort Frances dike swarm		F3 Folds Axis
	Granitoid Batholith		F2 Folds Axis
	Quartz Monzonite Stock		T2 Thrusts
	Metasediments, shale, mudstones		Claim Boundary
	Dacitic brecciated volcaniclastics		
	Dacitic Volcanics, unsubdivided		
	Aeromagnetic positive anomaly in Mafic Volcanics		
	Mafic Volcanics unsubdivided		



Rainy Lake Batholithic complex

0 1 2 3 4 5
Kilometres

Figure 7-2

newgold™ New Gold
 Rainy River Mine,
 Rainy River, ON, Canada

Bedrock Geology of the
 Rainy River Gold Mine

LITHOLOGY

LOWER MAFIC VOLCANIC SUCCESSION

The Rainy River area is primarily underlain by tholeiitic metavolcanic rocks of the western Wabigoon subprovince – the RRGB. Geochemically they are high-iron and high-magnesium basalts comprising coarse-grained massive lava flows, massive and pillow flows, and flow breccia. Subordinate dacitic tuff and intrusive quartz-feldspar porphyry dikes and sills are commonly noted interbedded or intruding respectively throughout the mafic volcanic rock.

INTERMEDIATE-FELSIC PORPHYRITIC INTRUSIVE ROCK

Swarms of porphyritic intermediate to felsic dikes cut through the Lower Mafic volcanic succession. They range in thickness up to several tens of metres. It has been suggested that these dikes may have been the conduits that fed the overlying intermediate succession hosting the mineralization. They have been variably interpreted and often described as dacitic tuffs due to their similar composition and appearance to units noted within the overlying intermediate succession. Historically, these complex and strongly deformed units have been denoted as the Georgeson/Feeder Porphyries.

UPPER FELSIC SUCCESSION

The Upper Felsic Succession overlies the Intermediate Succession along the southern boundary of Richardson Township. The Upper Felsic Succession is a few hundred metres thick and has been traced for four kilometres westwards from the Black Hawk Stock. It has been interpreted as a quartz-phyric rhyolite.

PINEWOOD SEDIMENT SUCCESSION

The Pinewood sedimentary rock package is composed of predominantly clastic intermediate derived wacke and argillite. The sequence conformably overlies the upper diverse mafic volcanic rocks, and the contact is typically marked by a pyritic heavy metal-bearing graphitic horizon. The upper contact of the succession is interbedded with the Upper Felsic Succession.

PYRITIC SEDIMENT SUCCESSION

Conformably overlying the Lower Mafic Volcanic Succession are a series of pyrite-bearing siliceous to chloritic wacke, interpreted as derived from intermediate to mafic volcanic sediments. These horizons are increasingly interbedded with homogenous and nondescript to quartz-eye dacite tuff horizons as the upper contact is approached, and these tuff horizons likely represent onset of the lateral equivalent of subsequent intermediate volcanism.

ULTRAMAFIC-MAFIC INTRUSION

Thin zones of ultramafic to mafic intrusions have been noted in drill core. They form dikes or sills intruding the volcanic stratigraphy at different times. Their sulphide content is typically below 2%. The 34 Zone is hosted in a late-stage mafic-ultramafic intrusion, crosscutting the 17 Zone. The main lithological units include dunite, pyroxenite, pyroxene-gabbro, and gabbro. The lowermost units contain significant sulphide mineralization enriched in copper, nickel, gold, and platinum group metals.

INTERMEDIATE FRAGMENTAL VOLCANIC SUCCESSION

The Intermediate Succession is complex. Immediately overlying the pyritic sediment horizon in Richardson Township, these volcanoclastic rocks are composed of fine-grained “quartz-eye” dacite and fine-grained ash horizons with subordinate interbedded coarse-grained lapilli tuff and localized sedimentary and exhalative horizons. A high proportion of what appear to be coarse volcanoclastic rocks may in fact be massive flows or tuffs overprinted by strong, anastomosing foliation and sericite alteration. Geochemically these intermediate rocks have been interpreted as calc-alkaline dacite with subordinate rhyolite and andesite. Some blocks of tuff breccia have been observed juxtaposed against the Black Hawk Stock which intrudes and notably alters the volcanoclastic rocks to the east. The rocks of the intermediate succession dip 50° to 70° to the south in the Richardson area and are the principal host of the mineralization in the ODM/17, 433, Beaver Pond, Western, and HS Zones.

BLACK HAWK STOCK

This quartz monzonitic to granodioritic stock consists of two phases and represents a topographic high to the east. The early phase forms the rim of the stock, and is a weakly foliated, notably magnetic, massive to pegmatitic quartz monzonite with minor subordinate granodiorite. The late phase consists of equigranular coarse grained granodiorite, and forms the central core of the stock. Associated magnetic aplitic to pegmatitic dikes compositionally similar to the early phase intrude the surrounding metavolcanic rocks.

MASSIVE LAVA FLOWS

Immediately overlying the intermediate fragmental volcanic rocks are a series of intermediate to mafic volcanic massive lava flows, ranging from fine-grained porphyritic quartz dacite, to massive magnetite-bearing mafic volcanic rocks, with localized pillowed mafic flows. These units are notably homogenous, and the intermediate volcanic units often show a diagnostic

deformed sericitic net-textured compression fracture pattern. Upper and lower contacts display a centimetre scale shear fabric at the margins.

UPPER DIVERSE MAFIC VOLCANICS

The upper diverse mafic volcanic succession is composed of a series of mafic tuffs, massive to glomeroporphyritic mafic flows, localized pillowed flows, interflow sediment and hyaloclastite, and minor subordinate intermediate volcanic tuffs. The rocks of the upper diverse mafic volcanics are the principal host of the CAP Zone mineralization.

PROTEROZOIC DIABASE DIKE

A northwest striking, steeply dipping diabase dike cross-cuts the ODM/17 Zone and extends across the entire Project area.

STRUCTURAL GEOLOGY

The volcano-sedimentary sequences of the Rainy River area and regional greenstone belt were affected by at least five main deformation episodes (Figure 7-3).

All rock types within the Intrepid mineralized zone, with the exception of the late diabase dykes, have a well-developed, moderately south dipping, penetrative foliation and a moderately southwest plunging stretching lineation (Figure 7-4).

These fabrics commonly obliterate the original layering and a primary folding event (F1), consisting of large recumbent and low plunging folds with a north-south axial plane parallel to the strike of a steep axial-plane foliation (S1). The folding event was also accompanied by localized T1 thrusts (potentially at regional scale) (Rankin, 2013). The pre-D1 mineralized veins are strongly folded and commonly transposed into the S1 foliation.

A second folding event (F2) consisting of east-southeast trending upright to overturned shallow plunging folds of variable intensity, refolded S1 and L1 with variable dips and plunges across the belt. The Rainy River auriferous zones lie within a moderate to steeply dipping F2 limb with S0/S1 trending 110/55 (average) and L1 steep south-southwestern plunge (~down-dip). This forms the northern limb of a regional F2. F2 folding was possibly accompanied by T2 thrust to high angle reverse faults, partitioning subdomains with varying D2 strain. A weak F2 axial-planar foliation is locally visible in both drill core and outcrop. F2 fold axes are typically subhorizontal to shallowly plunging (Rankin, 2013). Steeply plunging ore-shoots within the

mineralized zones probably represent localized F1 fold hinges, forming thickened zones of early veins. Termination of ore shoots down-plunge may locally be due to refolding of F1 about local F2 folds at an oblique angle.

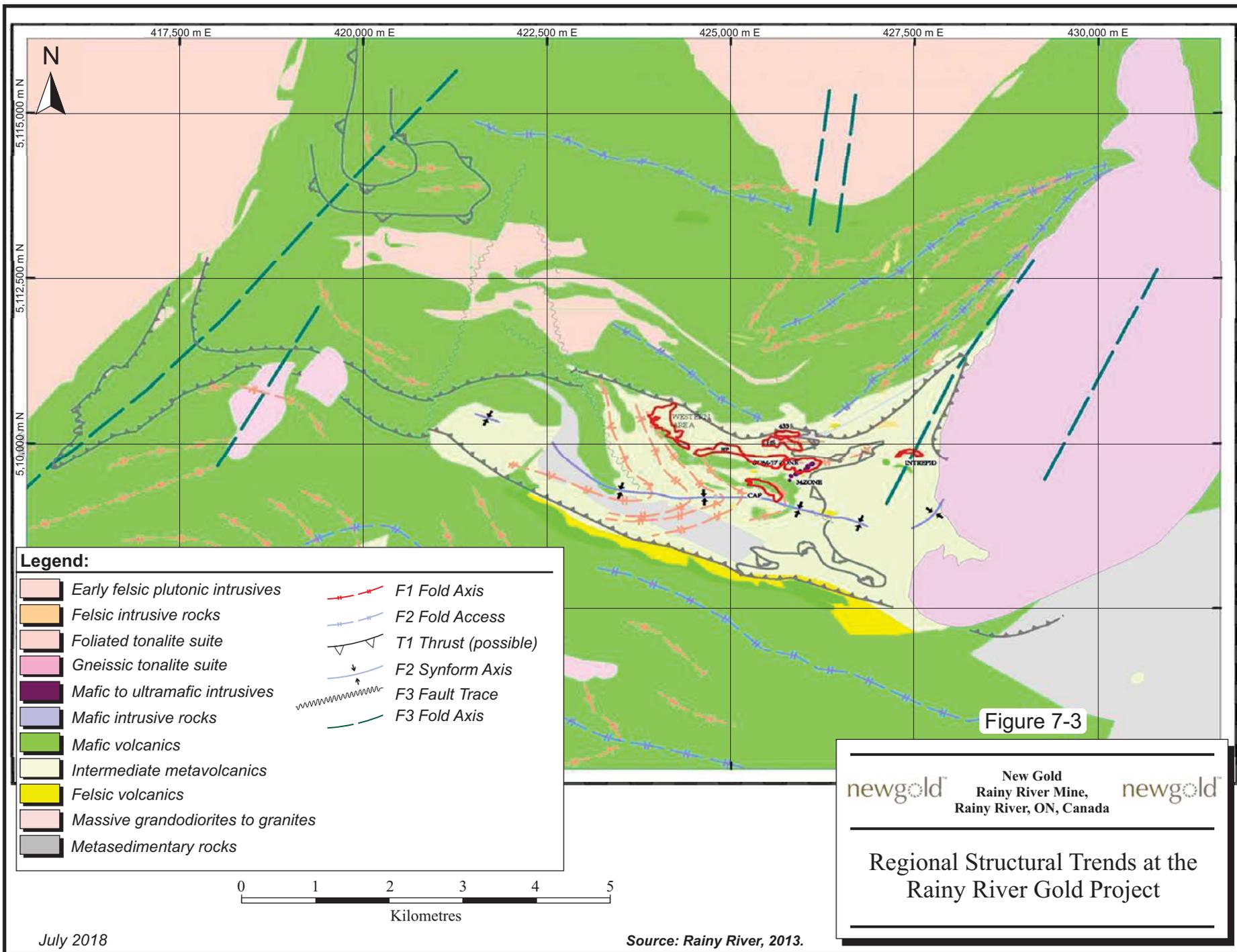
Broad-scale bends (D3) in the D1/D2 structural grain followed as kink folds in the greenstone belt. These trend north-northwest to northeast (with some conjugate kink geometry evident). F3 folds are associated with subvertical S3 spaced fracture cleavages to small scale faults (Figure 7-5).

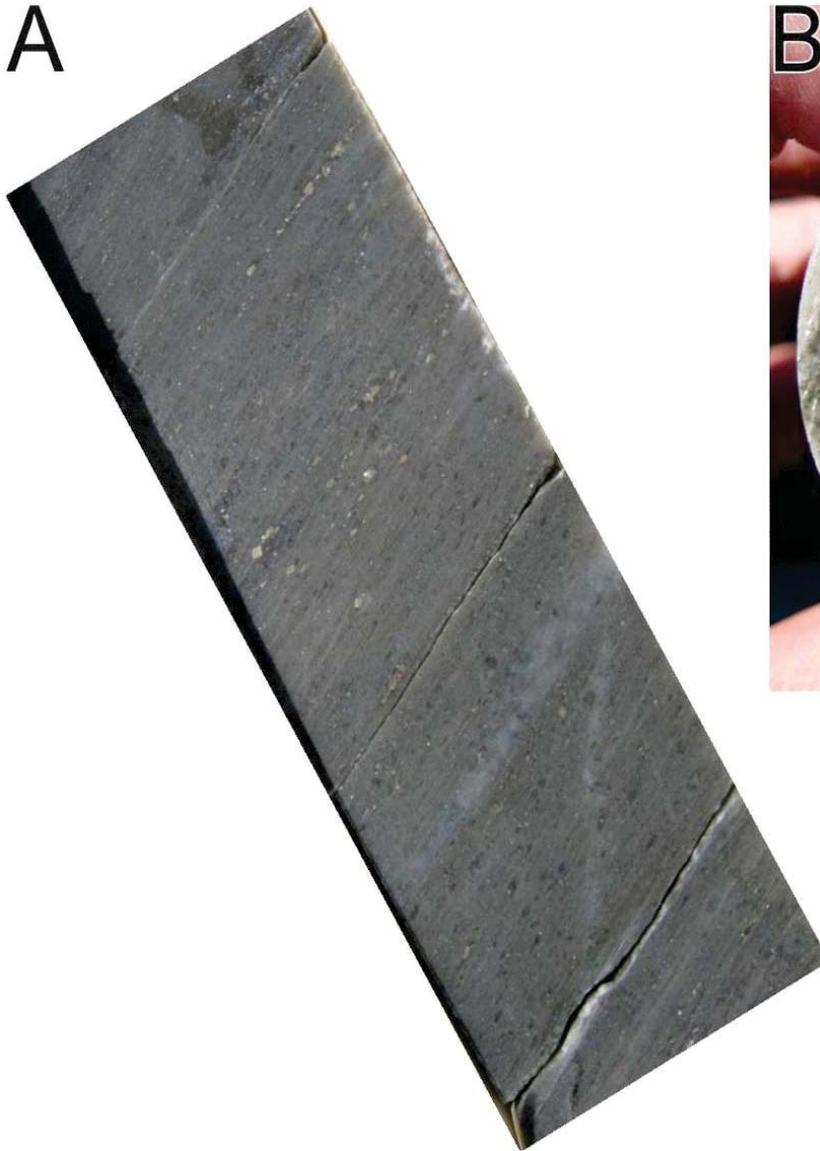
A consistent sinistral displacement along these structures may be due to progressive rotation of the compressive stress direction from D3 to D4. Small-scale F3 kinks are common within the layered sequences in outcrop and drill core. Very localized remobilization of quartz-sulphide (as veinlets) into the kink axial planes may have produced small zones of enriched mineralization. F3 fold axes typically plunge steeply, and occur where folds are steeply dipping (S0/S1 fabrics). Emplacement of north trending granitoid stocks east of the Rainy River Mine is interpreted to have occurred along F3 kink axes (possible reactivated basement faults) (Rankin, 2013).

D4 is represented by a late-stage north-northwest–south-southeast to north-south compressive episode causing broad warping of all pre-existing fabrics, including F3 mega-kink axial planes. D4 is interpreted to have also caused both flat-lying breccia bodies with late-stage kaolin-sericite alteration in the Intrepid area (subhorizontal tension gash structures), and a weak east-southeast trending foliation in the Black Hawk granitoid stock.

The final deformation event, D5, is represented by late stage (Proterozoic) emplacement of northwest trending mafic dykes (northeast-southwest extension).

- A. Strong, penetrative foliation in sericite-quartz-altered, quartz-phyric rocks. Photo is rotated to approximate dip of borehole (NR09399, 672.2 m).
- B. Southwest raking stretching lineation (NR09402, 245.9 m).





Legend:

A: Strong, penetrative foliation in sericite-quartz-altered, quartz-phyric rocks. Photo is rotated to approximate dip of borehole (NR09399, 672.2 m).

B: Southwest raking stretching lineation (NR09402, 245.9 m).

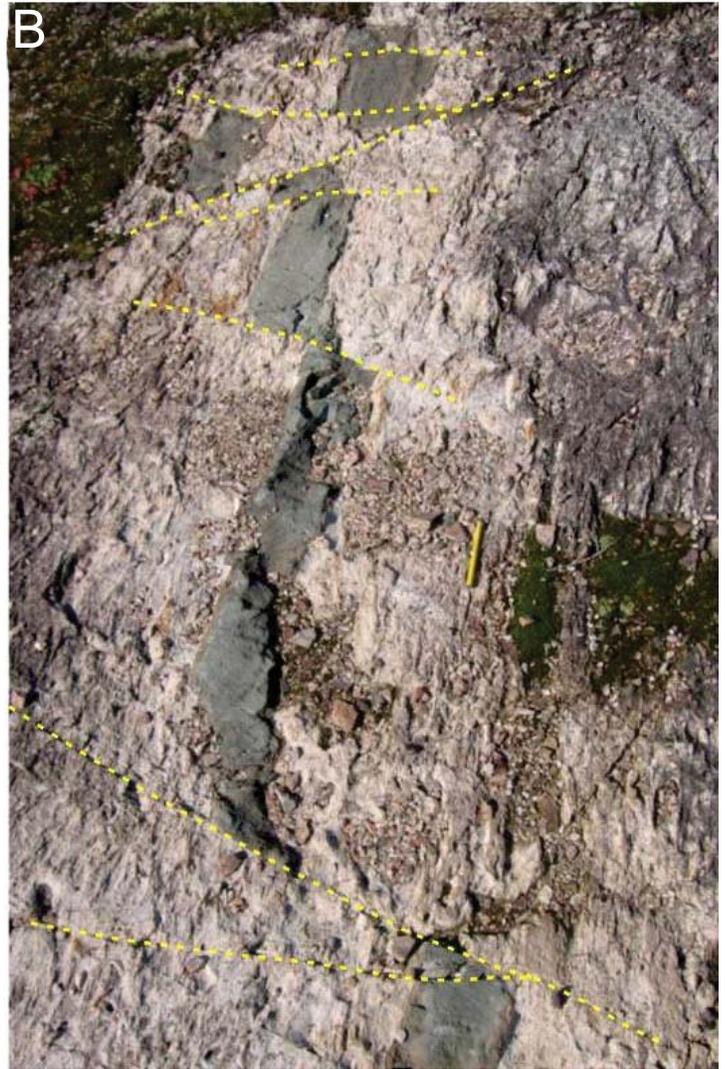
July 2018

Source: SRK, 2011.

Figure 7-4

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 Rainy River Mine,
 Rainy River, ON, Canada

Structural Fabrics at Rainy River



Facing West

Figure 7-5

Legend:

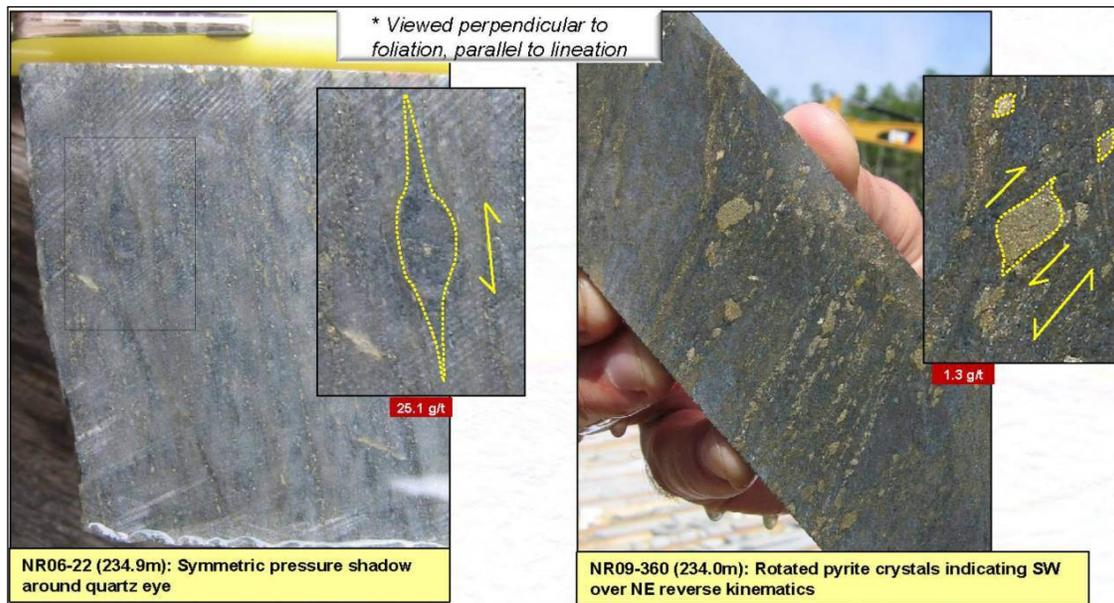
- A:** Sub-horizontal striations on calcite-filled, north-striking sub-vertical strike-slip faults (borehole NR0505, 150.6 m).
- B:** Outcrop (42,255 mE / 5,409,525 mN) showing set of 360° to 020° striking, subvertical faults offsetting mafic dike, predominantly with sinistral separation in this area.

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 Rainy River Mine,
 Rainy River, ON, Canada newgold[™]

Features at Rainy River showing
 Brittle Strike-Slip Faulting

A late-stage north-northwest–south-southeast to north-south compressive episode caused broad warping of all pre-existing fabrics, including F3 mega-kink axial planes (Figure 7-6). This D4 episode is interpreted to have also caused both flat-lying breccia bodies with late-stage kaolin-sericite alteration in the Intrepid Zone (subhorizontal tension gash structures) and a weak east-southeast trending foliation in the Black Hawk granitoid stock.

FIGURE 7-6 PRESSURE SHADOWS AROUND RIGID OBJECTS IN DACITIC ROCK



SRK structural analyses (Siddorn, 2007; Hrabi and Vos, 2010) have noted that the gold mineralization is strongly overprinted by subsequent deformation.

Key observations in core and outcrop include:

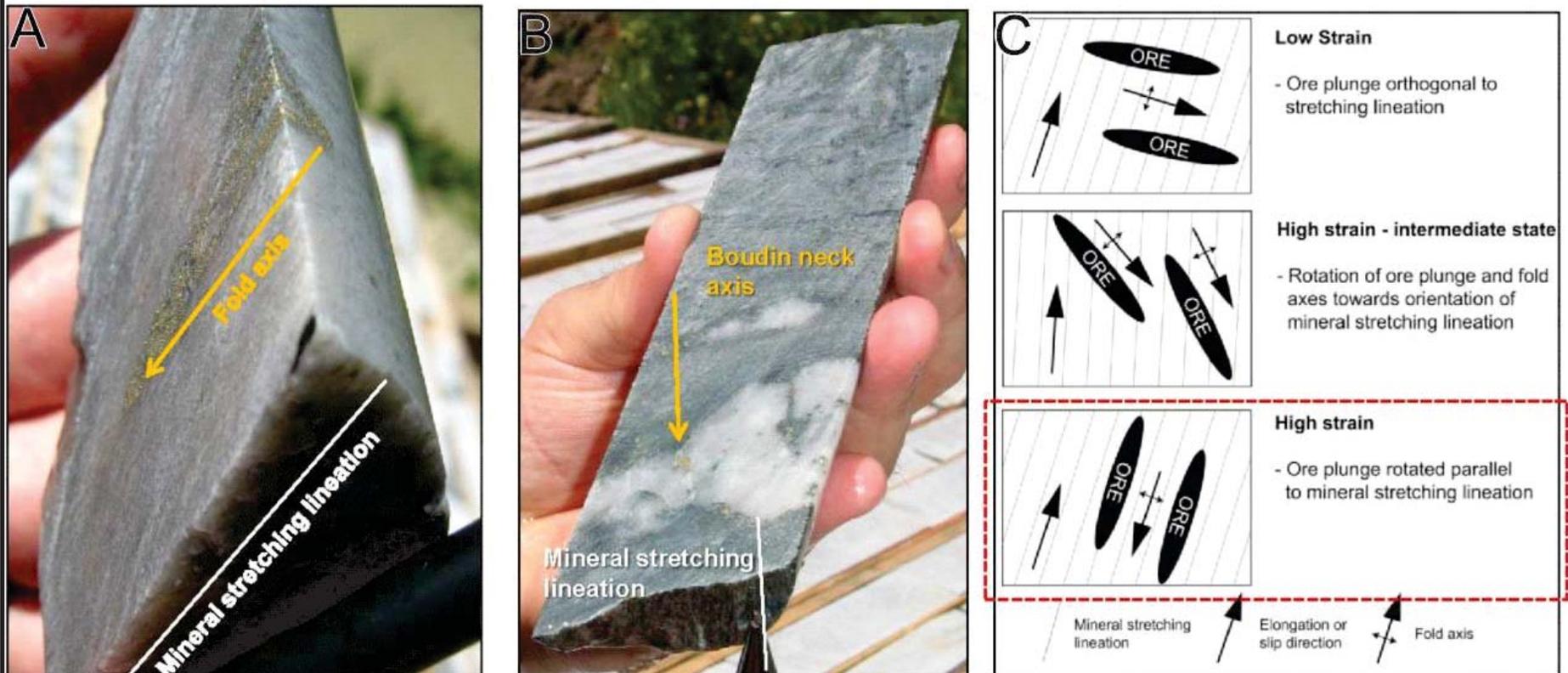
- Auriferous mineralization is aligned along the regional foliation.
- Fold axes of auriferous quartz veins and sulphide stringers are rotated subparallel to the stretching lineation.
- Fold axes, boudin necks, and stretching lineation are subparallel to the plunge of the gold mineralization.
- Early sulphide mineralization is deformed by folding (Figure 7-7).
- Later quartz-sulphide veins are variably deformed and overlap in time with the main regional deformation.

FIGURE 7-7 SULPHIDE MINERALIZATION DEFORMED BY FOLDING IN DRILL CORE FROM THE RAINY RIVER PROJECT



Source: SRK, 2011

This strongly suggests that the current geometry and plunge of the gold mineralization at Rainy River is the result of high strain deforming features associated with gold mineralization and rotating the ore plunge parallel to the stretching direction (Figure 7-8).



Legend:

- A:** Fold axis of pyrite stringer vein rotated parallel to mineral lineation (NR09408, 459.5 m).
- B:** Boudinaged quartz vein with boudin neck parallel to stretching lineation (NR09360, 766.4 m).
- C:** Diagram illustrating rotation of ore plunge in high strain deformation (modified from Robert and Poulsen, 2001).

July 2018

Source: Rainy River, 2013.

Figure 7-8

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Structural Control over the Plunge of
Gold Mineralization at the
Rainy River Project

MINERALIZATION

A total of four main styles of mineralization have been identified at Rainy River:

1. Moderately to strongly deformed, auriferous sulphide and quartz-sulphide stringers and veins in felsic quartz-phyric rocks (ODM/17, Beaver Pond, 433 and HS Zones);
2. Deformed quartz-ankerite-pyrite shear veins in mafic volcanic rocks (CAP/South Zone);
3. Deformed sulphide-bearing quartz veinlets in dacitic tuffs/breccias hosting enriched silver grades (Intrepid Zone); and
4. Copper-nickel-platinum group metals mineralization hosted in a younger mafic-ultramafic intrusion (34 Zone).

MINERALIZATION WITHIN AURIFEROUS SULPHIDE AND QUARTZ-SULPHIDE STRINGERS AND VEINS IN FELSIC QUARTZ- PHYRIC ROCKS

The bulk of the gold mineralization at Rainy River is contained in sulphide and quartz-sulphide stringers and veins hosted by felsic quartz-phyric rocks. Two main zones are recognized (ODM/17 and 433 Zones) with subsidiary zones (HS and New), which are mostly bounded by high strain zones.

ODM/17 ZONE

A total of three styles of gold mineralization can be observed in the ODM/17 Zone. Low grade intervals are characterized by tightly folded pyrite stringer veins and disseminated pyrite in sericite-quartz-chlorite altered host rocks.

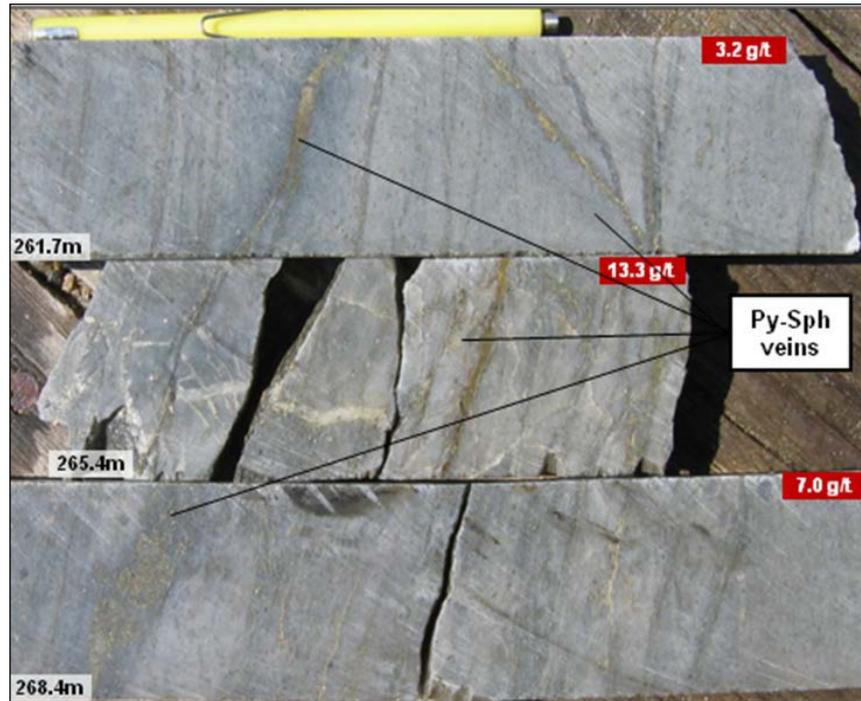
Low- to moderate-grade (up to approximately 10 g/t Au) intervals are characterized by tightly folded and foliation parallel pyrite-sphalerite and pyrite stringer veins, commonly associated with stronger silica and weak garnet alteration (Figure 7-9). High grade gold mineralization in this zone is associated with deformed quartz-pyrite-gold veinlets (Figure 7-10) that overprint other mineralization styles.

The low grade ODM/17 Zone is modelled over a strike length of approximately 1,600 m, over a vertical distance of approximately 975 m, and over a true width of up to 200 m.

The medium-grade Beaver Pond subdomain (Subdomain 114) is located 300 m to the west of the ODM/17 Zone and is included within the larger ODM/17 low grade domain. Mineralization is similar in style and character to the ODM/17 Zone, including the presence of deformed gold-

rich quartz veinlets, although generally the widths of auriferous drill intercepts are narrower. Mineralization in the ODM/17 Zone is open below the modelled depth.

FIGURE 7-9 ODM/17 ZONE GOLD MINERALIZATION



Deformed pyrite-sphalerite veins and stringers parallel to, or obliquely to foliation in quartz- sericite-chlorite altered rocks (Borehole NR0651 at downhole interval, as indicated).

FIGURE 7-10 ODM/17 HIGH GRADE GOLD MINERALIZATION



Deformed quartz-pyrite vein with visible gold emplaced along boudin neck (Borehole NR0651 at 251.1 m; 195.5 g/t gold over 1 m core length interval).

MINERALIZATION WITHIN 433, HS AND NEW ZONES

The style of gold mineralization in the 433 Zone is similar to that observed in the ODM/17 Zone, although some differences are apparent. These include an overall dominance of chlorite alteration (relative to dominant sericite in the ODM/17 Zone) of quartz-phyric host rocks, occurrences of chlorite-pyrite altered heterolithic conglomerates, and the occurrence of chalcopyrite and chlorite with high grade quartz-pyrite-gold veinlets (Figure 7-11).

FIGURE 7-11 433 HIGH GRADE GOLD MINERALIZATION

Deformed quartz-pyrite-chalcopyrite-chlorite-gold veins cross-cutting foliation and disseminated pyrite in quartz-sericite altered quartz-phyric rock (Borehole NR07-218 at 305.2 m; 4,159 g/t gold over 1 m core length interval).

Source: SRK, 2011

The modelled low grade 433 Zone has a flattened oblate shape that plunges moderately to the southwest. The zone extends over a strike length of approximately 325 m, over a vertical distance of approximately 820 m, and a true width of up to 125 m.

Several subsidiary zones of gold mineralization are identified at Rainy River, including the HS and New Zones. The HS and New Zones are located north of and structurally beneath the ODM/17 Zone and slightly above the 433 Zone. The full extent of the HS Zone has not been defined by drilling to date.

The HS Zone defines a plunging, flattened oblate shape subparallel to the ODM/17 and 433 Zones which hosts discontinuous, irregular low grade gold mineralization associated with chlorite-pyrite replacement of matrix in flattened, albitized heterolithic pebble conglomerates. The New Zone is more irregular in shape, comprising a number of small zones in the immediate hangingwall of the 433 Zone. The zones have a strike length of approximately 200 and 275 m respectively, and both extend a vertical distance of approximately 700 m.

Mineralization is open below the modelled depth for each of these zones.

THE WESTERN ZONE

The Western Zone occurs near surface approximately one kilometre northwest of the Beaver Pond Zone. It is composed of stockwork of discrete centimetre scale anastomosing, folded to linear quartz and quartz-carbonate veinlets, hosted predominantly by strongly deformed intermediate volcanic fragmental units, analogous to those that host the ODM/17 Zone, but also present in mafic volcanic flows in both the immediate footwall and hangingwall. The stratigraphy hosting the Western Zone shows a much higher degree of deformation than to the east and, combined with intense sericitic alteration and foliation, is often described as a pervasive shear fabric or approaching mylonitic texture. The veinlets are variably mineralized, with inclusions (in the order of frequency) of pyrite, anemic sphalerite, chalcopyrite, galena, native silver, electrum, and native gold.

Mineralization in the Western Zone is open below the modelled depth.

DEFORMED QUARTZ-ANKERITE-PYRITE SHEAR VEINS IN MAFIC VOLCANIC ROCKS

THE CAP ZONE

The CAP Zone is located approximately 200 m to the south of the ODM/17 Zone and defines a southwest plunging lens modelled over a strike length of approximately 400 m to a depth of approximately 400 m below surface. Higher-grade gold mineralization is associated with deformed quartz-ankerite-pyrite shear and extensional veins hosted by quartz-ankerite-pyrite altered mafic volcanic rocks (Figure 7-12). Relative to ODM/17 and 433 Zones, the CAP Zone has a higher pyrite-chalcopyrite content.

Mineralization in the Cap Zone is open below the modelled depth.

FIGURE 7-12 HIGHER GRADE GOLD MINERALIZATION WITHIN THE CAP ZONE



Borehole NR10-474 from 188.0 to 234.0 m.

Source: SRK, 2011

SILVER-RICH DEFORMED SULPHIDE-QUARTZ VEINS WITHIN TUFFACEOUS ROCKS

The so-called “Footwall Silver Zone” occurs in altered dacitic tuffs/tuff breccias immediately adjacent to the high strain zone at the northern contact of the ODM/17 Zone. The zone plunges to the southwest in similar orientation to the ODM/17 Zone, and is hosted by centimetre scale sulphide bearing quartz veinlets, typically appearing as millimetre scale fracture filling to dendritic native silver inclusions. Associated sulphides within these veinlets in order of frequency are: pyrite, sphalerite, chalcopyrite, and galena. Localized spessartine garnets have been noted. The presence of isoclinal folding of the veinlets gives the mineralization a relative timing of pre- to syn-deformational, and the zone is currently considered to be coeval with the ODM/17 Zone.

High grade gold and silver mineralization in the Intrepid Zone (Figure 7-13) is associated with deformed quartz-pyrite-gold, quartz-pyrite-silver, or quartz-pyrite-gold-silver veinlets that

overprint other mineralization styles. The gold-silver ratio is determined by their location within the base metal zonation controlling the low to moderate grade mineralization.

FIGURE 7-13 INTREPID ZONE GOLD MINERALIZATION



Deformed Pyrite-Sphalerite Veins and Stringers within borehole NR131542

Source: Rainy River, 2013

8 DEPOSIT TYPES

The following section has been summarized from Pelletier's M.Sc. thesis (2016) and it represents the latest update on deposit style and formation of the Rainy River mineralization.

The Rainy River deposit is an auriferous volcanogenic massive sulphide (VMS) system (Pelletier, 2016) with a primary syn-volcanic source and possibly a secondary syn-tectonic mineralization event (Langevin et al., 2015).

Wartman (2011) and Pelletier (2016) have proposed that gold mineralization was introduced alongside base metals prior to the main deformation event at Rainy River, through fluid flow associated with a syn-volcanic hydrothermal system.

Evidence to support an early gold precipitation event includes:

1. Spatial correlation of gold with base metals at the deposit scale;
2. Close spatial association between gold and zoned hydrothermal alteration;
3. Stacking of auriferous bodies in a restrained volcanic pile;
4. The presence of a gold-rich core and a barren rim of pyrite mineralization; and
5. Preferential association of alteration and auriferous zones with volcanoclastic rocks (control on fluid circulation by primary permeability of the host rock).

The peak hydrothermal activity and associated metal deposition is thought to have occurred during a volcanic activity hiatus during which fine-grained, pyrite-rich sediments were deposited on top of the dacite volcanic rocks that host the ODM zone, and before the deposition of tholeiitic basalts in the uppermost part of the host succession.

An early, pre-D2 origin for the alteration and sulphide zones is further supported by the strong control of the combined S2 and L2 fabrics on the shape of the mineralized zones and lithological contacts.

In VMS deposits, the main source of metals are the surrounding volcanic and/or sedimentary rocks, from which circulating hydrothermal fluids collect, enrich and transport the metals and

precipitate them in a zone of massive sulphide mineralization at or below seafloor (Franklin et al., 2005).

At Rainy River, gold and silver are the dominant metals and the base metal (Cu-Pb-Zn) sulphides, although good indicators of the presence of gold, represent less than 10%, by volume, of the host rock. This is in contrast with other VMS systems that generally contain large amounts of base metals. However, there are exceptions, i.e., gold-rich VMS deposits that often contain modest amounts of base metals relative to gold (Mercier-Langevin et al., 2015 and references therein).

Consistent with other gold-rich VMS deposits, the difference in metal budget between Rainy River and typical VMS systems suggests a different source than the surrounding host rocks, for example a magmatic input, and/or efficient precipitation mechanisms for gold.

In the scenario of a magmatic source of metals, specific petrogenetic processes related to specific geodynamic environments can be inferred (e.g., Hannington et al., 1999; Huston, 2000; Yang and Scott, 2003; Mercier-Langevin, 2005; Mercier-Langevin et al., 2007b; 2011; 2015).

DEPOSIT FORMATION

The following scenario has been proposed for the formation of the Rainy River deposit (Pelletier, 2016), and to distinguish the deposit from the typical VMS systems:

1. Deposit would have formed in an environment absent of sub-volcanic intrusions.
2. Different heat source driving the hydrothermal activity is required (rather than the typical level intrusive activity).

Although probable, this proposed volcanic environment for the Rainy River deposit remains speculative and would require further regional work to better constrain the stratigraphy and paleotectonic setting of the volcanic rocks found in the area.

Wartman (2011) and Pelletier (2016) interpret a volcanic evolution of the Rainy River host volcanic succession where:

1. Emplacement of dacitic units within the tholeiitic basalts as cryptodomes and associated breccias (Figure 8-1A).
2. Effusive activity of viscous dacites forming domes and associated breccias.
3. Dacites would eventually reach the sea floor, thus forming a positive topography (Figure 8-1B).
4. Voluminous episode(s) of dacitic volcanism (high eruption rate?), as evidenced by the rare earth element (REE) profiles of dacite samples showing a tight clustering, indicating minimal evolution of the source (magma chamber of significant size).
5. Deposition of pyrite rich, mafic volcanic dominated siltstones that envelope and constrain the dacite events.

A subaqueous depositional environment has been proposed since minor siltstone strata conformably sit above and below the dacitic body hosting the deposit. A subaqueous environment is also in agreement with the pillowed nature of the mafic volcanic rocks documented on the property.

The volcanic hiatus may have allowed for the development and optimal circulation of metal-bearing hydrothermal fluids within the volcanic units (Figure 8-1C).

Hydrothermal circulation was controlled by the primary porosity of the volcanic rocks, therefore metal-bearing fluids would have been channelled within the more porous volcanoclastic rocks, forming what are now the ODM, HS, and 433 zones. This is supported by the distribution of high grade gold zones with respect to volcanic facies as well as higher mass variations of volcanoclastic dacites versus coherent dacite samples.

Syn-volcanic faults could have been present and acted as fluid conduits, although the strong S2 fabric most likely blurred these pathways. It has been speculated that zones of muscovite schists below the ODM zone are remnants of a broad zones of muscovite alteration and indicate the location of preferential hydrothermal fluid circulation.

In a scenario where boiling would have occurred, hydraulic conductivities at grain boundaries in porous volcanoclastics and within fracture systems in coherent volcanic rocks would have been lower pressure zones where boiling could have occurred. The porosity of volcanoclastics and fractures in coherent dacites would also have provided necessary void space to accommodate volume increase intrinsic to boiling. In its turn, boiling can generate hydrofracturing, which further increases porosity and optimizes metal precipitation processes.

The extensive dacitic lava flows with high phenocryst content of the ODM hangingwall could have been favourable to the precipitation of Au below, where the thick and dense lava flow would have acted as an impermeable barrier focusing metal deposition.

Mineralization at Rainy River is distributed in several mineralized zones hosted in dacite of various volcanic facies and stacked perpendicular to S2. Metal associations between these mineralized zones display strong metal zonation that sits perpendicular to the stacking, and the deposit is centred on a hydrothermal alteration system also showing clear mineralogical and mass variation zonation (Figure 8-1A).

For these reasons, stacking of ore zones is interpreted as resulting from a short-lived system, precipitating sulphide phases and alteration minerals with respect to the temperature gradient resulting from a single main event.

Subsequent resuming of back arc tholeiitic volcanism would have further insulated the dacitic volcanic pile and part of the mineralization within the system would be deposited within the basalts now part of the CAP zone (Figure 8-1C). The fertile hydrothermal system would eventually wane and structural deformation would start, although it is uncertain whether the two overlap in time.

It is the QP's opinion that the geological model and concepts described here are valid, and form a good basis for understanding the Rainy River deposit style, formation, and mineralization.

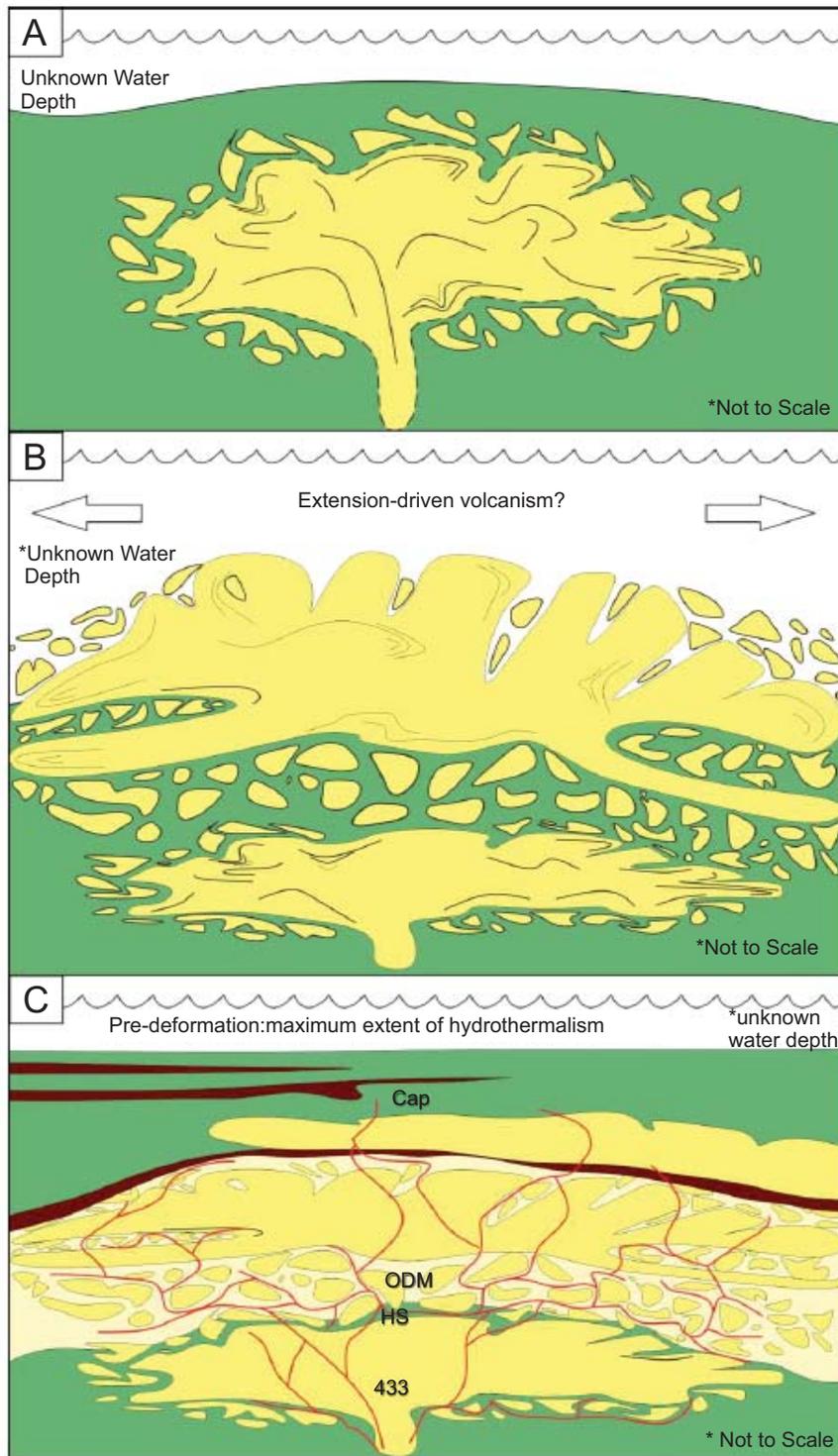


Figure 8-1

Legend:

- A:** Beginning of calc-alkaline volcanism by emplacement of dacite dome in the basalt units.
- B:** Prolonged dacite volcanism that eventually reaches the seafloor, forming lava domes and flows, and associated breccias.
- C:** Final volcanic activity and mineralization, prior to deformation. Note: layers in dark brown are sedimentary units.

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Potential Formation of the
Rainy River Deposit

9 EXPLORATION

In July 2013, New Gold began the process of acquiring RRR (ultimately completing the acquisition of the company in October 2013). Over the next several months, New Gold re-logged 56,000 m of drill core through key sections of the ODM zone and input all of the data into a relational database. In November 2013, SRK completed an updated mineral resource estimate for Rainy River which served as the basis for an updated Feasibility Study that was subsequently completed in February 2014. The November 2013 resource estimate was the ninth estimate completed for Rainy River.

During the same period, New Gold completed an MMI geochemical survey over five prospective areas identified within the greater area surrounding the known Mineral Resources, which were subsequently drill tested.

From 2014 to 2015, New Gold completed an MMI geochemical survey and a hyperspectral alteration study to determine potential vectors to gold mineralization. The results of this work were used to define prospective satellite targets to the known Mineral Resources.

In January 2015, New Gold acquired a 100% interest in three mineral properties located within the Rainy River area through the acquisition of Bayfield Ventures Corp. (Bayfield). The company subsequently re-logged 317 core holes totalling 102,380 m from the Burns Block claim located immediately east of the planned open pit. Geologic and assay data collected from the Burns Block drill core were integrated with the geologic and assay data for the broader project and incorporated into an updated Mineral Resource estimate completed by SRK in May 2015.

Additional exploration advances during 2016 included the completion of a detailed study of the geology of the Rainy River gold deposit completed as part of a M.Sc. thesis project by Ms. Mireille Pelletier (Pelletier, 2016). The thesis provides the most comprehensive description of deposit geology and controls to mineralization completed for Rainy River to date.

The exploration activities undertaken by New Gold for the period July 2013 through 2017 are summarized in Table 9-1. No exploration work has been conducted at Rainy River in 2018 to date.

TABLE 9-1 SUMMARY OF NEW GOLD EXPLORATION ACTIVITIES AT RAINY RIVER (EXCLUDING DRILLING)

Activity	Date	Performed by
2,085 sample MMI geochemical survey	July-October 2013	New Gold Geologists
56,000 m re-logging program within ODM	July-November 2013	New Gold Geologists
Ninth Mineral Resource estimate	November 2013	New Gold Geologists, SRK
Feasibility Study of the Rainy River Gold Mine: Re-addressed to New Gold	February 2014	BBA, AMEC, SRK, Golder Associates
862 sample MMI geochemical survey	May – July 2014	New Gold Geologists
Tenth Mineral Resource estimate	October 2014	New Gold Geologists, SRK
102,380 m re-logging program within Burns Block claim	January – May 2015	New Gold Geologists
Eleventh Mineral Resource estimate	June 2015	New Gold Geologists, SRK
5,000 m Hyperspectral alteration survey	April - November 2016	New Gold Geologists
1,992 sample SWIR spectral alteration survey	May 2015 – December 2016	New Gold Geologists
Twelfth Mineral Resource estimate Reported herein	December 2017	New Gold Geologists

10 DRILLING

RRR's and New Gold's drill programs have been designed and conducted by an experienced exploration team under the supervision of a Project Manager and a Vice President, Exploration. The drill procedures used by Nuinsco are not well documented, therefore New Gold cannot comment on the procedures used by Nuinsco.

This section describes the results of the diamond drilling programs only, as these were the only drill holes used to support the Mineral Resource estimate contained in this report.

Diamond drill holes completed on the principal Rainy River deposit and Intrepid Zone were performed by Bradley Bros. Ltd, Naicatchewenin Development Corporation (NDC) in partnership with C3 Drilling, Major Drilling Group International Inc., Rodren Drilling Ltd., and Orbit Garant Drilling. Ninety-seven percent of drilling used NQ core tools from surface collars. HQ (2.75%) and PQ (0.25%) make up the remaining 3% of drill holes.

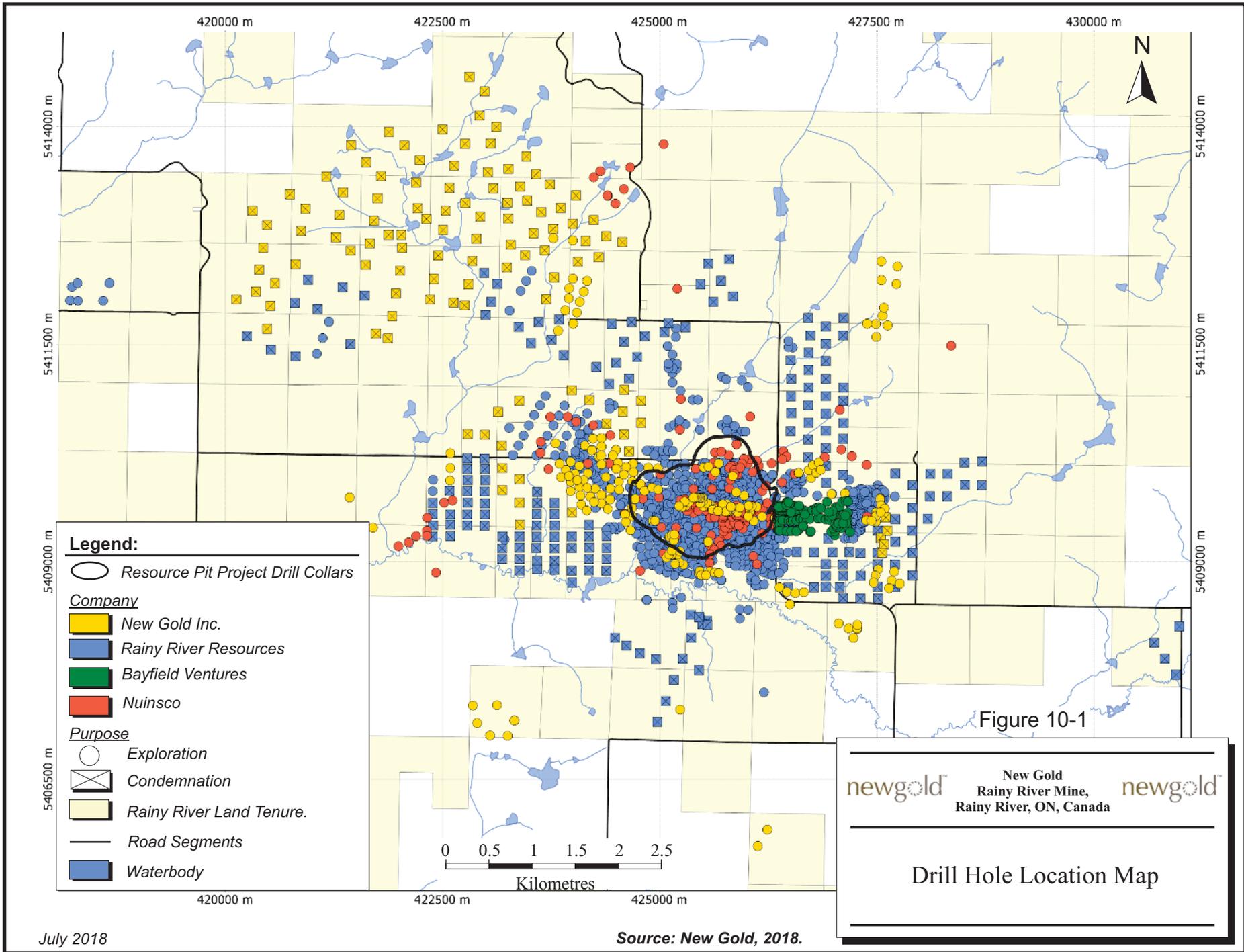
Rainy River drill holes are predominantly angled and drilled on northerly directed azimuths, predominantly at a dip angle of 50° to 65°. The main zones of gold mineralization have been drilled on a grid of at least 60 m by 60 m, with some areas drilled as closely as 12.5 m by 12.5 m. A complete summary of drilling at the Rainy River Mine is included in Table 10-1, and Figure 10-1 gives an overview of the location of the drill hole collars with respect to the property boundary.

TABLE 10-1 SUMMARY OF DRILLING AT RAINY RIVER

Company	Period	Exploration Holes		Condemnation Holes	
		Count	Metres	Count	Metres
Nuinsco	1994 – 2004	203	49,897		
RRR	2005 – 2013	1,407	688,645	190	42,628
Bayfield	2010 – 2014	317	102,380		
	2013	27	9,305	37	7,700
	2014	113	44,452	78	15,690
New Gold	2015	50	10,592		
	2016	37	5,871		
	2017	31	10,546		
	New Gold Total	258	80,766	115	23,390
All	Overall Total	2,185	921,688	305	66,018

Notes:

1. Excludes abandoned, geotechnical, and RC drill holes



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Drill Hole Location Map

COLLAR SURVEYING

A hand-held GPS is used initially to locate and prepare drilling pads in the field. After each hole is complete, a Differential Global Positioning System (DGPS) is used to survey the casing collar. The DGPS accuracy is validated using a known control station location.

The path of the core borehole is surveyed using a Reflex EZ-SHOT™ instrument, which is an electronic solid-state, single-shot drill hole survey tool, at downhole intervals of 50 m. The path typically flattens up with depth and wanders off section on the deeper boreholes.

Borehole deviation is regarded as a critical issue, as the average borehole length is approximately 400 m. In late 2011, RRR started using Tech Directional Drilling to help steer the deeper holes for better control and targeting of the zones. At the Intrepid Zone, 60 out of the 230 boreholes have been resurveyed with a Reflex Gyro at five-metre intervals. The initial orientation of the Gyro instrument was set using an azimuth pointing system (APS) at the collar location.

CORE LOGGING

Standardized logging procedures include the collection of lithological, structural, mineralization, and alteration features. Magnetic susceptibility readings are recorded every three metres. Core recovery is reported to be excellent, but was not measured until 2013 when procedures were upgraded to include core recovery and geotechnical parameters such as rock quality designation (RQD), joint/fracture analyses, material type, and rock strength. Structural and geotechnical logging is usually not based on orientated core. RRR and New Gold have submitted several samples per borehole for specific gravity analyses for both waste rock and mineralized zones.

Core was not routinely photographed, although significant intersections and features are photographically recorded. Diamond core is archived in secure, high quality storage facilities on the Rainy River Mine site where it is under security watch 24 hours a day, seven days a week.

RRR and New Gold used a well-designed procedure for logging the drill core and the subsequent integration of this information into the exploration database. Core logging is

recorded directly onto laptop computers equipped with Maxwell LogChief (previously with DHLogger) logging software which ensures that all relevant information is consistently captured and transferred to the main Maxwell DataShed database. Descriptive geological information is recorded with the appropriate validation procedures in place. After validation, logging information is transferred directly from the DataShed database into Maptek Vulcan software for 3D visualization, interpretation, and modelling.

SAMPLING

Initially, RRR selectively sampled each drill hole based on visible observation of mineralization and alteration. Core was marked for sampling at regular 1.5 m intervals and core was split, with one half retained in the core box as a record and the other half submitted for preparation and analysis. Since then, sampling is performed for the entire length of each hole. The maximum and most common sample interval is still 1.5 m. However, in 2016 and 2017 sampling was performed at regular one metre intervals. Shorter samples were collected to demarcate geological domains.

The sampling interval is the last item marked on the core and recorded in the log. A qualified geologist and/or geotechnician marks out sample intervals with a red grease pencil and places two sample tags at the beginning of each sample interval. A third copy of the sample tag remains in the sample booklet, along with “from” and “to” information recorded by the geologist. These tags are kept in the main office and filed with each individual hole.

The core boxes with samples marked are then prepared for cutting. Once a sample is cut, one half of the core is rinsed and placed into a sample bag and the second half is returned to the core box. One of the sample tags is placed in the sample bag, while the other remains in the core box for reference. The sample bags are stapled closed by the core cutting technician and also individually marked with each sample number. Five sample bags are normally placed into a labelled rice bag, which is then sealed and stored in a secured area prior to dispatch to the assaying laboratory. Each hole is separated by placing the rice bags on separate wooden pallets, never combining holes on one pallet.

Sample shipments are typically coordinated two days per week, to ensure the shipment is never left overnight or over weekends at the shipping yard. A photocopy of the sample

submission form is placed inside the first rice bag of each hole. The rice bags are transported directly to Gardewine North Shipping, in Fort Frances. A typical dispatch contains approximately 400 to 600 samples. Rice bags requiring overnight storage are securely stored inside a designated building.

Following completion of core cutting and sample packing, the core boxes containing the remaining half core are stored outdoors, on sheltered racks. Unsampled intervals in the Nuinsco boreholes were subsequently sampled by RRR and incorporated into the borehole database.

New Gold is of the opinion that the sampling methodology and procedures used by RRR and adopted by New Gold are appropriate for maintaining data integrity. The core samples were collected by competent personnel using procedures in line with generally accepted industry best practices.

11 SAMPLE PREPARATION, ANALYSES AND SECURITY

The sampling method and approach used by Nuinsco during its 1994 to 2004 drilling program is detailed in Mackie (2003). The following sections describe the approaches taken by RRR from 2005 and by New Gold from 2013.

From early 2005 to late 2006, Rainy River samples were prepared at the ALS Minerals Laboratories in Thunder Bay, Ontario (ALS–Thunder Bay) and analyzed at ALS Minerals Laboratories in North Vancouver, British Columbia (ALS–North Vancouver). The management system of the ALS Group Laboratories holds quality management accreditation from the International Organization for Standardization (ISO 9001:2000). The North Vancouver Laboratory holds accreditation for the competence of testing and calibration from the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC 17025:2005) for certain testing procedures, including those used to assay samples submitted from the Rainy River Mine, and is independent of RRR. ALS Laboratories also participated in international proficiency tests such as those managed by CANMET and Geostats Pty Ltd.

Between late 2006 and early 2011, samples from the Rainy River Mine were almost exclusively submitted to the Accurassay Laboratory (Accurassay) facility in Thunder Bay as a primary laboratory, which holds accreditations including ISO 9001:2000 and ISO/IEC 17025:2005 for the Mine's relevant analytical tests, and is independent of RRR.

RRR used the ALS-North Vancouver as an umpire laboratory to monitor the reliability of assaying results delivered by Accurassay during 2010.

In late 2009, RRR also used Activation Laboratories (Actlabs) in Thunder Bay, Ontario, to accelerate the delivery of a small amount of sample batches prior to the resource estimate update of March 2010. Actlabs holds accreditation ISO/IEC 17025 for certain testing procedures including gold and silver assaying using a fire assay procedure, and is independent of RRR.

In February 2011, RRR reverted to ALS-Thunder Bay and ALS-North Vancouver as the primary preparation and analytical laboratories for Rainy River. New Gold have continued to use these independent laboratories from 2013 to present.

The general sample preparation and analyses procedures used by ALS-Thunder Bay from 2005 to 2006 are described in a previous Technical Report (CCIC, 2008). A description of the sample preparation and analyses procedures adopted by Accurassay (2006 to 2011) and ALS (2011 to 2017) is provided in the following sections.

SAMPLE PREPARATION

ACCURASSAY LABORATORY (2006 – 2011)

Samples were first entered into a local information management system (LIMS). The protocol for sample preparation at Accurassay involved drying, crushing, splitting, pulverizing, and matting:

- **Drying:** Prior to the preparation of drill core, the samples are placed in a drying oven, if necessary (approximately 50°C), until dry.
- **Crushing:** The entire sample is crushed using a TM Engineering Rhino Jaw crusher to below 10 mesh.
- **Splitting:** Approximately 500 g subsamples are split-off using a Jones Riffle Splitter.
- **Pulverizing:** Samples are pulverized using a TM Engineering ring and puck pulverizer with 500 g bowls to 90% below 150 mesh (105 microns). The bowls are cleaned with silica sand between each sample.
- **Matting:** Pulverized samples are matted to ensure homogeneity.

The homogeneous sample was then sent to the fire assay laboratory or the wet chemistry laboratory, depending on the analysis required.

ALS MINERALS (2011 - 2013)

The sample was logged in the tracking system, weighed, dried, and finely crushed to better than 70% passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of up to 250 g was taken and pulverized to better than 85% passing a 75 micron (Tyler 200 mesh, US Std. No.

200) screen. ALS sample preparation method codes applied were: LOG-22, DRY-21, CRU-31, SPL-21, and PUL-31.

ALS MINERALS (2013 - 2017)

The sample was logged in the tracking system, weighed, dried, and finely crushed to better than 90% passing a 2 mm (Tyler 9 mesh, US Std. No.10) screen. A split of up to 1,000 g was taken and pulverized to better than 90% passing a 105 micron (Tyler 150 mesh, US Std. No. 140) screen. ALS sample preparation method codes applied were: LOG-21, DRY-21, CRU-32, SPL-22Y, and PUL-35n.

ANALYTICAL METHODS

ACCURASSAY LABORATORY (2006 – 2011)

PRECIOUS METALS ANALYSES

Precious metal analyses (gold, platinum, palladium, and/or rhodium) require that the sample be mixed with a lead-based flux and fused for one hour and 15 minutes. Each sample had a silver solution added to it prior to fusion, which allowed each sample to produce a precious metal bead after cupellation. The fusing process produced lead buttons that contained all of the precious metals from the sample as well as the silver that had been added.

The button was then placed in a cupelling furnace where all of the lead was absorbed by the cupel and a silver bead, which contained any gold, platinum, and palladium, was left in the cupel. The cupel was removed from the furnace and allowed to cool. Once the cupel had cooled sufficiently, the silver bead was placed in an appropriately labelled test tube and digested using aqua regia. The samples were bulked up with 1.0 mL of distilled de-ionized water and 1.0 mL of 1% digested lanthanum solution. The samples were allowed to cool and were mixed to ensure proper homogeneity of the solution.

Once the samples had settled, they were analyzed for gold, platinum, and palladium using atomic absorption spectroscopy (AAS). The AAS unit was calibrated for each element using the appropriate ISO 9002 certified standards in an air-acetylene flame. The results for the atomic absorption were checked by the technician and then forwarded to data entry by means

of electronic transfer and a certificate was produced. The laboratory manager checked the data, validated the certificates, and issued the results in the format requested by RRR.

BASE METAL ANALYSES

Base metal samples (copper, nickel, cobalt, lead, zinc, and silver) were weighed for geochemical analysis and digested using aqua regia. The samples were bulked to a final volume and mixed. Once the samples had settled, they were analyzed for copper, nickel, and cobalt using AAS. The AAS unit was calibrated for each element using the appropriate ISO 9002 certified standards in an air-acetylene flame. The results for the AAS were checked by the technician and then forwarded to data entry by means of electronic transfer and a certificate was produced. The laboratory manager checked the data and validated the certificates, and issued the results in the format requested by RRR.

ALS MINERALS (2011 - 2013)

PRECIOUS METALS ANALYSES

Sample decomposition was by fire assay fusion (ALS method codes FA-FUSO₁ and FA-FUSO₂) and the analytical method was AAS (ALS method code Au-AA23).

A prepared sample (30 g) was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents, as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead was digested in 0.5 mL dilute nitric acid in the microwave oven; 0.5 mL concentrated hydrochloric acid was then added and the bead was further digested in the microwave at a lower power setting. The digested solution was cooled, diluted to a total volume of 4 mL with demineralized water, and analyzed by AAS against matrix-matched standards.

Samples grading over 10 g/t Au were analyzed by gravimetric methods (ALS method code Au-GRA21).

BASE METAL ANALYSES

ALS also undertook multi-element analyses by inductively coupled plasma with atomic emission spectroscopy (ICP-AES).

Sample decomposition was by HF-HNO₃-HClO₄ acid digestion, HCl leach (ALS method code GEO 4A01), whereas the analytical method was ICP-AES or inductively coupled plasma - mass spectrometry (ICP-MS).

A prepared sample (0.25 g) was digested with perchloric, nitric, hydrofluoric, and hydrochloric acids. The residue was topped up with dilute hydrochloric acid and analyzed by ICP-AES. Following this analysis, the results were reviewed for high concentrations of bismuth, mercury, molybdenum, silver, and tungsten and diluted accordingly. Samples meeting this criterion were then analyzed by ICP-MS. Results are corrected for spectral inter-element interferences.

In May 2012 the decomposition method was changed to HNO₃ – HCl aqua regia digestion (ALS method code GEO-AR01). A prepared sample (0.50 g) was digested with aqua regia for 45 minutes in a graphite heating block. After cooling, the resulting solution was diluted to 12.5 mL with deionized water, mixed and analyzed by ICP-AES. The analytical results are corrected for inter-element spectral interferences.

ALS MINERALS (2013 - 2017)

PRECIOUS METALS ANALYSES

Sample decomposition was by fire assay fusion (ALS method codes FA-FUSO₁ and FA-FUSO₂) and the analytical method was AAS (ALS method code Au-AA24).

A prepared sample (50 g) was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents, as required, inquarted with 6 mg of gold-free silver and then cupelled to yield a precious metal bead. The bead was digested in 0.5 mL dilute nitric acid in the microwave oven; 0.5 mL concentrated hydrochloric acid was then added and the bead was further digested in the microwave at a lower power setting. The digested solution was cooled, diluted to a total volume of 4 mL with demineralized water, and analyzed by AAS against matrix-matched standards.

Samples grading over 10 g/t Au were analyzed by gravimetric methods (ALS method code Au-GRA22).

BASE METAL ANALYSES

ALS also undertook multi-element analyses by ICP-AES.

Sample decomposition was by HNO₃ – HCl aqua regia digestion (ALS method code GEO-AR01), whereas the analytical method was ICP-AES or ICP-MS.

A prepared sample (0.50 g) was digested with aqua regia for 45 minutes in a graphite heating block. After cooling, the resulting solution was diluted to 12.5 mL with deionized water, mixed and analyzed by ICP-AES. The analytical results were corrected for inter-element spectral interferences.

METALLURGICAL TESTING

RRR used the SGS Canada Minerals Services Lakefield Laboratory in Lakefield, Ontario (SGS-Lakefield) for metallurgical testwork. SGS-Lakefield is accredited to ISO/IEC 17025:2005 for certain testing procedures, including those used to test and assay samples submitted by RRR. The metallurgical testwork completed by SGS-Lakefield is discussed in more detail in Section 13.

DENSITY MEASUREMENTS

A total of 12,367 density measurements were completed by Accurassay, and more recently ALS, by pycnometry on pulverized split core samples selected as representative of each modelled geological domain.

CHAIN OF CUSTODY AND SECURITY

Once a sample is cut, one half of the core is rinsed and placed into a sample bag and the second half is returned to the core box. One of the sample tags is placed in the sample bag, while the other remains in the core box for reference. The sample bags are stapled shut by the core cutting technician and individually marked with a sample number. Five sample bags are normally placed into a labelled rice bag, which is then sealed and stored in a secured area prior to dispatch to the assaying laboratory. Each hole is separated by placing the rice bags on separate wooden pallets, never combining holes on one pallet.

Sample shipments are typically coordinated two days per week, in order to limit overnight or over weekend storage at the shipping yard. A photocopy of the sample submission form is placed inside the first rice bag of each hole. The rice bags are transported directly to Gardewine North Shipping, in Fort Frances. A typical dispatch contains from 400 to 600 samples. Rice bags requiring overnight storage are securely stored inside a designated building.

Following completion of core cutting and sample packing, the core boxes containing the remaining half core are stored outdoors, on sheltered racks.

QUALITY ASSURANCE AND QUALITY CONTROL

This section addresses the collection procedures, results, and analysis of quality assurance and quality control (QA/QC) data collected from 2015 to 2017 by New Gold, supporting open pit infill drilling, near deposit exploration underground infill drilling, and Bayfield extension programs run by the New Gold Exploration Group. QA/QC data collected by prior owners was analyzed by SRK (BBA, 2014) and was found to be sufficiently reliable to support resource estimation.

QA/QC PROCEDURES

New Gold geologists included certified reference material (CRM) samples at a rate of 1 in 20, and blank samples at a rate of 1 in 30 within every suite of samples submitted for assay. In addition, pulp and coarse duplicates are performed by the laboratory at a rate of 1 in 20. Pulps are repeated at a rate of 1 in 10. A summary of available data is collated in Table 11-1.

**TABLE 11-1 SUMMARY OF AVAILABLE GOLD AND SILVER QA/QC DATA
(APRIL 15, 2015 TO MAY 25, 2017)**

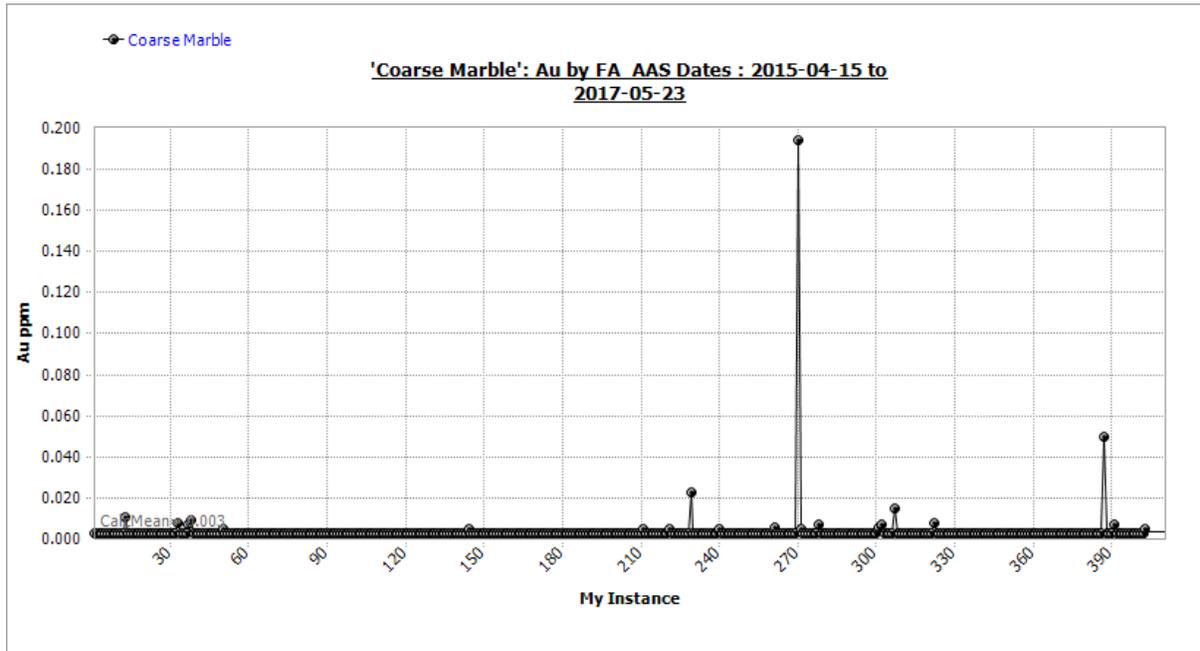
Year	2015	2016	2017	Total
No. Holes	50	37	31	118
Total Metres	10,592	5,871	10,546	27,009
No. Assays	5,869	4,683	3,550	14,102
No. Assays incl. QA/QC	7,512	5,991	4,776	18,279
QA/QC Samples				
Blanks				
Count	158	152	137	447
% of Assays	2.1	2.5	2.9	2.4
Gold CRM				
Count	246	229	165	640
% of Assays	3.3	3.8	3.5	3.5
Silver CRM				
Count	36	28	20	84
% of Assays	0.5	0.5	0.4	0.5
Pulp Repeat				
Count	613	517	400	1,530
% of Assays	8.2	8.6	8.4	8.4
Pulp Duplicate				
Count	306	198	261	765
% of Assays	4.1	3.3	5.5	3.9
Coarse Duplicate				
Count	284	184	243	711
% of Assays	3.8	3.1	5.1	3.9
Total QA/QC Assays				
Count	1,643	1,308	1,226	4,177
% of Assays	21.9	21.8	25.7	22.9

BLANKS

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors. From 2015 to 2017, a total of 447 coarse blank samples, were submitted alongside Rainy River drill core samples to assess contamination during sample preparation, and to identify sample numbering errors. Blank material was either a marble garden stone sourced from a local store (Coarse Marble), or a granite sourced from the nearby Black Hawk Stock (Coarse Blank). The use of the Coarse Blank was discontinued following the discovery that it was not barren of gold. Despite this, only three samples submitted during this period reported a gold value outside the accepted

failure limit set by New Gold at 10x the detection limit for the assay procedure. A performance chart of the Coarse Marble samples is shown in Figure 11-1.

FIGURE 11-1 COARSE MARBLE PERFORMANCE CHART



CERTIFIED REFERENCE MATERIAL

Results of the regular submission of CRMs (standards) are used to identify problems with specific sample batches, and biases associated with the primary assay laboratory.

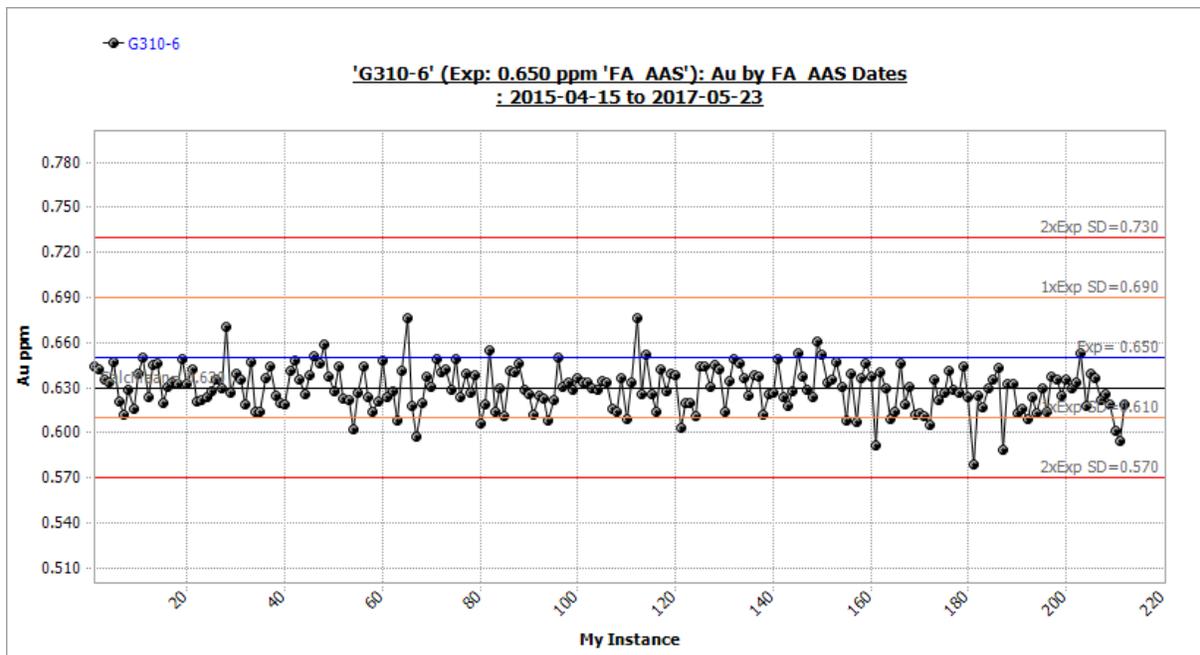
From 2015 to 2017, a total of 640 gold CRMs were submitted alongside Rainy River drill core samples to assess and identify problems with specific sample batches, and biases associated with the primary assay laboratory. Table 11-2 lists the CRMs, and their expected values and ranges used at the Rainy River Mine from 2015 to 2017.

**TABLE 11-2 SUMMARY OF SUBMITTED GOLD CRM
(APRIL 15, 2015 TO MAY 25, 2017)**

CRM	Expected Values (ppm)			STD Dev.	Count	No. Failures
	Expected Value	Lower Value	Upper Value			
Gold						
G308-7	0.27	0.23	0.31	0.008	229	4
G310-6	0.65	0.57	0.73	0.015	212	1
G311-8	1.57	1.41	1.73	0.037	148	0
G913-8	4.87	4.55	5.19	0.103	51	1
Silver						
GBM310-9	3.1	2.7	3.5	0.133	55	0
GBMS911-1	11.9	9.9	13.9	0.651	29	1

A low bias exists for gold CRMs between the means and their expected values (Figure 11-2). A similar bias is seen at other New Gold projects, where GeoStats CRMs are used at various assaying laboratories in BC. The consistent nature of the low bias across laboratories has led New Gold to conclude that this is an issue with the expected value of the CRMs, rather than the assaying or preparation methods at ALS. As with the other projects, the low bias for Au reduces as the grade of the CRMs increases.

FIGURE 11-2 G310-6 PERFORMANCE CHART



The observed bias in all cases has been reduced for the temporal period from 2015 to 2017 compared to 2013 to 2014.

DUPLICATE SAMPLES

A total of 1,530 pulp replicates (R1), 765 pulp duplicates (D1), and 711 coarse reject duplicates (D2), were inserted by ALS as part of the Rainy River QA/QC program from 2015 to 2017. Pulp and coarse duplicates are performed by the laboratory at a rate of 1 in 20. Pulps are repeated at a rate of 1 in 10. There is no duplicate sampling program in place at Rainy River that is blind to the assay laboratory.

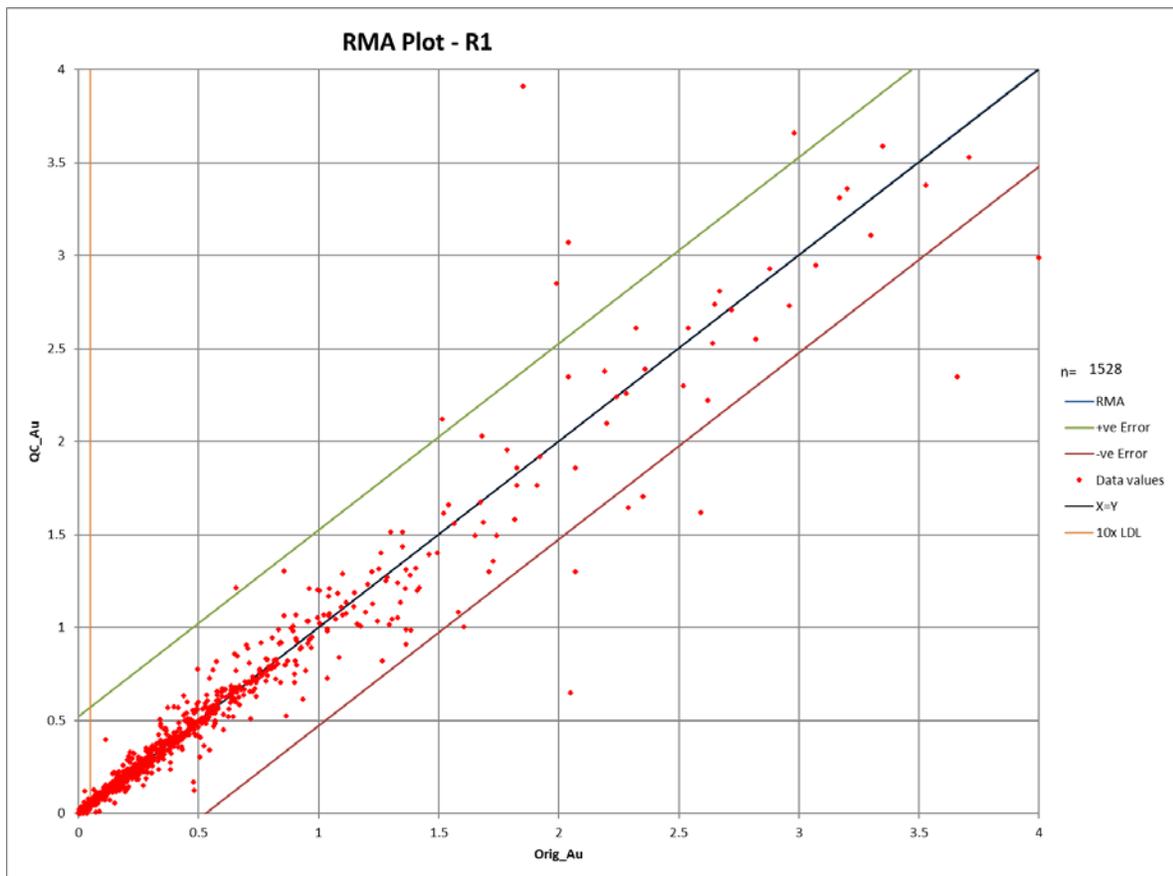
Table 11-3 shows the summary statistics for the duplicates run, with outliers removed. Outliers may indicate errors or represent higher grade data points with a large absolute difference, which have a disproportionate effect on the summary statistics.

**TABLE 11-3 SUMMARY STATISTICS OF SUBMITTED DUPLICATES
(APRIL 15, 2015 TO MAY 25, 2017)**

Duplicate Type	Count	Mean Orig	Mean Check	Bias %	CV _{AVR} %	Intercept	Slope	95% CI	R ²
R1 - Au	1,528	0.441	0.440	0	14.56	0.00	1.00	0.53	0.97
R1 - Ag	1,527	1.287	1.313	-2	19.17	-0.03	1.05	1.50	0.94
D1 - Au	763	0.376	0.375	0	13.72	0.00	1.01	0.62	0.93
D1 - Ag	762	1.189	1.194	0	18.70	-0.02	1.02	1.25	0.94
D2 - Au	710	0.479	0.483	-1	14.57	0.01	0.98	0.87	0.99
D2 - Ag	710	1.400	1.394	0	18.08	0.05	0.96	1.53	0.96

Figure 11-3 presents Reduced Major Axis (RMA) plot for Rainy River pulp replicates. The indicators of bias to look for in the RMA model are the range of possible slopes containing '1' and the range of possible intercepts containing '0'. All duplicates, excluding the Au repeats, display minor biases, with the slopes of the RMA model not crossing 1. None of these biases are large and other indicators such as the intercept of the RMA graph crossing 0, the R² value approaching 1, and low absolute bias from the mean show non-significant bias.

**FIGURE 11-3 PULP REPLICATE AU RMA CHART
(APRIL 15 2015 TO MAY 25 2017)**



The average Coefficient of Variation ($CV_{AVR}\%$) measures the relative precision error in the duplicate pairs. The CV values for Ag and Au are in keeping with previous drilling campaigns at Rainy River also.

Results from the duplicate samples indicate that no significant bias is introduced during the preparation or analysis of the samples.

EXTERNAL LABORATORY CHECK ASSAYS

External check assays were submitted during 2015 only. There are not enough samples to draw meaningful conclusions over the period of 2015 to 2017 and therefore they are not discussed further.

QA/QC CONCLUSIONS AND RECOMMENDATIONS

In the QP's (MDL) opinion, the sample preparation, analysis, and security procedures at Rainy River are adequate for use in the estimation of Mineral Resources.

Low bias observed in gold CRMs is considered by New Gold to be due to slight overestimation of expected value.

1. Results of gold and silver CRM indicate that the primary laboratory measures gold and silver in samples accurately.
2. Results of the blank sample program indicate that there is little to no contamination of samples during sample preparation.
3. Results of the duplicate sample program indicate that no significant bias is present in the preparation or analysis of the samples

With regard to the QAQC monitoring programs in place at Rainy River, New Gold proposes the following:

1. Submit samples to an external check assay laboratory at a rate of 1 in 50, accompanied by CRM and blank samples at a rate of 1 in 20 and 1 in 30, respectively.
2. Acquire or develop three or four matrix matched CRMs in all future drill programs that approximate the cut-off grade, average grade, and high grade material at the Mine and that are certified for silver and gold. Insert these at a rate of 1 in 25.
3. Submit pulp blanks as well as coarse blank material to assess contamination during analysis.

12 DATA VERIFICATION

VERIFICATION OF NUINSCO DATA

Considering the lack of documented exploration procedures adopted during the Nuinsco exploration program, CCIC (2008) recommended that RRR re-sample Nuinsco cores for verification of the assays. According to CCIC (2008), these re-sampled core assays compare well with original assay results reported by Nuinsco. Nuinsco logging and assay data was audited by New Gold as part of a full database review undertaken in 2013 and 2014. Original assay certificates were located for 84% of Nuinsco data and imported directly to the database to avoid potential historical import errors. The remaining 16% were imported from compilation files. No significant issues were found.

The QP is of the opinion that the data is of high quality, and as such suitable for inclusion to the project database to support Mineral Resource estimation.

DRILL HOLE DATABASE

The project database is built using the Maxwell Geoservices DataShed. It has built-in validation checks which assess for possible errors such as duplicate collar IDs, duplicate sample numbers, overlapping intervals, and logging and sampling beyond the maximum depth of the hole. These validations are run at the point of data entry, and errors cannot be entered into the database. In September 2013, the Rainy River database was migrated from DHLogger to DataShed and all these validations were run. Very few errors were encountered, and all were rectified. Subsequent data entry has all been directly to DataShed and the integrity of the data has been maintained.

Validations are run regularly during drilling programs to check for missing intervals in the logging and sampling tables. Any gaps are investigated and appropriate actions taken if required.

A review of drill hole collar coordinates revealed that a separate modelling database was being maintained at site, containing out of date collar coordinates (original versus DGPS surveyed)

for some drill holes. This issue, affecting 253 collars, was resolved and a plan was put in place to ensure a single database is maintained moving forward.

A check on the EZ-Shot downhole survey slip data entry error rate was conducted in 2014. Approximately 10% of the paper survey slips were checked for depth, dip, and magnetic azimuth readings against the database, and an error rate of 2% was found. This was above the desired 1%, so a 100% re-entry of paper survey slips was undertaken. All readings were manually entered and checked against the database. All discrepancies were investigated and the correct value entered into the database. Subsequently, all survey readings have immediately been entered into the database and visually checked in 3D.

Drill hole traces were generated in Vulcan and statistically and visually verified. Statistical verification of minimum and maximum assay values was carried out to make sure no data entry errors were present, such as percentage values above 100% or negative values. No such errors were identified. Visual checks of drill hole traces were performed to determine errors in the imported collar and downhole survey files. No errors were identified.

Assay certificates were delivered by analytical laboratories in a format specified by New Gold which allows for the direct import of all analytical results and metadata to the DataShed project database. The DataShed project database was populated from all original certificates when the database migrated from DHLogger to DataShed in September 2013, rather than flat loading the data directly from DHLogger. A direct comparison of the DHLogger database and the DataShed database was carried out, and all discrepancies investigated. In all cases the discrepancies were due to import errors in the DHLogger database.

The near fully automated assay certificate import process reduces the likelihood for any errors during import.

SITE VISIT

On April 11, 2018, full-time AMC employee Ms. Dinara Nussipakynova, P.Geo., visited the property to undertake the following verification steps:

1. Review several data collection, handling and manipulation procedures, including:
 - Sample collection;
 - Sample preparation for grade control;
 - Sample storage;
 - QA/QC; and
 - Geological interpretation.
2. Inspect the core shed.
3. Review selected logged and assayed drill core intersections. Table 12-1 lists the inspected drill holes.

TABLE 12-1 INSPECTED RAINY RIVER DRILL HOLES

Drill Hole ID	Inspected Interval
NR10-0596	251.0 m to 350.0 m
NR10-0563	410.0 m to 530.0 m
NR13-1565	324.0 m to 391.5 m

Under supervision of Ms. Nussipakynova, Simeon Robinson, P.Geo., of AMC undertook random cross-checks of assay results in the database with original assay results on the assay certificates returned from ALS for gold and silver. This verification included comparing 1,360 of the 24,227 assays for the drilling conducted from 2015 to 2017 (5%). No errors were identified.

In addition, verification was carried out using the normal routines in Datamine where the database was checked for collar, survey, and assay inconsistencies, overlaps, and gaps. AMC makes the following observations based on the data verification that was conducted:

- Site geologists are appropriately trained;
- Procedures for data collection and storage are well-established and adhered to;
- QA/QC procedures are adequate and give confidence in the assay results; and
- Cross-checking a sample set of the database with the original assay results revealed no errors.

In AMC's opinion, the database is fit-for-purpose and the geological data provided by New Gold for the purposes of Mineral Resource estimation was collected in line with the industry best standards as defined in the CIM Exploration Best Practice Guidelines and CIM Mineral

Resource, Mineral Resource Best Practice Guidelines. As such, the data are suitable for use in the estimation of Mineral Resources.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

METALLURGICAL SAMPLING AND COMPOSITE PREPARATION

INTRODUCTION

Metallurgical testing programs were performed to support each phase of project development beginning with the Preliminary Economic Assessment (PEA), issued in October 2012, and the Feasibility Study, issued on May 23, 2013 and then readdressed to New Gold at the end of July 2013. In August 2013, New Gold requested that BBA prepare an updated Feasibility Study from the original Feasibility Study issued on May 23, 2013. The Feasibility Study was prepared by BBA in a collaborative effort with New Gold and selected consultants. The technical report was prepared to support the updated Feasibility Study as disclosed in New Gold's press release entitled "New Gold Announces Its Rainy River Feasibility Study Results," dated January 16, 2014, and was issued on February 14, 2014 (BBA et al., 2014).

The updated Feasibility Study was prepared to ensure that the design criteria and assumptions used to evaluate the Rainy River Mine were consistent with those used for New Gold's other projects and operations and to incorporate New Gold's most recent Mineral Resource estimate (November 2, 2013) and project metallurgical testwork. The Mineral Resources used in the Feasibility Study are based on the updated drilling database and block model information as of August 16, 2013 (including the Intrepid Zone). The Feasibility Study is based on the development of a combined open pit (19,500 tpd) and underground mining operation (1,500 tpd), feeding a conventional process plant to recover gold and silver mineralization. The process plant would have a capacity of 21,000 tpd.

METALLURGICAL TESTING SUPPORTING THE PEA

Initial metallurgical testwork was carried out by SGS in Lakefield, Ontario from 2008 to 2011 and formed the basis for the PEA Technical Report published in October 2012. The testwork included:

- Mineralogy
- Comminution
- Gravity separation

- Flotation
- Flotation concentrate leaching
- Whole ore leaching

Work continued at SGS Lakefield in 2012 with variability sample collection and testing guided by SGS geometallurgical modelling.

The initial work focused on flotation and concentrate leaching. Whole ore leach tests were also performed to provide a second viable process option. The results of the studies indicated that the overall gold recovery for the flotation concentrate leach was approximately 88.5% with the flotation feed ground to an 80% passing (P_{80}) 150 μm and the flotation concentrate re-ground to a P_{80} of 15 μm . Recovery for the whole rock leach circuit was approximately 91.0% when ground to a P_{80} of 60 μm . No optimization of the grind size was performed at the time.

Though the overall recovery was lower, the flotation and concentrate leaching option was selected mainly due to the ability to separate the sulphide concentrates from the non-sulphide flotation tailings, which would be deposited in the tailings management area. The flotation tailings would be low in sulphur and free of cyanide resulting in a more benign tailing. Upon further review, neither option had a distinct economic or environmental advantage over the other, so the testing program continued in 2012 to provide more information to support the selection of a final flowsheet.

METALLURGICAL TESTING SUPPORTING THE FEASIBILITY STUDY

Metallurgical testwork was performed from October 2011 to November 2012 on the main pit and from November 2012 to November 2013 on the Intrepid Zone. The objective of the Intrepid Zone test program was to determine whether the Intrepid Zone material could be treated successfully using the same gravity-cyanidation flowsheet selected for the main pit ore, and the Intrepid Zone ore would not negatively impact plant performance when blended in low tonnages with the main pit ore. The testwork included comminution, gravity separation, cyanidation, carbon adsorption modelling, cyanide destruction, and solid-liquid separation.

SAMPLE SELECTION AND COMPOSITING

MASTER COMPOSITE SAMPLE – 2008 TO 2011 TESTING

Metallurgical samples were selected from drill core and assay rejects from drill core to represent each of the zones of mineralization in the deposit. The individual samples were combined into eight zone composites including CAP, Z-433, HS, NZ, ODM-1, ODM-2, ODM-3, and ODM-4. A Master composite was then created by combining individual samples from each zone in the proportion indicated in Table 13-1. The composite consisted of 80% ODM ore, with the balance coming from the remaining zones.

Additionally, composites were made from the high grade areas of the ODM-17 and Z433 zones. Two composites were made of each zone including an ODM-17 4gAu/t, an ODM-17 8gAu/t, a Z433 4gAu/t, and a Z433 8gAu/t composite.

TABLE 13-1 MASTER COMPOSITE SAMPLE PROPORTIONS

Zone Composite	Zone Composite Proportions, %	Total Proportion, %
CAP	2.0	20.0
Z-433	12.0	
HS	1.0	
NZ	5.0	
ODM-1	35.1	80.0
ODM-2	3.5	
ODM-3	31.4	
ODM-4	9.9	
Master		100.0

COMPOSITE SAMPLES FOR FLOWSHEET CONFIRMATION

Three composite samples were developed to represent the major ore types in the deposit and the blend of ore to be processed through the life of mine. These include the:

- ODM Master
- Initial Pit
- Remaining Life of Mine (RLOM)

A separate ODM composite was prepared as the ODM zone is the largest zone in the Initial Pit and the overall deposit. The Initial Pit composites and RLOM composites were selected to develop a better understanding of the metallurgical responses for the early years of mining compared to the later years.

The percentages of each zone, which occur in the Initial Pit, the RLOM and the Overall Pit, used in each composite are presented in Table 13-2. The values shown were determined in March 2012 and used for the metallurgical testing.

TABLE 13-2 PERCENTAGES BY ZONE FOR SAMPLE COMPOSITES

Zone	Initial Pit (%)	RLOM (%)	Overall Pit (%)
ODM	86.4	60.4	68.0
Z433	4.3	13.8	11.0
HS	0.4	5.5	4.0
NZ	4.4	5.2	5.0
CAP	4.6	15.1	12.0

Table 13-3 presents the percentages of each zone selected for use in the final Design Criteria prepared by AMEC. The percentages are comparable, however, the percent of ODM is significantly higher and the NZ zone is absent.

TABLE 13-3 PERCENTAGES BY ZONE SELECTED FOR DESIGN CRITERIA

Zone	Initial Pit (%)	RLOM (%)	Overall Pit (%)
ODM	82.0	71.0	78.0
Z433	10.0	6.0	8.0
HS	5.0	7.0	6.0
NZ	0.0	0.0	0.0
CAP	0.0	12.0	5.0
Other	3.0	4.0	4.0

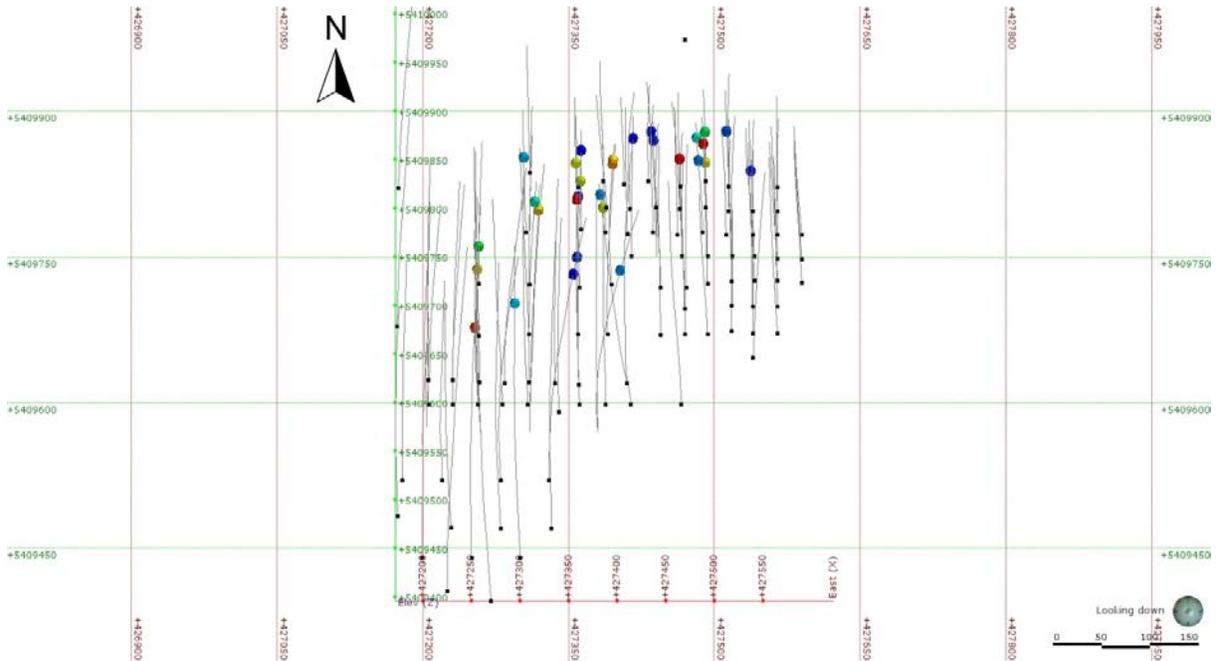
VARIABILITY SAMPLE SELECTION

Following flowsheet selection and development of the base test criteria, sample variability tests were performed. The variability test program included 162 comminution samples and 208 leaching samples from the main pit, and another 30 comminution and leaching samples from the Intrepid Zone. A geometallurgical model and statistical analyses, developed by SGS, were used to select the sample locations, drill hole intervals, and quantities of material for the variability samples. Geographic location, mineralization grade, and trend were the main variables used to classify and define the ore zones.

The borehole and sample locations in the main pit are presented in Figures 13-1 and 13-2 in plan view and cross-section respectively.

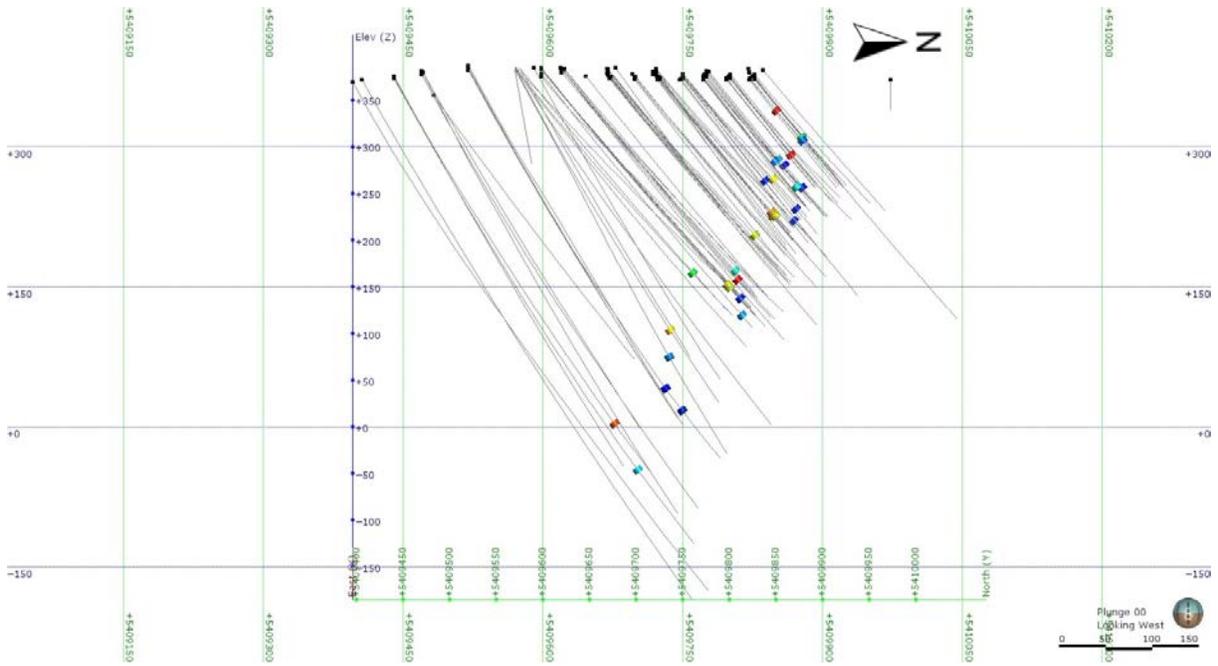
The sample locations for comminution and leach variability testing in the main pit are presented three-dimensionally in Figures 13-3 and 13-4.

FIGURE 13-1 PLAN VIEW OF DRILL HOLE AND SAMPLE LOCATIONS IN THE INTREPID ZONE WITH COLOUR INDICATING GRADE



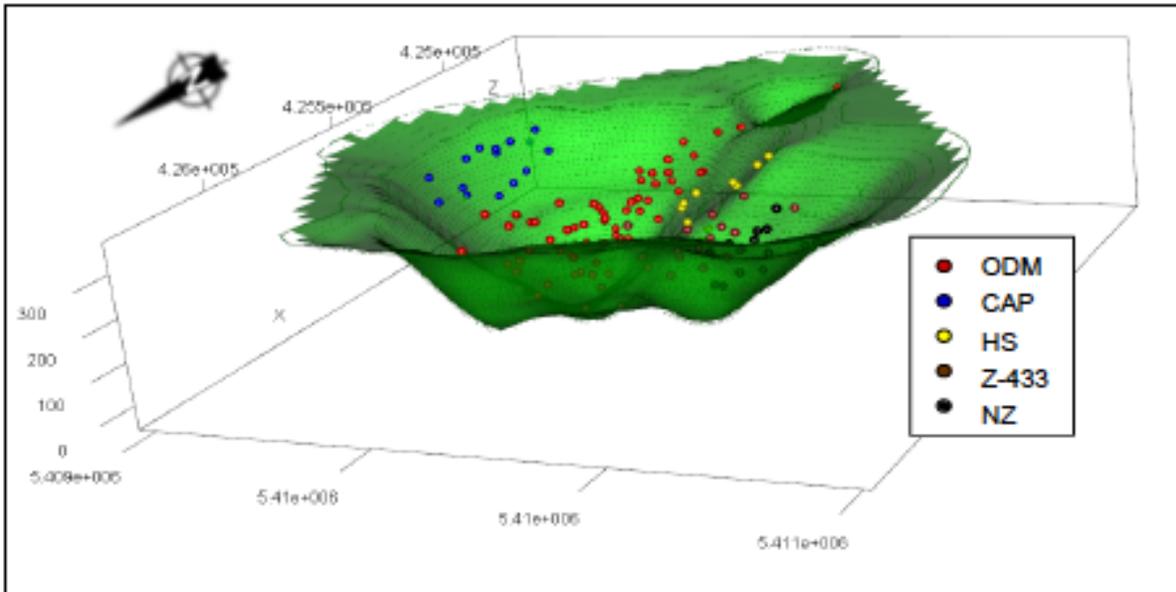
Source: New Gold, 2018

FIGURE 13-2 LOCATION OF INTREPID ZONE SAMPLES WITH COLOUR DENOTING GRADE (CROSS-SECTION LOOKING WEST)



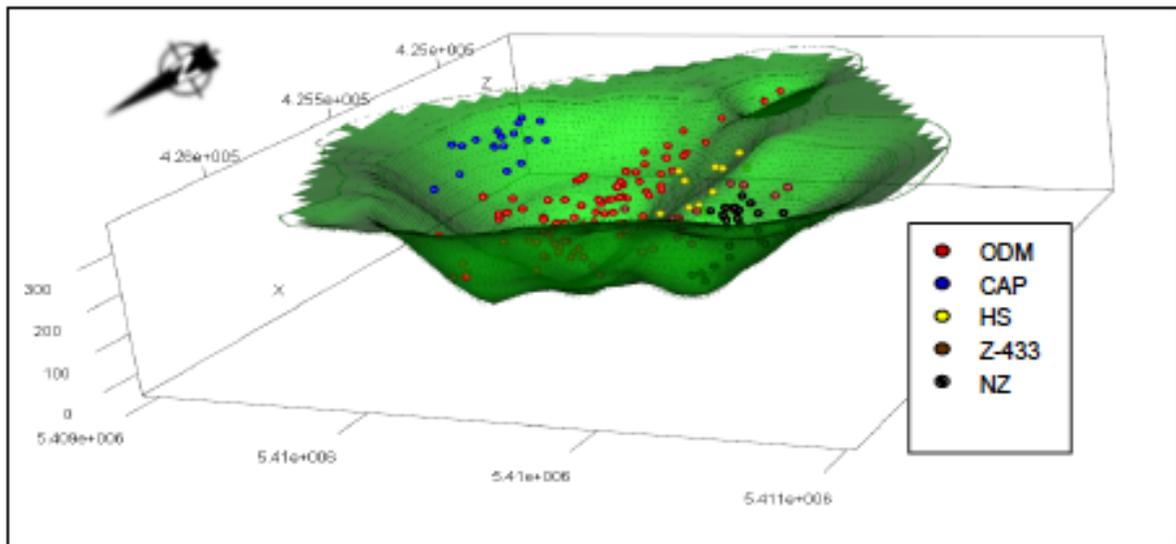
Source: New Gold, 2018

FIGURE 13-3 SAMPLE LOCATIONS FOR COMMINUTION VARIABILITY TESTWORK



Source: New Gold, 2018

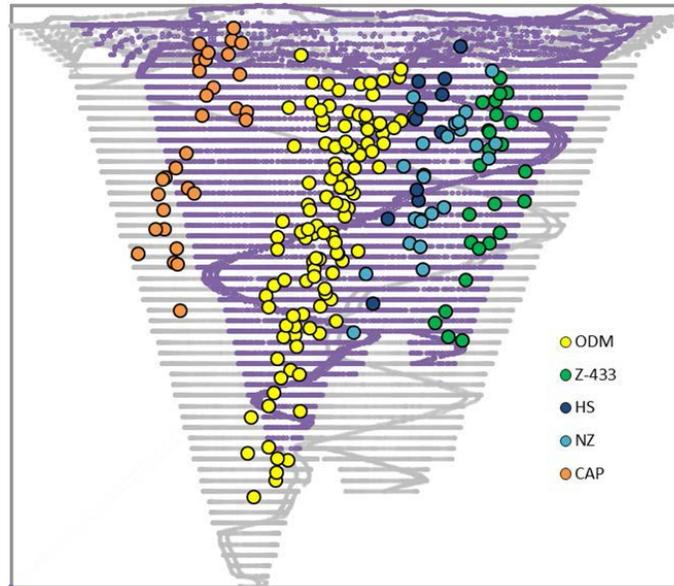
FIGURE 13-4 SAMPLE LOCATIONS FOR LEACHING VARIABILITY TESTWORK



Source: New Gold, 2018

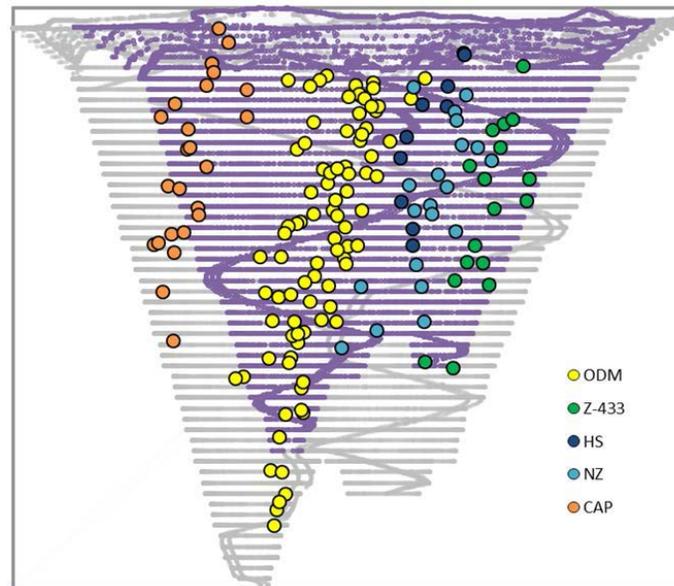
Some of the samples are located outside the proposed pit outline. This is due to a reduction in the size of the engineered pit from the December 2011 PEA to the August 2012 updated PEA, after the samples were collected. Figures 13-5 and 13-6 show the change in pit outlines and the sample locations for variability comminution testing and variability leach testing. The August 2012 outline is in purple and the December 2011 pit is in grey.

FIGURE 13-5 SAMPLE LOCATIONS FOR VARIABILITY COMMINUTION TESTWORK



Source: New Gold, 2018

FIGURE 13-6 SAMPLE LOCATIONS FOR VARIABILITY LEACHING TESTWORK



Source: New Gold, 2018

SAMPLE CHARACTERIZATION

The head grades and major impurities for the Master Composites and variability samples are presented in Table 13-4.

TABLE 13-4 HEAD ANALYSES FOR THE COMPOSITE AND VARIABILITY SAMPLES

Sample	Number of Samples	Au (Screen Met.) g/t	Ag (Direct) g/t	Cu %	S %	Zn %	Fe %
Zone Composites							
Master		0.9	3	0.034	2.22	0.081	3.1
ODM-1		0.83	<2	0.012	1.42	0.058	2.45
ODM-2		2.31	18	0.038	2.64	0.51	3.1
ODM-3		0.90	3	0.027	2.51	0.11	3.1
ODM-4		0.56	4	0.016	1.67	0.084	2.6
Z433		1.67	<2	0.12	1.88	0.005	3.3
HS		0.72	<2	0.042	2.22	0.017	3.2
NZ		1.07	2	0.039	2.94	0.16	4.7
CAP		0.83	5	0.031	5.11	0.15	10.0
Variability Samples							
ODM	117	1.04	4.04	0.010	2.07	0.13	2.74
Z433	27	1.12	2.03	0.041	2.22	0.06	4.20
HS	13	0.51	1.00	0.015	2.15	0.06	3.23
NZ	22	0.79	1.99	0.019	2.25	0.07	3.54
CAP	33	0.72	3.65	0.017	3.70	0.07	9.39
Intrepid Zone	30	1.64	14.9	0.009	2.27	0.11	2.37
Master Composites							
Initial Pit	-	0.90	2.57	0.016	2.05	0.15	3.13
Remaining Life of Mine	-	0.71	2.86	0.010	2.54	0.07	4.05
Intrepid Zone Master	-	1.45	13.8	0.009	2.19	0.10	2.34

The samples from the CAP Zone have significantly higher levels of sulphur and iron than the other zones. The Intrepid Zone has much higher silver levels than the other zones, however, the copper, iron, sulphur, and zinc levels are consistent with the other zones.

MINERALOGY

Four main styles of mineralization have been identified at the Mine:

- Moderately to strongly deformed, auriferous sulphide and quartz-sulphide stringers and veins in felsic quartzphyric rocks (ODM/17, Beaver Pond, 433 and HS Zones);
- Deformed quartz-ankerite-pyrite shear veins in mafic volcanic rocks (CAP/South Zone);

- Deformed sulphide bearing quartz veinlets in dacitic tuffs and tuff breccias hosting enriched silver grades (Intrepid Zone); and
- Copper-nickel-platinum group metals mineralization hosted in a younger mafic-ultramafic intrusion (34 Zone).

The bulk of the gold mineralization at the Mine is contained in sulphide and quartz-sulphide stringers and veins hosted by felsic quartz-phyric rocks. Two main zones are recognized (ODM/17 and 433 Zones) with subsidiary zones (HS and NZ), which are mostly bound by high strain zones.

Gold deportment studies were performed on each zone during the 2011 and 2012 metallurgical testing campaigns. Five ODM samples, two Z433 samples, one CAP sample, one HS sample, and one NZ sample were studied.

- The samples were composed mainly of non-opaque minerals, with minor amounts of pyrite present, ranging from 2.5% in one of the Z433 composites to 9.5% in the CAP composite.
- The gold mainly occurs as native gold, electrum, and kuestelite. Small amounts of petzite (Ag_3AuTe_2) were also noted. Other gold minerals including calaverite, aurostibite, auricuprite, hessite, and two unknown phases (AuAgHg and AuAgPb) were also observed occasionally in samples.
- The gold occurs as liberated, attached, and locked particles in all of the composite samples at a grind of 150 μm except for the CAP sample. In the CAP Zone composite, gold only occurs as inclusions locked in pyrite and non-opaque minerals.
 - Liberated and attached gold can be readily extracted with whole rock leaching at the 150 μm grind size.
 - The majority of the gold occurs as locked particles in sulphides and silicates. Those composites with locked particles would require very fine grinding to liberate the gold particles prior to whole rock cyanide leaching.
 - The CAP Zone composite contains gold particles as locked inclusions in pyrite and non-opaque minerals and would require fine grinding to liberate the gold particles prior to leaching.
- The gold grain size was relatively fine in all samples, with coarse gold, >100 μm , noted in only two of the composites. The HS samples and one of the Z433 samples contained coarse gold.
- The Z433 samples had the largest gold particles.
 - All other samples contained gold grains that were less than 10 μm .
 - The coarse particles tended to be liberated, while the fine particles tended to be encapsulated.

- Trace amounts of pyrrhotite were found in approximately half of the samples. Pyrrhotite contains loosely bound sulphur that will increase cyanide consumption by forming thiocyanate.

COMMINUTION TESTWORK

A large comminution test program was conducted with the Rainy River composites in support of the design of the crushing and grinding circuits.

- Crushing characteristics were determined by performing seven crushing work index tests at each of three vendor laboratories, for a total of 21 tests. Tests were performed by Metso, SGS, and FLSmidth. Only the seven tests performed at the Metso laboratory were selected by AMEC for use in the final design.
- A total of 160 Bond work index (BWi) tests were performed at SGS-Lakefield, including 140 Modified Bond Work Index (ModBWi) tests. Twenty full bond ball mill, work index tests were performed to calibrate the ModBWi tests.
- Unconfined compressive strength (UCS) testwork was performed at Queen's University in Kingston, Ontario. Most of the UCS samples failed along foliation lines, and as such are not considered to be particularly reliable.
- The Abrasion Index work was performed at SGS-Lakefield.
- Thirteen JK Drop Weight tests (DWT) and 175 Semi-autogenous Grinding (SAG) Mill Comminution (SMC) tests were performed at SGS-Durango.
- The 80th percentile value in each of the tests was used for design, unless otherwise noted.

CRUSHER WORK INDEX TESTS

Crusher work index tests were performed at three separate laboratories, including SGS, Metso (Supplier A) and FLSmidth (Supplier B). The results are presented in Table 13-5.

TABLE 13-5 CRUSHER WORK INDEX TEST RESULTS

Lab Zone	SGS/Phillips					Supplier A				Supplier B			
	ODM	Z-433	HS	NZ	CAP	ODM	Z-433	HS	CAP	ODM	Z-433	HS	CAP
No of Tests	4	2	1	1	1	6	1	2	2	4	1	1	1
No of Samples	69	38	20	17	20	60	10	20	20	40	10	10	10
Average	19.7	34.8	25.0	19.4	10.9	20.9	18.7	18.8	14.0	11.6	10.3	10.3	7.3
Minimum	8.8	17.2	17.1	13.7	6.6	11.1	12.0	10.0	10.2	2.9	6.4	6.9	3.7
Median	17.7	35.9	24.5	17.4	10.1	20.9	18.2	17.3	14.3	10.3	10.1	9.8	6.7
80 th Percentile	24.0	39.9	28.4	24.4	14.3	24.7	23.4	21.7	15.5	16.5	11.1	13.4	9.8
Maximum	52.1	50.3	30.9	27.6	18.3	36.6	27.8	39.4	20.0	30.2	20.9	15.1	11.6

It was determined that the most consistent test results were those of Supplier A, Metso, which were midway between the results of SGS and Supplier B. The results from Supplier A, Metso, were selected for use in the design. The 80th percentile value of 25 kWh/t was selected for design.

UNCONFINED COMPRESSIVE STRENGTH TESTS

UCS tests were performed at Queen's University to determine the competency of the selected rock samples. Four ODM samples, one Z-433 sample, one HS sample, and one CAP sample were tested in duplicate for a total of 14 samples.

Ten of the 14 samples had partial failure occur along foliation lines, including all of the ODM samples. The values from all the tests ranged from 34.5 MPa to 109.4 MPa with an average of 66.3 MPa. The average compressive strength of the samples that did not fail along foliation lines was 87.2 MPa. As the majority of the samples had low results due to failure along foliation lines, the results were deemed unsuitable for design purposes.

BOND BALL MILL WORK INDEX TESTS

The ball mill work index testing program consisted of 160 ModBWi and 20 standard BWi tests on material from all five zones of the deposit, within the main pit.

The ModBWi Test is an open circuit milling test using the standard Bond ball mill. The test is run for a specific amount of time after which the feed and product size distributions are determined. The target P₈₀ product screen size for the Rainy River samples was 200 mesh or 74 µm. The ModBWi results were calibrated by comparing the ModBWi and standard BWi test results. The results of the tests are presented in Table 13-6. The ModBWi results for the ODM, Z-433, and CAP zones were within 5% of the BWi. The HS sample had a slightly higher variance of 6.7% and the NZ and Intrepid samples had the highest variances at 11.1% and 17.4% of the BWi respectively. The ModBWi was considered validated and the remainder of the variability test program was performed using the ModBWi procedure.

TABLE 13-6 RESULTS OF BOND WORK INDEX AND MODIFIED BOND WORK INDEX TESTS

Description Zone	Bond Work Index, 75 μ m kWh/t					
	ODM	Z-433	HS	NZ	CAP	Intrepid
BWi, 75 μm (kWh/t)						
Number of Tests	5	4	2	2	3	8
Average	13.6	15.6	16.2	13.0	15.2	16.7
Minimum	12.6	15.2	16.1	12.1	14.8	13.2
Median	13.8	15.7	16.2	13.0	14.9	15.6
80 th Percentile	14.2	15.9	16.2	13.5	15.6	19.0
Maximum	15.0	15.9	16.3	13.8	16.1	21.5
ModBWi, 75 μm (kWh/t)						
Number of Tests	89	17	10	20	24	30
Average	13.8	15.1	14.9	14.1	14.7	15.1
Minimum	11.6	12.9	14.1	11.1	13.0	13.4
Median	13.8	15.3	15.0	14.2	14.8	15.1
80 th Percentile	14.7	15.4	15.2	15.0	15.5	15.7
Maximum	16.0	15.8	15.5	16.2	15.8	17.3
Variance ModBWi / BWi, 80th Percentile, %	3.2%	-3.14%	-6.17	11.11	0.64	17.37

It can be seen that at the 80th percentile all the zones are similar in terms of ModBWi. The 80th percentile weighted average ModBWi value of 15 kWh/t was selected for use in the design criteria.

The Intrepid Zone tests were performed after completion of the test program for the main pit. It can be seen that, at the 80th percentile, the BWi and ModBWi vary with values of 19.0 and 15.7 kWh/t, respectively. This is due to two samples that had considerably higher BWi values. When ignoring these two results, the 80th percentile of the BWi tests is 15.7 kWh/t, identical to the ModBWi results. Overall the Intrepid Zone has slightly higher BWi and ModBWi values, indicating that the zone is harder than the zones from the main pit.

The rock in the ODM and NZ Zones was softer than the other zones and had a wider range of values. The 80th percentile BWi values for each zone are relatively close ranging from 14.7 kWh/t to 15.7 kWh/t. A weighted average value of 15.0 kWh/t was used for the design.

BOND ABRASION INDEX

Twenty-four Bond abrasion index tests were performed. The results indicated a large amount of variability in the samples with values ranging from 0.09 to 0.38, which corresponds to the 10th and 90th percentile hardness in the data set. The ore is considered to be moderately abrasive when compared to the SGS database. The value used in the design was 0.25. Abrasion index data will be used to calculate the wear material consumptions for the operating costs.

The results are presented in Table 13-7.

TABLE 13-7 BOND ABRASION INDEX TEST RESULTS

Description	Zone				
	ODM	Z-433	HS	NZ	CAP
Number of Tests	12	4	2	2	4
Average	0.20	0.27	0.32	0.11	0.15
Minimum	0.05	0.14	0.21	0.11	0.08
Median	0.15	0.21	0.32	0.11	0.15
80 th Percentile	0.26	0.33	0.38	0.11	0.19
Maximum	0.66	0.51	0.43	0.11	0.21

JK DROP WEIGHT TESTING AND SMC TESTING

The SMC test program consisted of 13 JK DWT and 175 SMC tests on samples of the main pit, and an additional two samples from the Intrepid Zone. The JK DWT tests were performed on PQ (85 mm) core drilled specifically for the comminution program, while the SMC tests were performed on core samples selected from the exploration drilling program. The SMC samples were selected by the SGS geometallurgy group, by dividing the deposit into one million tonne blocks (domains) and selecting a sample from each domain. Using this method, 162 total samples from the main pit were selected for SMC testing, in addition to the 13 samples that were selected for the JK DWT testing, for a total of 175 tests.

The JK DWT results consist of A and b factors that measure the resistance to impact breakage and a t_a value, which measures the resistance to abrasion. A lower A x b value indicates a higher resistance to impact breakage while the higher t_a values indicate a material that is more resistant to abrasion breakage. The JK DWT results were also used to calibrate the SMC test results by zone. The SMC tests generate A and b factors similar to the JK DWT tests, along

with M_{ia} , M_{ic} , M_{ih} , and density values. The M_{ia} value is the coarse grinding work index, M_{ic} is the crushing work index, and M_{ih} is the high pressure grinding roller (HPGR) work index. All SMC tests were performed on the -22.4 + 19.2 mm size fraction.

SMC tests were performed on the reject material from each JK DWT test to provide a direct comparison between the results of the JK DWT and the SMC tests. The objective was to confirm that the SMC results are consistent with the JK DWT results and that the method is acceptable for use in the variability testing program. The results from the JK DWT tests and SMC tests performed on fractions of the same sample are presented in Table 13-8.

TABLE 13-8 JK DROP WEIGHT AND CORRESPONDING SMC TEST RESULTS

Zone	JK DWT					SMC			
	A	b	A x b	t_a	ρ (g/cm ³)	A	b	A x b	A x b % Difference
HS	76.4	0.30	22.9	0.32	2.79	75.4	0.33	24.9	8.7
	66.4	0.37	24.6	0.31	2.81	58.0	0.56	27.6	12.2
ODM	66.2	0.37	24.5	0.45	2.77	68.9	0.35	24.1	-1.6
	50.8	0.61	31.0	0.46	2.82	55.0	0.60	33.0	6.4
	54.9	0.55	30.2	0.48	2.83	54.2	0.64	34.7	12.9
	53.2	0.59	31.4	0.47	2.83	54.9	0.57	31.3	-0.3
	55.2	0.67	37.0	0.57	2.80	56.4	0.70	39.5	6.7
CAP	50.0	0.79	39.5	0.43	2.75	60.8	0.65	39.5	0.0
	67.0	0.37	24.8	0.35	3.02	58.6	0.45	26.4	6.4
Z-433	59.5	0.40	23.8	0.21	2.92	79.1	0.34	26.9	13.0
	60.6	0.41	24.8	0.44	2.81	69.5	0.35	24.3	-2.0
NZ	60.1	0.42	25.2	0.28	2.82	70.5	0.36	25.4	0.8
	35.0	0.81	28.4	0.46	2.73	64.7	0.45	29.1	2.5
Intrepid	65.9	1.36	89.6	1.04	2.63	64.9	1.60	104	16.1
	100	0.23	23.0	0.28	2.72	83.4	0.60	38.0	65.2
Average Main Pit									5.1
Average Including Intrepid Zone									9.8

The SMC tests are slightly higher than the corresponding JK DWT for the same sample, indicating that the SMC results will yield slightly lower resistance to breakage than the JK DWT. The SMC tests were confirmed to be acceptable for use in the variability testing program, rather than using the full JK DWT.

The variance in parameter values in the Intrepid Zone was higher than in the main pit samples. This indicates significant variances in hardness within the Intrepid Zone and that the Intrepid Zone will have a higher resistance to breakage than the main pit samples. The Intrepid Zone material will be blended with the main pit material so the differences may not have a significant effect on production rates.

The distributions of the Z-433, HS, and CAP zones are in a narrow range with A x b values ranging from 20 to 35 with the majority of the values between 20 and 25. The ODM and NZ zones have wider ranges of values with A x b values ranging from 20 to 60, with the majority between 20 and 45. The ODM and NZ ores are less resistant to breakage than the Z-433, HS and CAP zones, which are consistently harder (Table 13-9).

TABLE 13-9 SMC A X B VALUES AND CORRESPONDING M_{IA} VALUES

Description Zone	A x b, and M _{ia} (kWh/t)					
	ODM	Z-433	HS	NZ	CAP	Waste
	A x b					
Number of Tests	95	19	12	21	26	2
Average	32.9	23.7	22.0	28.3	23.2	21.6
Minimum	62.6	38.6	24.9	63.3	34.7	22.0
Median	32.4	22.7	22.1	26.0	22.3	21.6
80 th Percentile	26.6	20.7	20.8	21.8	20.3	21.3
Maximum	20.7	19.0	19.0	20.0	18.0	21.1
	M_{ia} (kWh/t)					
Average	23.6	30.0	31.5	27.0	30.3	31.9
Minimum	13.8	19.9	28.5	13.5	21.6	31.4
Median	23.2	30.4	31.1	27.4	30.6	31.9
80 th Percentile	27.0	32.5	33.0	31.4	33.2	32.1
Maximum	32.8	35.6	35.2	34.6	37.4	32.3

Based on the reference and industrial data, all zones tested are considered to be very hard. The ODM Zone is slightly less resistant to coarse breakage while the other zones and waste rock samples have much higher resistance.

A x b values of 25.8 and 24.3 at the 80th percentile were estimated from JK DWT and SMC tests for the Initial Pit and RLOM, respectively, using the proportions from Table 13-2. The A x b and t_a values were used in the JK SimMet simulation program to estimate SAG mill sizing and energy requirements. The A x b value used in the design was 24.2.

GRINDING CIRCUIT DESIGN

Several different design methods were used to size the SAG mill – ball mill circuit. The 80th percentile of the crushing and grinding parameters obtained from metallurgical testing were used in the design to provide sufficient power to process the majority of the ores being mined.

The following methods were used to calculate the size and power requirements of the grinding circuit:

- Morrell's Equation
- JK SimMet with the Bond Equation
- JK SimMet with the Phantom Cyclone
- SAG Design
- Orway Mineral Consultants (OMC)

To calculate the power requirement of the SAG and ball mill pinion power, the following design criteria were used:

- Simulations were performed at a nominal tonnage of 906 tonnes per hour (tph) or 20,000 tpd;
- Energy requirements (operating work indices) were then used to determine the operating power and required installed power for the SAG mill and ball mill for a nominal tonnage of 21,000 tpd;
- Variable transfer size (T80), calculated by the method used;
- Final circuit particle size distribution, P_{80} of 75 μm ;
- A x b value of 24.2, t_a value of 0.35;
- BWi value of 15.0 kWh/t;
- Mia value of 29.3 kWh/t.

The results of the simulations are presented in Table 13-10.

TABLE 13-10 SAG MILL AND BALL MILL SIMULATION RESULTS

Method	Units	Morrell's	JK SimMet+	JK SimMet	SAG	OMC
		Equations	Bonds	+ Phantom	Design	
		80 th	Equation	Cyclone	80 th	80 th
		Percentile	80 th	80 th	Percentile	Percentile
			Percentile	Percentile		
Parameters						
F ₈₀	µm	162,500	162,500	162,500	152,000	<150,000
T ₈₀	µm	750	2400	2400	1300	Unknown
Final P ₈₀	µm	75	75	75	75	75
Energy Requirements (Operating Work Indices)						
SAG Mill	kWh/t	15.26	13.23	13.23	12.56	13.70
Ball Mill	kWh/t	12.92	13.03	12.20	12.89	12.60
Subtotal	kWh/t	28.18	26.26	25.43	25.45	26.30
Pebble Crusher	kWh/t	0.46	0.37	0.37	-	0.57
Total	kWh/t	28.64	26.63	25.79	25.45	26.87
Operating Power Requirement (21,000 tpd)						
SAG Mill	kWh/t	14,510	12,580	12,580	11,948	13,033
Ball Mill	kWh/t	12,289	12,395	11,603	12,262	12,143

Notes:

1. Simulations were performed at 20,000 tpd. Operating powers for 21,000 tpd were calculated using the same operating work index (kWh/t).
2. The 79th percentile used for the SAGDesign simulations was based on seven samples only.

The results of the various SAG mill calculation and simulation methods yielded similar power requirements. The highest power requirement was obtained using the Morrell equations and the lowest, using the SAGDesign method. It was decided to use the JK SimMet + Bond Work Index method to determine the SAG mill sizing for the Feasibility Study.

Based on these results, 15 MW dual pinion drives were selected for both the SAG mill and the ball mill to process a mill fresh feed throughput of 951 tph. The SAG mill and drive were sized with an operating to installed power ratio of 90% and a safety factor of 5% was added. The mill was sized for a 13% ball charge and a maximum charge of 16%. The nominal mill load is 25% and the maximum load is 30%. The ball mill drive size was selected to match the SAG mill drive to minimize the spare part requirements. An 11 m x 6.1 m (5.6 m EGL) SAG mill and a 7.9 m x 12.3 m (12.2 EGL) ball mill were selected based on equipment sizing software and discussions with mill suppliers.

Subsequent simulations performed at 21,000 tpd (951 tph) indicated that the T₈₀ of the SAG mill circuit would be 2,800 µm rather than 2,400 µm.

GRAVITY RECOVERABLE GOLD

Two gravity recoverable gold (GRG) tests were performed by FLSmidth, Knelson, on ODM and Z-433 Zone composites. The test results are presented in Table 13-11.

TABLE 13-11 GRAVITY RECOVERABLE GOLD TEST RESULTS

Sample	Grind Size P ₈₀ (µm)	Product	Mass g	Mass %	Assay Au g/t	Au g	Distribution %	
ODM Master	650	Stage 1 Conc	79.1	0.4	46.7	3,691	18.8	
	542	Sampled Tails	188.8	1.0	0.62	118	0.6	
	275	Stage 2 Conc	76	0.4	48.7	2,698	18.8	
	211	Sampled Tails	205.7	1.1	0.72	149	0.8	
	141	Stage 3 Conc	98.6	0.5	27.1	2,676	13.6	
	90	Sampled Tails	18,339	96.6	0.51	9,311	47.7	
	Total (Head)			18,987	100	1.03	19,643	100
	Final Concentrate			254	1.3	39.7	10,065	51.2
Z-433	612	Stage 1 Conc	80.2	0.4	56	4,529	20.6	
	546	Sampled Tails	204.5	1.02	0.89	182	0.83	
	260	Stage 2 Conc	87.9	0.44	52	4,568	20.8	
	247	Sampled Tails	194.5	0.97	0.75	145	0.66	
	132	Stage 3 Conc	109.8	0.55	35.8	3,927	17.9	
	92	Sampled Tails	19,323	96.6	0.45	8,609	39.2	
	Total (head)			20,000	100	1.10	21,960	100
	Final Concentrate			277.9	1.39	46.9	13,024	59.3

The test results indicated that for samples ground to 90 µm, 51.2% of the gold in the ODM master composite is recoverable by gravity concentration and 59.3% of the gold in the Z433 composite is recoverable by gravity.

The gravity circuit is designed to treat primary cyclone feed slurry, with a P₈₀ of 1,000 µm so the gravity recovery will be closer to the values in the coarser range of the test. At 612 µm 18.8% of the gold is recoverable by gravity in ODM master composite and 20.8% in the Z433 composite.

GRAVITY VARIABILITY TESTWORK

In addition to the GRG tests, gravity separation was also performed during the variability testing program using 2 kg samples. The gravity recoveries of the variability tests ranged from 1% to 77% with an average of 27% for the non-CAP zones excluding the Intrepid Zone. The

gravity gold recovery from the CAP zone was considerably lower, with an average recovery of 9%. The Intrepid Zone also had lower gravity recoveries, averaging 16% for gold.

Gold recovery by gravity is dependent on gold particle liberation which is a function of the gold particle size, mineral particle size after grinding, and head grade. In both the ODM master composite and the Z433 samples, the best recovery was from the -90 µm fraction, 47.7% for the ODM composite and 39.2% for the Z433 sample.

CYANIDE LEACHING

GRAVITY CONCENTRATION AND LEACHING OF ROCK GRAVITY TAILINGS

Gravity gold recovery followed by leaching of the gravity tailings with cyanide was performed on samples of the ODM master composite as part of the trade-off study between flotation and concentrate leaching, and gravity concentration and leaching of the whole rock gravity tailings.

Cyanide leaching tests were performed on the ODM composite samples using the following baseline conditions:

- Grind sizes ranging from 50 µm to 119 µm
- Pre-aeration for 0.5 hours with air
- Pulp density of 50% solids
- Pulp pH was maintained between 10.5 and 11.0
- Cyanide concentration was varied between 0.5 g/L and 1.0 g/L NaCN
- Retention time was varied between 6 hours and 48 hours

The gravity tailings leach results for gold and silver are presented in Tables 13-12 and 13-13 respectively. Results presented for grind sizes with more than one test are averaged values.

TABLE 13-12 GRAVITY TAILINGS LEACH TEST RESULTS FOR GOLD

Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery, %						Au Assays	
		NaCN	CaO	6h	24h	36h	48h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
1	119	0.08	0.39	77.7	83.5	85.6	85.8	29.1	89.9	0.1	0.98
1	95	0.12	0.38	76.9	86.3	85.3	86.7	29.1	90.6	0.1	0.98
3	68	0.16	0.39	78.7	88.3	84.6	89.3	29.1	92.4	0.08	0.98
1	50	0.34	0.40	79.2	87.9	88.4	89.8	29.1	92.8	0.08	0.98
3	94	0.09	0.34	77.2	87.6	87.0	88.1	25.7	91.1	0.1	1.05
2	75	0.10	0.31	79.3	89.9	87.8	90.1	25.7	92.6	0.08	1.05
4	62	0.14	0.36	79.3	87.5	88.1	89.6	25.7	92.3	0.08	1.05
3	51	0.18	0.37	78.8	90.6	88.6	90.8	25.7	93.2	0.07	1.05

TABLE 13-13 GRAVITY TAILINGS LEACH TEST RESULTS FOR SILVER

Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Ag Recovery, %						Ag Assays	
		NaCN	CaO	6h	24h	36h	48h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
1	119	0.08	0.39	53.4	61.5	64.7	66.5	4.6	68.0	1.20	3.80
1	95	0.12	0.38	54.5	64.5	67.3	68.9	4.6	70.3	1.10	3.80
3	68	0.16	0.39	54.4	64.4	63.2	68.8	4.6	70.2	1.13	3.80
1	50	0.34	0.40	52.9	63.5	65.7	68.0	4.6	69.5	1.20	3.80
3	94	0.09	0.34	60.8	70.6	73.0	74.7	6.7	76.4	0.87	3.80
1	75	0.10	0.31	64.9	75.4	74.2	78.8	6.7	80.2	0.70	3.80
4	62	0.14	0.36	59.9	69.6	72.3	72.8	6.7	74.6	0.96	3.80
3	51	0.18	0.37	56.2	67.1	68.6	71.6	6.7	73.5	1.05	3.80

Test results yielded leach gold recoveries ranging from 85.8% at 119 µm to 90.8% at 51 µm and total gold recoveries from 89.9% at 119 µm to 93.2% at 51 µm. Gold recovery at the design grind size of 75 µm was 90.1% for leaching and 92.6% for leaching plus gravity recovery. Gold liberation and recovery are dependent on grind size. NaCN consumption increased from 0.08 kg/t NaCN at 119 µm to 0.18 kg/t at 51 µm. Gravity recovery was 29.1% for the first set of samples and 25.7% in the second set of samples.

Silver cyanide leaching recoveries increased from 66.5% at 119 µm to 78.8% at 69 µm and then dropped for the 62 µm and 51 µm tests. The silver recoveries increased with a decrease in particle size initially. The decrease in recovery may be due to a deficiency of cyanide at the finer grind. NaCN consumption increased from 0.08 kg/t NaCN at 119 µm to 0.34 kg/t at 50

µm. Gravity recovery was 4.6% for the first set of samples and 6.7% for the second set of samples.

GRAVITY TAILINGS LEACHING

Gravity tailings leach tests were performed on samples from the Initial Pit and the RLOM composites. The tests were performed for fixed times, with residue assays measured for each time duration.

Thirty-six tests were performed for each composite to help determine leach time and final grind size using the following criteria:

- Four leach times were used for each composite.
 - Initial Pit: 18, 24, 30, and 36 hours.
 - RLOM: 12, 18, 24, 30 hours
- Three grind sizes were tested, 110 µm, 85 µm, and 70 µm
- Triplicate tests were performed on each sample for each grind size and sample time for a total of 36 tests.

The gravity tailings leach test results are presented in Tables 13-14 and 13-15 for gold and silver, respectively. The results are presented as averages of the 12 tests performed per grind size per composite.

TABLE 13-14 INITIAL PIT AND RLOM GRAVITY TAILINGS LEACH TEST RESULTS FOR GOLD

Comp Name	Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery, %						Au Assays	
			NaCN	CaO	12h	18h	30h	36h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
Initial Pit	12	110	0.03	0.32	-	82.6	82.6	83.9	33.1	89.2	0.12	1.07
	12	85	0.04	0.33	-	84.8	85.2	86.4	33.1	90.9	0.10	1.07
	12	70	0.05	0.35	-	85.8	86.7	86.5	33.1	90.9	0.10	1.07
RLOM	12	110	0.02	0.31	79.7	79.7	82.2	-	29.6	87.5	0.10	0.83
	12	85	0.02	0.32	82.1	82.6	84.2	-	29.6	88.9	0.09	0.83
	12	70	0.01	0.32	84.1	85.2	85.7	-	29.6	90.0	0.08	0.83

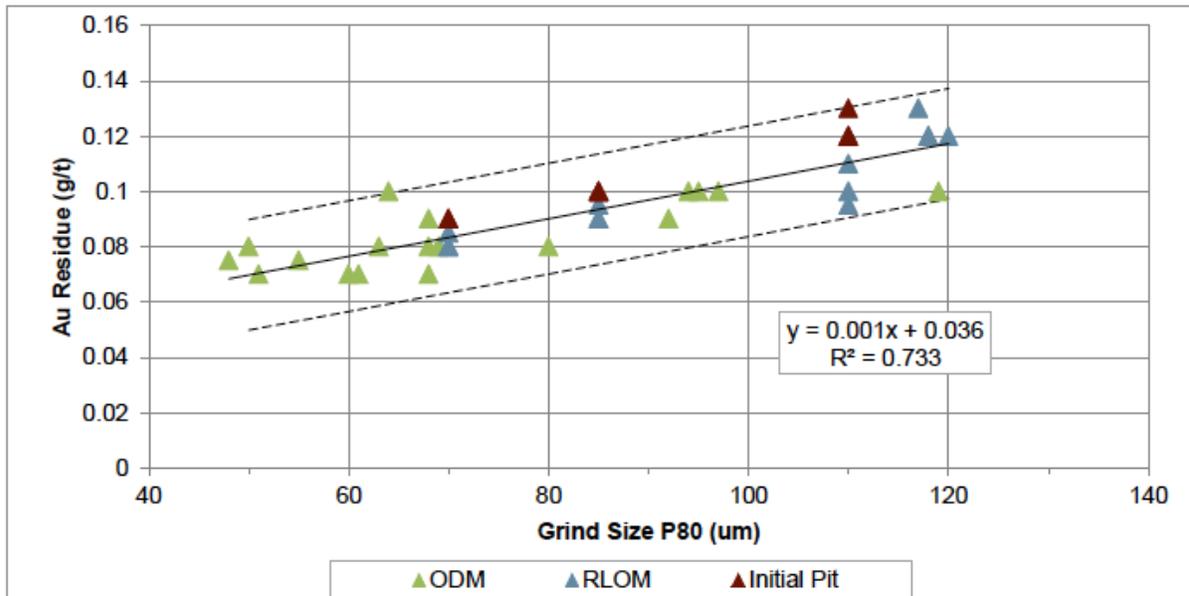
TABLE 13-15 INITIAL PIT AND RLOM GRAVITY TAILINGS LEACH TEST RESULTS FOR SILVER

Comp Name	Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Ag Recovery, %							Ag Assays	
			NaCN	CaO	12h	18h	24h	30h	36h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
Initial Pit	12	110	0.03	0.32	-	62.3	61.9	69.9	61.2	7.4	64.1	1.07	2.80
	12	85	0.04	0.33	-	59.4	62.4	70.5	62.8	7.4	65.5	1.07	2.80
	12	70	0.05	0.35	-	58.0	65.3	72.1	61.8	7.4	64.6	1.10	2.80
RLOM	12	110	0.02	0.31	61.1	66.4	68.1	68.9	-	6.1	70.8	0.80	2.80
	12	85	0.02	0.32	65.1	68.9	70.8	71.3	-	6.1	73.1	0.77	2.80
	12	70	0.01	0.32	66.2	72.1	70.7	70.8	-	6.1	72.6	0.80	2.80

The test results show that the gold and silver recoveries increase with a decrease in particle size indicating that the gold and silver grains are not fully liberated. Gold recovery did not improve beyond 30 hours. Initial Pit silver recovery increased up to 30 hours but then dropped between 30 and 36 hours. The decrease could indicate cyanide starvation on these particular samples at 36 hours.

The gravity tailings residue assays for the ODM, Initial Pit, and RLOM composite leach test results were plotted versus the grind size of the feed material and presented in Figure 13-7. The dotted lines represent the sensitivity of the assay technique of 0.02 g/t Au.

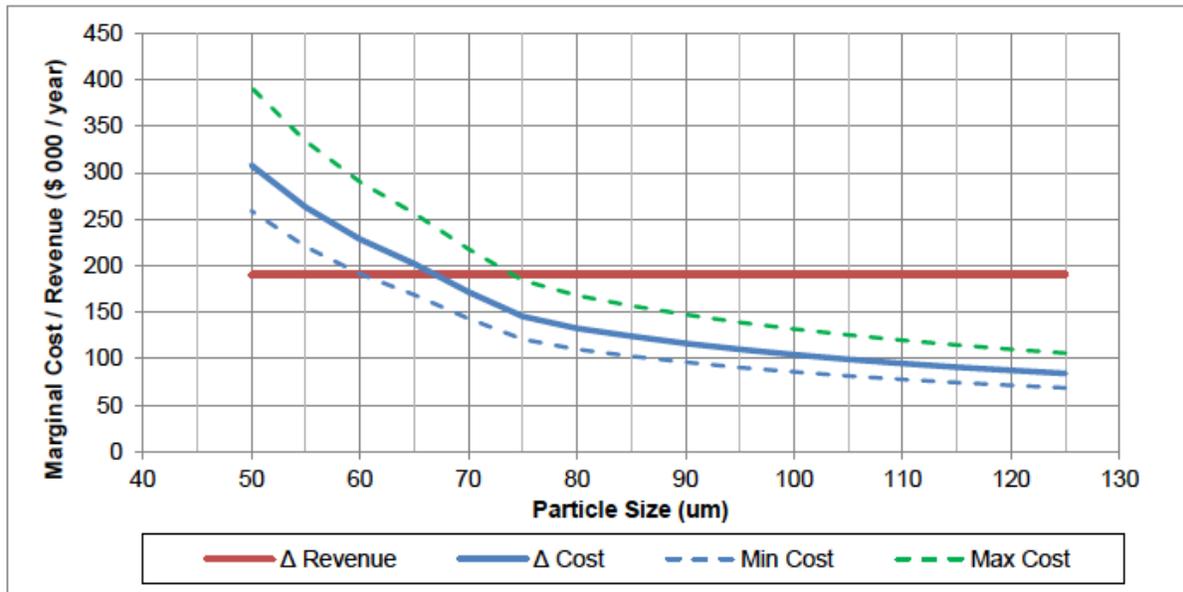
FIGURE 13-7 GRAVITY TAILINGS LEACH RESIDUE VERSUS GRIND SIZE



To determine a final P₈₀ for the variability test program, a cost versus revenue study was performed during the Feasibility Study. The costs included cyanide consumption, grinding energy at a fixed production rate and estimated media wear, while the revenue was calculated based on the residue equation from Figure 13-8. High and low cost scenarios were investigated in addition to the nominal costs. The cost of sodium cyanide, steel, and energy were varied to generate the high and low cost scenarios.

The results are presented in Figure 13-8.

FIGURE 13-8 COST AND REVENUE ANALYSIS BY GRIND SIZE



The results show that for the average costs of the listed parameters, grinding to 65 µm is still economical, however, when using the higher costs, it is only economical to grind to 75 µm.

Based on these results, a grind size of 75 µm and a retention time of 36 hours were selected for the variability test program.

EFFECT OF CYANIDE CONCENTRATION ON GOLD RECOVERY

The effect of cyanide concentration on gold recovery was investigated using RLOM composites samples. The cyanide concentrations were varied between 0.15 g/L and 0.5 g/L NaCN. The tests were run for 36 hours and samples were collected at time intervals.

The results of the tests are presented in Table 13-16.

TABLE 13-16 EFFECT OF CYANIDE CONCENTRATION ON GOLD RECOVERY

Comp Name	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery, %								Au Assays	
		NaCN Conc g/t	CaO	12h	18h	24h	30h	36h	Grav	Grav + CN	Residue, g/t	Head Grade, g/t	
RLOM	118	0.50	0.11	0.40	77	80	83	81	82.8	16.4	85.6	0.12	0.67
	117	0.30	0.08	0.37	71	77	82	81	81.9	16.4	84.9	0.13	0.69
	120	0.20	0.06	0.40	74	78	82	82	82.3	16.4	85.2	0.12	0.65
	118	0.15	0.06	0.41	70	77	81	80	82.3	16.4	85.2	0.12	0.68

Gold recoveries and initial leaching rates improved with an increase in NaCN concentration, though the differences were not large. The best result was obtained with a cyanide concentration of 0.5 g/L NaCN.

EFFECT OF PRE-AERATION ON GOLD RECOVERY

Pre-aeration tests were performed on sets of samples from the Initial Pit and the RLOM composites. The tests were performed for 36 hours and samples were taken at intervals throughout the test. Gold recovery, cyanide, and lime consumptions were measured. The results of the tests are presented in Table 13-17.

TABLE 13-17 EFFECT OF PRE-AERATION ON GOLD LEACH RECOVERY

Comp Name	Pre-Aeration	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery, %						Au Assays	
			NaCN	CaO	6h	12h	24h	36h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
Initial Pit	Y	100	0.07	0.36	79	83	83	83.5	31.4	88.7	0.12	0.73
	Y	100	0.08	0.36	73	79	80	82.7	31.4	88.1	0.13	0.72
	N	100	0.22	0.30	74	82	86	84.0	31.4	89.0	0.12	0.72
	N	100	0.19	0.31	75	82	81	85.0	31.4	89.7	0.12	0.77
RLOM	Y	118	0.08	0.36	75	75	81	80.8	16.4	84.0	0.14	0.70
	Y	118	0.07	0.36	76	82	83	82.1	16.4	85.0	0.13	0.70
	N	118	0.18	0.33	72	77	80	81.5	16.4	84.5	0.13	0.70
	N	118	0.25	0.29	70	70	77	78.8	16.4	82.3	0.15	0.71

The tests with pre-aeration had significantly lower cyanide and lime consumptions than those without pre-aeration. The results were nearly the same for both sets of samples. Cyanide consumption increased from 0.07 – 0.08 kg/t with pre-aeration to 0.18 - 0.25 kg/t without pre-

aeration. The gold recoveries were similar in both cases. Based on these tests, the variability tests were run using pre-aeration.

OXYGEN VERSUS AIR FOR PRE-AERATION

The effect of the use of oxygen in the cyanide leach tests was investigated. The results of the tests are presented in Table 13-18. The difference between oxygen and air for pre-aeration did not have a significant effect on cyanide and lime consumption. Lead nitrate did not reduce cyanide and lime consumption below that achieved using air. Based on these results, the variability tests included pre-aeration with air.

TABLE 13-18 EFFECT OF O₂, AIR AND LEACH NITRATE ON LEACH GOLD TEST RESULTS

Comp Name	Aeration	Lead Nitrate	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery, %						Au Assays	
				NaCN	CaO	6h	12h	24h	36h	Gravity	Grav + CN	Residue, g/t	Head Grade, g/t
Initial Pit	O ₂	N		0.04	0.37	82	-	-	-	29.0	87.1	0.12	0.97
	O ₂	N	54	0.04	0.36	-	86	-	-	29.0	90.2	0.09	
	O ₂	N	52	0.11	0.41	-	-	89	-	29.0	92.2	0.07	
	O ₂	N	61	0.06	0.38	-	-	88	-	29.0	91.8	0.10	
	O ₂	N	55	0.12	0.38	-	-	-	87	29.0	91.0	0.09	
	O ₂	N	59	0.04	0.39	-	-	-	87	29.0	90.8	0.10	
	O ₂	Y	59	0.16	0.50	-	-	-	88	29.0	91.5	0.08	
	O ₂	Y	45	0.05	0.52	-	-	-	87	29.0	90.9	0.08	
	Air	Y	48	0.14	0.56	-	-	-	88	29.0	91.3	0.08	
	Air	Y	59	0.06	0.51	-	-	-	87	29.0	90.9	0.09	
RLOM	O ₂	N	66	0.05	0.36	84	-	-	-	38.5	90.5	0.08	0.89
	O ₂	N	59	0.05	0.41	-	87	-	-	38.5	91.9	0.07	
	O ₂	N	79	0.06	0.33	-	-	87	-	38.5	92.2	0.09	
	O ₂	N	68	0.07	0.40	-	-	84	-	38.5	90.3	0.08	
	O ₂	N	57	0.08	0.41	-	-	-	85.3	38.5	91.0	0.08	
	O ₂	N	66	0.08	0.41	-	-	-	85.5	38.5	91.1	0.08	
	O ₂	Y	70	0.06	0.53	-	-	-	84.0	38.5	90.2	0.08	
	O ₂	Y	71	0.03	0.53	-	-	-	84.6	38.5	90.5	0.08	
	Air	Y	72	0.06	0.50	-	-	-	82.2	38.5	89.0	0.11	
	Air	Y	71	0.08	0.49	-	-	-	84.1	38.5	90.2	0.10	

INTREPID ZONE LEACHING KINETICS

The leaching kinetics of gold and silver from samples of the Intrepid Zone composites were investigated. The conditions for the tests were:

- Leach time of 96 hours with sampling at 30, 36, 48 and 72 hours.
- Target grind size P₈₀ of 75 µm
- Cyanide concentration of 0.5 g/L NaCN
- 30 -minute pre-aeration
- pH of 10.5 – 11.0

The results of the Intrepid Zone leach tests are presented in Tables 13-19 and 13-20.

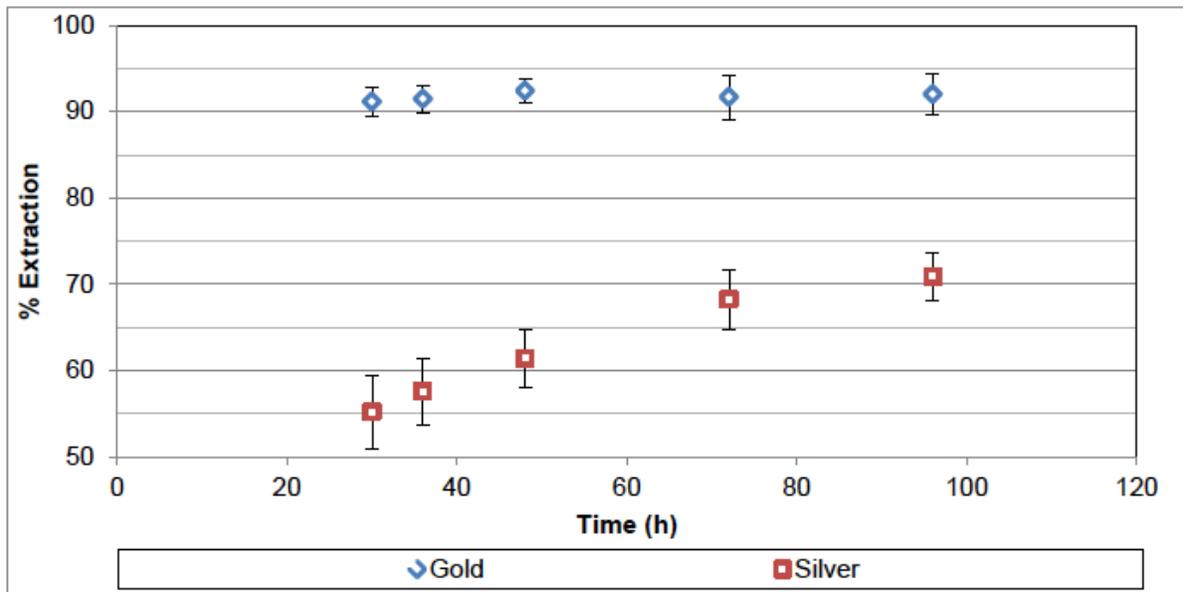
TABLE 13-19 INTREPID ZONE LEACH TESTING FOR GOLD

Test Number	P ₈₀ , µm	NaCN Conc g/t	Reagent Consumptions, kg/t		Au Recovery, %							Au Assays	
			NaCN	CaO	30h	36h	48h	72h	96h	Grav	Total	Residue, g/t	Head Grade, g/t
1	78	0.5	0.19	0.36	88	89	90	88	89.3	NA	89.3	0.17	1.45
1R	80	0.5	0.18	0.31	91	90	93	95	95.5	NA	95.5	0.10	1.45
2	74	1.0	0.31	0.32	92	93	93	92	92.2	NA	92.2	0.13	1.45
3	74	0.5	0.20	0.37	91	92	93	92	90.9	16.9	92.4	0.13	1.45
4	72	1.0	0.30	0.33	93	93	93	92	92.1	16.9	92.4	0.12	1.45

TABLE 13-20 INTREPID ZONE LEACH TESTING FOR SILVER

Test Number	P ₈₀ , µm	NaCN Conc g/t	Reagent Consumptions, kg/t		Ag Recovery, %							Ag Assays	
			NaCN	CaO	30h	36h	48h	72h	96h	Grav	Total	Residue, g/t	Head Grade, g/t
1	78	0.5	0.19	0.36	56	58	62	68	70.1	NA	70.1	4.3	13.8
1R	80	0.5	0.18	0.31	52	55	60	67	69.7	NA	69.7	4.6	13.8
2	74	1.0	0.31	0.32	58	61	64	70	72.5	NA	72.5	4.1	13.8
3	74	0.5	0.20	0.37	50	52	56	64	67.5	3.8	68.7	4.5	13.8
4	72	1.0	0.30	0.33	60	61	65	73	74.5	3.8	75.5	3.6	13.8

Figure 13-9 is a plot of the average leach extraction for gold and silver as a function of time to analyze the reaction kinetics. The data includes results with and without gravity separation

FIGURE 13-9 INTREPID ZONE GOLD AND SILVER CYANIDE LEACHING KINETICS

The kinetics for the extraction of silver from the Intrepid Zone samples is relatively slow and silver extraction is still increasing after 96-hours of leaching. Gold extraction is much faster and is essentially complete after 30 hours. Based on this information, it was decided to use the criteria established for the main pit to treat the Intrepid Zone.

VARIABILITY TESTING

Cyanide leaching was performed on 208 samples from the main pit and 30 samples from the Intrepid Zone. The results were used to develop grade-recovery curves for both gold and silver.

All of the tests were performed under the following conditions:

- Leach time of 36 hours with samples taken at 30 and 36 hours;
- Target grind size P_{80} of 75 μm ;
- Cyanide concentration of 0.5 g/L NaCN;
- 30-minute pre-oxidation with air;
- pH of 10.5 to 11.0.

Tables 13-21 and 13-22 contain summaries of the leach test results.

TABLE 13-21 AVERAGED VARIABILITY LEACH TEST RESULTS FOR GOLD

Zone	Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Au Recovery %				Au Assays		
			NaCN	CaO	30h	36h	Grav	Overall	Overall Recalc	Residue, g/t	Direct Head Grade, g/t
ODM	138	95	0.06	0.37	78.1	78.7	25.8	83.8	90.1	0.12	1.19
Z-433	30	82	0.10	0.41	82.8	84.4	35.6	89.5	93.8	0.08	1.21
HS	13	86	0.06	0.36	84.4	86.1	24.2	89.1	90.8	0.05	0.51
NZ	24	86	0.08	0.40	82.1	82.7	27.0	87.0	91.2	0.07	0.75
Intrepid	30	75	0.10	0.31	86.1	86.9	16.1	88.0	92.0	0.13	1.57
NonCAP	235	89	0.07	0.37	80.5	81.3	25.7	85.7	90.8	0.10	1.17
CAP	40	92	0.11	0.62	71.5	71.5	8.7	73.9	76.8	0.16	0.67
Total	275	89	0.08	0.41	79.1	79.9	23.4	84.0	89.8	0.11	1.09

TABLE 13-22 AVERAGED VARIABILITY LEACH TEST RESULTS FOR SILVER

Zone	Number of Tests	P ₈₀ , µm	Reagent Consumptions, kg/t		Ag Recovery %				Ag Assays		
			NaCN	CaO	30h	36h	Grav	Overall	Overall Recalc	Residue, g/t	Direct Head Grade, g/t
ODM	138	95	0.06	0.37	57.4	58.8	10.0	62.7	67.1	1.24	3.77
Z-433	30	82	0.10	0.41	49.1	51.4	12.8	57.6	55.2	0.60	1.34
HS	13	86	0.06	0.36	47.8	48.2	8.5	52.8	51.1	0.51	1.04
NZ	24	86	0.08	0.40	55.8	56.1	8.5	59.5	62.2	0.53	1.39
Intrepid	30	75	0.10	0.31	58.5	60.2	5.3	60.5	55.5	6.6	14.9
NonCAP	235	89	0.07	0.37	55.8	57.2	9.5	60.9	61.4	1.73	4.8
CAP	40	92	0.11	0.62	63.8	65.1	3.0	66.4	67.5	0.86	2.65
Total	275	89	0.08	0.41	57.0	58.3	8.6	61.9	61.9	1.60	4.21

Gold results:

- The gold cyanide leaching recoveries in the variability tests ranged from 78.7% in the ODM samples, which were ground to 95 µm, to 86.9% in the Intrepid Zone, which was ground to 75 µm.
- Gravity gold recoveries ranged from 16.1% in the Intrepid Zone to 35.6% in the Z-433 zone.
- Overall gold recoveries ranged from 90.1% in the ODM samples to 93.8% in the Z-433 Zone. The overall gold recovery for the non-CAP ore was 90.8%.
- The average residue value for all the samples was 0.10 g/t Au for the non-CAP zones including the Intrepid Zone and 0.16 g/t Au for the CAP Zone.

- The leaching performance was relatively consistent with the majority of the variance driven by the grind size and the gravity recovery, which may be related. Gold leaching was complete after 30 hours.
- The average P₈₀ of 89 µm was coarser than the target P₈₀ of 75 µm. The P₈₀ ranged from 30 µm to 260 µm with the same grinding times, indicating that there are large variations in ore hardness in the deposit.

Silver results:

- The silver cyanide leaching recoveries in the variability tests ranged from 48.2% in the HS samples, which were ground to 86 µm, to 60.2% in the Intrepid Zone, which was ground to 75 µm.
- Gravity silver recoveries ranged from 5.3% in the Intrepid Zone to 12.8% in the Z-433 Zone.
- Overall silver recoveries ranged from 51.1 in the HS samples to 67.1% in the ODM zone. The overall silver recovery for the non-CAP ore was 61.4%. The overall silver recovery for the CAP samples was 67.5% for an overall weighted average total silver recovery of 61.9%.
- The average silver residue value for all the samples was 1.73 g/t Ag for the non-CAP zones including the Intrepid Zone and 0.86 g/t Ag for the CAP Zone.
- The leaching performance was relatively consistent with the majority of the variance driven by the grind size, gravity separation, and leaching retention time. The silver was still leaching at 36 in all of the tests.
- The average P₈₀ of 89 µm was coarser than the target P₈₀ of 75 µm. The P₈₀ ranged from 30 µm to 260 µm with the same grinding times, indicating that there are large variations in ore hardness in the deposit.

DIAGNOSTIC LEACH TESTWORK

Diagnostic leach tests were performed on the leach tailings from three ODM samples and three CAP samples to determine the reasons that the residual gold did not leach.

The diagnostic leach test procedure includes the following steps:

- Intensive cyanide leach: Extraction of gold that is readily available and is an indication that more retention time was required to complete reaction.
- Hydrochloric acid leach followed by intensive cyanidation leach: Extraction of gold that is associated with pyrrhotite, calcite, ferrites, etc. This is done by leaching the tailings using hydrochloric acid to dissolve the pyrrhotite and other minerals, then performing the intensive cyanide leach to extract the liberated gold.
- Aqua regia leach: Extraction of gold associated with or encapsulated in sulphide minerals such as pyrite and arsenopyrite.

The final residue from these tests is considered to be locked in silicates or associated with fine sulphides that are locked in silicates.

The results from the diagnostic leach tests indicated that most of the residual gold is associated with pyrite, arsenopyrite, or other sulphide minerals for both the CAP and the ODM samples.

- The amount of the residual gold recovered by the aqua regia leach was estimated to be between 62% and 92%.
- Little to no gold was readily recoverable using intensive cyanide leaching, with four of the six samples having gold pregnant leach solution tenors below the detection limits and the other two being at the detection level.
- Higher percentages of the residual gold were recovered using the HCl leach followed by intensive cyanide leaching with approximately 8% to 24% of the residual gold being leached using this method.
 - Three of the six samples had final residual gold below detection limit while the other three samples were at the detection limit of 0.02 g/t Au.

MERCURY ASSAYS

Mercury assays were performed on two composite samples, measuring mercury levels in the feed, residue, loaded carbon, and barren solution streams after undergoing leaching and gold adsorption (Carbon-in-Pulp (CIP)). The objective of the testwork was to determine if any mercury leached into solution and adsorbed onto the carbon. All assays were below detection level except for one carbon reading, which had an assay of 0.06 g/t Hg.

CYANIDE DESTRUCTION

The SO₂ – air cyanide destruction process was investigated on the leach solutions from three composites: Initial Pit, RLOM, and Intrepid Zone. The Intrepid Zone sample was tested after completion of the main pit testwork. The first series of tests on the Intrepid Zone sample yielded high residual cyanide levels, however, a repeat of the test showed results in line with those from the main pit samples. One large bulk cyanide destruction and three continuous tests were conducted for each composite.

The cyanide destruction test results are presented in Table 13-23.

TABLE 13-23 CYANIDE DESTRUCTION TEST RESULTS

Sample	Pulp Density, %	Retention Time, Min	Solution Phase						Reagent Addition, g/gCN _{WAD}		
			pH	CN _T , mg/L	CN _{WAD} , Standard, mg/L	CN _{WAD} , Picric, mg/L	Cu, mg/L	Fe, mg/L	SO ₂	Lime	Cu
Feed	-	-	10.7	152	117	-	9.4	1.8	-	-	-
					Batch						
Initial Pit											
CND 3	50	90	8.6	-	-	<0.1	-	-	7.52	3.48	0.13
Continuous											
CND 3-1	50	75	8.6	3.1	0.19	0.40	0.08	0.1	5.33	3.33	0.12
CND 3-2	50	81	8.6	4.2	0.49	0.67	0.47	0.43	5.28	2.57	0.0
CND 3-3	50	80	8.6	5.2	0.12	0.12	0.73	0.58	4.66	1.89	0.0
Remaining Life of											
Feed	-	-	11.1	128	123	-	11.0	-	-	-	-
Batch											
CND4	50	180	8.5	-	-	0.4	-	-	12.7	14.9	0.24
Continuous											
CND 4-1	50	88	8.5	3.5	<0.1	0.38	0.07	0.1	4.46	4.47	0.23
CND 4-2	50	85	8.5	3.9	<0.1	0.25	<0.05	0.13	4.17	6.71	0.25
CND 4-3	50	99	8.5	5.8	0.13	0.29	0.10	0.52	4.24	1.79	0.0
Intrepid Zone											
Feed	-	-	10.7	151	77.4	-	20.0	2.22	-	-	-
Batch											
CND 2	50	150	8.6	-	-	0.26	-	-	11.9	7.68	0.13
Continuous											
CND 2-1	50	58	8.5	0.13	<0.1	4.1	18.0	0.2	4.64	2.36	0.13
CND 2-2	50	116	8.6	0.11	<0.1	0.94	7.3	0.2	4.64	2.36	0.13
CND 2-2	50	58	8.5	<0.1	<0.1	0.45	5.1	0.3	5.69	3.64	0.12
CND 2-2	50	116	8.5	<0.1	<0.1	<0.1	1.1	0.2	5.69	3.64	0.12

The results show that this process is effective at lowering the weak acid dissociable cyanide (CN_{WAD}) to levels well below 5 ppm CN. The average reagent consumptions were 4.7, 3.5, and 0.1 g/g CN_{WAD} for SO₂, lime and copper for the main pit, respectively. The reagent consumptions for the Intrepid Zone sample were comparable to these values. These consumptions are considered to be in agreement with standard industrial practices.

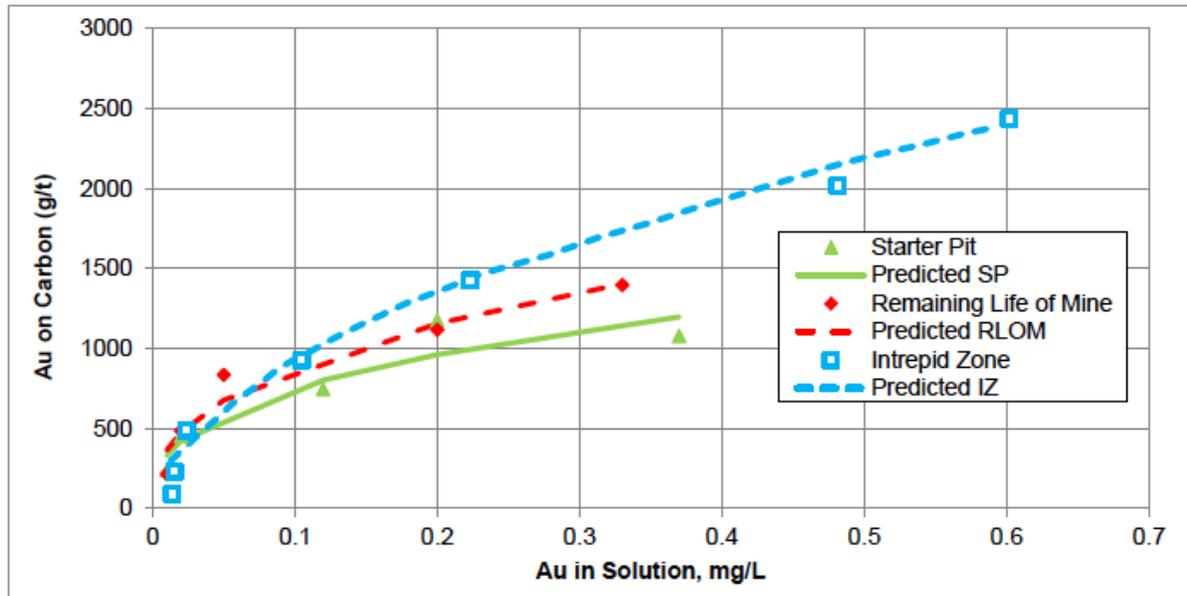
CARBON IN PULP MODELING

CIP modelling work was performed by SGS to validate the design of the CIP circuit. This technique is typically used for modelling of conventional CIP circuits, but was modified to model the kinetics of a carousel-style pump cell CIP circuit. Only gold was modelled by SGS.

The Initial Pit, RLOM, and Intrepid Zone master composites were used for the CIP modelling testwork.

The isotherms from the testwork are presented in Figure 13-10.

FIGURE 13-10 CIP ISOTHERMS USED FOR MODELLING



Using the isotherms it is possible to model the kinetics for gold adsorption onto the carbon in CIP. The adsorption kinetics are modelled using a kK value that is the product of the model output kinetic constant k and the model output equilibrium constant K . The kK values from the testwork were 69, 79, and 90 for the Initial Pit, RLOM, and Intrepid Zone composites respectively. The values are slightly lower than the reference value of 100, which is usually used as a cut-off from slow the fast kinetics.

Modelling was performed by SGS to determine the number of CIP tanks in series, frequency of carbon movement, and size of CIP tanks required. The simulations yielded solution losses of between 0.007 mg/L and 0.035 mg/L, depending on the configuration. The results indicated that a seven or eight tank configuration is required to achieve acceptable gold adsorption efficiency, and that the ability to transfer carbon every day is beneficial. Based on these results, the CIP circuit was designed to have seven tanks in series and the stripping circuit was

sized to be able to strip and regenerate one full tank, 20 tonnes of carbon, every two days or one half tank, 10 tonnes of carbon per day.

It was recommended that constant residual solution gold tenor of 0.007 mg/L to 0.008 mg/L Au be used for the financial analysis based on the kinetics observed for the three composites.

THICKENER SIZING TESTWORK

FLOCCULANT SCREENING

Flocculant screening was performed by a flocculant supplier, prior to performing sedimentation rate testing with the selected thickener vendors. Pre-leach and pre-detox thickener feed material was generated and tested with using two sample composites, Initial Pit, and RLOM. The target grind size was 75 µm. Pre-screening was performed on the four different flocculant types presented in Table 13-24.

TABLE 13-24 FLOCCULANT TYPES TESTED

Flocculant	Charge	Molecular Weight	Charge Density
Flocculant 1	Anionic	~13-16	5%
Flocculant 2	Anionic	~13-16	10%
Flocculant 3	Non-ionic	~9-11	Low
Flocculant 4	Anionic	~14-17	20%

The test results indicated that Flocculant 3, the low molecular weight, non-ionic flocculant had the most consistent performance in terms of settling rates and overflow clarity.

- Flocculant 1 also showed good performance but did not compete with Flocculant 3.
- Flocculant 2 had poor overflow clarity.
- Flocculant 4 had poor settling and poor clarity.

Flocculant 3 or an equivalent was selected for vendor sedimentation testing.

It was found that two stages of flocculant addition were required for certain samples to achieve acceptable overflow clarity. A pH of 10.5 or higher significantly improved the settling rates and overflow clarity.

SEDIMENTATION TESTWORK

Sedimentation testing was performed at three different suppliers' laboratories to size the pre-leach thickener. The sedimentation test results are presented in Table 13-25.

TABLE 13-25 RESULTS OF SUPPLIER SEDIMENTATION TESTWORK

Sample	Description	Units	Supplier A	Supplier B	Supplier C
Design Feed Rate (Dry)		tph	951	951	951
Initial Pit	Settling Rate	tph/m ²	0.65	0.90	0.61-1.05
	Rise Rate	m/h	<7	-	3.4-5.9
	Flocculant Dosage	g/t	30-35	40	20-40
	Overflow Clarity	ppm	<200	<150	10-86
Remaining Life of Mine	Settling Rate	tph/m ²	-	1.0	0.65-1.14
	Rise Rate	m/h	-	-	3.6-6.3
	Flocculant Dosage	g/t	-	25	19-40
	Overflow Clarity	ppm	-	<200	50-145
Recommended Diameter		m	45	39	46

Based on the test results, the recommended thickener diameter is between 39 m and 46 m. The lowest settling rates were observed by Supplier C, while the highest were from Supplier B. A 45 m diameter pre-leach thickener was selected.

The flocculant dosages required ranged from 19 g/t to 40 g/t, with an average of the three suppliers of approximately 32 g/t.

INTREPID ZONE

Static and dynamic settling tests were performed by SGS on the Intrepid Zone samples. The settling rates were found to be lower than the Initial Pit and RLOM samples at 0.42 tph/m² to 0.61 tph/m². The flocculant addition rates were similar at approximately 25 g/t in the dynamic tests and 20 g/t for the static tests. Good overflow clarity was achieved in both types of tests.

SLURRY RHEOLOGY TESTS

Slurry rheology tests were performed by SGS on the Initial Pit and RLOM composites using a concentric cylinder viscometer. The objective of the testwork was to determine the critical solids density (CSD) and to predict the maximum underflow solids density during thickener operation.

It was determined that the CSD was 62% solids and 63.5% solids for the Initial Pit and RLOM composites respectively. The design for the pre-leach and pre-detox thickeners is 61% and 60% respectively.

GOLD AND SILVER RECOVERY CURVES

Recovery curves for both gold and silver were developed based on variability testwork performed at SGS-Lakefield. To develop the recovery curves, residue versus head grade curves were developed and the recovery was back-calculated from the curves. Solution losses were taken into account when calculating the recovery. A total of 208 samples were tested, with another 37 samples repeated, either for quality control or due to missed target grind size.

The ore body was divided into two zones to develop the grade-recovery curves: non-CAP and CAP. This is because the CAP Zone behaved significantly differently than the other zones in the main pit. Material from the Intrepid Zone is also considered to be non-CAP.

The general equations for the recovery curves are as follows:

Gold Recovery Curve:

$$Rec_{Au} = \left\{ \left[\frac{(x_{Au} - A_{(Au \text{ res})})}{x_{Au}} \right] \cdot 100 - B_{(Au \text{ Gravity Rec})} \right\} \cdot CIP_{\text{eff-Au}} \cdot EW_{\text{eff}} + B_{(Au \text{ Gravity Rec})} \cdot IL_{\text{eff}}$$

Silver Recovery Curve:

$$Rec_{Ag} = \left\{ \left[\frac{(x_{Ag} - A_{(Ag \text{ res})})}{x_{Ag}} \right] \cdot 100 - B_{(Ag \text{ Gravity Rec})} \right\} \cdot CIP_{\text{eff-Ag}} \cdot EW_{\text{eff}} + B_{(Ag \text{ Gravity Rec})} \cdot IL_{\text{eff}}$$

Where:

Rec_{Au} is the overall gold recovery

Rec_{Ag} is the overall silver recovery

x_{Au} is the gold head grade

x_{Ag} is the silver head grade

$A_{(Au \text{ res})}$ is the gold residue

$A_{(Ag \text{ res})}$ is the silver residue

$B_{(Au \text{ Gravity Rec})}$ is the gold gravity recovery

$B_{(Ag \text{ Gravity Rec})}$ is the silver gravity recovery

$CIP_{\text{eff-Au}}$ is the CIP adsorption efficiency for gold

CIP_{eff-Ag} is the CIP adsorption efficiency for silver

EW_{eff} is the CIP stripping solution electrowinning recovery

IL_{eff} is the intensive cyanidation and dedicated electrowinning efficiency

The CIP_{eff-Au} was calculated using a fixed discharge solution gold tenor of 0.007 mg/L based on testwork performed by SGS. Testwork for silver modelling was not performed, therefore the model was based on discussions with CIP suppliers. The silver adsorption efficiency was estimated to be 96.6%. EW_{eff} and IL_{eff} values were assumed to be 100% since all residual solids and solutions from these circuits are recycled to the process.

The specific equations used for developing the recovery curves are as follows:

Non-CAP Zones Gold Recovery Curve:

$$\text{Re}_{\text{CAu}} = [(x_{\text{Au}} - (0.0937 \cdot x_{\text{Au}}^{0.4223} - 0.007)) / x_{\text{Au}}] \cdot 100$$

Non-Cap Zones Silver Recovery Curve:

$$\text{Re}_{\text{CAg}} = [((x_{\text{Ag}} - (0.148 \cdot x_{\text{Ag}}^2 + 0.294 \cdot x_{\text{Ag}})) / x_{\text{Ag}}) \cdot 100 - 9.8] \cdot 0.966 + 9.98$$

CAP Zone Gold Recovery Curve:

$$\text{Re}_{\text{CAu}} = [(x_{\text{Au}} - (0.2497 \cdot x_{\text{Au}}^{1.015} - 0.007)) / x_{\text{Au}}] \cdot 100$$

CAP Zone Silver Recovery Curve:

$$\text{Re}_{\text{CAg}} = [((x_{\text{Ag}} - (0.0363 \cdot x_{\text{Ag}}^2 + 0.245 \cdot x_{\text{Ag}})) / x_{\text{Ag}}) \cdot 100 - 3.6] \cdot 0.966 + 3.6$$

The expected gold and silver recoveries as a function of head grade are presented in Table 13-26.

TABLE 13-26 GOLD AND SILVER RECOVERY VERSUS HEAD GRADE

Head Grade, g/t	Gold Recovery		Silver Recovery	
	Non-CAP	CAP	Non-CAP	CAP
0.2	72.8	72.1	68.2	72.4
0.4	82.3	73.6	67.9	71.7
0.6	86.2	74.1	67.6	71.0
0.8	88.5	74.2	67.4	70.3
1.0	89.9	74.3	67.1	69.6
1.2	91.0	74.4	66.8	68.9
1.4	91.8	74.4	66.5	68.2
1.6	92.4	74.4	66.2	67.5
1.8	92.9	74.4	65.9	66.8
2.0	93.4	74.4	65.6	66.1
2.5	94.2	74.4	64.9	64.3
3.0	94.8	74.4	64.2	62.6
3.5	95.3	74.4	63.5	60.8
4.0	95.6	74.3	62.8	59.1
4.5	95.9	74.3	62.1	57.3
5.0	96.2	74.3	61.4	55.6

TESTWORK INTERPRETATION

- The results from the SGS testing program formed the basis for the Mineral Reserve estimate and updated Feasibility Study.
- Gravity separation followed by whole rock leaching was chosen over flotation and concentrate leaching due to lower recoveries, higher cyanide consumptions, and the energy costs associated with fine grinding the concentrate.
- Grinding tests indicate significant variation in ore hardness in the ODM Zone.
- The Intrepid Zone ore can be treated using the same flowsheet as the main pit ores. The high silver values will increase the load on the CIP and elution circuit if not blended.
- A gravity circuit was included in the plant design.
- The process is expected to yield gold recoveries of approximately 90% to 91% and silver recoveries of 66% to 67% over the life of mine, without solution losses.
- The grind size chosen was 75 µm.
- As per the mine plan schedule, CAP Zone material, when mined, will be placed in the low grade ore stockpile and treated toward the end of the mine life. Non-CAP Zone material will be processed in years 1 to 8.5, resulting in elevated recovery for those years.

- It is KSN's opinion that the metallurgical test program for the Rainy River deposit has been comprehensive and has included the major ore types and the mine plan into consideration when developing the composite samples for testing. The types of tests performed were appropriate and provided sufficient information for preparing the designs for the process facilities. The selection of data for use in the engineering design criteria was decided by AMEC and by the vendors for sizing of major equipment such as the crushers and grinding mills.

The average gold and silver recoveries based on the mine plan are shown in Table 13-27.

TABLE 13-27 AVERAGE ANNUAL GOLD AND SILVER HEAD GRADE AND PROJECTED RECOVERY

Year	Gold		Silver	
	Au Head Grade g/t	Au Recovery %	Ag Head Grade g/t	Ag Recovery %
1	1.31	90.6	2.21	56.1
2	1.00	91.0	2.02	56.1
3	1.27	92.3	2.82	56.1
4	1.77	93.9	2.75	56.1
5	1.08	91.4	4.72	56.1
6	0.98	90.8	4.00	56.1
7	1.22	91.1	4.81	56.1
8	1.90	93.2	3.48	56.1
9	1.14	90.7	3.12	56.1
10	0.71	88.6	2.47	56.1
11	0.68	88.2	2.78	56.1
12	0.68	88.2	3.12	56.1
13	0.70	808.4	2.78	56.1
14	0.71	87.6	2.80	56.1
LOM	1.09	90.8	3.20	56.1

14 MINERAL RESOURCE ESTIMATE

The Mineral Resource block model estimate for the Rainy River Mine, with the exception of the Intrepid and 34 zones, has been prepared by Mr. Mauro Bassotti (formerly of New Gold). The estimates for the Intrepid and 34 zones have been prepared by Ms. Dorota El-Rassi (formerly of SRK). Ms. Dinara Nussipakynova, P.Geo., of AMC, has reviewed the methodologies and data used to prepare the Mineral Resource estimates and is satisfied that they comply with reasonable industry practice. Ms. Nussipakynova takes responsibility for these estimates. The Mineral Resource estimate conforms to Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Reserves dated May 10, 2014 (CIM (2014) definitions).

A summary of the timing, authorship, and responsibility of the current mineral resource estimates contained in this report is shown in Table 14-1. The data used for both the 2017 and 2015 block model estimates include the results of all drilling and updated geologic interpretation carried out on the property to December 31, 2017. The models have been depleted to reflect remaining Mineral Resources as of June 30, 2018.

TABLE 14-1 CURRENT MINERAL RESOURCE ESTIMATES AT RAINY RIVER

Area	Year of Estimate	Author	Effective Date	Responsibility
Intrepid	2015	El-Rassi	June 30, 2018	Nussipakynova
34 Zone	2015	El-Rassi	June 30, 2018	Nussipakynova
Main Zone	2018	Bassotti	June 30, 2018	Nussipakynova

The Mineral Resource estimate of the Main Zone, completed in 2017, was performed using Maptrek's Vulcan software. The Mineral Resource estimates of the Intrepid and 34 zones were carried out using GEMS software. Interpolation of gold and silver grades over all areas was completed using ordinary kriging (OK). Density values were interpolated in the Main Zone using inverse distance squared (ID²), and were assigned based on rock type at Intrepid and 34 Zone.

A summary of Mineral Resources at Rainy River is presented in Table 14-2. Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves

and do not have demonstrated economic viability. Definitions for resource categories used in this report are consistent with those defined by CIM (2014) and adopted by NI 43-101.

Open pit Mineral Resources are reported at cut-off grades of 0.3 g/t and 0.5 g/t gold equivalent (AuEq) for low grade material and for direct processing material, respectively. Underground Mineral Resources are reported at a cut-off grade of 2.0 g/t AuEq. Measured and Indicated Mineral Resources are estimated to total 63.109 million tonnes at grades of 1.06 g/t Au and 3.8 g/t Ag, containing 2,142,000 ounces of gold and 7,651,000 ounces of silver. Inferred Mineral Resources are estimated to total 8.871 million tonnes at grades of 1.10 g/t Au and 2.4 g/t Ag, containing 313,000 ounces of gold and 672,000 ounces of silver.

TABLE 14-2 MINERAL RESOURCES – EFFECTIVE JUNE 30, 2018

Class	Tonnes (000s)	Grades, g/t		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Direct Processing					
Open Pit (OP)					
Measured	2,996	1.14	5.7	109	550
Indicated	26,541	1.12	3.4	957	2,921
OP Measured and Indicated	29,537	1.12	3.7	1,066	3,471
Inferred	3,697	1.06	3.2	126	385
Underground (UG)					
Measured	-	-	-	-	-
Indicated	7,934	3.06	8.6	780	2,206
UG Measured and Indicated	7,934	3.06	8.6	780	2,206
Inferred	1,215	3.59	2.7	140	107
Low Grade (LG) - Stockpile					
Measured	2,462	0.35	3.3	28	261
Indicated	23,175	0.36	2.3	268	1,713
LG Measured and Indicated	25,637	0.36	2.4	296	1,974
Inferred	3,959	0.37	1.4	47	180
Combined Mineral Resources					
Measured	5,458	0.78	4.6	137	811
Indicated	57,651	1.08	3.7	2,005	6,840
Total Measured and Indicated	63,109	1.06	3.8	2,142	7,651
Total Inferred	8,871	1.10	2.4	313	672

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.

2. Mineral Resources are estimated using long-term metal prices of US\$1,375 per ounce gold, US\$19.00 per ounce silver and a C\$/US\$ exchange rate of 0.77. Metal recoveries of 95% for Au and 70% for Ag were used.
3. Mineral Resources are reported at cut-off grades for direct processing material of 0.50 g/t AuEq for open pit and 2.0 g/t AuEq for underground and between 0.30 g/t and 0.5 g/t AuEq for low grade resources. The gold equivalency formula is as follows: $AuEq\ g/t = Au\ g/t + ((Ag\ g/t * 19 * 70) / 1375 * 95)$
4. Bulk density ranges from 2.70 t/m³ to 3.08 t/m³.
5. Mineral Resources are exclusive of Mineral Reserves
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
7. Open pit Mineral Resources are constrained by a conceptual pit shell.
8. Totals may not add due to rounding.

AMC is not aware of any environmental, permitting, legal, title, taxation, socioeconomic, marketing, political, or other similar factors that could materially affect the stated Mineral Resource estimates.

RESOURCE ESTIMATION PROCEDURES

Since acquiring the Rainy River project in 2013, New Gold has made significant progress in understanding the geology and controls to gold mineralization at Rainy River. This work has resulted in the development of a three-dimensional (3D) geological model that encompasses the project area and serves as the underlying framework for the Mineral Resource estimate. In connection with this work, some of the borehole collar locations and downhole surveys have been updated using the Trimble Differential GPS system, resulting in the shift of several borehole positions. New Gold has revised its interpretation of deposit geology and mineral domains using the new and more accurate borehole locations.

The evaluation of Mineral Resources at the Main Zone involved the following steps:

1. Database compilation and verification.
2. Construction of wireframe models for major lithological units, using stratigraphy, structural trends, and an array of appropriate geochemical indices.
3. Definition of mineralization domains.
4. Data conditioning (compositing and capping) for geostatistical analysis and variography.
5. Selection of estimation strategy, estimation, and block model parameters.
6. Grade interpolation of gold, silver, and density.
7. Validation, classification, and tabulation.
8. Assessment of “reasonable prospects for eventual economic extraction” and selection of reporting cut-off grades.
9. Preparation of the Mineral Resource Statement.

Additionally, estimates for calcium and sulphur have been incorporated into the current block model to support waste rock characterization for long term mining and closure plans.

The 3D geological and mineralization domains were prepared onsite at Rainy River using Leapfrog software. The shapes were exported as DXF files and imported into Vulcan for Mineral Resource estimation. Vulcan software was used to prepare assay data for geostatistical analysis, construct the block model, prepare composite samples, estimate metal and density grades, and validate and tabulate the Mineral Resources. The geostatistical software Snowden Supervisor was used for variography, geostatistical analysis, and validation.

RESOURCE DATABASE

The Rainy River Mineral Resource database is comprised of a series of Microsoft Excel files and includes drill hole collar locations, downhole survey, assay, and lithology data from 2,116 core boreholes (911,168.4 m) drilled by New Gold, RRR, Bayfield, and Nuinsco. A summary of records directly related to the Rainy River Mineral Resource models are provided in Table 14-3.

TABLE 14-3 SUMMARY OF RESOURCE DATABASE

Item	Record Count/Details
Drill Holes	2,116
Total Length (m)	911,169
Downhole Survey Entries	36,028
Lithology Entries	499,768
Assay Entries	492,281
Assay Length (m)	712,009
Topographic Surface	1
Lithology Wireframes	50
Wireframes of Mineralization	32
Dilution Envelope Wireframes	1

All exploration information is located using the local UTM grid (NAD 83 datum, Zone 15). Resource modelling was conducted in this UTM coordinate space.

Upon receipt of the digital drilling data, AMC undertook the following validation step:

- Checked minimum and maximum values for each quality field and confirmed/edited those outside of expected ranges;
- Checked for inconsistencies in lithological unit terminology and/or gaps in the lithological table;
- Checked for gaps, overlaps, and out of sequence intervals for both assays and lithology tables;
- Checked that collar locations plot in the correct location against the topography and there are no collars that are above or below the surface. Below topography collars were confirmed to match with open pit pre-stripping activities;
- Checked that all downhole survey dips are negative (no upward holes present);
- Checked that downhole survey azimuth readings are all in range of expected drilling deviation and not impacted by any erroneous effects; and
- Checked the 2017 drilling file against the 2015 drilling file (in Vulcan) to validate that collars and drill traces are the same between the two files (Main Zone only)

GEOLOGICAL INTERPRETATION AND 3D SOLIDS

As it is currently defined by exploration drilling, the Rainy River deposit comprises a cluster of eight distinct zones of gold-silver mineralization, collectively referred to as the Main Zone (and includes the 34 Zone). Intrepid represents a satellite deposit located one kilometre to the east of the Main Zone. A top-of-bedrock plan view of local geology and known zones of mineralization is presented in Figure 14-1.

In 2017, New Gold updated the geological model for the deposit. The model comprises three dimensional wireframes delineating the major lithological units and zones of significant gold and silver mineralization. Lithologic domains were modelled in Leapfrog, and mineralization domains were modelled in GEMS, guided by drill hole data and interpreted cross sections spaced 25 m apart. The final Main Zone model is comprised of 50 discrete lithologic domains and 32 mineralization domains. Main Zone mineralization domains are shown in plan and isometric views in Figures 14-2 and 14-3, respectively. The wireframes delineating the Intrepid and 34 zones remain unchanged since the previous geologic model prepared by New Gold in 2015. Mineralization domains defining Intrepid are shown in Figure 14-4.

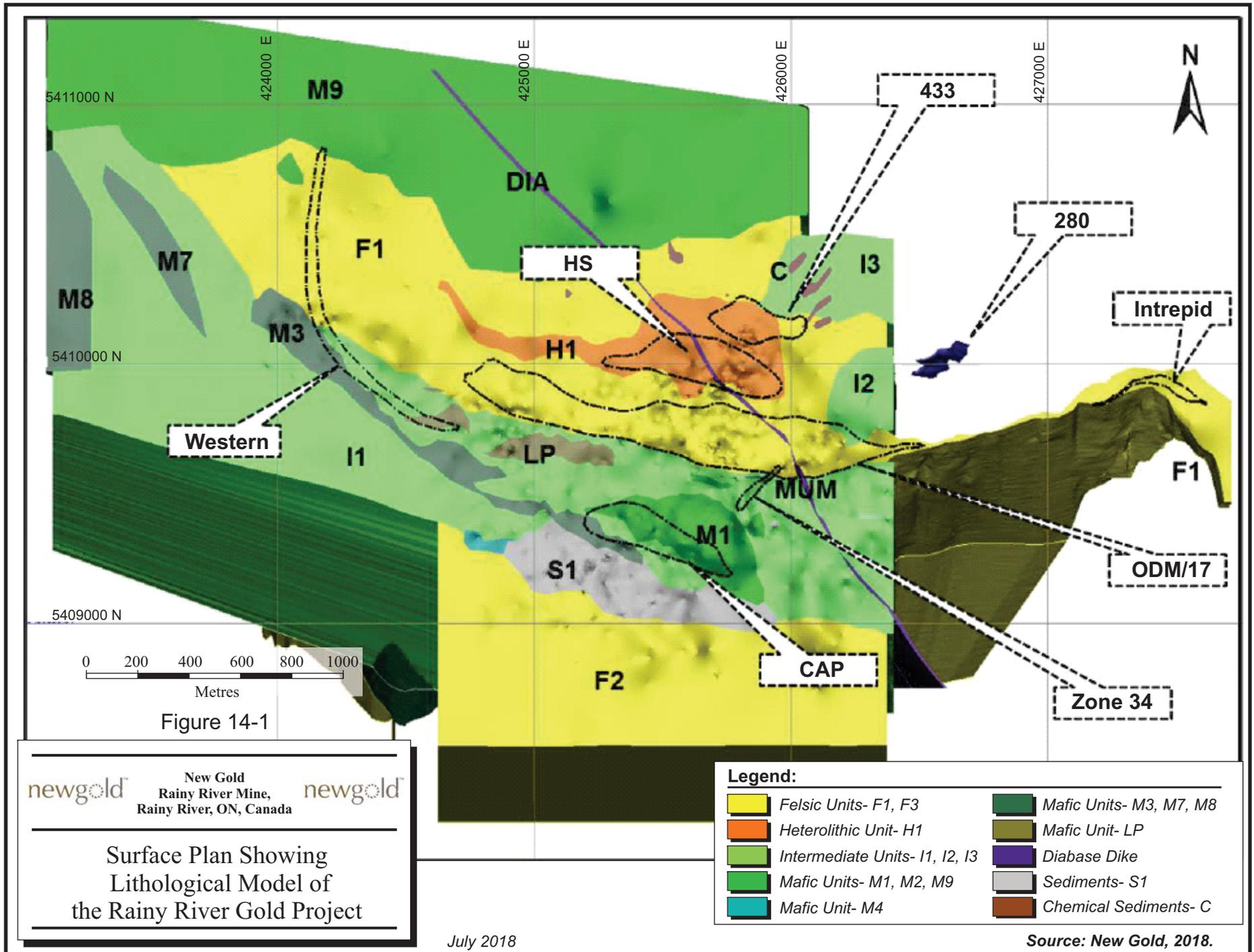


Figure 14-1

newgold New Gold Rainy River Mine, Rainy River, ON, Canada newgold

Surface Plan Showing Lithological Model of the Rainy River Gold Project

Legend:

Felsic Units- F1, F3	Mafic Units- M3, M7, M8
Heterolithic Unit- H1	Mafic Unit- LP
Intermediate Units- I1, I2, I3	Diabase Dike
Mafic Units- M1, M2, M9	Sediments- S1
Mafic Unit- M4	Chemical Sediments- C

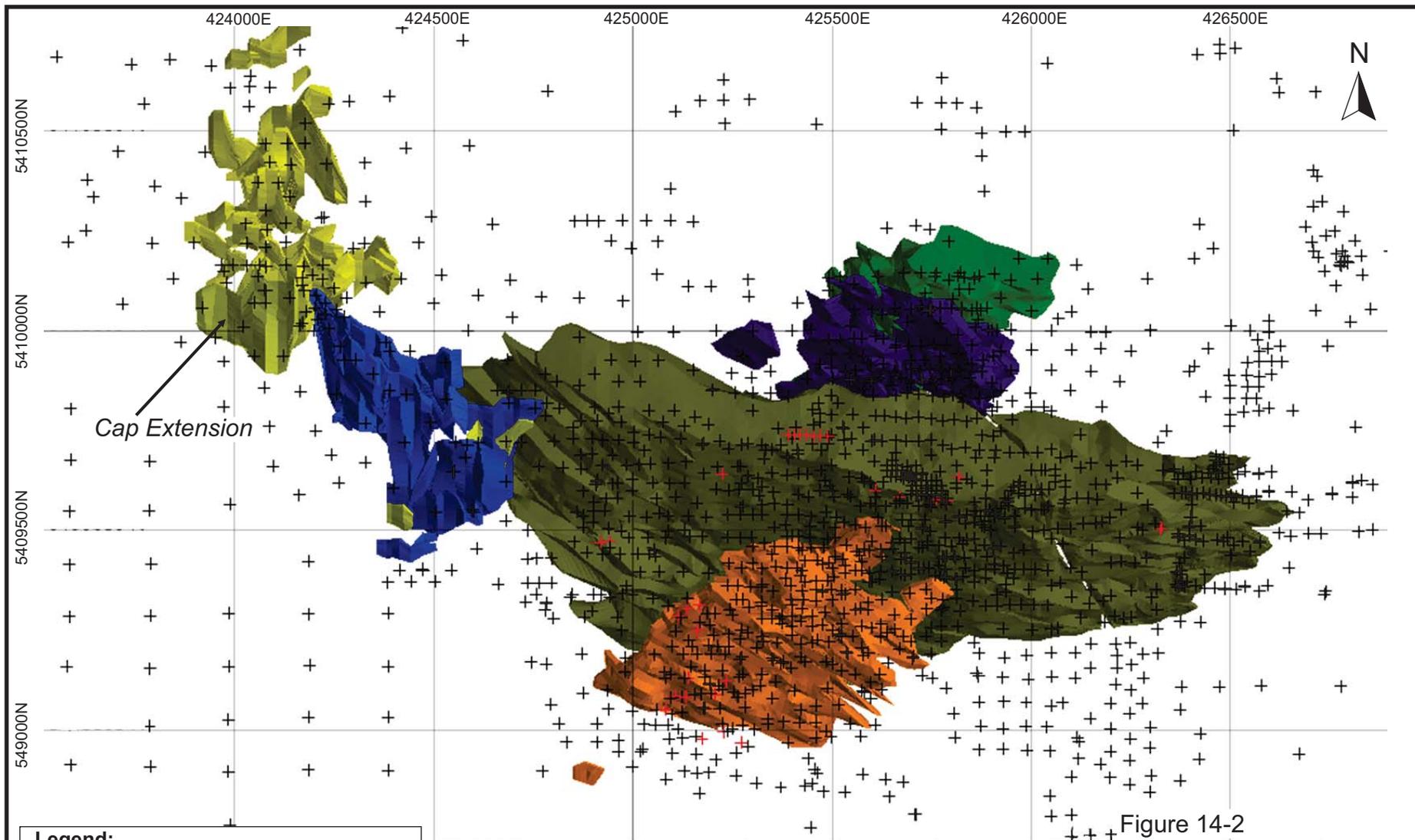


Figure 14-2

Legend:

Domains	Drill Hole Collar
HS	1994-2015
CAP	2016-2017
433	
Western Zone- Tuffs	
Western Zone- Intermediate Volcanics	
ODM/17 (including Silver- not shown)	

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 Rainy River Mine,
 Rainy River, ON, Canada newgold™

Plan View of Main Zone
 Mineralization Domains

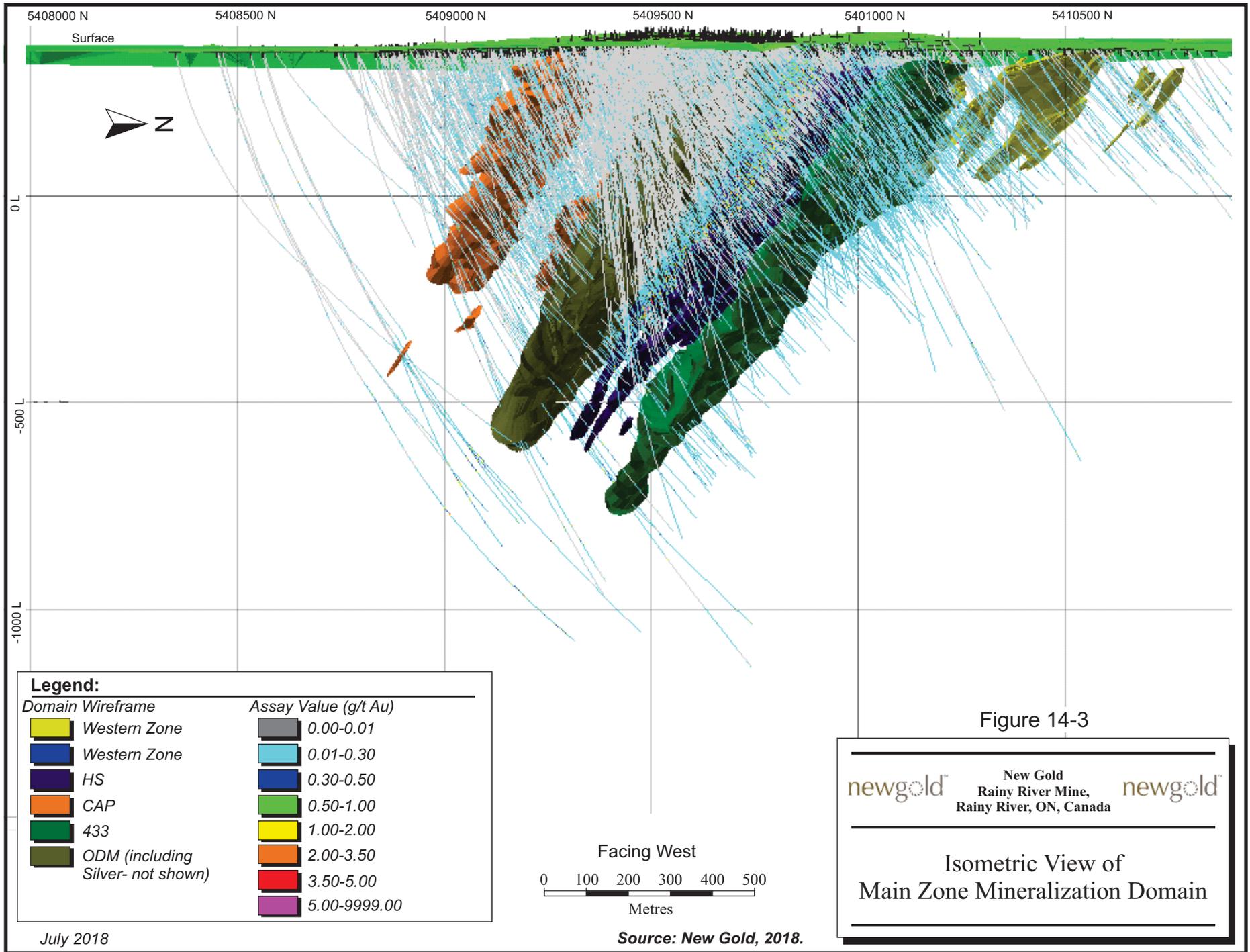


Figure 14-3

ODM/17 ZONE

The ODM/17 Zone is interpreted as a generally east-west trending, southwest plunging zone of mineralization within the Main Zone, cross-cut by numerous north-northeast striking faults. A combination of alteration indices and gold grade shells suggests a stacked pattern of slightly oblique zones that resemble tight folds occurring within the ODM/17 Zone, however, a lack of available outcrop and current density of exploration drilling precludes a more definitive interpretation of controls to gold mineralization within the zone. The overall outline of the ODM/17 Zone was based on the broad extent of a sericite index (K/Al cationic based) larger than 0.7. The outlines were guided by a 3D model of the sericite index and a 0.2 g/t Au grade shell.

The hangingwall of the zone coincides with the top of a felsic fragmental volcanoclastic unit that hosts much of the ODM/17 Zone. This rock package is separated from mafic volcanic and intermediate to felsic volcanic units to the south by a curved but generally east-west trending magnetic lineament. This lineament was modelled and used to define the hangingwall boundary of the ODM/17 Zone. This contact becomes cryptic to the east, but was projected parallel to the magnetic lineament. The ODM/17 domain was modelled on inclined sections oriented perpendicular to the southwesterly plunge of mineralization (azimuth 233 degrees plunge of 47 degrees) and subdivided into three grade subdomains based on the following divisions:

- High Grade: Greater than 0.9 g/t Au;
- Medium Grade: From 0.5 g/t to 0.9 g/t Au;
- Low Grade: From 0.2 g/t to 0.5 g/t Au.

The geometry of the medium and high grade subdomains is modelled parallel to the south dipping footwall of the overall ODM/17 domain or slightly oblique to it, consistent with the geometry of observed high strain zones bounding the subdomains and strain foliation orientation observed within them.

433 AND HS ZONES

The 433 and HS zones form two zones of gold mineralization in the Main Zone. They are located within the footwall of the ODM/17 Zone, and hosted by massive and fragmental felsic

to intermediate volcanics. The boundaries of these zones are not as well defined as for the ODM/17 Zone, but the southwesterly plunge to gold mineralization is similar.

Accordingly, the boundaries for the 433 and HS zones were modelled on inclined sections following the same orientation. The sericite index used to define the outer limits of the ODM/17 domain does not clearly define the 433 Zone. Instead, local disseminated chalcopyrite and sphalerite associated with gold mineralization has been used to define its domain boundaries, based on a copper-to-zinc ratio of 0.8. Similar to the ODM/17 Zone, the 433 Zone was subdivided into three grade subdomains based on the following divisions:

1. High Grade: Greater than 0.9 g/t Au;
2. Medium Grade: From 0.5 g/t to 0.9 g/t Au;
3. Low Grade: From 0.2 g/t to 0.5 g/t Au.

No geochemical or lithological criteria were incorporated into the delineation of the HS Zone. The HS Zone was defined using the interpreted extent of a 0.2 g/t Au threshold (based on 3 m composites) and guided by 0.2 g/t Au Leapfrog grade shells.

CAP ZONE

The CAP Zone occurs in the hangingwall of the ODM/17 Zone within the upper, predominantly mafic, volcanic sequence within the Main Zone. On the surface, the zone is associated with a number of quartz-carbonate vein sets and south dipping shear zones that are superimposed on the pervasive south dipping foliation. The orientation of the quartz-carbonate veins is also highly variable. Northeast to northwest striking sulphide veinlets anastomose across several surface outcrops. In drill core, individual high grade gold intersections are associated with increased sulphide mineralization, particularly chalcopyrite, within and adjacent to shear hosted quartz-carbonate veins.

Low grade gold mineralization in intermediate rocks within the CAP Zone is similar to the ODM/17 Zone, with a noticeably shallower plunge to the south-west. On north-south vertical sections, high grade gold intersections are aligned along south dipping planes. In plan view, high grade gold intersections show continuity along a west-northwest strike. Low grade mineralization shows good continuity when observed in cross-sections oriented perpendicular

to the slightly shallower plunge. The CAP Zone domain was modelled on vertical sections using a 0.2 g/t Au threshold guided by this preferred geometry.

WESTERN ZONE

The Western Zone represents a northwesterly extension of the ODM/17 Zone. Gold mineralization is more sporadic and discontinuous than in the ODM/17 Zone, but can be subdivided into at least two styles of mineralization:

1. Early (low to moderate grade) gold mineralization associated with sulphide (pyrite-sphalerite-chalcopyrite-galena) stringers and veins and disseminated pyrite in quartz-phyric volcanoclastic rocks and conglomerate; and
2. Late (high grade) gold mineralization associated with quartz-carbonate-pyrite-gold veins and veinlets, and rarely as native gold veins.

This hybrid style of mineralization consists of an early gold-rich volcanogenic sulphide mineralization overprinted by shear-hosted mesothermal gold mineralization. Gold mineralization is commonly associated with increased sericite and chlorite alteration. Mineralization also appears to have a strong association with strain. Increased strain, characterized by kink folds, boudinage, and strong kinematic fabric, is commonly associated with increased gold grade. At very high strain, however, mylonitic textures appear and gold grade diminishes to background levels. The Western Zone domain was defined on vertical sections guided by 0.2 g/t Au Leapfrog shells. As presently defined, gold mineralization in the Western Zone appears erratic and discontinuous, offering low potential for the delineation of a near surface gold resource.

34 ZONE

The 34 Zone was modelled by New Gold in 2015 using logged drill hole data and Leapfrog software. The zone represents a late stage mafic-ultramafic dike that cross-cuts the ODM/17 Zone and post-dates gold mineralization. It has been modelled as a distinct zone to constrain estimation of gold resources within the 2017 block model.

SILVER ZONE

The Silver Zone (not shown) occurs in the footwall of the ODM/17 Zone in dacitic tuff and breccias, immediately adjacent to a high strain zone located at the northern contact of the ODM/17 Zone. The Silver Zone plunges to the southwest in similar orientation to the ODM/17

Zone, and is associated with centimetre-scale sulphide bearing quartz veinlets that typically contain dendritic native silver inclusions. The Silver Zone domain was outlined by New Gold using a 19 g/t Ag cut-off grade (3.0 m composites; less than 4.0 m waste), on inclined cross-sections oriented perpendicular to the southwesterly plunge of the silver mineralization.

INTREPID ZONE

The Intrepid Zone was modelled on 17 vertical sections spaced at 25 m intervals which were subsequently linked into a series of 3D wireframes to define the limits of gold and silver mineralization. Three nested grade domains were defined based on the gold and silver content:

1. High Grade: Above 2.0 g/t Au;
2. Medium Grade: From 0.8 g/t to 2.0 g/t Au;
3. Low Grade: From 0.3 g/t to 0.8 g/t Au.

The Intrepid Zone has remained unchanged in this resource estimation, with no new drilling data available. The general shape of the Intrepid Zone is shown in Figure 14-4.

Table 14-4 lists the associated domain codes for the different mineralization zones and grade domains at Rainy River.

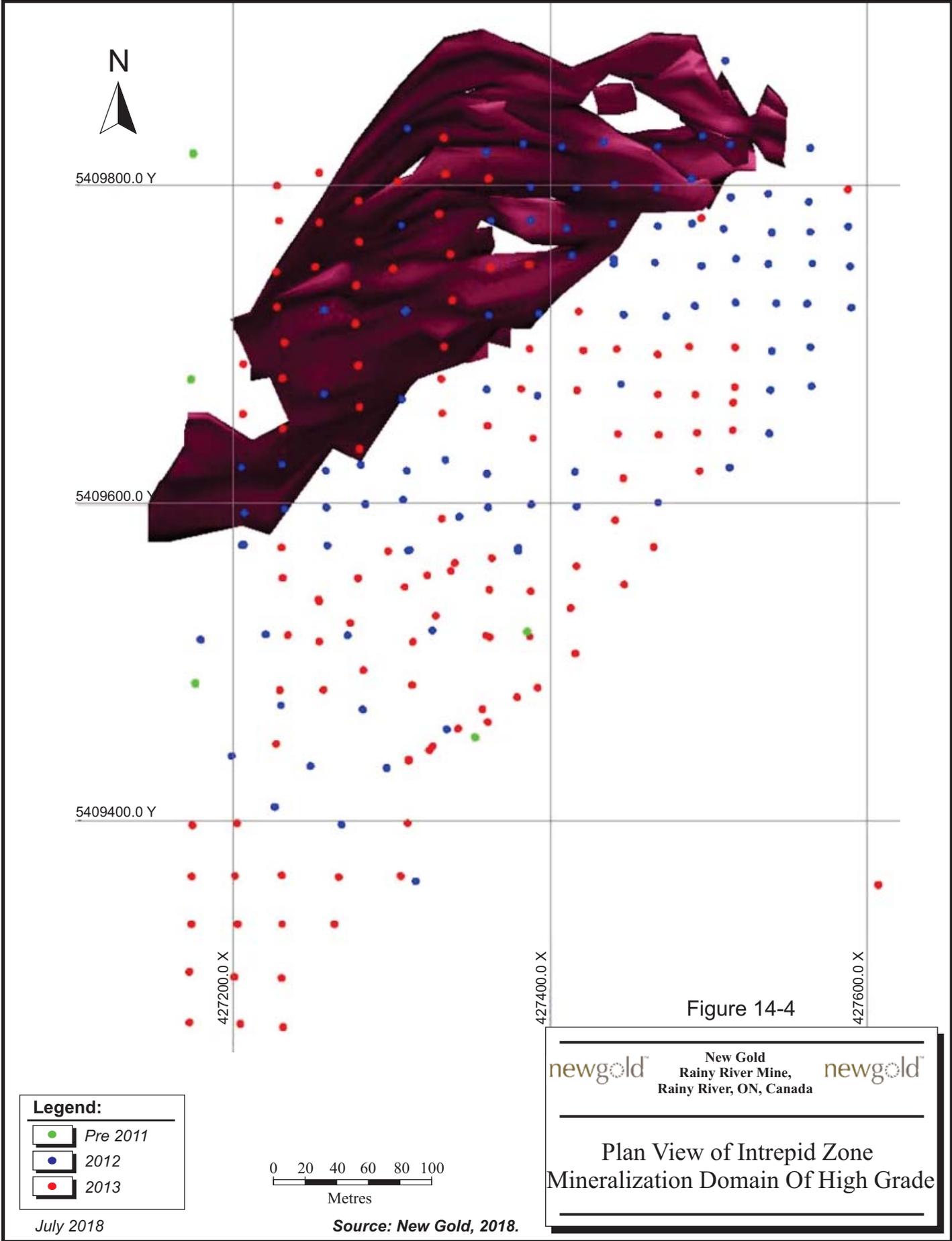


Figure 14-4


 New Gold
 Rainy River Mine,
 Rainy River, ON, Canada
 

Plan View of Intrepid Zone
 Mineralization Domain Of High Grade

- Legend:**
- Pre 2011
 - 2012
 - 2013

0 20 40 60 80 100
Metres

July 2018

Source: New Gold, 2018.

TABLE 14-4 MINERALIZATION AND LITHOLOGY DOMAIN CODES

Zone	Domain	Domain Code
MINERALIZATION	ODM/17	
	Low grade	101
	Medium grade	110 to 116
	High grade	120 to 126
	34 Zone	200
	Zone 280	280
	Zone 433	
	Low grade	300
	Medium grade	310
	High grade	320
	HS	400
	CAP	500
	Intrepid	
	Low grade	700
	Medium grade	710
	High grade	720
	Western	801-803
	Silver	901 to 904
	LITHOLOGY	Felsic Units
Heterolithic Unit		2001
Intermediate Units		3001/3002
Mafic Units		4001 to 4011
Mafic Units - LP		5001
Mafic Intrusion		6001
Diabase Dike		7001
Sediments		8001
Chemical Sediments		9001

EXPLORATORY DATA ANALYSIS

ASSAYS

Gold and silver assays located inside the wireframe models were tagged with domain identifiers and exported for statistical analysis. Results were used to help verify the modelling process. Descriptive statistics by domain are summarized in Tables 14-5 and 14-6 for gold and silver, respectively.

TABLE 14-5 STATISTICAL SUMMARY OF GOLD ASSAY DATA

Grade Domain	Domain Code	Count	Minimum	Maximum	Mean	StdDev	CV	Variance
ODM/17 Zone								
Low	101	76,384	0.00	448.56	0.24	2.20	9.22	4.85
	110	4,319	0.00	104.51	0.67	2.46	3.66	6.07
	111	5,887	0.00	168.50	0.72	3.38	4.69	11.45
	112	4,739	0.00	166.00	0.72	3.01	4.18	9.08
Medium	113	2,943	0.00	125.73	0.73	2.76	3.78	7.64
	114	1,103	0.03	1,080.00	1.97	33.69	17.12	1,135.06
	115	1,028	0.00	84.40	1.25	4.51	3.62	20.36
	116	804	0.00	7.49	0.65	0.82	1.25	0.66
	120	2,693	0.01	746.33	1.96	9.38	4.79	87.97
	121	4,457	0.00	1,221.19	2.35	21.97	9.37	482.85
High	122	4,457	0.01	482.00	2.68	12.99	4.85	168.83
	123	798	0.01	2,559.00	2.54	45.20	17.79	2,043.17
	124	2	0.03	1.54	0.79	0.92	1.17	0.85
	125	104	0.10	31.03	2.05	2.99	1.46	8.92
	126	324	0.01	281.00	7.57	21.36	2.82	456.31
34 Zone								
	200	246	0.00	10.00	0.22	0.68	3.17	0.47
280 Zone								
	280	269	0.01	51.68	0.62	2.97	4.79	8.83
433 Zone								
Low	300	11,456	0.00	1,000.00	0.29	5.65	19.80	31.92
Medium	310	3,069	0.00	121.20	0.88	4.01	4.57	16.10
High	320	1,113	0.01	4,158.63	5.68	108.86	19.17	11,850.68
HS Zone								
	400	10,114	0.00	707.80	0.58	7.50	12.95	56.31
CAP Zone								
	500	12,102	0.00	192.72	0.43	1.52	3.57	2.31
Intrepid Zone								
Low	700	2,691	0.00	37.60	0.40	0.97	2.45	0.95
Medium	710	1,385	0.01	25.80	1.11	1.90	1.72	3.62
High	720	1,053	0.02	528.00	4.28	18.01	4.21	324.19

Domain Code	Count	Minimum	Maximum	Mean	StdDev	CV	Variance
Western Zone							
801	125	0.01	13.40	0.35	0.78	2.24	0.60
802	713	0.01	14.90	0.47	0.84	1.81	0.71
803	1,256	0.00	1335.00	1.83	38.44	21.04	1477.48
Silver Zone							
901	227	0.00	9.56	0.28	0.91	3.18	0.82
902	86	0.11	1,088.45	18.13	122.04	6.73	14,894.45
903	215	0.00	28.87	0.98	2.42	2.46	5.84
904	537	0.00	18.44	0.45	0.92	2.03	0.84
Lithological Domains							
1001	69,371	0.00	255.00	0.09	1.21	13.57	1.46
1002	10,461	0.00	74.10	0.04	0.88	24.91	0.77
2001	38,743	0.00	188.00	0.13	1.35	10.60	1.82
3001	66,413	0.00	48.79	0.04	0.27	6.04	0.07
3002	8,356	0.00	15.91	0.04	0.18	4.76	0.03
4001	13,362	0.00	79.60	0.08	0.74	9.14	0.54
4002	3,101	0.00	7.83	0.06	0.19	3.11	0.04
4003	6,460	0.00	7.39	0.10	0.19	1.93	0.04
4004	755	0.00	1.02	0.02	0.06	3.33	0.00
4007	269	0.00	0.12	0.00	0.01	1.68	0.00
4009	11,884	0.00	32.80	0.07	0.37	5.39	0.14
4011	2,168	0.00	2.42	0.05	0.10	2.06	0.01
5001	8,052	0.00	8.53	0.07	0.20	3.09	0.04
6001	352	0.00	0.51	0.03	0.06	1.88	0.00
7001	1,562	0.00	8.07	0.09	0.28	3.32	0.08
8001	13,987	0.00	3.78	0.02	0.09	3.54	0.01
9001	4,391	0.00	8.56	0.10	0.28	2.85	0.08

TABLE 14-6 STATISTICAL SUMMARY OF SILVER ASSAY DATA

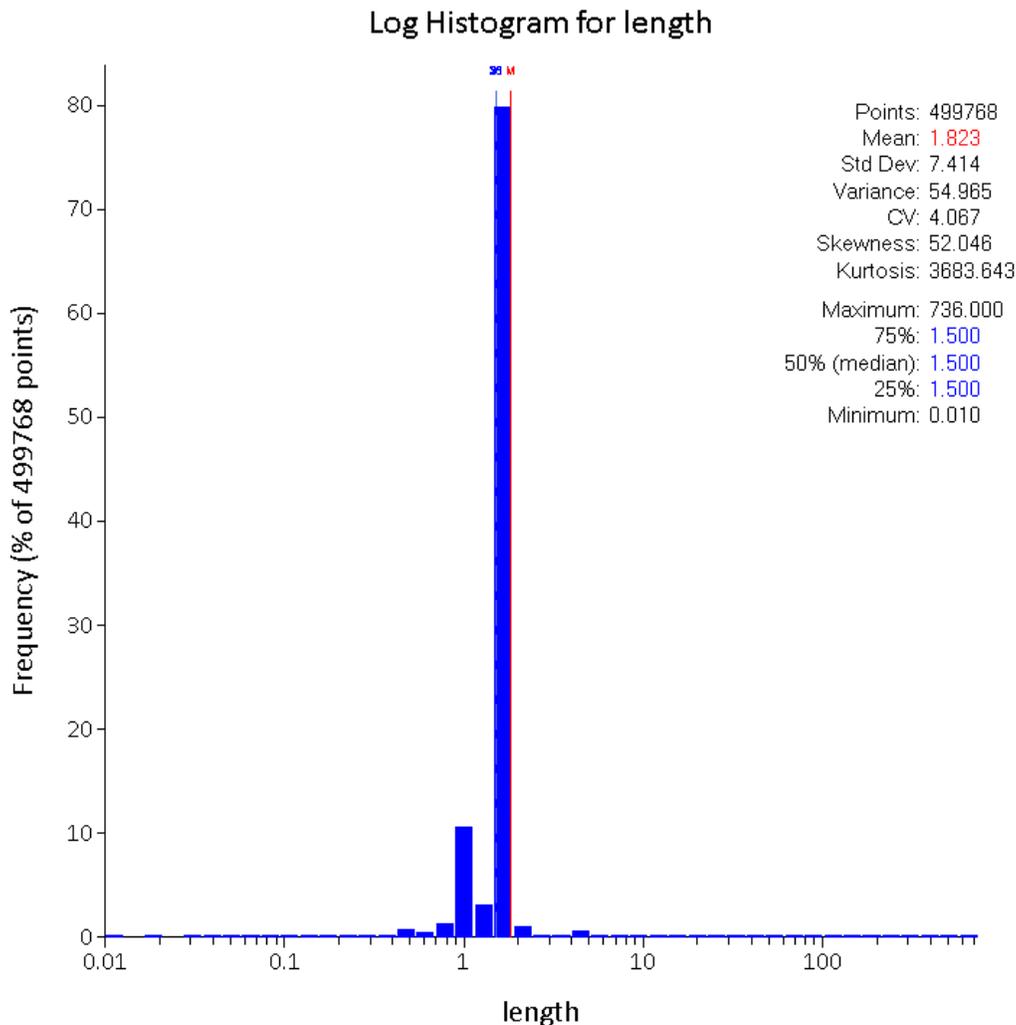
Domain Code	Count	Minimum	Maximum	Mean	StdDev	CV	Variance	
ODM/17 Zone								
Low	101	75,786	0.01	2,020.00	2.03	9.03	4.45	81.54
	110	4,318	0.03	430.00	2.94	8.33	2.84	69.32
	111	5,887	0.09	181.00	1.48	3.68	2.48	13.51
	112	4,739	0.09	65.00	1.44	2.53	1.76	6.38
Medium	113	2,905	0.08	135.00	2.75	6.28	2.29	39.45
	114	1,103	0.22	256.00	4.84	10.64	2.20	113.19
	115	1,007	0.13	1,760.00	13.85	67.40	4.87	4,542.20
	116	776	0.50	773.00	14.38	32.92	2.29	1,083.99
	120	2,687	0.10	332.00	4.80	12.45	2.60	155.05
	121	4,457	0.10	230.00	2.26	4.91	2.17	24.13
	122	4,457	0.09	190.00	2.39	5.11	2.14	26.11
High	123	788	0.10	655.00	3.21	12.68	3.95	160.80
	124	2	0.80	20.80	10.80	12.25	1.13	150.00
	125	104	0.50	100.00	9.77	16.10	1.65	259.08
	126	316	0.60	2,580.00	77.20	202.79	2.63	41,122.55
34 Zone								
	200	230	0.10	59.00	2.45	5.60	2.29	31.39
280 Zone								
	280	269	0.10	11.40	0.95	1.34	1.41	1.80
433 Zone								
Low	300	11,419	0.01	294.00	0.78	2.23	2.87	4.98
Medium	310	3,069	0.01	100.00	0.99	3.20	3.22	10.21
High	320	1,113	0.10	439.00	1.62	12.03	7.45	144.74
HS Zone								
	400	10,114	0.03	1,000.00	1.27	8.83	6.97	77.97
CAP Zone								
	500	12,101	0.04	1,288.98	2.29	8.91	3.89	79.31
Intrepid Zone								
Low	700	2,691	0.10	139.00	5.40	8.54	1.58	72.85
Medium	710	1,385	0.10	207.00	12.36	17.35	1.40	301.14
High	720	1,053	0.30	464.00	26.61	42.93	1.61	1,843.25

Domain Code	Count	Minimum	Maximum	Mean	StdDev	CV	Variance
Western Zone							
801	125	0.10	14.20	0.70	0.93	1.33	0.87
802	713	0.06	48.40	1.03	2.80	2.71	7.84
803	1,255	0.03	166.00	2.39	9.80	4.10	96.13
Silver Zone							
901	227	0.45	1,050.00	58.31	95.38	1.64	9,096.66
902	86	0.50	312.00	24.57	43.73	1.78	1,911.85
903	214	0.40	384.00	18.44	31.60	1.71	998.70
904	534	0.50	437.00	18.75	31.31	1.67	980.04
Lithological Domains							
1001	68,960	0.01	920.00	0.88	4.15	4.73	17.26
1002	10,462	0.01	33.00	0.34	0.94	2.76	0.87
2001	38,699	0.01	182.00	0.70	1.71	2.43	2.93
3001	64,343	0.01	875.00	0.51	4.08	8.08	16.61
3002	7,397	0.01	18.00	0.50	0.77	1.54	0.58
4001	13,361	0.01	1,398.00	0.81	12.51	15.51	156.44
4002	3,100	0.01	150.00	0.83	3.33	4.01	11.07
4003	6,460	0.01	48.50	0.64	1.09	1.69	1.18
4004	755	0.03	30.00	0.40	1.16	2.92	1.33
4007	269	0.01	12.30	0.16	0.54	3.40	0.29
4009	11,736	0.02	45.70	0.69	1.47	2.13	2.16
4011	2,159	0.03	25.10	0.80	0.99	1.23	0.98
5001	8,042	0.01	21.00	0.58	0.98	1.68	0.95
6001	329	0.10	6.20	1.04	1.14	1.10	1.31
7001	1,543	0.03	274.00	1.45	8.93	6.18	79.68
8001	13,932	0.01	86.10	0.59	1.42	2.41	2.01
9001	4,391	0.01	39.50	0.86	1.48	1.71	2.18

DRILL SAMPLE COMPOSITES

Prior to grade interpolation, the assay data was composited to 1.5 m intervals, broken at domain boundaries. The composite length was chosen based on the analysis of the predominant sampling length, style of mineralization, and continuity of grade. A histogram of raw sample lengths is shown in Figure 14-5.

FIGURE 14-5 HISTOGRAM OF SAMPLE LENGTHS AT RAINY RIVER



GRADE CAPPING

Extreme high grade values can lead to overestimation of grade in a block model. Capping of composite gold grades was performed to limit the influence of high grade outlier values. Grade capping thresholds were determined for gold and silver separately within each mineralization

domain and any subdomains therein. Capping thresholds for gold and silver are summarized in Table 14-7, and cumulative gold distribution plots by zone domain are shown in Figure 14-6. No capping was applied to calcium or sulphur.

TABLE 14-7 SUMMARY OF GOLD AND SILVER CAPPING THRESHOLDS

Zone	Domain	Cap (g/t)	Gold Percentile	No. Capped	Cap (g/t)	Silver Percentile	No. Capped
ODM/17	101	40.00	99.98%	12	250.00	99.99%	7
	110	25.00	99.82%	7	70.00	99.87%	5
	111	40.00	99.92%	4	28.00	99.86%	7
	112	30.00	99.86%	6	30.00	99.93%	3
	113	30.00	99.92%	2	50.00	99.70%	7
	114	70.00	99.81%	2	40.00	99.43%	6
	115	50.00	99.68%	3	250.00	99.24%	7
	116	5.00	99.44%	4	100.00	99.13%	6
	120	90.00	99.87%	3	85.00	99.73%	6
	121	95.00	99.89%	4	60.00	99.92%	3
	122	120.00	99.82%	7	60.00	99.87%	5
	123	30.00	99.52%	3	30.00	99.36%	4
	124	NC	100.00%	0	NC	100.00%	0
	125	7.00	98.04%	2	45.00	98.04%	2
	126	80.00	98.84%	3	600.00	97.64%	6
34	200	3.00	99.03%	2	35.00	99.49%	1
280	280	9.00	98.87%	3	6.00	98.12%	5
433	300	25.00	99.97%	3	30.00	99.93%	8
	310	30.00	99.63%	11	30.00	99.80%	6
	320	120.00	99.62%	4	30.00	99.52%	5
HS	400	65.00	99.97%	3	100.00	99.98%	2
CAP	500	15.00	99.94%	7	100.00	99.96%	4
INTREPID	700	7	99.85%	4	90	99.89%	3
	710	15	99.44%	8	150	99.86%	2
	720	80	99.71%	3	250	99.43%	6
WESTERN	801	2.00	99.20%	1	2.50	98.40%	2
	802	3.00	99.07%	7	8.00	99.07%	7
	803	30.00	99.84%	2	90.00	99.75%	3
SILVER	901	5.00	98.40%	3	280.00	97.33%	5
	902	7.00	94.19%	5	80.00	91.86%	7
	903	4.50	96.83%	6	100.00	97.88%	4
	904	3.00	98.63%	6	115.00	98.39%	7
Felsic Volcanics	1001	25.00	99.99%	8	100.00	99.99%	8
	1002	12.00	99.97%	3	17.00	99.96%	4
Heterolithic	2001	25.00	99.98%	8	100.00	100.00%	1

Zone	Domain	Cap (g/t)	Gold Percentile	No. Capped	Cap (g/t)	Silver Percentile	No. Capped
Intermediate	3001	6.00	99.99%	5	60.00	99.99%	6
Volcanics	3002	2.00	99.95%	4	15.00	99.96%	3
Mafic Units	4001	4.00	99.94%	8	30.00	99.95%	7
	4002	2.00	99.90%	3	25.00	99.83%	5
	4003	4.00	99.98%	1	15.00	99.95%	3
	4004	1.02	100.00%		4.00	99.48%	4
	4007	0.05	100.00%		3.52	100.00%	0
	4009	5.00	99.97%	4	30.00	99.93%	8
	4011	1.00	99.86%	3	5.00	99.37%	13
Mafic Units - LP	5001	4.00	99.96%	3	12.00	99.94%	5
Mafic Intrusion	6001	0.30	99.43%	2	4.00	96.86%	10
Diabase Dike	7001	12.00	100.00%		157.38	100.00%	0
Sediments	8001	2.00	99.96%	5	30.00	99.98%	3
Chem. Sediments	9001	5.00	99.91%	4	15.00	99.91%	4

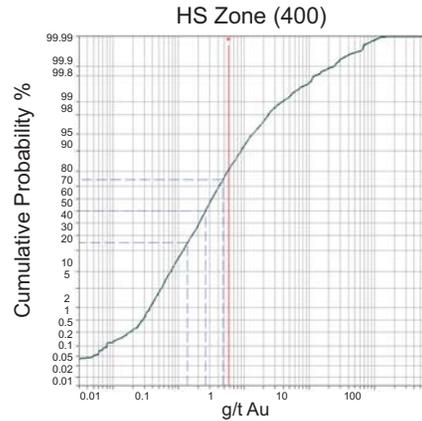
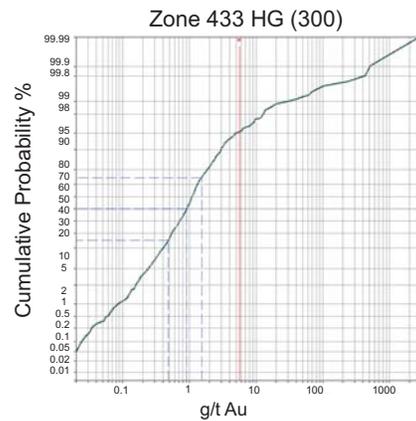
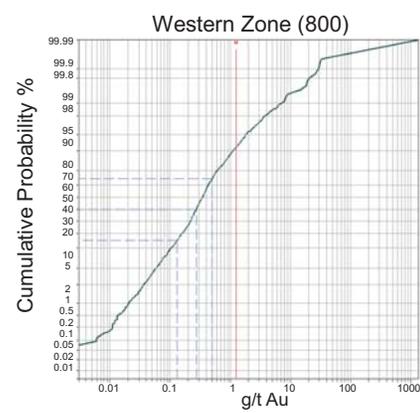
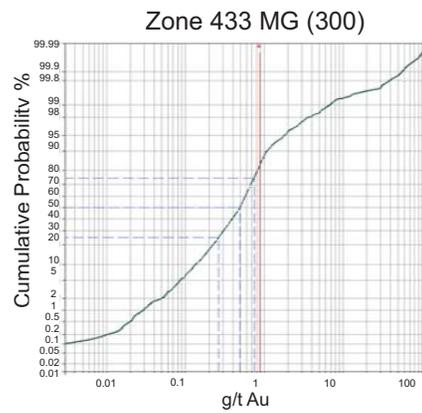
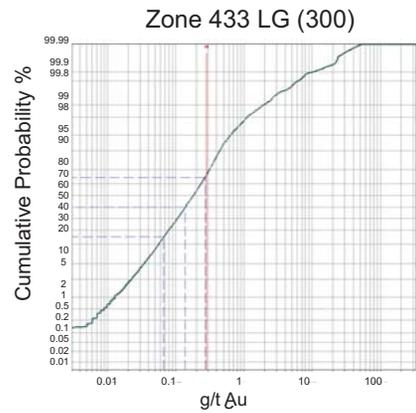
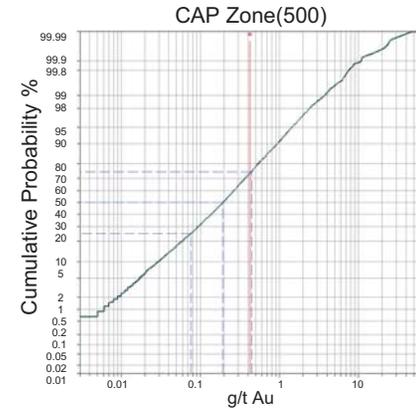
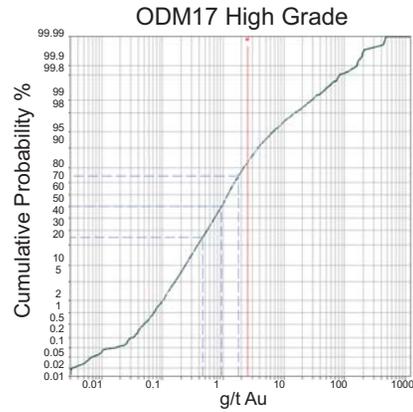
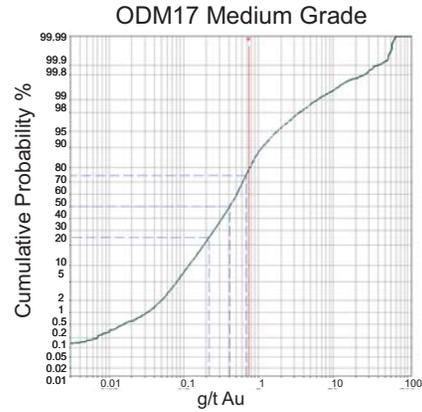


Figure 14-6

newgold™ **New Gold**
Rainy River Mine, **newgold™**
Rainy River, ON, Canada

**Cumulative Frequency Plots
of Gold Distribution by Domain**

Basic statistics for the composite and capped composite data for gold and silver within all Mineral Resource domains are summarized in Table 14-8 and 14-9.

TABLE 14-8 STATISTICAL SUMMARY OF GOLD COMPOSITES

Zone	Domain	Code	Count	Min (Au g/t)	Max (Au g/t)	Mean (Au g/t)	Cut Mean (Au g/t)	CV	Cut CV
ODM17	Low	101	70,268	0.00	448.56	0.24	0.23	8.86	3.56
		110	3,841	0.00	55.60	0.68	0.65	3.05	2.22
		111	4,903	0.00	112.40	0.72	0.70	3.73	2.81
		112	4,354	0.00	66.99	0.72	0.70	3.24	2.52
	Medium	113	2,377	0.00	61.00	0.73	0.70	3.10	2.33
		114	1,045	0.03	1,080.00	1.97	0.99	17.12	4.30
		115	934	0.00	60.73	1.25	1.23	3.32	3.17
		116	709	0.01	6.86	0.65	0.64	1.13	1.09
	High	120	2,245	0.01	195.29	1.96	1.84	3.85	2.78
		121	3,596	0.00	1,221.19	2.35	1.99	9.13	2.69
		122	3,967	0.01	482.00	2.68	2.51	4.50	3.24
		123	625	0.01	462.05	2.53	1.73	7.65	1.80
		124	2	0.03	1.54	0.79	0.79	1.17	1.17
		125	102	0.10	24.31	2.05	1.86	1.36	0.88
		126	259	0.01	164.37	7.52	6.99	2.35	2.02
Zone 34		200	207	0.00	6.13	0.22	0.20	2.65	2.07
280		280	266	0.01	24.13	0.62	0.53	3.25	2.25
433	Low	300	11,059	0.00	333.97	0.29	0.25	11.61	3.09
	Medium	310	2,938	0.00	121.20	0.88	0.79	4.35	2.68
	High	320	1,041	0.02	2,772.67	5.67	2.31	15.81	3.97
HS		400	9,654	0.00	707.80	0.58	0.51	12.93	3.51
CAP		500	11,367	0.00	64.77	0.43	0.42	2.71	1.97
Intrepid	Low	700	2,680	0.00	37.60	0.40	0.39	2.41	1.54
	Medium	710	1,377	0.01	25.80	1.11	1.11	1.67	1.52
	High	720	1,026	0.02	528.00	4.28	3.93	4.16	1.83
Western		801	125	0.02	5.78	0.35	0.32	1.68	1.08
		802	751	0.01	14.90	0.47	0.43	1.81	1.14
		803	1,222	0.00	1,335.00	1.83	0.75	21.04	3.04
Silver		901	187	0.00	6.51	0.28	0.27	2.87	2.67
		902	86	0.11	1,088.45	18.13	1.69	6.73	1.14
		903	189	0.01	28.87	0.98	0.81	2.42	1.24
		904	439	0.00	7.82	0.46	0.43	1.57	1.30
Lithological Domains		1001	68,301	0.00	255.00	0.09	0.08	12.85	4.98
		1002	10,329	0.00	74.10	0.04	0.03	23.16	10.57
		2001	37,925	0.00	188.00	0.13	0.12	9.95	4.34
		3001	68,444	0.00	42.35	0.04	0.04	5.52	2.92

Zone	Domain	Code	Count	Min (Au g/t)	Max (Au g/t)	Mean (Au g/t)	Cut Mean (Au g/t)	CV	Cut CV
		3002	8,663	0.00	9.58	0.04	0.04	3.89	2.67
		4001	13,124	0.00	58.27	0.08	0.07	7.76	2.78
		4002	3,020	0.00	5.22	0.06	0.06	2.53	2.02
		4003	6,372	0.00	7.39	0.10	0.10	1.81	1.63
		4004	765	0.00	1.02	0.02	0.02	3.17	3.17
		4007	265	0.00	0.05	0.00	0.00	1.16	1.16
		4009	11,700	0.00	21.93	0.07	0.07	4.33	2.91
		4011	2,084	0.00	1.26	0.05	0.05	1.81	1.76
		5001	8,031	0.00	4.91	0.07	0.07	2.85	2.80
		6001	350	0.00	0.35	0.03	0.03	1.68	1.65
		7001	1,679	0.00	12.00	0.08	0.08	3.46	3.46
		8001	13,764	0.00	3.42	0.02	0.02	3.26	2.98
		9001	4,344	0.00	8.56	0.10	0.10	2.73	2.41

TABLE 14-9 STATISTICAL SUMMARY OF SILVER COMPOSITES

Zone	Domain	Code	Count	Min (Ag g/t)	Max (Ag g/t)	Mean (Ag g/t)	Cut Mean (Au g/t)	CV	Cut CV
ODM17	Low	101	69,817	0.01	1,039.20	2.03	2.00	3.83	2.66
		110	3,840	0.03	235.05	2.94	2.83	2.48	1.75
		111	4,903	0.09	121.20	1.48	1.42	2.24	1.35
		112	4,354	0.09	46.03	1.44	1.43	1.66	1.59
	Medium	113	2,361	0.08	75.13	2.74	2.70	1.95	1.83
		114	1,045	0.22	256.00	4.84	4.47	2.17	1.27
		115	920	0.14	1,205.37	13.85	11.55	4.20	2.60
		116	692	0.50	519.27	14.33	13.25	1.93	1.21
		120	2,245	0.10	292.00	4.79	4.63	2.30	1.91
		121	3,596	0.10	106.21	2.27	2.24	1.80	1.59
	High	122	3,967	0.09	100.00	2.39	2.36	1.93	1.74
		124	624	0.10	121.30	3.30	3.05	2.15	1.45
		123	2	0.80	20.80	10.80	10.80	1.13	1.13
		125	102	0.50	99.67	9.77	8.98	1.58	1.35
126		254	0.60	1,632.10	76.73	68.64	2.23	1.81	
Zone 34		200	198	0.10	46.86	2.54	2.48	2.21	2.08
280		280	266	0.10	11.40	0.95	0.91	1.35	1.17
433	Low	300	11,022	0.01	98.42	0.78	0.76	2.26	1.67
	Medium	310	2,938	0.01	100.00	0.99	0.95	2.91	1.95
	High	320	1,041	0.10	292.83	1.62	1.27	6.12	2.12
HS		400	9,654	0.03	667.55	1.27	1.21	5.83	2.43
CAP		500	11,367	0.05	440.70	2.29	2.24	2.82	1.96

Zone	Domain	Code	Count	Min (Ag g/t)	Max (Ag g/t)	Mean (Ag g/t)	Cut Mean (Au g/t)	CV	Cut CV
Intrepid	Low	700	2,680	0.10	139.00	5.40	4.48	1.55	1.79
	Medium	710	1,377	0.10	207.00	12.37	12.68	1.36	1.33
	High	720	1,026	0.30	464.00	26.61	26.22	1.58	1.38
Western		801	125	0.10	6.47	0.70	0.66	1.07	0.82
		802	751	0.07	48.40	1.03	0.86	2.68	1.37
		803	1,222	0.03	166.00	2.39	2.25	3.91	3.30
Silver Zones		901	187	0.50	759.67	58.14	54.89	1.45	1.20
		902	86	0.50	312.00	24.57	19.79	1.76	1.21
		903	189	0.50	204.85	18.47	17.15	1.53	1.25
		904	436	0.50	326.00	19.12	17.87	1.54	1.21
Lithological Domains		1001	67,829	0.01	437.27	0.88	0.87	3.84	2.80
		1002	10,329	0.01	23.00	0.34	0.34	2.43	2.32
		2001	37,886	0.01	137.00	0.70	0.70	2.26	2.15
		3001	63,888	0.01	875.00	0.51	0.49	7.91	2.12
		3002	7,339	0.01	17.52	0.50	0.49	1.49	1.47
		4001	13,123	0.01	1,398.00	0.81	0.69	15.38	1.80
		4002	3,020	0.01	100.10	0.83	0.79	3.00	1.77
		4003	6,372	0.01	32.77	0.64	0.64	1.58	1.45
		4004	765	0.03	30.00	0.40	0.36	2.91	1.12
		4007	265	0.01	3.52	0.16	0.16	1.91	1.91
		4009	11,554	0.02	45.70	0.69	0.69	2.03	1.94
		4011	2,073	0.03	15.17	0.80	0.78	1.14	0.95
		5001	8,017	0.01	20.00	0.59	0.59	1.62	1.54
		6001	318	0.10	5.00	1.01	0.99	1.08	1.03
	7001	1,667	0.03	157.38	1.38	1.38	5.13	5.13	
	8001	13,682	0.01	75.83	0.59	0.59	2.24	1.93	
	9001	4,344	0.01	39.50	0.86	0.85	1.69	1.41	

SPECIFIC GRAVITY

The specific gravity database contains 10,591 measurements completed by Accurassay via pycnometry on representative split drill core samples selected for each lithologic and mineralized domain. Table 14-10 summarizes the statistics of specific gravity data for each domain.

TABLE 14-10 STATISTICAL SUMMARY OF SPECIFIC GRAVITY

Zone	Domain	Code	Count	Min	Max	Mean	Std Dev	CV
ODM17	Low	101	3,093	2.46	3.72	2.80	0.14	0.05
		110	440	2.47	3.73	2.85	0.18	0.06
		111	863	2.29	3.88	2.81	0.14	0.05
		112	537	2.50	3.93	2.85	0.23	0.08
	Medium	113	86	2.55	3.39	2.84	0.17	0.06
		114	57	2.66	3.13	2.87	0.08	0.03
		115	84	2.49	2.99	2.79	0.10	0.04
		116	-	-	-	-	-	-
	High	120	267	2.52	3.25	2.85	0.11	0.04
		121	919	2.50	3.87	2.82	0.13	0.05
		122	613	2.50	3.94	2.81	0.18	0.07
		124	54	2.52	3.53	2.86	0.16	0.06
		123	-	-	-	-	-	-
125		19	2.74	3.03	2.89	0.09	0.03	
126	-	-	-	-	-	-	-	
Zone 34		200	7	2.81	2.96	2.88	0.06	0.02
280		280	3	2.77	2.92	2.84	0.07	0.02
433	Low	300	597	2.50	3.90	2.85	0.20	0.07
	Medium	310	366	2.51	3.85	2.84	0.20	0.07
	High	320	144	2.51	3.82	2.86	0.26	0.09
HS		400	265	2.51	3.29	2.81	0.13	0.05
	CAP	500	885	2.51	3.95	2.94	0.21	0.07
Intrepid	Low	700	134	2.63	3.17	2.85	0.09	0.03
	Medium	710	95	2.62	3.01	2.83	0.07	0.03
	High	720	105	2.62	3.03	2.81	0.08	0.03
Western		801	-	-	-	-	-	-
		802	7	2.98	3.19	3.08	0.09	0.03
		803	42	2.74	3.06	2.84	0.08	0.03
Silver Zones		901	54	2.65	2.97	2.84	0.07	0.02
		902	11	2.76	3.01	2.85	0.08	0.03
		903	41	2.64	3.35	2.90	0.16	0.06
		904	7	2.64	2.88	2.70	0.08	0.03
Lithological Domains		1001	157	2.60	3.42	2.78	0.12	0.04
		1002	-	-	-	-	-	-
		2001	138	2.48	3.46	2.81	0.15	0.05
		3001	268	2.50	3.14	2.75	0.12	0.04
		3002	100	2.42	2.98	2.76	0.09	0.03
		4001	66	2.56	3.59	2.92	0.17	0.06
		4002	1	2.77	2.77	2.77	0.00	0.00
		4003	27	2.59	3.16	2.89	0.14	0.05
		4004	-	-	-	-	-	-
		4007	-	-	-	-	-	-

Zone	Domain	Code	Count	Min	Max	Mean	Std Dev	CV
		4009	-	-	-	-	-	-
		4011	13	2.78	3.10	2.94	0.11	0.04
		5001	14	2.59	3.04	2.80	0.09	0.03
		6001	-	-	-	-	-	-
		7001	1	2.55	2.55	2.55	0.00	0.00
		8001	11	2.70	3.10	2.89	0.13	0.05
		9001	-	-	-	-	-	-

BLOCK MODEL PARAMETERS

Two block models, representing the open pit and underground volumes within the Main Zone, were created using Vulcan software over the Main Zone. The block models are unrotated with respect to true north and horizontal reference plane, and sub-blocking along domain boundaries was applied to assure accurate estimation of volumes for individual domains. Table 14-11 lists the block model definition parameters. The Intrepid Zone block model prepared by SRK in 2015, unchanged since the previous resource estimation update, was created using GEMS software.

TABLE 14-11 BLOCK MODEL PARAMETERS

Model	Direction	Size (m)	Sub-block (m)	Minimum	Maximum
Open Pit	West	10	2	423,700	427,075
	North	10	2	5,408,750	5,411,125
	Vertical	10	2	-1,200	450
Underground	West	5	1	423,700	427,075
	North	5	1	5,408,750	5,411,125
	Vertical	5	1	-1,200	450
Intrepid	West	5		427,075	427,675
	North	5		5,409,500	5,409,950
	Vertical	5		-180	420

The sub-blocked model for open pit resources was regularized to a 10 m x 10 m x 10 m block model to support estimation of open pit Mineral Reserves.

As part of its review of the methodologies and data used to prepare the Mineral Resource estimates for the Rainy River Mine, AMC imported all block models into Datamine software and integrated them into a single unified block model. The integrated block model has been used as the basis for the Mineral Resource estimate reported herein (Table 14-12). Prior to

the integration of the different block models, the prototype parameters were extended to the east by 600 m in order to combine the Intrepid block model with the integrated Main Zone block model. The parent block size is 10 m x 10 m x 10 m. Additional attributes, including mineral resource and reserve pit shells, underground stopes, and infrastructure were assigned to the integrated block model.

TABLE 14-12 INTEGRATED BLOCK MODEL PARAMETERS

Model	Direction	Size (m)	Sub-block (m)	Minimum	Maximum
Integrated	West	10	0.5	423,700	427,680
	North	10	0.5	5,408,750	5,411,130
	Vertical	10	0.5	-1,200	450

VARIOGRAPHY

Variogram model parameters are unchanged from the previous resource estimate (SRK, 2015), and are listed in Tables 14-13 and 14-14 for gold and silver, respectively. Example gold variogram models for the four principal mineralization domains comprising the Main Zone are shown in Figure 14-7. Variogram models were also completed by lithology domain (irrespective of mineralization domain) for calcium and sulphur (not shown).

TABLE 14-13 MAIN ZONE GOLD VARIOGRAM MODELS

Domain	Nugget	Sill	Type	X1	X2	X3	Sill	Type	X2	Y2	Z2	Sill	Type	X3	Y3	Z3
101	0.2	0.20	Exp	10	15	10	0.30	Exp	80	60	70	0.30	Sph	500	500	70
110	0.2	0.70	Exp	42	50	8	0.10	Sph	150	90	50					
111	0.2	0.60	Exp	10	15	5	0.20	Sph	140	80	25					
112	0.3	0.60	Exp	15	15	7	0.10	Sph	100	90	50					
113	0.2	0.50	Exp	25	0	10	0.10	Sph	25	90	40	0.20	Sph	110	90	40
114	0.25	0.75	Exp	70	70	8										
115	0.2	0.65	Exp	40	40	25	0.15	Sph	130	130	40					
116	0.3	0.40	Exp	15	25	5	0.30	Sph	130	130	40					
120	0.2	0.60	Exp	15	15	5	0.20	Sph	70	70	25					
121	0.2	0.65	Exp	15	15	6	0.05	Sph	15	60	13	0.10	Sph	140	60	19
122	0.2	0.50	Exp	15	15	4	0.15	Sph	50	70	30	0.15	Sph	160	70	30
123	0.2	0.60	Exp	15	15	5	0.20	Sph	70	70	25					
125	0.2	0.65	Exp	40	40	15	0.15	Sph	130	130	40					
126	0.3	0.40	Exp	15	25	5	0.30	Sph	130	45	12					
200	0.15	0.25	Exp	10	10	10	0.60	Exp	75	55	35					
300	0.1	0.40	Sph	10	30	8	0.25	Exp	100	45	25					

Domain	Nugget	Sill	Type	X1	X2	X3	Sill	Type	X2	Y2	Z2	Sill	Type	X3	Y3	Z3
310	0.2	0.60	Sph	15	35	6	0.20	Exp	200	60	20					
320	0.2	0.45	Sph	10	10	4	0.35	Exp	60	30	8					
280	0.2	0.80	Exp	20	20	20										
400	0.2	0.80	Exp	40	55	5										
500	0.2	0.55	Sph	15	15	5	0.25	Exp	110	50	5					
700E	0.2	0.80	Exp	110	70	3										
700W	0.2	0.55	Exp	50	20	3	0.25	Sph	60	50	3					
710E	0.3	0.40	Exp	30	40	3	0.30	Sph	40	80	3					
710W	0.3	0.45	Exp	20	10	3	0.25	Sph	80	70	3					
720E	0.3	0.40	Exp	60	40	3	0.30	Sph	80	50	3					
720W	0.3	0.45	Exp	40	40	6	0.25	Sph	50	50	6					
800	0.25	0.55	Sph	70	70	12	0.20	Exp	80	80	20					
901-904	0.2	0.80	Sph	60	60	12										
1001	0.3	0.55	Sph	30	15	5	0.15	Exp	280	120	60					
1002	0.2	0.35	Sph	30	30	4	0.30	Sph	30	30	25	0.15	Exp	240	240	120
2001	0.3	0.60	Sph	25	20	4	0.10	Exp	280	100	40					
3001	0.25	0.25	Sph	20	5	5	0.25	Sph	100	25	20	0.25	Exp	400	300	100
3002	0.3	0.45	Sph	45	20	5	0.20	Sph	200	150	10	0.05	Exp	350	280	80
4001	0.25	0.60	Sph	20	10	5	0.05	Exp	260	20	10	0.10	Exp	260	200	10
4002	0.15	0.60	Sph	10	10	4	0.25	Exp	100	60	60					
4003	0.3	0.60	Sph	20	20	5	0.10	Exp	200	200	25					
4004	0.2	0.60	Sph	10	10	5	0.20	Exp	120	45	35					
4009	0.3	0.50	Sph	50	40	8	0.20	Exp	400	400	150					
4011	0.3	0.60	Sph	40	20	10	0.10	Exp	200	100	30					
5001	0.3	0.55	Sph	40	40	5	0.15	Exp	90	90	20					
6001	0.3	0.30	Sph	5	5	5	0.25	Sph	30	30	15	0.15	Exp	300	300	120
7001																
8001	0.2	0.60	Sph	15	15	5	0.10	Exp	200	120	10	0.10	Exp	400	250	75
9001	0.35	0.35	Sph	25	15	5	0.18	Sph	60	20	10	0.12	Exp	200	200	120

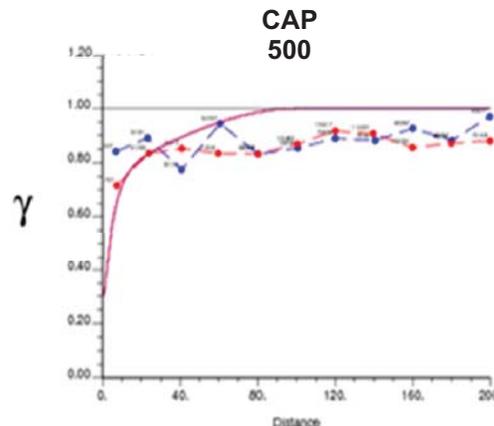
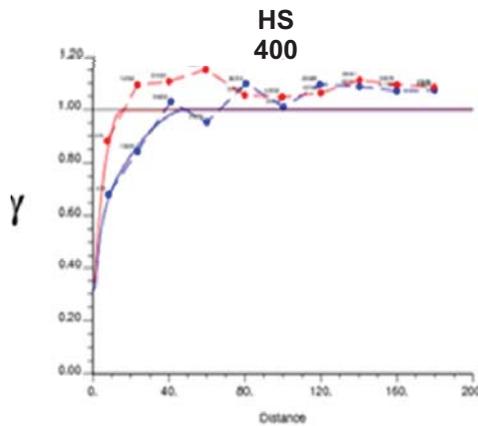
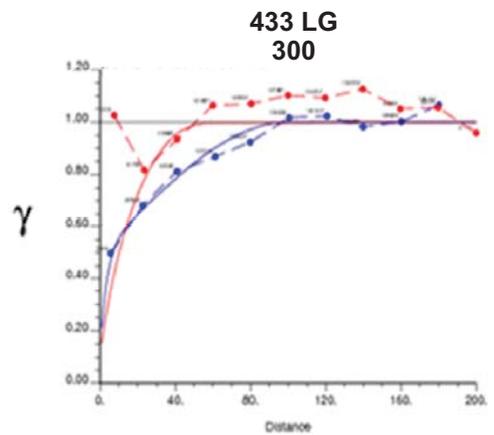
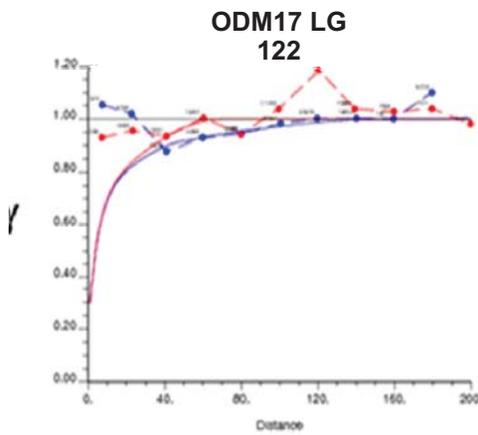
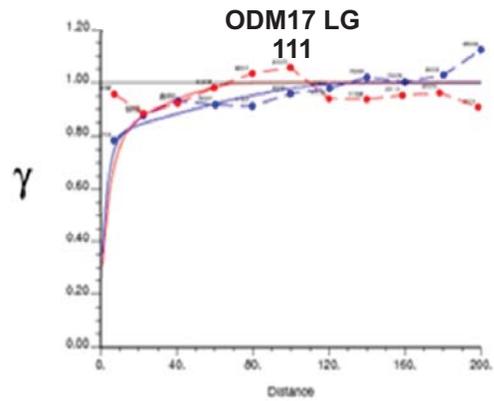
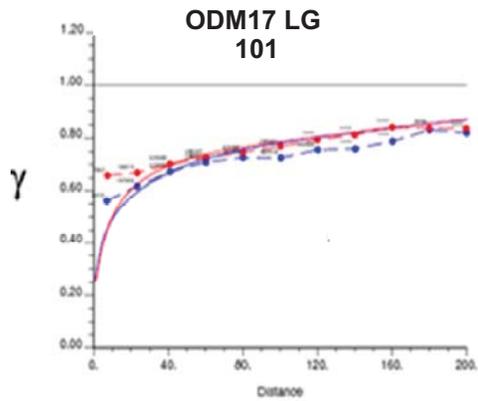


Figure 14-7

newgold[™] New Gold
 Rainy River Mine,
 Rainy River, ON, Canada newgold[™]

Example Gold Variogram
 Models at Rainy River

TABLE 14-14 MAIN ZONE SILVER VARIOGRAM MODELS

Domain	Nugget	Sill	Type	X1	X2	X3	Sill	Type	X2	Y2	Z2	Sill	Type	X3	Y3	Z3
101	0.2	0.40	Sph	35	30	20	0.30	Sph	35	30	110	0.25	Sph	300	300	110
110	0.2	0.35	Sph	50	50	20	0.45	Exp	300	300	40					
111	0.2	0.45	Sph	25	10	15	0.35	Exp	70	110	70					
112	0.2	0.55	Sph	5	5	5	0.25	Exp	35	35	35					
113	0.2	0.60	Sph	35	10	20	0.20	Exp	70	40	40					
114	0.2	0.80	Sph	50	50	20										
115	0.2	0.30	Sph	30	30	30	0.50	Exp	100	100	40					
116	0.3	0.40	Exp	10	10	5	0.30	Sph	130	45	12					
120	0.2	0.40	Sph	45	150	30	0.40	Exp	400	300	30					
121	0.25	0.45	Sph	35	60	8	0.30	Exp	80	100	45					
122	0.2	0.50	Sph	10	10	4	0.30	Exp	60	60	35					
123	0.2	0.35	Sph	35	35	7	0.45	Exp	50	50	10					
125	0.2	0.30	Exp	30	30	30	0.50	Sph	100	100	40					
126	0.3	0.40	Exp	10	10	7	0.30	Sph	50	50	7					
200	0.2	0.20	Sph	10	10	15	0.30	Sph	80	80	40	0.30	Sph	220	220	40
300	0.2	0.30	Sph	5	5	10	0.25	Exp	60	60	100	0.30	Sph	120	120	100
310	0.2	0.20	Sph	45	15	8	0.40	Sph	45	15	60	0.20	Sph	25	90	60
320	0.2	0.20	Sph	10	35	20	0.60	Exp	110	35	20					
280	0.2	0.80	Exp	20	20	20										
400	0.2	0.65	Sph	15	30	5	0.15	Exp	200	100	50					
500	0.2	0.50	Sph	15	40	5	0.15	Exp	160	40	5	0.15	Exp	160	100	20
700E	0.20	1	Sph	110	80	3.00										
700W	0.20	1	Exp	70	135	6.00	0.25	Sph	180	135	6					
710E	0.30	0	Exp	30	35	3.00	0.40	Sph	110	65	3					
710W	0.30	0	Exp	20	10	3.00	0.25	Sph	80	70	3					
720E	0.30	1	Exp	35	35	3.00	0.15	Sph	70	45	3					
720W	0.30	0.45	Exp	30	15	3.00	0.25	Sph	90	45	8					
800	0.25	0.55	Exp	70	70	12	0.20	Sph	140	140	20					
901-904	0.2	0.80	Sph	55	55	12										
1001	0.2	0.57	Sph	25	20	12	0.23	Exp	330	130	90					
1002	0.25	0.60	Sph	40	25	9	0.15	Exp	250	140	70					
2001	0.3	0.54	Sph	20	10	8	0.16	Exp	300	150	100					
3001	0.2	0.33	Sph	25	5	5	0.20	Exp	40	20	20	0.27	Exp	350	200	100
3002	0.3	0.45	Sph	45	20	5	0.20	Sph	100	100	10	0.05	Exp	200	280	35
4001	0.25	0.60	Sph	20	20	5	0.05	Exp	260	100	10	0.10	Exp	260	200	10
4002	0.2	0.70	Sph	35	20	15	0.10	Exp	250	250	70					
4003	0.2	0.50	Sph	55	25	5	0.30	Exp	400	300	25					
4004	0.15	0.55	Sph	30	25	10	0.30	Exp	120	45	60					
4009	0.2	0.60	Sph	40	40	10	0.20	Exp	400	400	150					
4011	0.15	0.65	Sph	20	20	5	0.20	Exp	50	50	30					
5001	0.3	0.55	Sph	40	40	5	0.15	Exp	60	60	15					
6001	0.25	0.35	Sph	40	40	6	0.28	Sph	50	40	40	0.12	Exp	220	140	70

Domain	Nugget	Sill	Type	X1	X2	X3	Sill	Type	X2	Y2	Z2	Sill	Type	X3	Y3	Z3
7001																
8001	0.2	0.60	Sph	15	15	9	0.10	Exp	200	20	10	0.10	Exp	300	200	80
9001	0.2	0.60	Sph	50	20	8	0.20	Exp	300	150	60					

INTERPOLATION PARAMETERS

Gold and silver grade interpolation used OK from capped composite data. Grade interpolation was completed in two or three successive passes using search ellipse orientations and dimensions as described in Table 14-15 and composite sample selection and limits as described in Table 14-16. Interpolation parameters have remained largely unchanged from the previous resource estimation (SRK, 2015) with only a slight adjustment to the width of the search ellipse in the low grade ODM domain (domain 101). This change was implemented to minimize grade smearing across the domain in locations of wide drilling density. Both calcium and sulphur were interpolated according to lithology domain using a three-pass approach and search ellipse and orientations based upon variogram models.

TABLE 14-15 MAIN ZONE GOLD AND SILVER SEARCH ORIENTATION AND RANGES

Domain	Bearing	Plunge	Dip	Pass 1			Pass 2			Pass 3		
				Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor
101	250	-40	42	200	100	5	200	200	25			
110	240	-40	32	100	60	35	200	120	70			
111	255	-40	55	95	55	20	190	110	40			
112	250	-40	42	70	60	35	140	120	70			
113	240	-40	32	75	60	30	150	120	60			
114	230	-40	22	50	50	10	100	100	20	150	150	30
115	240	-40	32	90	90	30	180	180	60			
116	355	60	-5	75	30	7	150	60	14			
120	240	-40	32	55	55	25	110	110	50			
121	250	-40	42	95	40	15	190	80	30			
122	245	-40	37	110	50	25	220	100	50			
123	240	-40	35	55	55	25	110	110	50			
125	5	80	5	90	90	30	180	180	60			
126	190	-45	40	30	75	7	60	150	14			
200	85	36	48	135	135	45	270	270	90			
280	240	-40	32	20	20	20	40	40	40	60	60	60
300	-160	-50	0	70	40	20	140	80	40			
310	-165	-50	0	135	40	15	270	80	30			
320	-160	-45	0	60	30	20	120	60	40			
400	10	50	0	40	55	5	80	110	15	120	165	30

Domain	Bearing	Plunge	Dip	Pass 1			Pass 2			Pass 3		
				Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor
500	15	55	0	100	40	5	200	80	20	300	120	30
801	250	-40	26	80	80	20	160	160	40			
802	250	-40	26	80	80	20	160	160	40			
803	250	-40	26	80	80	20	160	160	40			
901	-140	-55	0	60	60	12	120	120	24			
902	-160	-45	0	60	60	12	120	120	24			
903	-175	-55	0	60	60	12	120	120	24			
904	-160	-60	0	60	60	12	120	120	24			
1001	0	60	0	60	25	10	60	25	10	200	120	60
1002	180	-55	0	55	55	28	55	55	28	200	200	60
2001	185	-50	0	40	20	5	40	20	5	200	100	40
3001	190	-55	0	160	95	45	160	95	45	200	200	100
3002	340	60	-10	80	50	10	80	50	10	200	200	80
4001	30	50	0	50	20	5	50	20	5	200	200	10
4002	30	58	0	35	35	25	35	35	25	100	60	60
4003	40	48	0	60	60	8	60	60	8	200	200	25
4004	0	52	0	65	25	20	65	25	20	120	45	35
4009	0	60	0	120	120	45	120	120	45	200	200	150
4011	5	55	0	65	40	13	65	40	13	200	100	30
5001	0	50	0	63	63	25	63	63	25	200	200	125
6001	175	48	-35	35	35	6	35	35	6	90	90	20
8001	195	-55	0	95	60	60	95	60	60	200	200	75
9001	160	-50	0	60	20	10	60	20	10	200	200	120

Blocks within the Main Zone were estimated using hard boundaries between the different lithologic domains and mineralized zones, and semi-soft boundaries between the high, medium, and low grade subdomains where they occurred. For example, within the ODM/17 Zone, composites within both the high grade and medium grade domains informed blocks within the medium grade domains, and medium grade and low grade composite samples informed blocks within the low grade domains. The high grade domains were estimated using hard boundaries.

TABLE 14-16 BLOCK MODEL INTERPOLATION PARAMETERS

Interpolation Parameters		1 st Pass	2 nd Pass	3 rd Pass
	Search type	Octant	Ellipsoidal	Ellipsoidal
Open Pit (OK)	Min. No. octants	2	-	-
	Max. No. comps per octant	5	-	-
	Min. No. comps.	7	5	2
	Max. No. comps.	12	12	15

Interpolation Parameters		1st Pass	2nd Pass	3rd Pass
	Max. No. comps per Drill Hole	5	3	-
Underground (OK)	Search type	Octant	Ellipsoidal	-
	Min. No. octants	2	-	-
	Max. No. comps per octant	5	-	-
	Min. No. comps.	3	2	-
	Max. No. comps.	8	15	-
	Max. No. comps per drill hole	2	-	-
Intrepid (OK)	Search type	Octant	Ellipsoidal	Ellipsoidal
	Min. No. octants	2	-	-
	Max. No. comps per octant	5	-	-
	Min. No. comps.	5	3	2
	Max. No. comps.	10	15	15
	Max. No. comps per drill hole	3	2	-

Rock density was interpolated into the Main Zone mineralization domains using a single pass, ID² interpolation, a 500 m x 500 m x 500 m search ellipse, and minimum and maximum composite sample limits of two and six, respectively, using hard boundaries for the domains. Where there were insufficient composites to support interpolation, a default density value was assigned for the affected domain (i.e., all blocks within Western, Silver, 34, and 280 zones and unestimated blocks in all other domains). Default values are listed in Table 14-17.

TABLE 14-17 MAIN ZONE DEFAULT DENSITY VALUES

Domain	Density (t/m³)	Domain	Density (t/m³)
Overburden (22)	1.80	1002	2.80
101 - 126	2.85	2001	2.81
200	3.00	3001	2.76
280	2.85	3002	2.76
700	2.84	4001	2.95
710	2.93	4002	2.77
720	2.82	4003	2.90
801	2.90	4004	2.90
802	3.08	4007	2.90
803	2.85	4008	2.90
901	2.84	5001	2.81
902	2.88	6001	2.94
903	2.84	7001	2.78
904	2.70	8001	2.91
1001	2.80	9001	2.95

Gold and silver search orientations and ranges for the Intrepid Zone are listed in Table 14-18. These parameters remain unchanged since the previous estimate prepared by SRK in 2015.

TABLE 14-18 INTREPID ZONE GOLD AND SILVER SEARCH ORIENTATION AND RANGES

Domain	Bearing	Dip	Plunge	Pass 1			Pass 2			Pass 3		
				Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor	Major Axis	Semi Major	Minor
Gold												
100West	165	-58	75	60	50	3	120	100	6	180	150	9
100East	60	58	-60	110	70	3	220	140	6	330	210	9
200West	190	-58	75	80	70	3	160	140	6	240	210	9
200East	60	58	-60	40	80	3	80	160	6	160	160	9
300West	190	-58	75	50	80	3	100	160	6	150	240	9
300East	40	58	-60	80	50	3	160	100	6	240	150	9
Silver												
100West	190	-58	75	120	110	6	240	220	12	240	220	12
100East	60	58	-60	110	80	3	220	160	6	220	160	6
200West	190	-58	75	80	70	3	160	140	6	160	140	6
200East	60	58	-60	85	50	3	170	100	6	170	100	6
300West	190	-58	75	90	45	8	180	90	16	180	90	16
300East	60	58	-60	70	45	3	140	90	6	140	90	6

NEW GOLD BLOCK MODEL VALIDATION

New Gold validated various modelling aspects of the Main Zone estimation. A list of the block model validations is provided below:

- Validation of wireframes;
- Volume comparison by domain between wireframes and block models;
- Validation of OK estimate by comparison to inverse distance cubed (ID³) and nearest neighbour (NN) results;
- Swath plots;
- Visual inspection;
- Graphical comparison (histograms plots) of gold grades in block model and composites; and,
- Comparison of block model and composite statistics.

All validation methods showed satisfactory results.

Selected comparative statistics are shown for gold in Figure 14-8.

FIGURE 14-8 AVERAGE GRADE OF DECLUSTERED COMPOSITES AND OK AND NN BLOCKS BY DOMAIN WITHIN THE MAIN ZONE

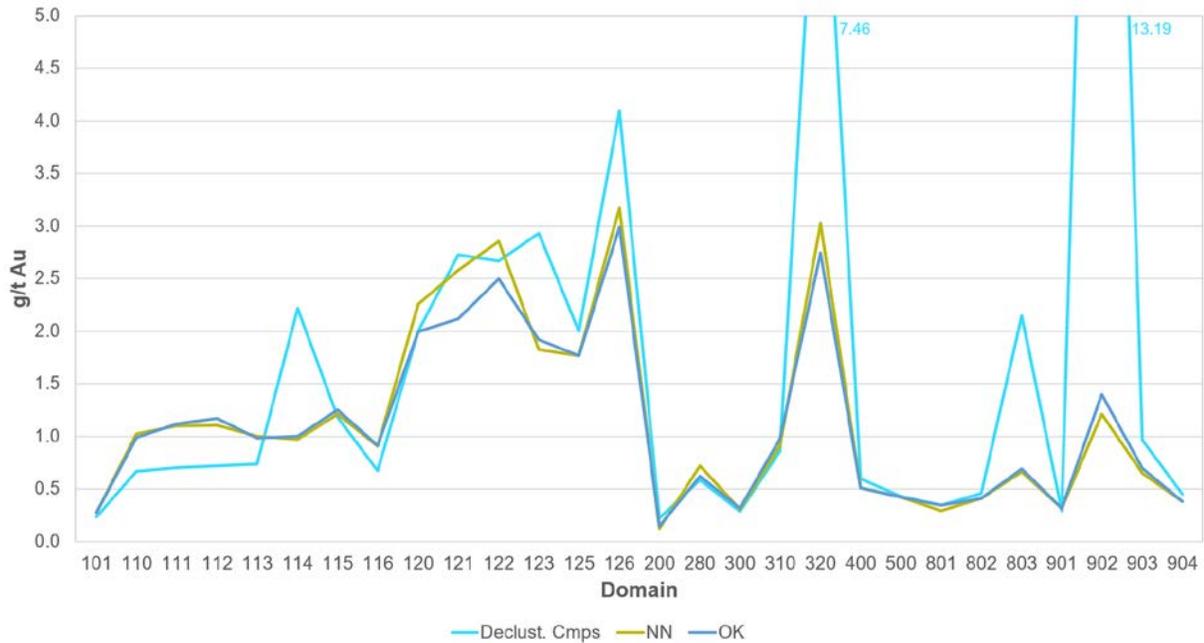
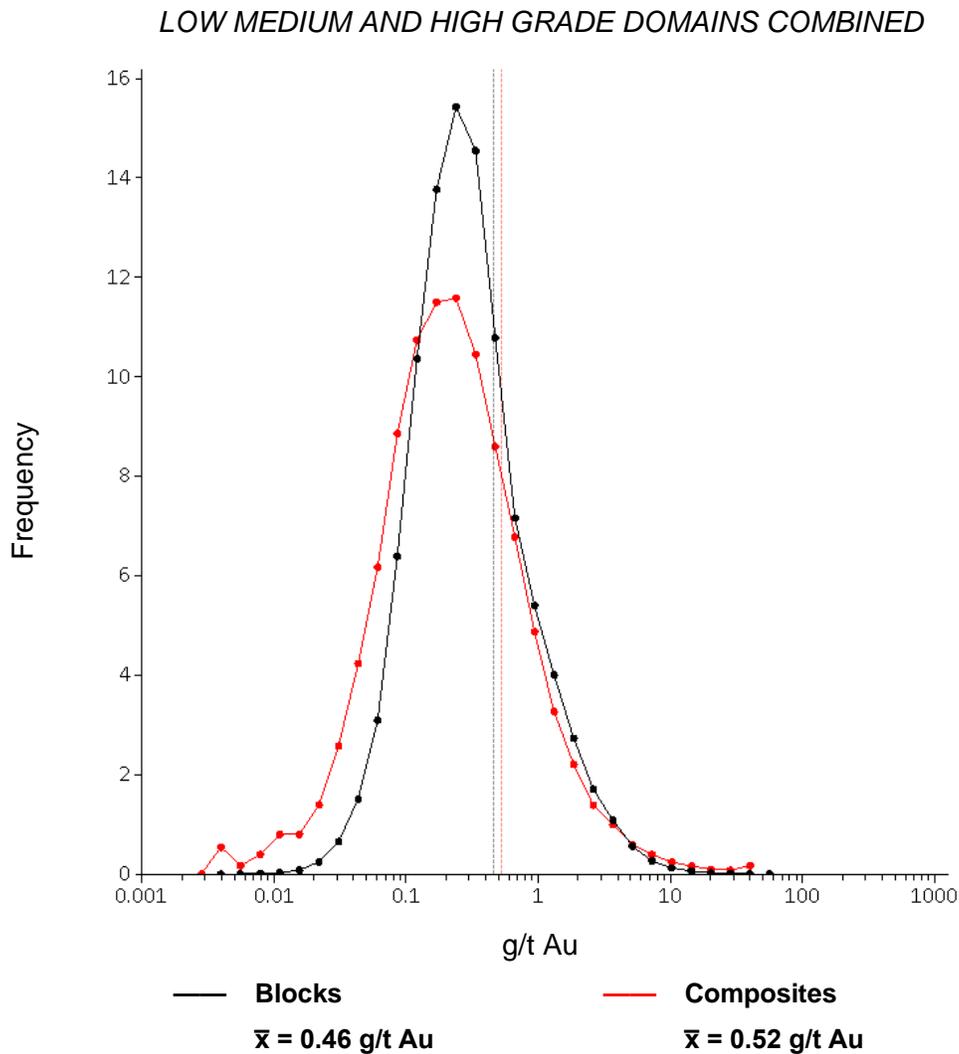


Figure 14-9 shows a histogram of gold values from both blocks and composites within the ODM/17 Zone, including the low, medium, and high grade domains.

FIGURE 14-9 GOLD HISTOGRAM OF BLOCKS AND COMPOSITES WITHIN THE ODM17 ZONE



AMC BLOCK MODEL VALIDATION

In addition to reviewing the validation undertaken by New Gold, AMC has independently conducted the following validation checks:

- Validation of drill hole database
- Validation of wireframes and digital terrain mapping (DTM) topographic surfaces;
- Review and checking of the statistics of selected raw samples and composites; and
- Validation of block models by visual comparisons, statistics, and swath plots.

The Main Zone open pit and underground block models were further validated via the integration of the models by AMC to ensure no overlaps exist between the models that could lead to inadvertent double accounting of volumes during resource and reserve estimation and reporting.

DRILL HOLES

Drill hole database files were provided in Excel format (collars, surveys, assays and lithology) and are effective as of December 31, 2017.

Validation of drill hole data included the following checks:

- collar coordinates outside of range;
- inconsistent FROM and TO values;
- combined assay values greater than 100% or less than detection;
- gaps in assaying where gaps should not exist;
- duplicate records;
- duplicate holes; and
- downhole surveys.

AMC is of the opinion that the drill hole database is valid and suitable to estimate Mineral Resources for the Mine.

MINERALIZED DOMAINS

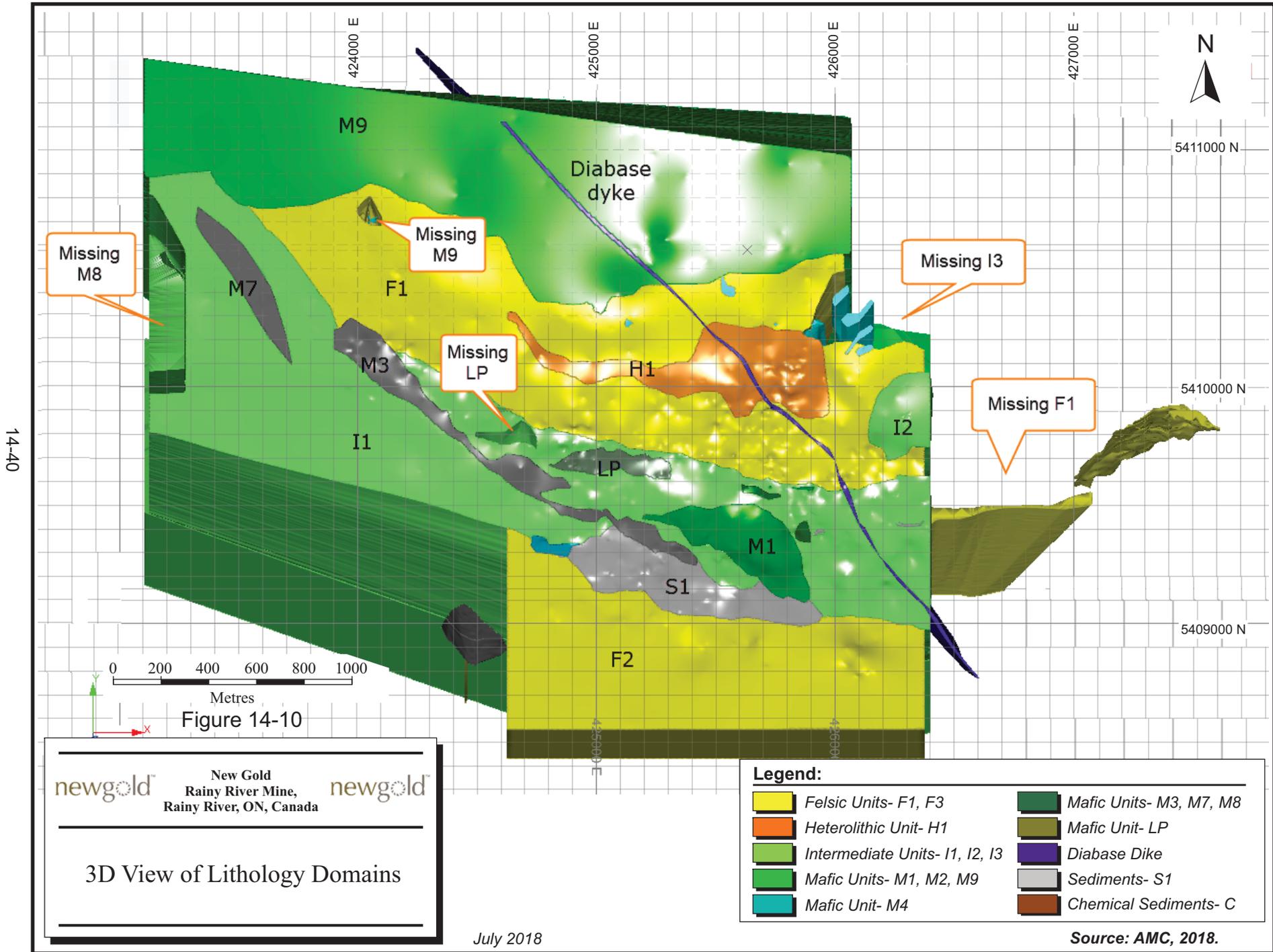
Validation of the wireframes comprising the geologic model included the following checks:

- Verifying the mineralization domains for intercept, crossovers and duplicates;
- Verifying the domaining code name; and
- Comparing volumes of solids with volumes in the block model.

New Gold has provided 15 wireframe solids of mineralized domains. AMC found that the file of ODM zone domain 126 is duplicating Domain 124. As Domain 124 was not estimated, there is no impact to the resource estimation. AMC is of the opinion that there are no domain flagging errors in the block model and that the block model domains are volumetrically representative of their informing wireframes.

LITHOLOGY DOMAINS

New Gold provided a total of 59 separate lithology domains for the ten principal lithologic units in the Rainy River deposit. AMC identified five small lithology domains in unmineralized areas which were missing and had not been assigned to the block model. AMC is of the opinion that this will not have a material impact and that the lithology model is reasonable and appropriate to support Mineral Resource estimation. Figure 14-10 highlights the location of the missing units.



newgold™ New Gold Rainy River Mine, Rainy River, ON, Canada newgold™

3D View of Lithology Domains

Legend:

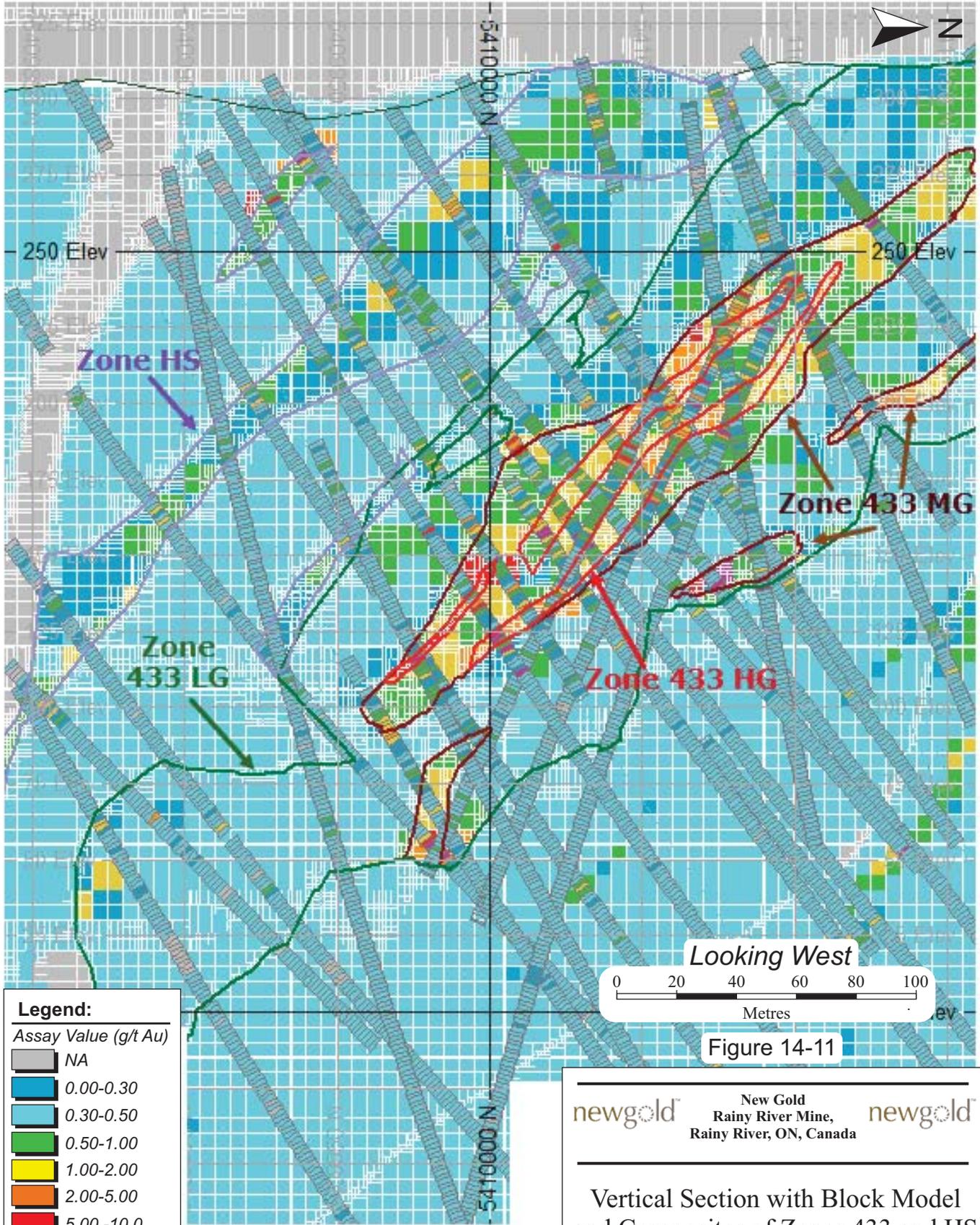
Felsic Units- F1, F3	Mafic Units- M3, M7, M8
Heterolithic Unit- H1	Mafic Unit- LP
Intermediate Units- I1, I2, I3	Diabase Dike
Mafic Units- M1, M2, M9	Sediments- S1
Mafic Unit- M4	Chemical Sediments- C

July 2018

Source: AMC, 2018.

MAIN ZONE MODEL VALIDATION

AMC conducted a visual comparison of composite and block gold grades over the Main Zone. Good agreement between the composite and block gold grades was observed. Figure 14-11 shows an example of the de-clustered drill hole composite gold grades compared to the estimated block grades for the HS and 433 Zone domains.



Legend:

Assay Value (g/t Au)

Grey	NA
Light Blue	0.00-0.30
Medium Blue	0.30-0.50
Green	0.50-1.00
Yellow	1.00-2.00
Orange	2.00-5.00
Red	5.00 -10.0
Purple	>10.0

Looking West
 0 20 40 60 80 100
 Metres

Figure 14-11

newgold New Gold
 Rainy River Mine,
 Rainy River, ON, Canada newgold

Vertical Section with Block Model
 and Composites of Zones 433 and HS

July 2018

Source: AMC, 2018.

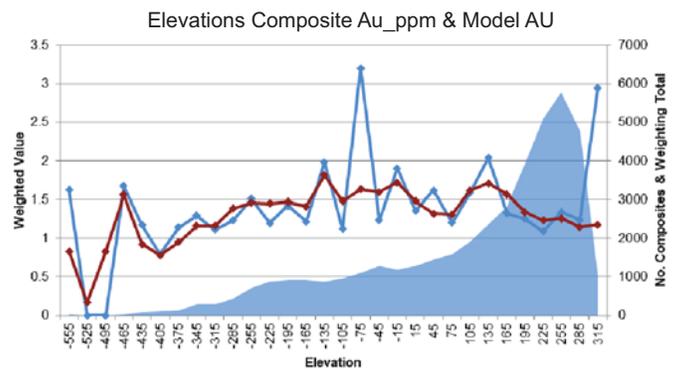
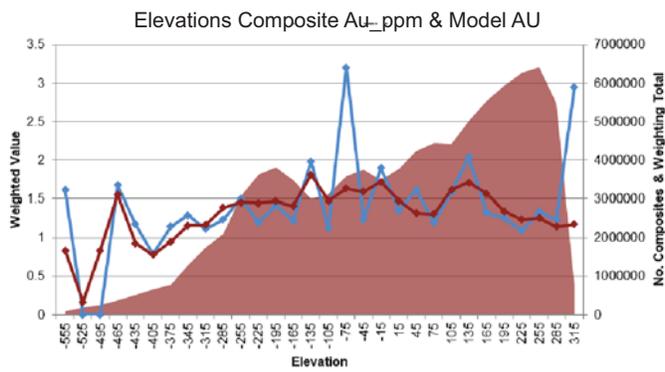
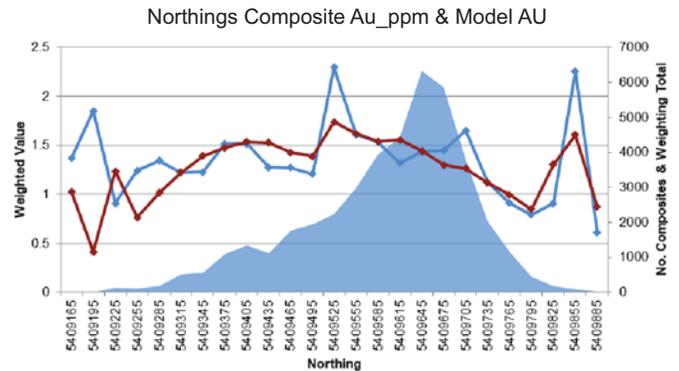
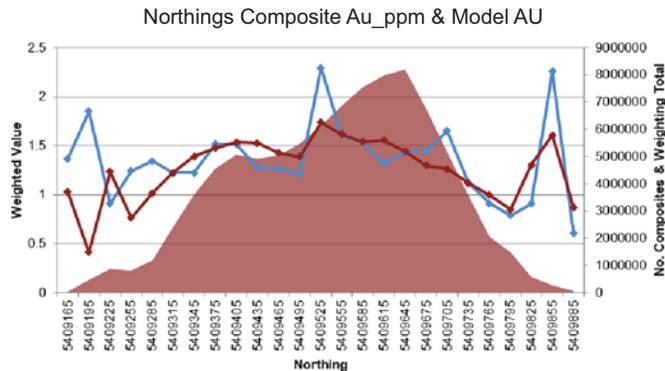
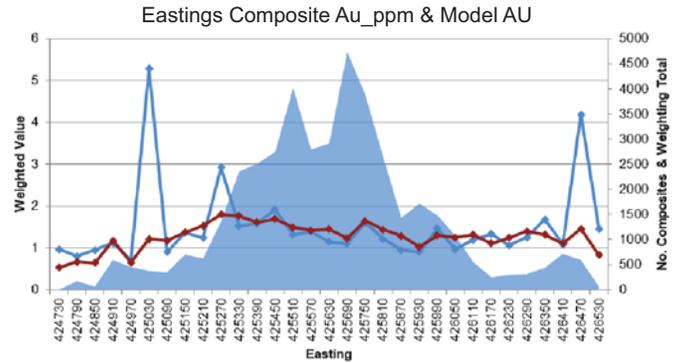
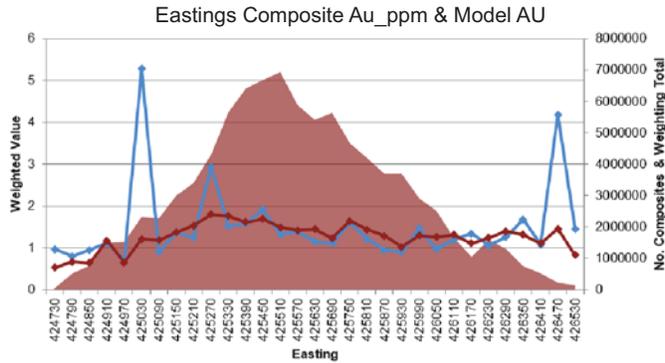
AMC compared the average composite and block gold and silver grades by domain (Table 14-19) and found them to show good agreement.

TABLE 14-19 COMPARISON OF AVERAGE COMPOSITE AND BLOCK GOLD GRADES BY DOMAIN

Domain	Mean (Au g/t)		Mean (Ag g/t)	
	Composite	Model	Composite	Model
101	0.24	0.28	2.03	2.09
110	0.68	0.98	2.94	3.82
111	0.72	1.09	1.48	1.61
112	0.72	1.15	1.44	1.99
113	0.73	0.97	2.74	2.99
114	1.97	0.96	4.84	5.27
115	1.25	1.23	13.85	11.63
116	0.65	0.81	14.33	15.93
120	1.96	1.95	4.79	5.01
121	2.35	2.09	2.26	2.31
122	2.67	2.60	2.39	2.43
123	2.53	1.93	3.30	3.09
124	0.79	0.00	10.80	0.00
125	2.05	1.76	9.77	9.30
126	7.52	2.46	76.73	33.20
200	0.22	0.15	2.54	1.83
280	0.62	0.19	0.95	0.90
300	0.29	0.26	0.78	0.91
310	0.88	0.94	0.99	1.18
320	5.67	2.71	1.61	1.80
400	0.58	0.43	1.27	1.31
500	0.43	0.36	2.29	2.44
700	0.40	0.39	5.40	5.46
710	1.11	1.04	12.37	11.92
720	4.28	3.78	26.61	26.95
801	0.35	0.24	0.70	0.63
802	0.46	0.35	1.03	0.84
803	1.83	0.64	2.39	2.40
901	0.28	0.31	58.14	47.43
902	18.13	1.51	24.57	17.04
903	0.98	0.70	18.47	17.27
904	0.46	0.38	19.12	17.04

AMC compared gold and silver grades of declustered composites and blocks spatially through the preparation and review of east, north, and vertically trending swath plots, by domain.

Figures 14-12 and 14-13 show swath plots for gold and silver grade distribution in the high and medium grade domains of the ODM/17 Zone. The swath plots show good grade agreement. AMC is of the opinion that the Mineral Resource estimate at the Main Zone is in line with reasonable industry practice.



Legend:

- Number of Composites
- Model Tonnes
- Declustered Composite Grade
- Model Grade

Figure 14-12

New Gold
 Rainy River Mine,
 Rainy River, ON, Canada

Swath Plots of Gold Grades for ODM/17 Zone

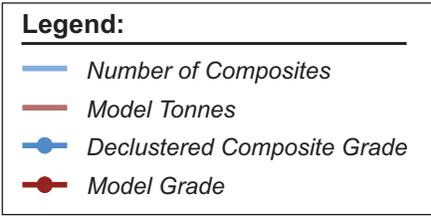
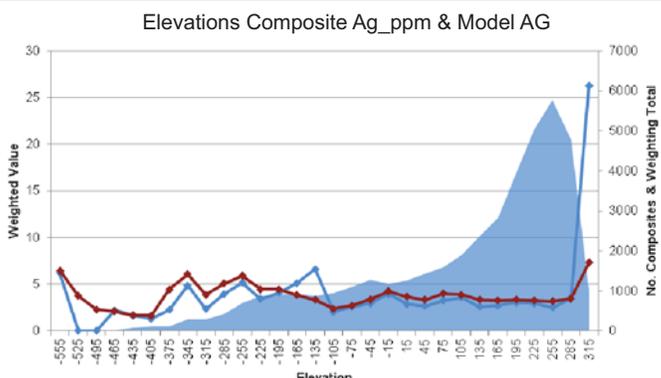
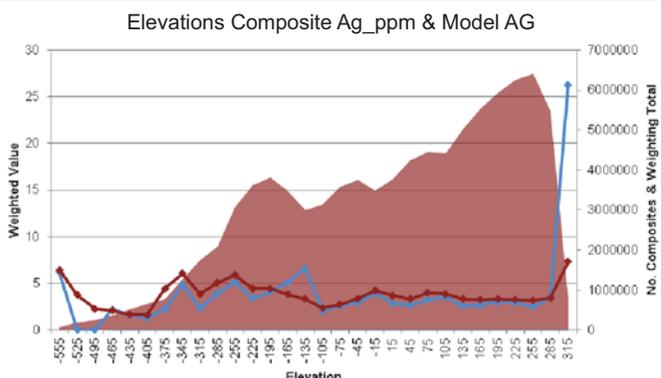
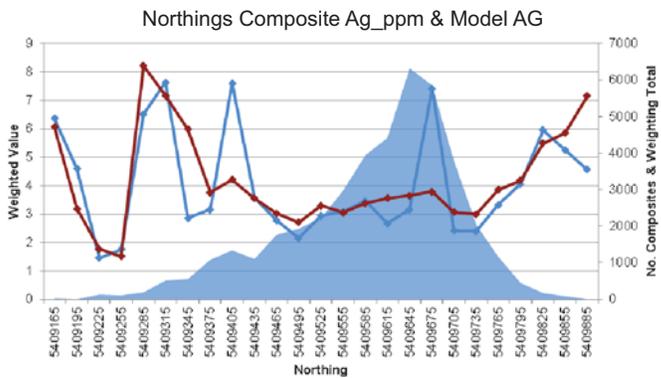
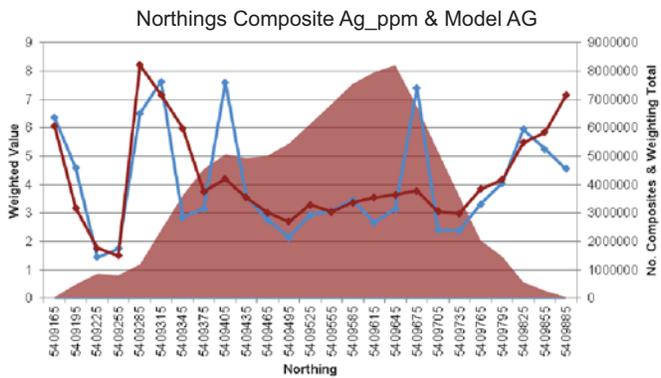
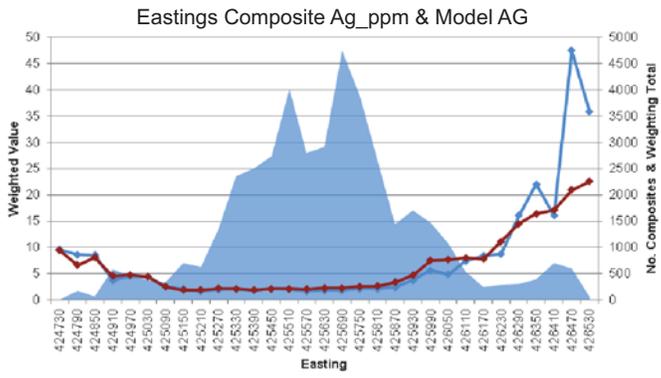
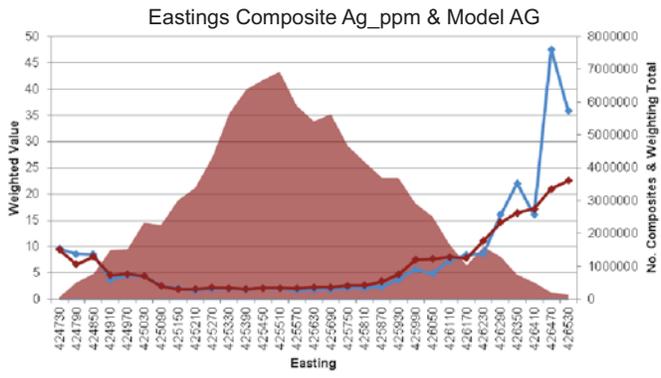


Figure 14-13

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Rainy River, ON, Canada

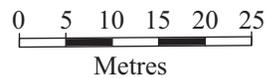
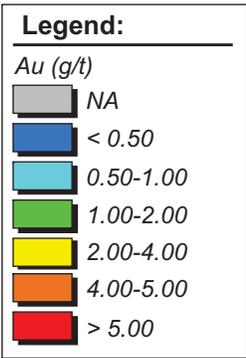
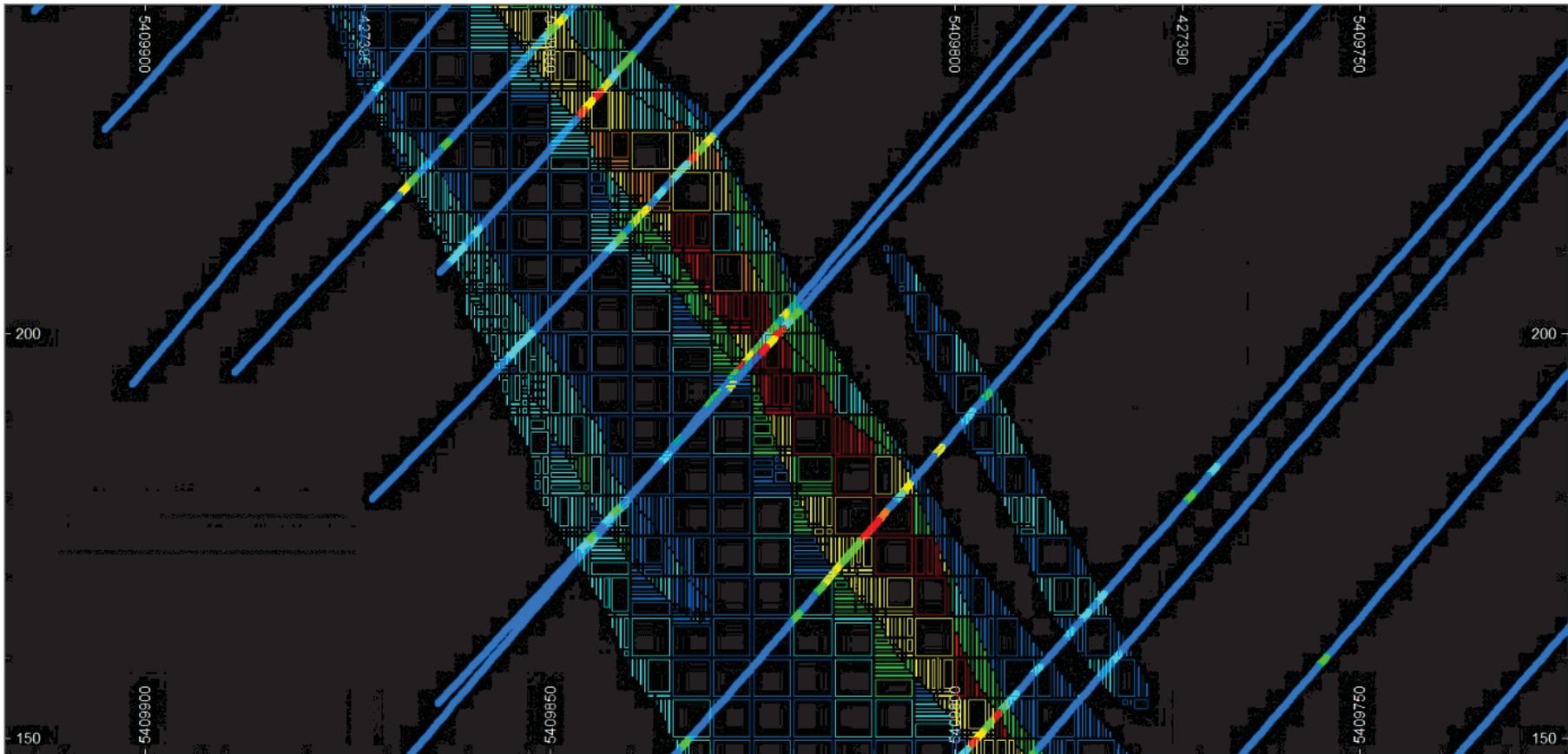
Swath Plots of Silver
Grades for ODM/17 Zone

INTREPID MODEL VALIDATION

AMC's review of the Intrepid block model included the following:

- Drill hole validation;
- Wireframe checks, including checks for open edges and triangle cross-overs;
- Block model checks, comprising checks for:
 - Cell overlaps;
 - Unexpected gaps, holes, or voids internally within the block model;
 - Negative values;
 - Cell size suitability for data spacing;
 - Grade distribution consistency with drill hole data and mineralization style;
 - Reasonability of the interpolation method for the volume of data and style of mineralization;
 - Classified blocks with absent grade values; and
 - Agreement of block grades with supporting drill hole data.

In general, AMC found there to be good visual agreement between the block model and drill hole grades (Figure 14-14). Swath plots of the raw data (composited and capped data were not provided) were compared to block model values for gold and silver for the high grade zone. These showed good agreement between the model and raw assays as shown in Figure 14-15.



Looking East

Figure 14-14

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 Rainy River Mine,
 Rainy River, ON, Canada

Vertical Section Showing Gold
 in Block Model and Drill
 Holes at Intrepid Zone

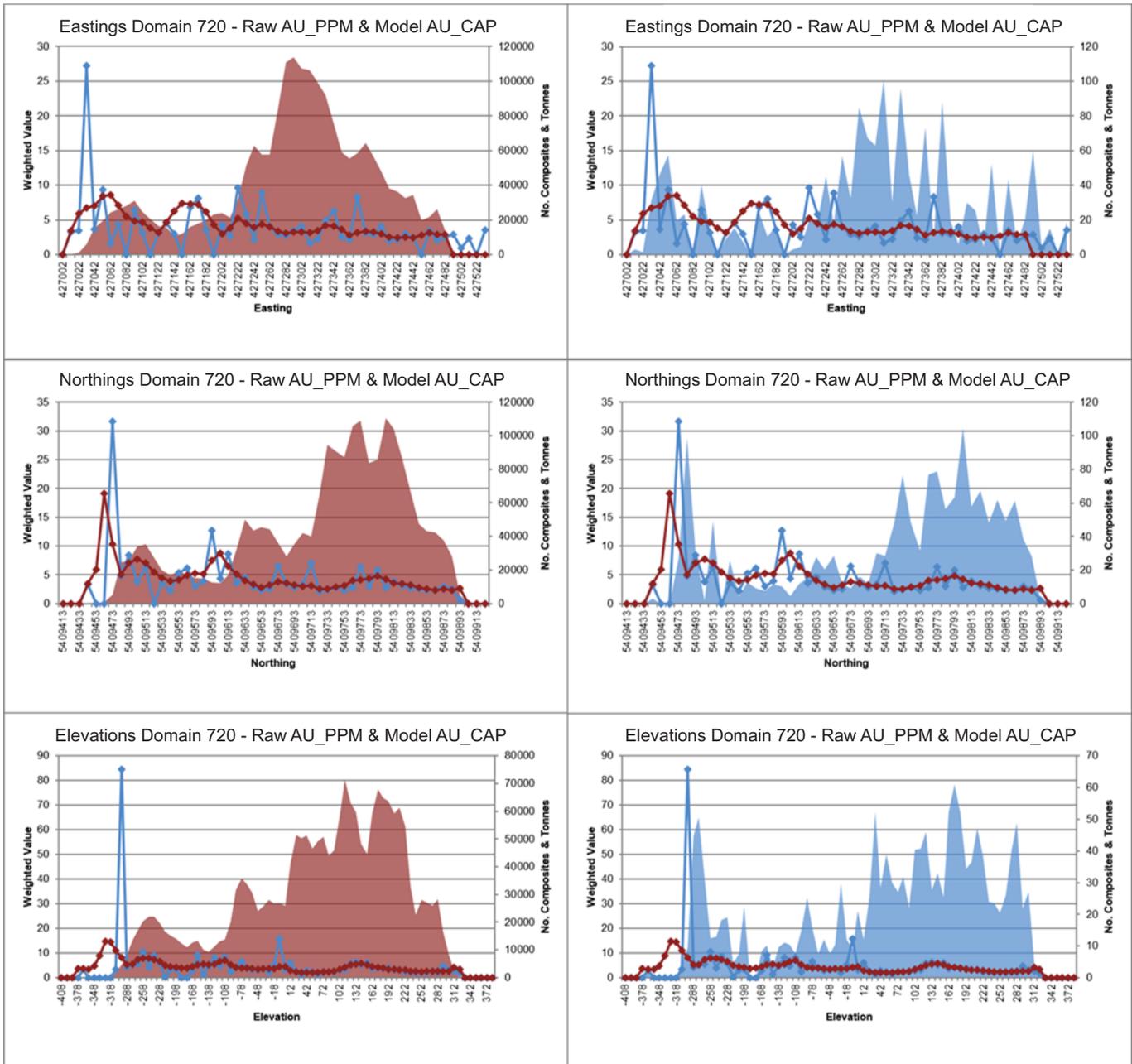


Figure 14-15

Legend:

- Number of Composites
- Model Tonnes
- Declustered Composite Grade
- Model Grade

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 Rainy River, ON, Canada

Swath Plots of Gold Grades for Intrepid Zone

AMC did not find any significant errors that would have an adverse material impact on Mineral Resources. AMC is of the opinion that the Mineral Resource estimate at Intrepid is in line with reasonable industry practice.

MINERAL RESOURCE CLASSIFICATION

Mineral Resources are classified primarily on the basis of an estimated block's distance from the nearest informing drill hole sample composites and corresponding local gold variogram results, with additional consideration given to local geology and gold grade continuity.

New Gold has assigned Measured classification where both drill hole density and rock density measurements provide a high level of confidence in the geologic interpretation, grade continuity, and local grade and rock density estimates. Currently, the ODM/17 and 433 Zones are the only areas with sufficient exploration drilling to support the classification of Measured Mineral Resources. The parameters used for Measured classification are summarized in Table 14-20.

TABLE 14-20 MEASURED BLOCK CLASSIFICATION CRITERIA

Interpolation Parameters	Criteria
Zone	ODM17 / 433 Zone
Interpolation Method	Ordinary Kriging
Search Type	Octant (25x25x25)
Minimum Number of Octants	3
Maximum Number of Composites per Octant	4
Minimum Number of Composites	5
Maximum Number of Composites	8
Maximum Number of Composites per Drill Hole	2

Indicated classification is assigned to blocks estimated during the first estimation pass, where the search ellipse size is equal to 95% of the variogram sill. Inferred classification is assigned to all blocks estimated during the second or third estimation passes. New Gold also considered the confidence in the geological interpretation during the classification process. A vertical section displaying block class is shown in Figure 14-16. AMC is of the opinion that the classification criteria used to categorize blocks at Rainy River is reasonable.

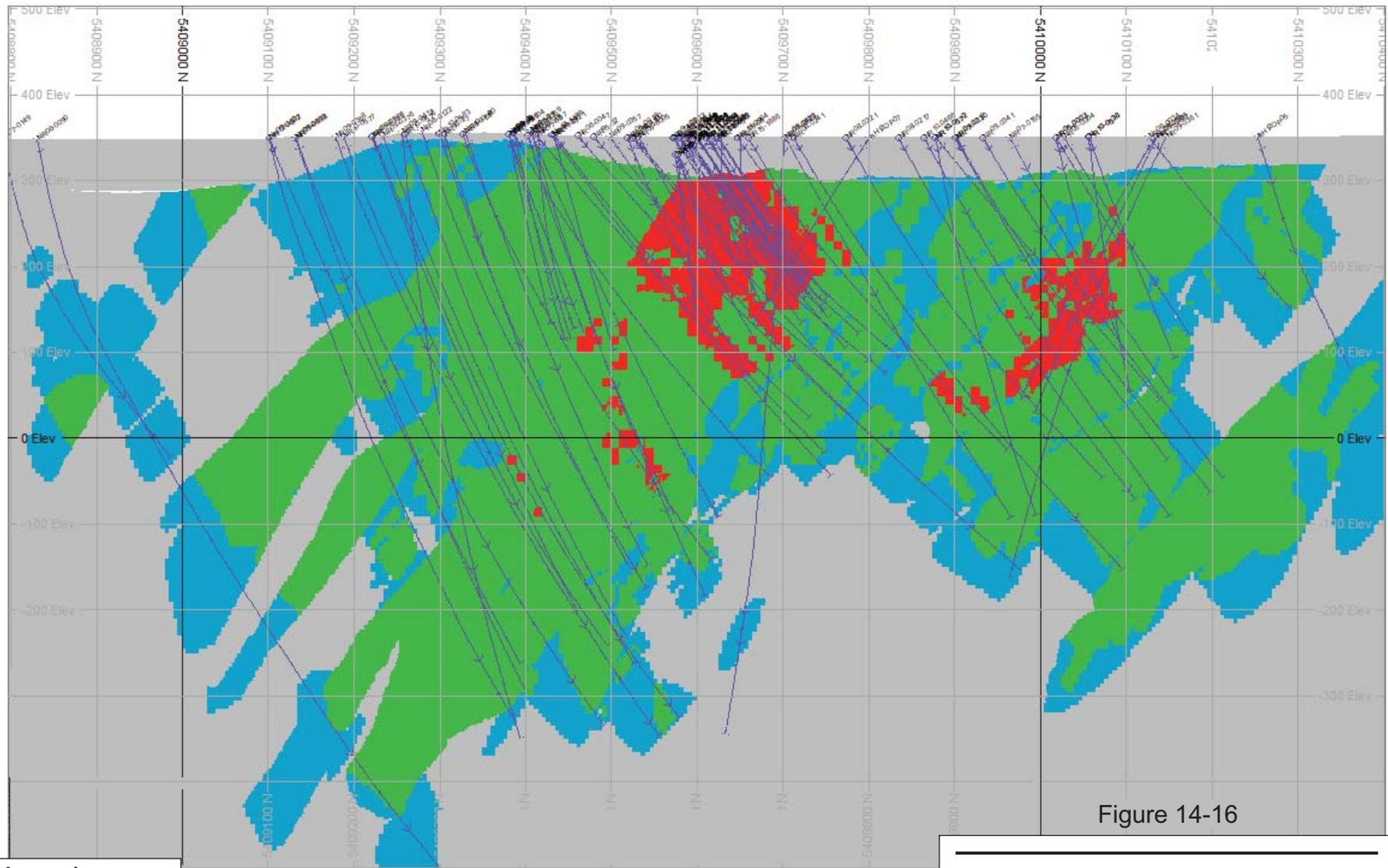
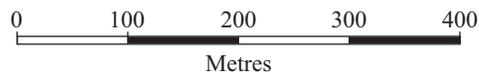


Figure 14-16

Legend:

Classification

- NA
- Measured
- Indicated
- Inferred



Section 425775 E Looking West

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Rainy River Mine,
Rainy River, ON, Canada

Vertical Section Showing
Block Model Classification

CUT-OFF GRADE

The Mineral Resource cut-off grade is expressed as a gold equivalent grade. The gold equivalency formula used to calculate cut-off grades is provided below:

$$\text{AuEq} = \text{Au g/t} + ((\text{Ag g/t} * 19 * 70) / (1375 * 95))$$

Where:

Gold price = \$1375 per ounce

Gold recovery = 95%

Silver price = \$19 per ounce

Silver recovery = 70%

The assumptions for gold and silver prices and recoveries are discussed in more detail in Section 15.

COMPARISON TO PREVIOUS BLOCK MODEL

New Gold prepared grade tonnage curves (Figure 14-17) comparing the 2015 block model with the 2017 block model detailed in this report. For consistency in the comparison, both block models have been constrained by the 2017 Open Pit Resource Shell.

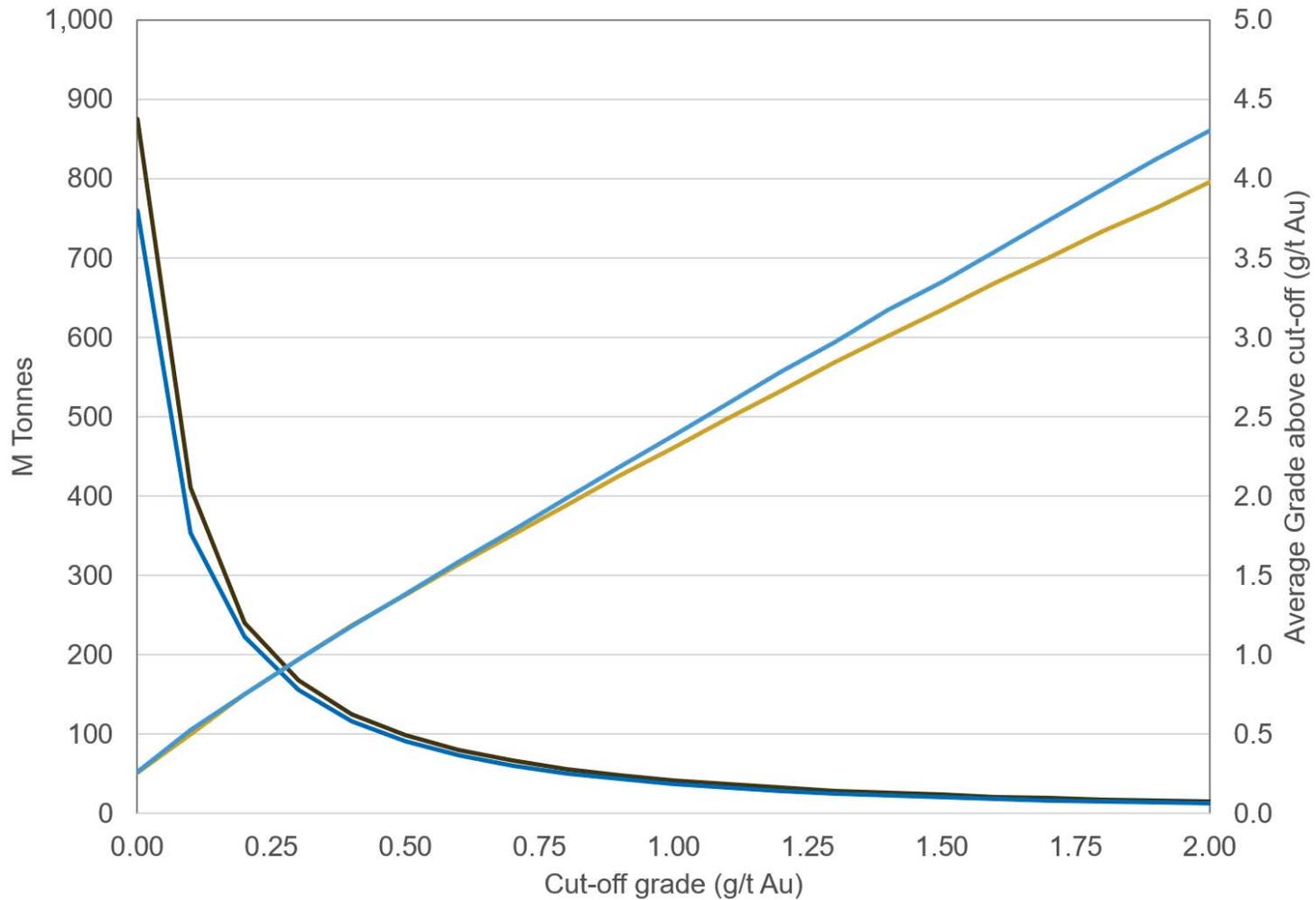


Figure 14-17

Legend:

- 2017 Tonnes
- 2015 Tonnes
- 2017 Gold Grade
- 2015 Gold Grade

July 2018

Source: New Gold, 2018.

New Gold
Rainy River Mine,
Rainy River, ON, Canada

Grade Tonnage Curves Comparing 2017
Sub-Blocked Block Model and 2015 Block
Model Constrained within
the 2017 Open Pit Resource Shell

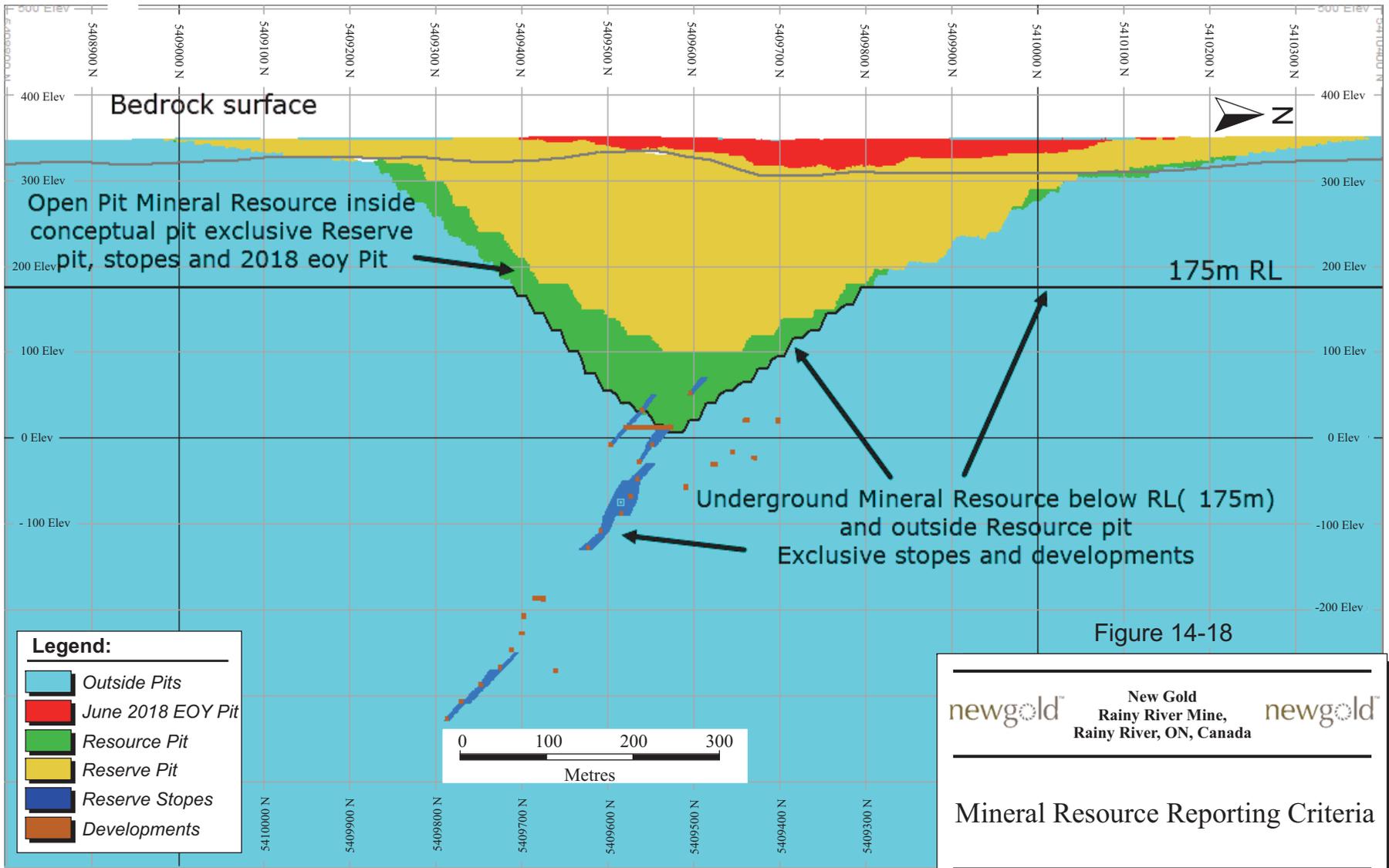
Below a cut-off grade of 1.00 g/t Au, the 2017 model reports very similar average grades, and slightly higher tonnes and consequently, slightly higher ounces than the 2015 model. Above a cut-off grade of 1.00 g/t Au, the 2017 reports very similar tonnages and lower average grades at increasing cutoffs and therefore slightly lower total contained ounces compared to the 2015 model. These differences are largely attributable to the 2017 sub-blocked Vulcan model more accurately representing the volumes of individual domains, especially the higher grade subdomains of the ODM/17 and 433 Zones.

MINERAL RESOURCE SUMMARY

Mineral Resources for the Rainy River Mine have been updated to an effective date of June 30, 2018. They are reported based on gold equivalent cut-off grades consistent with the mining methods envisioned for possible extraction in the future. A summary of the Mineral Resources at Rainy River is presented in Table 14-21. The Mineral Resources reported herein supersede the mineral resources reported previously in New Gold's 2017 Annual Information Form dated March 31, 2018 and available of SEDAR. Mineral Resources are reported exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability.

Open pit Mineral Resources reported here are constrained by a conceptual open pit shell that has been defined based on metal prices of \$1,375 per ounce for gold and \$19 per ounce for silver, metal recoveries of 95% for gold and 70% for silver, and mining, processing, and general and administrative (G&A) costs consistent with the current operation. The open pit Mineral Resource is also reported based on higher grade direct processing material and lower grade material likely to be stockpiled for future processing. Underground Mineral Resources are reported below the RL 175 m reference elevation and peripheral to and below the conceptual resource pit shell.

Figure 14-18 provides a schematic vertical section of the constraining limits of the open pit and underground Mineral Resources reported for the Rainy River Mine.



New Gold
Rainy River Mine,
Rainy River, ON, Canada

Mineral Resource Reporting Criteria

TABLE 14-21 MINERAL RESOURCES – EFFECTIVE JUNE 30, 2018

Class	Tonnes (000s)	Grades, g/t		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Direct Processing					
Open Pit (OP)					
Measured	2,996	1.14	5.7	109	550
Indicated	26,541	1.12	3.4	957	2,921
OP Measured and Indicated	29,537	1.12	3.7	1,066	3,471
Inferred	3,697	1.06	3.2	126	385
Underground (UG)					
Measured	-	-	-	-	-
Indicated	7,934	3.06	8.6	780	2,206
UG Measured and Indicated	7,934	3.06	8.6	780	2,206
Inferred	1,215	3.59	2.7	140	107
Low Grade (LG) - Stockpile					
Measured	2,462	0.35	3.3	28	261
Indicated	23,175	0.36	2.3	268	1,713
LG Measured and Indicated	25,637	0.36	2.4	296	1,974
Inferred	3,959	0.37	1.4	47	180
Combined Mineral Resources					
Measured	5,458	0.78	4.6	137	811
Indicated	57,651	1.08	3.7	2,005	6,840
Total Measured and Indicated	63,109	1.06	3.8	2,142	7,651
Total Inferred	8,871	1.10	2.4	313	672

Notes:

1. CIM (2014) definitions were followed for Mineral Resources.
2. Mineral Resources are estimated using long-term metal prices of US\$1,375 per ounce gold, US\$19.00 per ounce silver and a C\$/US\$ exchange rate of 0.77. Metal recoveries of 95% for Au and 70% for Ag were used.
3. Mineral Resources are reported at cut-off grades for direct processing material of 0.50 g/t AuEq for open pit and 2.0 g/t AuEq for underground and between 0.30 g/t and 0.5 g/t AuEq for low grade resources. The gold equivalency formula is as follows: $AuEq\ g/t = Au\ g/t + ((Ag\ g/t * 19 * 70) / 1375 * 95)$
4. Bulk density ranges from 2.70 t/m³ to 3.08 t/m³.
5. Mineral Resources are exclusive of Mineral Reserves
6. Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
7. Open pit Mineral Resources are constrained by a conceptual pit shell.
8. Totals may not add due to rounding.

COMPARISON TO PREVIOUS MINERAL RESOURCE ESTIMATE

A comparison between the current Mineral Resource estimate, which is effective June 30, 2018 (based on the 2017 block model), and the Mineral Resource Statement dated December 31, 2016 (based on the 2015 block model) is presented in Table 14-22. Principal changes since the 2015 block model include:

- Additional resource definition drilling totalling 24,263 m in 105 drill holes;
- Updated AuEq cut-off grades based on updated gold and silver prices and recoveries

TABLE 14-22 COMPARISON OF MINERAL RESOURCE WITH PREVIOUS ESTIMATION

Resource Estimate	Class	Quantity	Grade		Metal	
		'000 t	Au g/t	Ag g/t	Au '000 oz	Ag '000 oz
Direct Processing Material						
Open pit in situ Resource						
December 2016	Measured	3,638	1.11	2.8	130	329
	Indicated	28,976	1.16	3.7	1,079	3,485
	Measured & Indicated	32,614	1.15	3.6	1,209	3,814
	Inferred	5,808	1.01	2.8	188	528
June 2018	Measured	2,996	1.13	5.71	109	550
	Indicated	26,541	1.12	3.42	957	2,921
	Measured & Indicated	29,537	1.12	3.66	1,066	3,471
	Inferred	3,697	1.06	3.24	126	385
Difference %	Measured	-18	2	104	-16	67
	Indicated	-8	-3	-7	-11	-16
	Measured & Indicated	-9	-3	2	-12	-9
	Inferred	-36	5	16	-33	-27
Underground in situ Resource						
December 2016	Measured					
	Indicated	5,035	3.71	10.4	601	1,678
	Measured & Indicated	5,035	3.71	10.4	601	1,678
	Inferred	5,130	3.53	2.8	583	467
June 2018	Measured					
	Indicated	7,934	3.06	8.65	780	2,206
	Measured & Indicated	7,934	3.06	8.65	780	2,206
	Inferred	1,215	3.59	2.74	140	107
Difference %	Measured					
	Indicated	58	-18	-17	30	31
	Measured & Indicated	58	-18	-17	30	31
	Inferred	-76	2	-2	-76	-77

Resource Estimate	Class	Quantity	Grade		Metal	
		'000 t	Au g/t	Ag g/t	Au '000 oz	Ag '000 oz
Low Grade Material						
Open Pit in situ Resource						
December 2016	Measured	2,490	0.36	2.8	29	223
	Indicated	34,984	0.43	2.4	483	2,694
	Measured & Indicated	37,474	0.42	2.42	512	2,917
	Inferred	8,916	0.40	1.5	114	435
June 2018	Measured	2,462	0.36	2.80	28	261
	Indicated	23,175	0.43	2.40	268	1,713
	Measured & Indicated	25,637	0.36	2.39	296	1,974
	Inferred	3,959	0.37	1.41	47	180
Difference %	Measured	-1	0	0	-4	17
	Indicated	-34	0	0	-44	-36
	Measured & Indicated	-32	-16	-1	-42	-32
	Inferred	-56	-7	-6	-59	-59
Combined Direct Processing and Low Grade Mineral Resources						
December 2016	Measured	6,128	0.81	2.8	159	552
	Indicated	68,995	0.97	3.5	2,163	7,857
	Measured & Indicated	75,123	0.96	3.5	2,322	8,409
	Inferred	19,854	1.39	2.2	885	1430
June 2018	Measured	5,458	0.78	4.6	137	811
	Indicated	57,651	1.08	3.7	2,005	6,840
	Measured & Indicated	63,109	1.06	3.8	2,142	7,651
	Inferred	8,871	1.10	2.4	313	672
Difference %	Measured	-11	-3	65	-14	47
	Indicated	-16	11	4	-7	-13
	Measured & Indicated	-16	10	8	-8	-9
	Inferred	-55	-21	5	-65	-53

Notes for the 2016 Estimate:

1. Mineral resources were estimated in accordance with the CIM Definition Standards, which are incorporated by reference in NI 43-101.
 2. The mineral resource is effective December 31, 2016.
 3. Mineral resources have been estimated based on the following metal prices \$1,350 oz Au, \$17.00 oz Ag.
 4. All metal prices are quoted in US dollar at an exchange rate of \$1.00 US to \$1.25 Canadian.
 5. Open pit direct processing mineral resources are reported at a cut-off grade of 0.30-0.45 g/t AuEq, Open pit stockpile mineral resources are reported at a cut-off grade of 0.30g/t AuEq, underground mineral resources are reported at a cut-off grade of 2.50 g/t AuEq.
 6. Mineral resources are exclusive of Mineral Reserves.
 7. Numbers may not add due to rounding.
 8. Underground mineral resources are reported below a larger mineral resource pit shell, which has been defined based on a \$1,350/oz gold.
- (Source: New Gold AIF March 29, 2017)

Notes for the 2018 Estimate:

1. CIM (2014) definitions have been followed for the estimation and reporting of mineral resources.
2. Mineral resources are estimated based on long-term metal prices of US\$1,375 per ounce gold, US\$19.00 per ounce silver and a C\$/US\$ exchange rate of 0.77. Metal recoveries of 95% for gold and 70% for silver were used.

3. Mineral resources are reported at cut-off grades of 0.50 g/t AuEq for open pit and 2.0 g/t AuEq for underground direct processing material respectively. Low grade resources are reported between cut-offs of 0.30 g/t and 0.5 g/t of AuEq. $AuEq\ g/t = Au\ g/t + ((Ag\ g/t * 19 * 70) / 1375 * 95)$
 4. Bulk density ranges from 2.70 t/m³ to 3.08 t/m³.
 5. Open pit mineral resources are constrained by a conceptual pit shell.
 6. Totals may not add due to rounding
- (Source: AMC)

Comparison of the current and previous resource estimates shows the following for the open pit mineral resources:

- Open pit Measured and Indicated Mineral Resource has a decrease by 9% in tonnage and by 3% in gold grades, but the grades of silver have increased by 2%. The metal content of gold and silver has decreased by 12% and 9% respectively.
- Open pit Inferred Mineral Resource tonnes have decreased by 36%. As a result of only slight increases of gold grades (5%) and silver grades (16%), the metal content of gold and silver has decreased by 33% and 27% respectively.

Comparison of the current and previous resource estimates shows the following for underground mineral resources:

- Underground Indicated Mineral Resource tonnes increased by 58% while gold and silver grades decreased by 18% and 17% respectively. Contained metal increased by 30% for gold and 31% for silver.
- Both underground Inferred tonnes and contained metal decreased by approximately 76%. The grades of gold increased by 2% and the silver grades decreased by 2%.

Comparison of the current and previous resource estimates for stockpile mineral resources indicates the following:

- Stockpile Measured and Indicated Mineral Resource tonnes have decreased by 32%, the gold grades have decreased by 16% and silver grades have decreased by 1% only. Contained gold metal has decreased by 42% and silver metal has decreased by 32%.
- The Inferred Mineral Resource tonnes have decreased by 56%. The grades for gold and silver decreased by 7% and 6% respectively. Both the gold metal content and silver metal content decreased by 59%.

Comparison of the current and previous resource estimates indicates the following for total combined mineral resources:

- Total Measured and Indicated Mineral Resource tonnes have decreased by 16%, while gold grades have increased by 10% and the silver grades have increased by 8% resulting in contained gold metal decreasing by 8% and silver metal decreasing by 9%.

- Total Inferred Mineral Resource tonnes have decreased by 55%, gold grades decreased by 21% and silvers grade increased by 5%. The metal content of gold and silver decreased for 65% and 53% respectively.

To summarize, the principal reasons for differences in tonnes, grades, and metal include:

1. Reporting at revised cut-off grades
2. Updated gold equivalent formula
3. Addition of 105 drill holes totalling 24,263 m to the drill hole database
4. The use of different open pit reserve shells and open pit resource shells because of different input parameters

15 MINERAL RESERVE ESTIMATES

Mr. Nicholas Kwong, P.Eng., of New Gold has overseen, and is the QP responsible for, the preparation of the Mineral Reserve estimate for the Rainy River Mine as a whole, using the Mineral Resource estimate to December 31, 2017, which has since been depleted to reflect production from December 31, 2017 to June 30, 2018. The Mineral Reserve estimate conforms to CIM (2014) definitions.

The open pit Mineral Reserve estimate has been prepared by Mr. Binsar Sirait, P.Eng., a mining engineer and employee of New Gold, who takes QP responsibility for the open pit Mineral Reserve estimate. The underground Mineral Reserves have been prepared by New Gold engineers, under the guidance of Mr. Herbert A. Smith, P.Eng., a mining engineer and independent consultant employed by AMC, who takes QP responsibility for the underground Mineral Reserve estimate. Messrs. Kwong, Sirait, and Smith are QPs as this term is defined in NI 43-101.

A summary of the Mineral Reserve estimates at Rainy River is presented in Table 15-1.

TABLE 15-1 MINERAL RESERVES – EFFECTIVE JUNE 30, 2018

	Tonnes (000s)	Grades, g/t		Contained Ounces (000s)	
		Au	Ag	Au	Ag
Direct Processing Reserves					
Open Pit (OP)					
Proven	21,468	1.22	2.5	842	1,739
Probable	50,409	1.15	3.1	1,860	5,069
Total OP P&P (direct processing)	71,878	1.17	2.9	2,701	6,807
Underground (UG)					
Proven					
Probable	8,954	3.55	9.5	1,021	2,728
Total UG P&P (direct processing)	8,954	3.55	9.5	1,021	2,728
Low Grade Reserves (OP)					
Proven	7,407	0.38	2.0	90	478
Probable	26,900	0.36	2.4	308	2,086
Total OP LG P&P	34,307	0.36	2.3	398	2,564
Stockpiles					
Proven	5,313	0.58	1.8	99	299
Probable					
Total Stockpile P&P	5,313	0.58	1.8	99	299
Combined OP & UG					
Proven	34,189	0.94	2.3	1,031	2,516
Probable	86,263	1.15	3.6	3,189	9,882
Total OP & UG P&P	120,451	1.09	3.2	4,220	12,398

Notes:

- CIM (2014) definitions were followed for Mineral Reserves.
- Effective date of Mineral Reserves is June 30, 2018; estimated by Nicholas Kwong, P. Eng., an employee of New Gold Inc.
- The open pit Mineral Reserves were estimated using an operating pit design determined from Measured and Indicated Mineral Resources and supporting a mine plan of 27 ktpd. The design is based on a gold price of US\$1,275 per troy ounce and a silver price of US\$17 per troy ounce.
- Open pit Mineral Reserves were estimated at a cut-off grade of 0.5 g/t AuEq for direct processing and 0.3 g/t AuEq for low grade material for stockpiles (see note 6 below for basis of AuEq).
- Underground Mineral Reserves were estimated at a cut-off grade of 2.2 g/t AuEq for stoping and 0.53 g/t AuEq for development (see note 6 below for basis of AuEq).
UG cut-off grade assumptions:
 - Au price US\$1,275 per troy ounce, Ag price US\$17 per troy ounce
 - Exchange rate 1.3 C\$: 1 US\$ (1 C\$ = 0.77 US\$)
 - Planned hangingwall and footwall dilution of 0.5 m and 0.25 m respectively; unplanned dilution 4.3%
 - Average mining recovery: 92%
 - Mill Au and Ag recovery: 95% and 60% respectively
 - Breakeven cost of \$107/t of mill feed, inclusive of costs for mining, processing, G&A, refining & transport, royalties and sustaining capital allowance.
- AuEq is equal to Au g/t + 0.0084*Ag g/t.
- Totals may not add due to rounding.
- Tonnes and grades are in metric units.

The QP is not aware of any mining, metallurgical, infrastructure, permitting, or other relevant factors that could materially affect the Mineral Reserve estimates.

OPEN PIT MINERAL RESERVE ESTIMATES

The pit optimization analyses for the mine plan were run on the Measured and Indicated Mineral Resources to determine the economics of extraction by open pit methods using metal prices of US\$1,275/oz Au and US\$17.00/oz Ag. Only blocks classified as Measured or Indicated Resources were included in the pit optimization process.

The final pit design was developed from the pit optimization results and included 35 m wide ramps at a 10% grade. Open pit Mineral Reserves are reported within the final pit design at cut-off grades of 0.3 g/t and 0.5 g/t AuEq for Low Grade and Direct Processing ores, respectively.

OPEN PIT OPTIMIZATION

Open pit optimization was conducted on the Mineral Resources using US\$1,275/oz Au and US\$17.00/oz Ag for the resource pit. MineSight Economic Planner was used for open pit optimization.

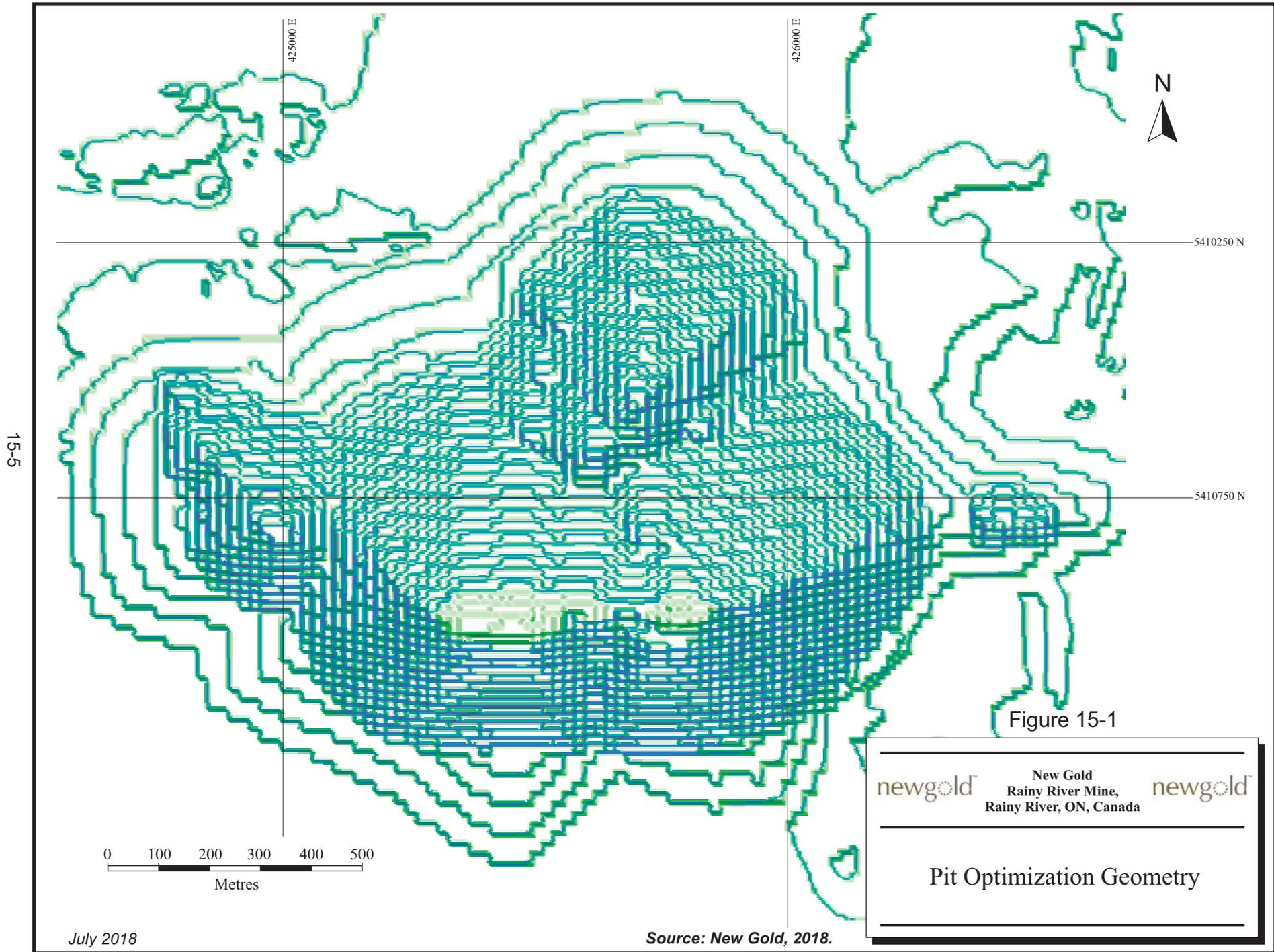
The optimization parameters used for open pit optimization are listed in Table 15-2. These parameters were used in the generation of the pit shell for Mineral Reserves and may differ somewhat from the final economic parameters used in the economic model. The pit optimization was also used by BGC Engineering Inc. (BGC) for the completion of a geotechnical assessment to evaluate pit slope recommendations from June 2017 (BGC, 2017).

The pit optimization was completed by New Gold based on the December 31, 2017 Mineral Resource estimate. Only blocks classified as Measured or Indicated Resources were included in the pit optimization process.

TABLE 15-2 OPEN PIT OPTIMIZATION PARAMETERS

Pit Optimization Parameter	Units	Values
MineSight Block Size	m	10 x 10 x 10
Overburden	H:V	8:1
Hard Rock	°	40-54
Gold Price	US\$/oz	1275
Silver Price	US\$/oz	17
CIP Process		
Au Dore treatment charge/refining charge (TCRC)	C\$/t processed	0.13
Gold Recovery	%	90
Silver Recovery	%	65
Royalty	C\$/t processed	1.34
Costs		
Reference Mining Cost	C\$/t mined	2.59
Process	C\$/t processed	8.86
G&A Cost	C\$/t processed	3.9

Figure 15-1 presents the pit optimization geometry. A series of pit optimizations were run based on gold price sensitivities. These results can be seen in Table 15-3.



15-5

Figure 15-1

newgold[™] New Gold Rainy River Mine, Rainy River, ON, Canada newgold[™]

Pit Optimization Geometry

TABLE 15-3 PIT OPTIMIZATION RESULT

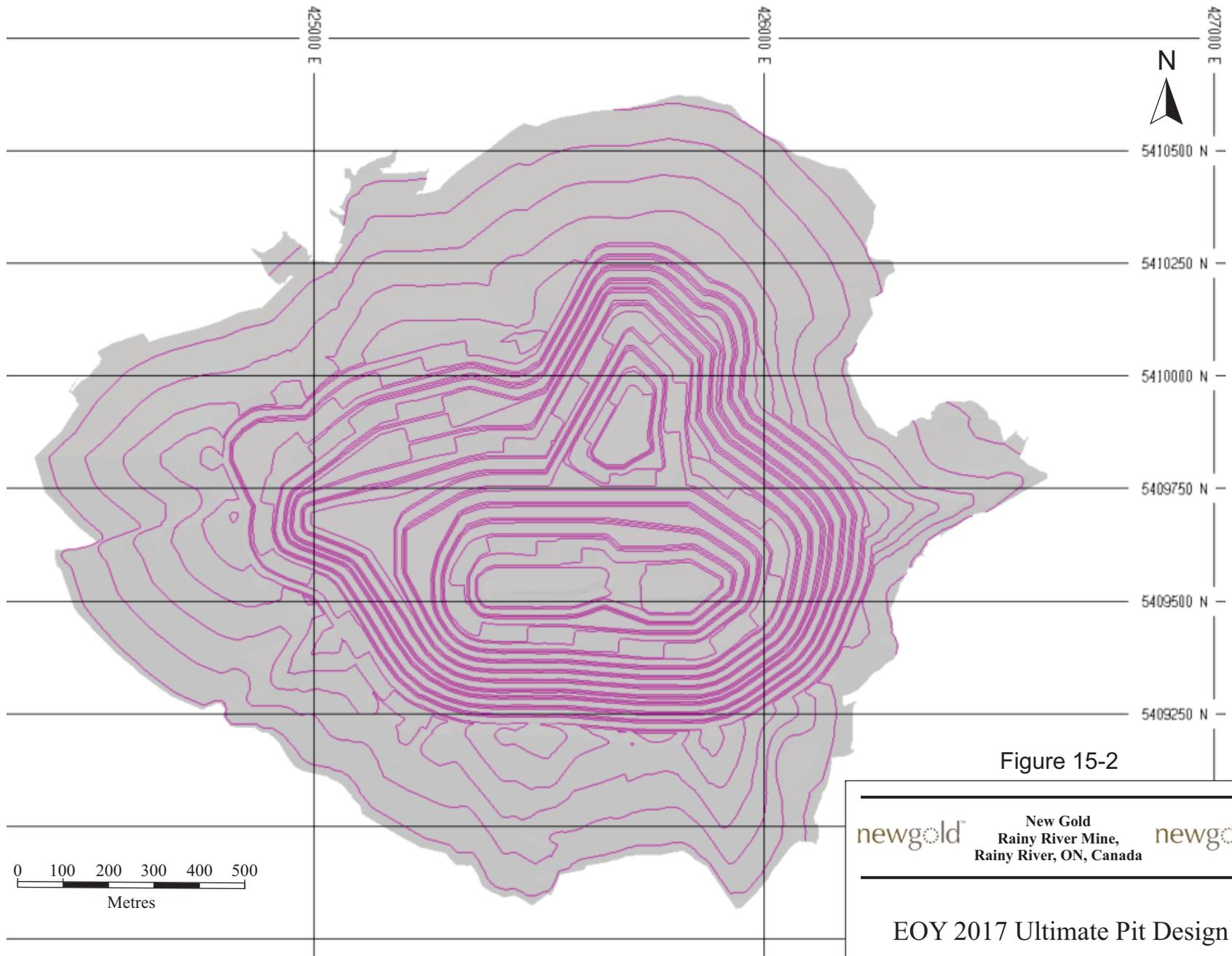
Pit #	Au Price (US\$)	Total Mined (Mt)	Waste Mined (Mt)	Process Input (Mt)	Au Grade (g/t)	Ag Grade (g/t)
1	400	21,858,472	19,266,837	2,591,635	3.62	3.24
2	450	35,426,812	30,765,595	4,661,217	3.11	2.91
3	500	38,353,294	32,651,614	5,701,680	2.86	2.79
4	550	38,806,321	32,256,136	6,550,185	2.65	2.70
5	600	56,108,227	47,160,738	8,947,489	2.46	2.59
6	650	60,151,390	49,758,630	10,392,760	2.30	2.53
7	700	78,426,240	65,807,517	12,618,723	2.20	2.60
8	750	115,211,240	98,210,832	17,000,408	2.05	3.03
9	800	149,524,909	128,053,144	21,471,765	1.94	2.83
10	850	269,449,435	237,156,642	32,292,793	1.83	2.77
11	900	276,130,056	241,033,743	35,096,313	1.76	2.73
12	950	280,703,581	242,969,100	37,734,481	1.70	2.71
13	1,000	289,674,711	248,965,259	40,709,452	1.64	1.57
14	1,050	294,618,897	251,371,078	43,247,819	1.59	2.76
15	1,100	304,876,554	258,684,814	46,191,740	1.54	2.79
16	1,150	312,332,187	263,679,829	48,652,358	1.50	2.77
17	1,200	315,148,048	264,167,166	50,980,882	1.46	2.76
18	1,250	348,719,434	293,470,109	55,249,325	1.42	2.81
19	1,275	456,847,838	391,946,260	64,901,578	1.37	3.13
20	1,300	471,239,523	404,199,225	67,040,298	1.36	3.12
21	1,350	476,639,389	406,627,322	70,012,067	1.32	3.10
22	1,375	479,401,957	408,042,544	71,359,413	1.31	3.09
23	1,400	494,017,261	420,314,750	73,702,511	1.29	3.08
24	1,450	498,466,651	422,130,802	76,335,849	1.57	3.08
25	1,500	605,723,524	519,083,566	86,639,958	1.23	3.06

It can be noted that pit shells after pit 19 will continue increasing both the process input and the total waste mined. As shown in Table 15-3, however, pit 19 (generated at a revenue factor of 1.0) was selected to maximize the net revenue processing material that adds economic value to the project under current assumptions. Analyzing the Specified Case, the net revenue does not increase significantly after pit 19.

RESERVE PIT DESIGN

Pit 19 was selected as a guideline to design the smooth pit at a revenue factor of 1.0 on gold (US\$1,275/oz Au) and silver price (US\$17/oz Ag). An updated pit slope design recommendation from BGC (BGC, 2017) was used to design a new ultimate pit design. Figure

15-2 shows the end of year (EOY) 2017 ultimate pit design. The reserve pit spans approximately 1,700 m east to west and 1,400 m north to south and has a maximum depth of approximately 400 m. The total material within this pit, including waste, is approximately 511 million tonnes.



UNDERGROUND MINERAL RESERVE ESTIMATES

Underground Mineral Reserves were estimated by the application of mine development and stoping plans to convert the Indicated Mineral Resources to Probable Mineral Reserves. The underground Mineral Reserve estimates are based upon the use of mechanized long hole stoping with a combination of open stopes and backfilled stopes.

DILUTION

Stope dilution within mining shapes (planned dilution) has been assessed using the empirical estimation of wall slough after Clark (2002). Relative to the ore zone, stope hangingwall, dip, and depth, a 0.25 m footwall and 0.5 m hangingwall of Equivalent Linear Overbreak/Sloughs (ELOS) was estimated, for a sublevel spacing of 20 m and a strike length of 20 m. This estimate has been reviewed and further substantiated by more recent work, also using the ELOS method, and results from 2D and 3D modelling.

Cemented aggregate backfill dilution from stope walls (and floors where mucking on a fill floor) has been estimated at 0.3 m based on bench-marking to similar operations.

Average overall unplanned stope dilution, inclusive of backfill dilution, has been estimated at 4.3%.

Development dilution has been estimated as 0.1 m of overbreak.

EXTRACTION RATIO

Based upon recent geotechnical review and update of long hole open stope designs, and assessment of allowable spans and related rib-pillar requirements, mining extraction in each zone is projected as shown in Table 15-4.

TABLE 15-4 UNDERGROUND DESIGN EXTRACTION

Zone	Design Extraction
Intrepid Upper	86.6%
Intrepid Lower	95.4%
17E	95.0%
ODMW	92.8%
ODME	97.7%
ODMM	96.0%

In addition, the mining recovery of sill pillars, where such are needed, is estimated to be 50%.

A mucking extraction of 95% has been applied to the estimates, resulting in an overall average mining recovery factor of 92%.

The minimum mining width in the stope design was 3.5 m.

CUT-OFF GRADE

The underground cut-off grade was calculated using the 2018 pre-budget cost data and referencing metal prices and exchange rate as summarized below in Table 15-5. A cut-off grade of 2.2 g/t AuEq was used for the estimation of Mineral Reserves. Previous mine plans used a cut-off grade that was higher than a breakeven cut-off grade. For the current mine plan, it was determined that at the higher cut-off grade (3.5 g/t AuEq) there was insufficient feed for an optimum economic return. It was also noted that, using 2018 budget projections, a breakeven cut-off grade of 1.7 g/t AuEq could be considered; however, the underground tonnage would extend beyond the open pit life and underground mining would have to account for the site overhead costs for that period, which would not provide an optimum economic scenario.

A cut-off grade of 0.53 g/t AuEq was used for material encountered in development. This is based on incremental mining for material which must be mined and brought to surface.

TABLE 15-5 CUT-OFF GRADE CALCULATION

Parameter	Unit	Pre-budget 2018	COG Used	2018 Budget
Gold Price	US\$/oz	1,275	1,275	1,326
Exchange Rate	C\$:US\$	0.77	0.77	0.80
Gold Price	C\$/oz	1,657.5	1,657.5	1,658
Gold Recovery		100%	95%	95%
Mining Cost	C\$/t	90.84	90.84	72.88
Processing Cost	C\$/t	11.41	11.41	11.41
G&A Cost	C\$/t	1.25	1.25	1.25
Royalties	C\$/t	3.11	3.11	2.05
Total Cost	C\$/t	106.61	106.61	87.59
Break even Cut-off Grade	g/t AuEq	2.1	2.2	1.8

The cut-off grade is expressed as a AuEq grade. The factors in the equivalence calculation are:

- Gold price \$1,275/oz
- Silver price \$17.00/oz
- Gold recovery 95%
- Silver recovery 60%.

The cut-off grade calculation is considered to be appropriate for the deposit based upon the assumptions used. There may be an opportunity to increase the underground tonnage by future reductions in the cut-off grade and after consideration of the strategy for the mine as a whole.

Metal prices used for Mineral Reserves have been adopted after consideration of recent market values, long term forecasts from banks, financial institutions and other sources, and also considering prices used in recent NI 43-101 Technical Reports. For Mineral Resources, metal prices used are slightly higher than those for Mineral Reserves.

CONVERSION OF MINERAL RESOURCES TO MINERAL RESERVES

Table 15-6 outlines the proportion of total Measured and Indicated Mineral Resources that has been converted to Mineral Reserves in terms of contained gold ounces.

**TABLE 15-6 MINERAL RESOURCE TO MINERAL RESERVE
CONVERSION RATE**

Mineral Resources	Contained Gold (000 oz)	Mineral Reserves	Contained Gold (000 oz)	Conversion Rate
DIRECT PROCESSING				
Open Pit				
Measured	926	Proven	842	91%
Indicated	<u>3,054</u>	Probable	<u>1,860</u>	61%
O/P M&I	3,980	O/P P&P	2,701	68%
Underground				
Measured	-	Proven	-	-
Indicated	<u>1,544</u>	Probable	<u>1,021</u>	66%
U/G M&I	1,544	U/G P&P	1,021	66%
Total Direct Processing M&I	5,523	Total Direct Processing P&P	3,722	67%
LOW GRADE				
Open Pit				
Measured	228	Proven	189	69%
Indicated	<u>643</u>	Probable	<u>308</u>	48%
O/P Low Grade M&I	871	O/P Low Grade P&P	497	57%
Total M&I	6,394	Total P&P	4,219	66%

16 MINING METHODS

The Rainy River Mine is an open pit and underground gold-silver mining project. The open pit mine is a conventional truck and shovel open pit mining operation, which utilizes hydraulic shovels and 220-tonne trucks as the primary mining equipment. The open pit mine operates at a rate of 180,000 tpd of ore and waste and has an overall strip ratio of 3.7:1.0 (W:O). The underground mine is a mechanized ramp access mine that will use long hole stoping to exploit the underground Mineral Reserves at a rate of approximately 2,300 tpd after 2021.

OPEN PIT MINING

GEOTECHNICAL ASSESSMENTS

A geotechnical assessment evaluating the geotechnical slope stability for the open pit and stockpiles was carried out on Rainy River in June 2017 by BGC (BGC, 2017). Recommended pit slope angles from this geotechnical assessment are presented in Figure 16-1 and Table 16-1.

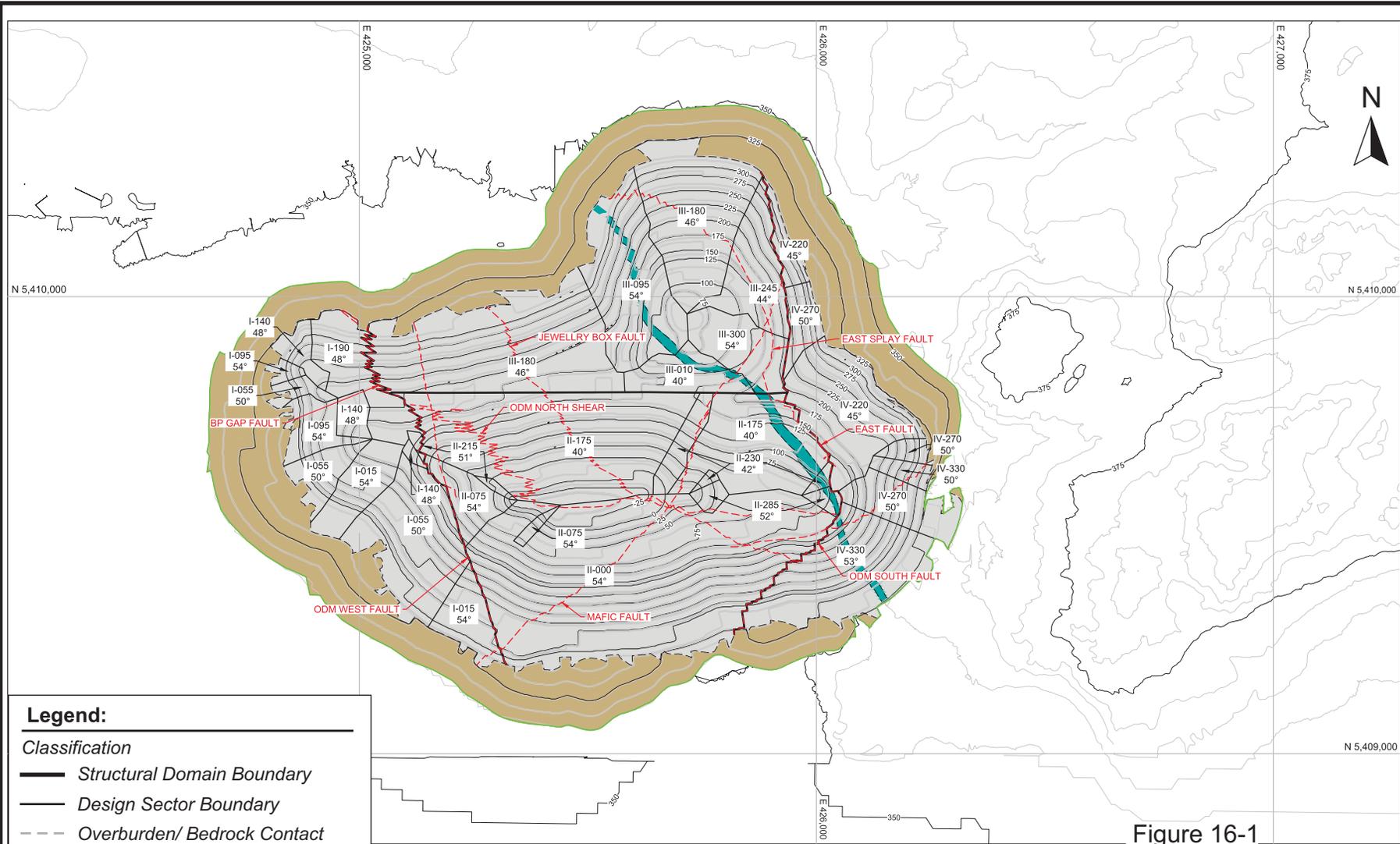


Figure 16-1

Legend:

- Classification**
- Structural Domain Boundary
 - Design Sector Boundary
 - Overburden/ Bedrock Contact
 - Pit Crest
 - Fault
- IV-270 Kinematic Sector ID**
- 50 ° Recommended Inter-Ramp Angle
 - Overburden
 - Diabase Dike
 - Bed Rock

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Rainy River, ON, Canada

BGC Geotechnical Domain

TABLE 16-1 BGC PIT SLOPE RECOMMENDATIONS

Kinematic Sector	Slope Dip Direction		Slope Azimuth		IRA (°)	BGC Recommended Slope Design			
	Start (°)	End (°)	Start (°)	End (°)		BFA (°)	Effective BFA (°)	Bench Width (m)	Bench Height (m)
I-015	000	030	180	210	54	73	70	10.5	30
I-055	030	075	210	255	50	73	70	14.5	30
I-095	075	110	255	290	54	73	70	10.5	30
I-140	110	170	290	350	48	73	70	16	30
I-190	170	210	350	030	48	65	62	11	30
II-000	320	040	140	220	54	73	70	10.5	30
II-075	040	110	220	290	54	73	70	10.5	30
II-125	110	140	290	320	51	73	70	13.5	30
II-175	140	210	320	030	40	65	62	20	30
II-230	210	250	030	070	42	65	62	17.5	30
II-285	250	320	070	140	52	70	67	10.5	30
III-010	330	045	150	225	40	73	70	25	30
III-095	045	140	225	320	54	73	70	10.5	30
III-180	140	220	320	040	46	65	62	13	30
III-245	220	270	040	090	44	65	62	15	30
III-300	270	330	090	150	54	73	70	10.5	30
IV-220	180	255	000	075	45	67	64	15.5	30
IV-270	255	285	075	105	50	67	64	10.5	30
IV-330	285	020	105	200	53	73	70	11.5	30

PIT PHASES DESIGN

Pit phases are designed by selecting the highest grade first, limiting the stripping ratio and following minimum mining width constraints. By selecting the highest grade first, metal content in early years is maximized which provides the best sequence for the discounted cash flow. The phase sequence was derived using Whittle pit shells as a guideline, starting from lower revenue factors, maintaining minimum mining width, and keeping continuity in the bench progression.

Phase designs, including access ramps, were developed using MineSight 3D. The final pit phase designs are presented in Figure 16-2.

The final pit design includes 35 m ramp widths graded by 10%. The glacial till has slopes of 8:1 (H:V), and the harder rock zones vary with inter-ramps slopes of 40° to 54°, which results in overall slope angles ranging from 34° to 48°.

Figure 16-3 shows the mine rock stockpile locations. The mine rock stockpiles have sufficient capacity to accommodate the 402 million tonnes of waste required by the mine production schedule. There is also capacity for 15 million tonnes as backfill.

Table 16-2 presents the mine rock stockpile capacity based on 1.8 t/m³ loose density. Waste dump designs are based on 20 m berms for every 50 m lift and face angles of 35°. Dumps are located within the economic zone boundary and at a distance of more than 100 m from the final pit design.

TABLE 16-2 MINE ROCK STOCKPILE CAPACITY

Rock Stockpile	Volume (Mm³)	Capacity (Mt)
West	75	164
East	75	164
Overburden	50	75
Total	200	403

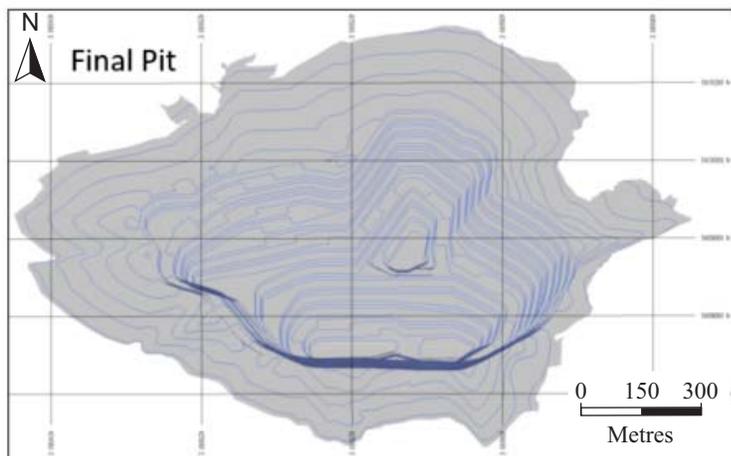
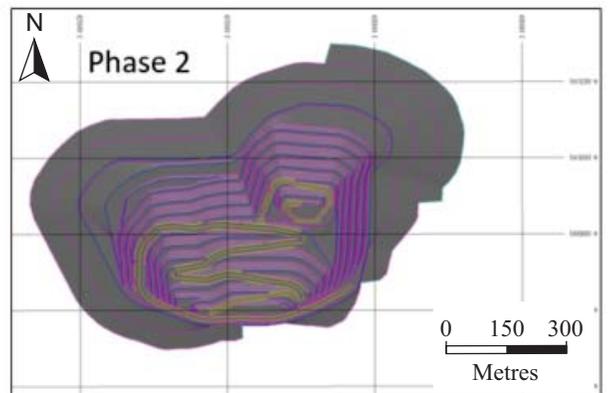
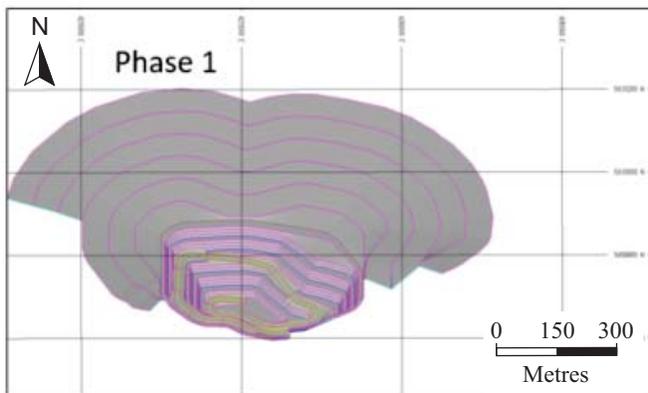
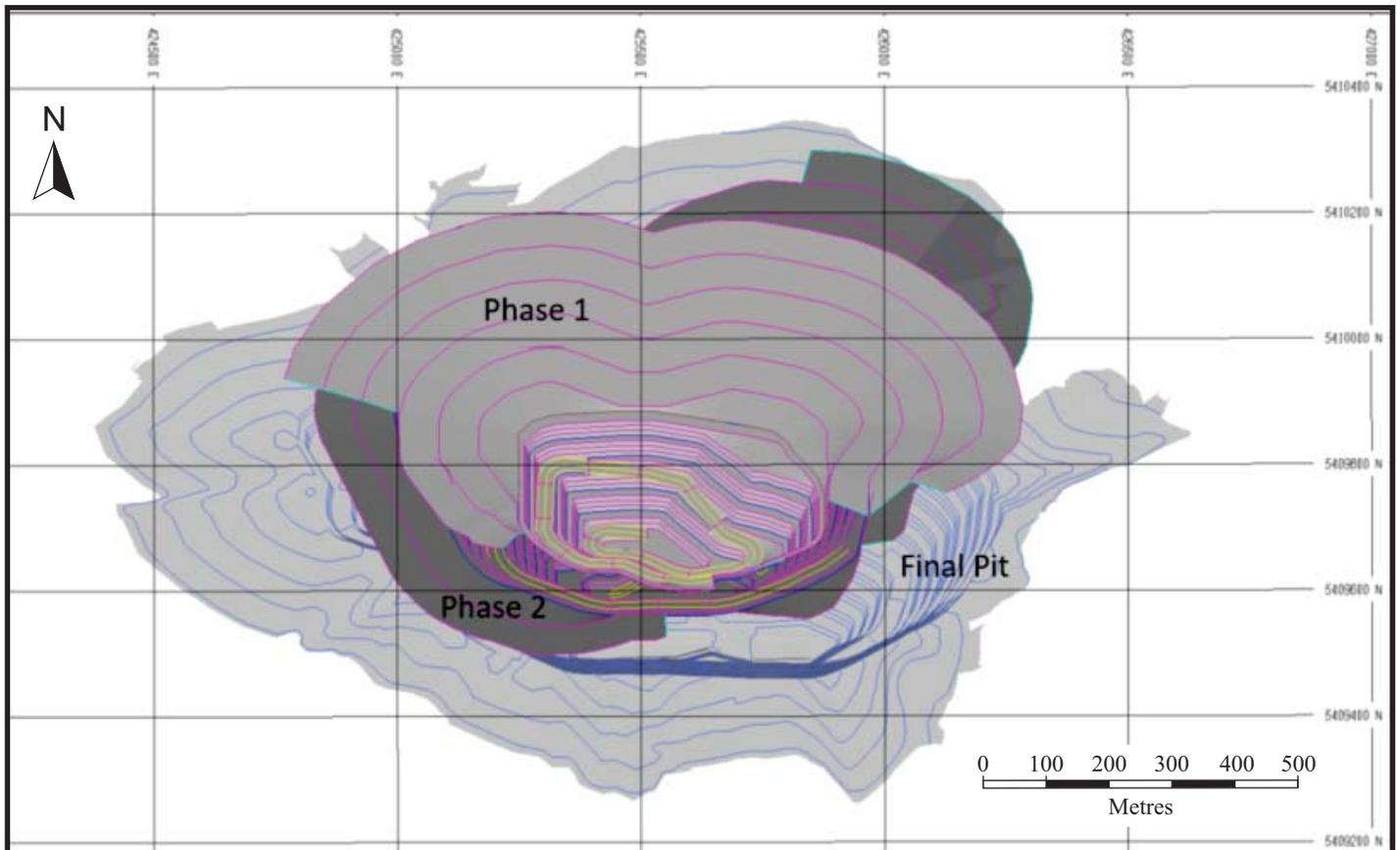


Figure 16-2

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 Rainy River, ON, Canada newgold™

Pit Phase Design

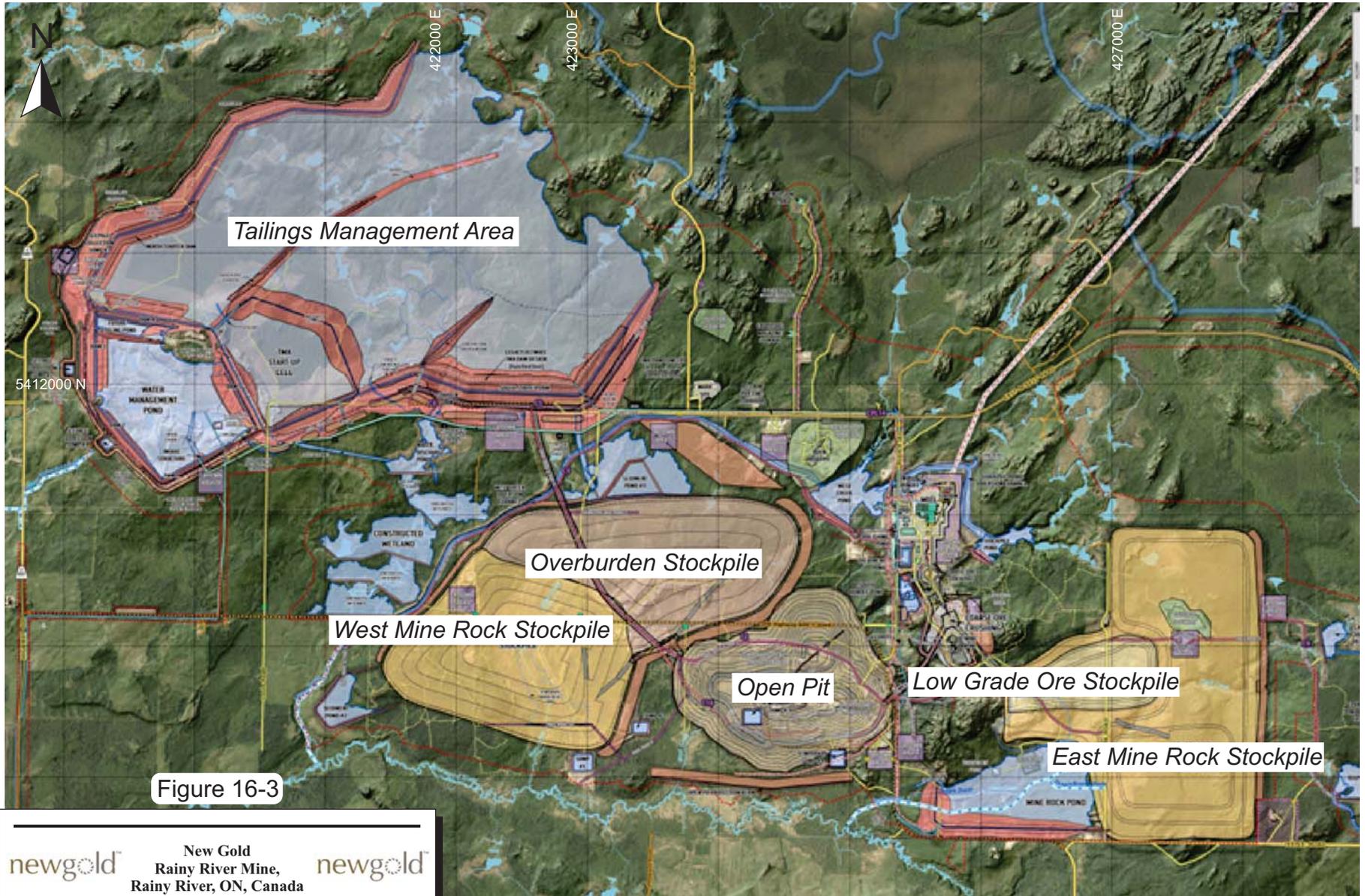


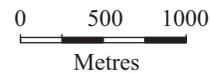
Figure 16-3

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Rainy River Mine,
Rainy River, ON, Canada

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Final Pit Design and
Mine Rock Stockpiles



PRODUCTION SCHEDULE

Mine production was scheduled to be carried out at a maximum mining rate ranging from 125,000 tpd to 180,000 tpd of total material. Stripping ratios are expected to average 3.69 over the LOM plan.

The open pit is scheduled to be mined out by 2026. Mill production will continue with underground and stockpile feed until 2032.

Table 16-3 presents the mine production schedule and Table 16-4 presents the processing plant production schedule.

TABLE 16-3 OPEN PIT MINE PRODUCTION SCHEDULE

Year	Ore Tonnes (000s)	Grade		Contained Metal		Waste Tonnes (000s)	Total Tonnes (000s)	Strip Ratio W:O
		Au (g/t)	Ag (g/t)	Gold (oz)	Silver (oz)			
2018	7,916	1.01	1.96	256,000	498,000	19,653	27,569	2.48
2019	8,903	0.89	1.84	256,000	527,000	50,357	59,260	5.66
2020	14,794	0.89	2.34	424,000	1,115,000	57,173	71,967	3.86
2021	16,103	1.13	2.20	587,000	1,141,000	53,297	69,400	3.31
2022	11,966	0.71	3.90	272,000	1,499,000	57,523	69,489	4.81
2023	9,882	0.65	3.94	206,000	1,253,000	58,004	67,886	5.87
2024	11,478	0.79	3.65	291,000	1,347,000	56,267	67,744	4.90
2025	25,393	1.00	2.48	816,000	2,021,000	43,490	68,884	1.71
2026	-	-	-	-	-	-	-	-
2027	-	-	-	-	-	-	-	-
2028	-	-	-	-	-	-	-	-
2029	-	-	-	-	-	-	-	-
2030	-	-	-	-	-	-	-	-
2031	-	-	-	-	-	-	-	-
Total OP	106,435	0.91	2.75	3,107,000	9,401,000	395,764	502,199	3.72

Notes:

1. Totals may not add due to rounding.

TABLE 16-4 PROCESS PRODUCTION SCHEDULE

Year	Mill Feed			Ore Rehandle			UG Direct Feed			OP Direct Feed		
	Tonnes	Au (g/t)	Ag (g/t)	Tonnes	Au (g/t)	Ag (g/t)	Tonnes	Au (g/t)	Ag (g/t)	Tonnes	Au (g/t)	Ag (g/t)
2018	4,080,000	1.32	2.22	166,612	1.29	2.03				3,913,388	1.32	2.23
2019	8,857,667	1.05	2.12	3,044,497	0.79	2.02	59,855	2.96	-	5,753,315	1.17	2.20
2020	8,942,500	1.27	2.82				166,711	3.29	23.75	8,775,789	1.23	2.42
2021	9,307,500	1.77	2.75				497,852	3.15	11.41	8,809,648	1.70	2.26
2022	9,307,500	1.08	4.72	1,406,166	0.62	2.12	699,602	3.46	12.74	7,201,732	0.94	4.45
2023	9,307,500	1.22	4.81	1,565,423	0.62	2.12	856,817	3.47	12.41	6,885,260	1.08	4.48
2024	9,307,500	0.98	4.00	2,956,051	0.62	2.12	840,103	2.90	7.94	5,511,345	0.89	4.42
2025	9,307,500	1.90	3.48				837,901	3.66	7.21	8,469,599	1.72	3.11
2026	9,307,500	1.14	3.12	8,470,549	0.88	2.50	836,951	3.81	9.33			
2027	9,307,500	0.71	2.47	8,459,015	0.41	2.25	848,485	3.69	4.68			
2028	9,307,500	0.68	2.78	8,462,003	0.37	2.24	845,497	3.76	8.19			
2029	9,307,500	0.68	3.12	8,413,408	0.37	2.24	894,092	3.59	11.46			
2030	9,307,500	0.70	2.78	8,426,771	0.37	2.24	880,729	3.83	7.97			
2031	5,635,947	0.71	2.80	5,057,139	0.37	2.24	578,808	3.67	7.69			
Total/Avg	120,591,114	1.09	3.20	56,427,633	0.50	2.25	8,843,404	3.55	9.31	55,320,077	1.29	3.18

Notes:

- Totals may not add due to rounding.

The mine operates on a two 12-hour shift per day schedule, 7 days per week for a total of 14 shifts per week.

MINE EQUIPMENT

Mine equipment requirements were developed from the annual mine production schedule, based on the mine operation schedule, equipment availability, and equipment productivities. The primary mine equipment fleet includes 42 m³ and 30 m³ hydraulic shovels, 18 m³ wheel loaders, 220-tonne class haul trucks, and 229 mm and 171 mm diameter track-mounted rotary drills.

Equipment productivities were determined for drills, shovels, and loaders. Haul truck productivity was dependent on annual cycle times. Production hours were calculated for the trucks, loaders, and support equipment. Annual operating requirements, such as fleet size, fleet utilization, and labor requirements, were then output from the production hours. Annual operating requirements for auxiliary equipment were based on haul truck hours for graders and water trucks, while dozer support was based on operating shifts and loader hours. A summary of the total fleet requirements for the major mine equipment is presented in given in Table 16-5.

TABLE 16-5 MAJOR MINE EQUIPMENT REQUIREMENTS

Fleet	2018	2019	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029-2032
Komatsu 830E	21	24	24	25	25	25	28	29	25	3	3	3
Komatsu PC5500	2	2	2	2	2	2	2	2	2	0	0	0
Komatsu PC8000	1	1	1	1	1	1	1	1	1	0	0	0
Komatsu PC3000	1	1	1	1	1	1	1	1	1	0	0	0
Komatsu WA1200	1	1	1	1	1	1	1	1	1	1	1	1
Komatsu WA900	1	1	1	1	1	1	1	1	1	0	0	0
Sandvik DR580	2	2	2	2	2	2	2	2	2	0	0	0
Sandvik DR461i	2	3	3	3	3	3	3	3	2	0	0	0
Sandvik DP1500i	2	2	2	2	1	1	1	1	1	0	0	0
Komatsu D475	2	2	2	2	2	2	2	1	1	0	0	0
Caterpillar D10T	3	3	3	3	3	3	2	2	2	0	0	0
Caterpillar D9T	5	5	5	5	5	5	3	3	3	2	2	0
Caterpillar D8T	2	2	2	2	0	0	0	0	0	0	0	0
Caterpillar 16M	2	2	2	2	2	2	2	2	2	1	1	1
Caterpillar 24M	1	1	1	1	1	1	1	1	1	0	0	0
Komatsu HD785 H ₂ O	2	2	2	2	2	2	2	2	2	2	2	2
Komatsu WD600	1	1	1	1	1	1	1	1	1	0	0	0

UNIT OPERATIONS AND PRODUCTIVITY

DRILLING AND BLASTING

The blast hole drills consist of a fleet of crawler-mounted diesel-powered units. The Sandvik 460i drills use a 229 mm drill bit, while the Sandvik 580 drills use a 171 mm drill bit. The Sandvik DP1500i is used for controlled blasting.

The Sandvik D460i drills a 6.2 m by 7.2 m drill pattern and the Sandvik D580 drills a 5.8 m by 4.9 m drill pattern. All areas are sampled, however, till and overburden are not blasted. Powder factors are 0.26 kg/t to 0.27 kg/t.

The drill productivity and production tonnage were used to calculate the number of hours required in a given time period. Drill utilization was not allowed to exceed 80% to reflect lost time during a shift for blast moves.

LOADING

The primary mine loading fleet consists of 30 m³ and 42 m³ capacity hydraulic shovels and 18 m³ wheel loaders.

Hydraulic Shovel (30 m³)

There are two Komatsu PC5500 diesel-powered hydraulic shovels equipped with standard 30 m³ rock buckets which are used to load material into rear dump haul trucks. Four passes are used to load the 220-tonne trucks. Hourly production is 1,900 tph.

Hydraulic Shovel (42 m³)

One Komatsu PC8000 diesel-powered hydraulic shovel equipped with standard 42 m³ rock bucket is used to load material into rear dump haul trucks. Three passes are used to load the 220-tonne trucks. Hourly production is 2,900 tph.

Wheel Loader (18 m³)

One Komatsu WA1200 rubber-tired front-end loader equipped with a standard 18 m³ rock bucket is used to load the fleet of 220-tonne capacity haul trucks. Hourly production is 1,000 tph. Six to eight passes are used to load the 220-tonne trucks. Wheel loaders provide flexibility and mobility for mining and also provide pit wall clean when required.

HAULING

A single haulage fleet consisting of Komatsu 830E electric drive rear-dump haul trucks in the 220-tonne payload class are used to minimize mining costs while still providing selectivity. The trucks match up with the 30 m³ and 42 m³ class hydraulic shovels with a nominal four and three passes per truck, respectively, and with the 18 m³ class wheel loaders with a nominal six passes per truck. The rated payload capacity is a nominal 210 dry tonnes.

Haulage requirements were calculated based on an average annual truck cycle time. Cycle times were calculated based the annual production requirement. Haulage profiles were calculated for each bench based on the reserves for the bench and the destination of the material. Separate profiles were measured for each material type and destination. Truck cycle times provided the input to determine the number of truck hours required.

Haul road widths are presently designed at 35 m.

UNDERGROUND MINING

In addition to the open pit Mineral Reserves, there are Mineral Reserves at Rainy River amenable to underground mining. Underground Mineral Reserves have been identified in ten zones. Several of the zones are extensions of the open pit ore bodies while others are up to one kilometre from the open pit. The underground deposits exist from near surface to approximately 600 m below surface within an area approximately 2.8 km long and 0.9 km in width.

The ore grade mineralization occurs in subvertical horizons from three metres to 20 m thick. Widths over 15 m are rare and the weighted average thickness is eight metres. A three metre minimum width is used for Mineral Reserves. The ore zones generally dip at 60° or greater but can flatten locally to 45°. The planned mining method relies upon gravity ore flow along the footwall. In areas where the dip is less than 55° there is the potential for higher costs, higher dilution and/or higher ore loss than planned. As detailed stope planning information becomes available, footwall geometry is planned to be reviewed.

The underground Mineral Reserves are summarized in Table 16-6 and a schematic view of the planned underground is shown in Figure 16-4.

TABLE 16-6 UNDERGROUND MINERAL RESERVES AS AT JUNE 30, 2018

Zone	Tonnes (000)	Au (g/t)	Ag (g/t)	Au (oz)	Ag (oz)
Intrepid Upper	777,582	3.72	28.56	93,062	713,940
Intrepid Lower	689,188	3.72	17.77	82,396	393,821
433	891,710	4.38	1.49	125,603	42,803
ODM East	919,462	3.22	5.46	95,215	161,372
ODM Upper	1,948,459	4.02	2.29	251,606	143,737
ODM Lower	2,362,274	3.23	2.38	245,285	180,924
ODM West	279,581	3.37	7.07	30,269	63,529
17E Lower	1,071,420	2.80	29.65	96,326	1,021,375
17E Upper	13,996	2.89	14.64	1,298	6,587
Total	8,953,674	3.55	9.48	1,021,060	2,728,088

Notes:

1. CIM (2014) definitions were followed for Mineral Reserves.
2. Effective date of Reserves is June 30, 2018; estimated by Herbert Smith, P. Eng., an employee of AMC.
3. Underground Mineral Reserves were estimated at a cut-off grade of 2.2 g/t Au for stoping and 0.53g/t for development. UG cut-off grade assumptions:
 - Au price US\$1,275 per troy ounce, Ag price US\$17 per troy ounce
 - Exchange rate 1.3 C\$: 1 US\$ (1 C\$ = 0.77 US\$)
 - Planned hangingwall and footwall dilution of 0.5 m and 0.25 m respectively; unplanned dilution 4.3%
 - Average mining recovery: 92%
 - Mill Au and Ag recovery: 95% and 60% respectively
 - Breakeven cost of \$107/t of mill feed, inclusive of costs for mining, processing, G&A, refining & transport, royalties and sustaining capital allowance.
4. Totals may not add due to rounding.
5. Tonnes and grades are in metric units.

MINE DESIGN

The Feasibility Study mine design was generated by AMC and subsequently revised in 2016 to reflect a project wide re-survey of all drill hole information and the addition of new drilling. The current mine designs were prepared by New Gold with a lower cut-off grade applied in the stope design. AMC is of the opinion that the current underground mine designs are appropriate.

The use of backfill has been significantly reduced in the current mine design compared to the Feasibility Study mine designs. Additionally, the stoping plan has been converted to focus on longitudinal long hole open stoping with pillars in some areas and with rock fill and cemented aggregate fill in other areas. Fill will be used in approximately half of the voids created by ore

extraction. The fill is planned to be 40% rock fill sourced from development waste and 60% cemented rock fill with 4% cement added as a binder. The design and plan are based upon the use of open stoping for areas less than 10 m wide and filled stopes for areas greater than 10 m wide.

Cavity monitoring surveys are recommended for all stopes as a part of the production management and reconciliation process. The monitoring of pillars in the open stopes and the open stope hangingwalls will facilitate assessment of the ongoing stability of both individual pillars and the overall hangingwall geometry.

Ore will be loaded by manual and remote controlled 7 m³ (17 t) load, haul, dump (LHD) loaders in the stope and will then be loaded to 45 t haul trucks for transportation to surface. The underground mine will be operated at a production rate of 2,300 tpd simultaneously with the open pit over the nine-year period from 2022 to 2030. Underground ore production will commence in 2019 and be completed in 2031.

The lateral and vertical development designs are shown in Figure 16-4.

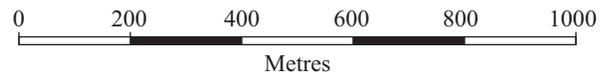
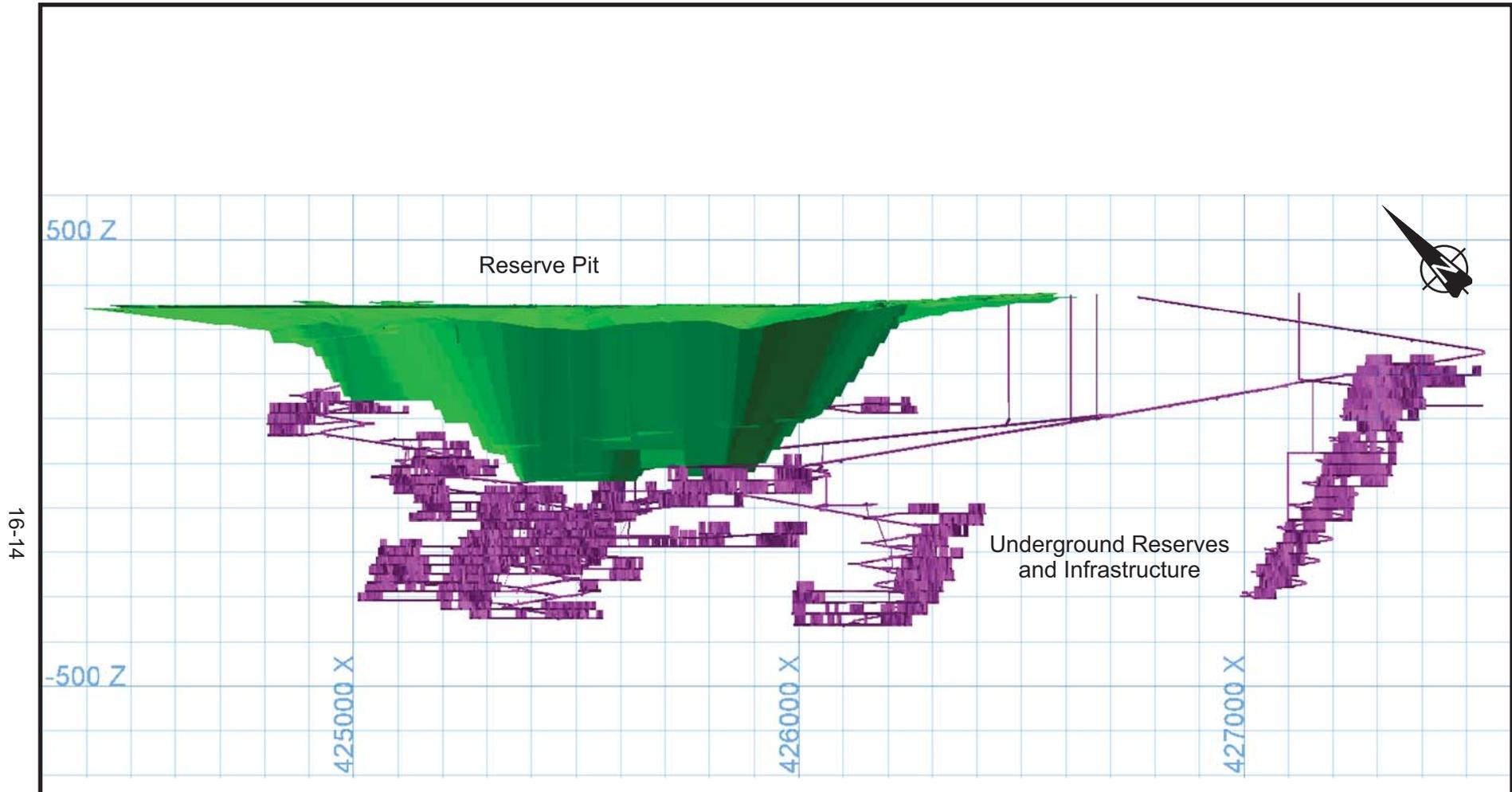


Figure 16-4

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Schematic View of the Underground Design

GEOTECHNICAL CONSIDERATIONS

Site investigations and initial Feasibility Study level design work for underground stopes, access development, ground support, and backfill were performed and reported on by AMEC (BBA, 2014) as part of the updated Feasibility Study for the Mine completed in 2014 on behalf of New Gold.

During the 2012 drilling campaign (BBA, 2014), three main zones of the ODM17 were intercepted: the West (BH12-UG-01), Central (BH12-UG-02) and East (BH12-UG-03) zones, while the 433 North zone was delineated with the deeper borehole sections of BH12-OP-05 & 06. In addition, New Gold performed orientation of cores for four boreholes in the Intrepid Zone. These holes, including other select exploration cores, were subsequently geomechanically logged by AMEC to provide supporting data for geomechanical designs (BBA, 2014).

In 2016 and 2017, BGC completed Feasibility Study level studies for the open pit (BGC, 2017); much of the data and information collected and analyzed during their study is relevant to the underground mining zone and has been incorporated into the current Feasibility Study update.

In 2017, North Rock Mining Solutions Inc. (NRMS) was retained by New Gold to assist with advancing the Mine through the mine development phase. NRMS's work scope included:

- A Site Visit and QP Inspection including geotechnical logging of select representative core intervals and mapping of open pit, quarry walls, and portal site.
- Data compilation and discussion with mine technical services.
- A detailed review of previous studies.
- A review and update of the stope designs, stability graph inputs, and assumptions.
- Review and update of the ELOS Dilution Graph inputs and assumptions.
- Review of AMEC numerical stress model and results with confirmatory 2D and 3D stress modelling of updated Feasibility Study mining shapes.
- Review and update of ground support recommendations.
- Portal design and related recommendations for initial underground development.

NRMS subsequently provided mine design criteria and recommendations to New Gold which were incorporated into the underground mine design.

UNDERGROUND GEOTECHNICS

Typical rock RQDs were calculated by AMEC and confirmed by NRMS to be excellent, ranging from 90% to 100% throughout all stoping domains. With respect to the Modified NGI Q-system, Q' (after Barton et. al., 1974, 2013), average values of 23, 17, and 19, were obtained characterizing the hangingwall, ore zone, and footwall domains of the largest west zone, respectively. Typical rock masses in the Intrepid Zone had average Q' values of 21, 22 and 17, in the hangingwall, ore zone, and footwall domains, respectively.

General rock mass conditions in all domains can be characterized as “Fair” to predominantly “Good” (Barton et. al, 1974, 2013), however, there is an apparent slight decrease in the quality in the central zone of the ODM based on the present data. Additionally, above and to the east of the Intrepid Zone, there is a zone of brecciated rock that is found to be developed in sub-horizontal structures that terminate rapidly. These also have a lower RQD in the range of 10 to 70 (average of 40), and an average Q' of approximately 4; however, mining does not intersect these zones.

A summary of Rock Mass Classification data by mining zone (Q', RMR₇₆ and GSI) is provided in Table 16-7.

TABLE 16-7 RAINY RIVER UNDERGROUND ROCK MASS CLASSIFICATION SUMMARY

Zone	Zone Length		Run (#)	Q'				RMR ₇₆				GSI			
	From (m)	To (m)		Avg	Std Dev	Min	Max	Avg	Std Dev	Min	Max	Avg	Std Dev	Min	Max
Hangingwall	0	50	49	19.2	7.5	5.8	50.3	70	4	60	79	70	6	58	82
Ore Zone	0	34*	36	16.1	6.8	5.2	36.9	68	4	59	76	70	5	54	81
Footwall	0	50	50	17.7	7.5	6.6	40.0	69	4	61	77	72	4	61	80
Ore Zone + Footwall	0	84*	86	17.0	7.2	5.2	40.0	69	4	59	77	71	5	54	81

* Average Length

Laboratory testing, consisting of 30 UCS, 27 triaxial tests and 24 Brazilian tensile tests, was used to determine strength characteristics and develop rock mass failure criteria. The overall average UCS results for the hangingwall, ore zone, footwall and ore zone + footwall, were 87

MPa, 125 MPa, 104 MPa and 114 MPa respectively, indicating strong to very strong rocks. Field assessments by NRMS confirmed these ranges as accurate and representative.

NRMS reviewed the current mine plan in consideration of the recently collected data and has provided updated stope design guidance using the same experienced-based empirical and numerical modelling techniques as the Feasibility Study. Updated permissible and optimized 'Supported' and 'Unsupported' stoping dimensions based on typical conditions (Q' 10-40) and lower rock quality, moderately adverse conditions (Q' 3-10) are provided in Tables 16-8 and 16-9, representing the rock classes anticipated underground.

The results of NRMS's assessment indicate that increased and optimized stope dimensions compared to those used in the Feasibility Study mine plan may be viable, including 25 m to 30 m level spacing, 25 m to 30 m stope strike lengths, or 20 m (h) x 40 m (l) stopes or similar alternatives. These potential opportunities further support the feasibility of the current designs.

TABLE 16-8 PERMISSIBLE/OPTIMIZED SUPPORTED AND UNSUPPORTED STOPPING DIMENSIONS - TYPICAL ROCKMASS CONDITIONS

Hydraulic Radius	Unsupported		Supported		
	Q'=10	Q'=40	Q'=10	Q'=40	
BACK	3.5	6.5	8.0	11.0	
HW	8.0	13.0	13.0	16.0	

BACK		W (metre)										
L (metre)	HYD RAD	10	15	20	25	30	35	40	45	50	55	60
	5	1.7	1.9	2.0	2.1	2.1	2.2	2.2	2.3	2.3	2.3	2.3
10	2.5	3.0	3.3	3.6	3.8	3.9	4.0	4.1	4.2	4.2	4.2	4.3
15	3.0	3.8	4.3	4.7	5.0	5.3	5.5	5.6	5.8	5.9	6.0	6.0
20	3.3	4.3	5.0	5.6	6.0	6.4	6.7	6.9	7.1	7.3	7.5	7.5
25	3.6	4.7	5.6	6.3	6.8	7.3	7.7	8.0	8.3	8.6	8.8	8.8
30	3.8	5.0	6.0	6.8	7.5	8.1	8.6	9.0	9.4	9.7	10.0	10.0
35	3.9	5.3	6.4	7.3	8.1	8.8	9.3	9.8	10.3	10.7	11.1	11.1
40	4.0	5.5	6.7	7.7	8.6	9.3	10.0	10.6	11.1	11.6	12.0	12.0
45	4.1	5.6	6.9	8.0	9.0	9.8	10.6	11.3	11.8	12.4	12.9	12.9
50	4.2	5.8	7.1	8.3	9.4	10.3	11.1	11.8	12.5	13.1	13.6	13.6
55	4.2	5.9	7.3	8.6	9.7	10.7	11.6	12.4	13.1	13.8	14.3	14.3
60	4.3	6.0	7.5	8.8	10.0	11.1	12.0	12.9	13.6	14.3	15.0	15.0

HW		W (metre)										
H (metre)	HYD RAD	10	15	20	25	30	35	40	45	50	55	60
	20	3.3	4.3	5.0	5.6	6.0	6.4	6.7	6.9	7.1	7.3	7.5
25	3.6	4.7	5.6	6.3	6.8	7.3	7.7	8.0	8.3	8.6	8.8	8.8
30	3.8	5.0	6.0	6.8	7.5	8.1	8.6	9.0	9.4	9.7	10.0	10.0
35	3.9	5.3	6.4	7.3	8.1	8.8	9.3	9.8	10.3	10.7	11.1	11.1
40	4.0	5.5	6.7	7.7	8.6	9.3	10.0	10.6	11.1	11.6	12.0	12.0
45	4.1	5.6	6.9	8.0	9.0	9.8	10.6	11.3	11.8	12.4	12.9	12.9
50	4.2	5.8	7.1	8.3	9.4	10.3	11.1	11.8	12.5	13.1	13.6	13.6
55	4.2	5.9	7.3	8.6	9.7	10.7	11.6	12.4	13.1	13.8	14.3	14.3
60	4.3	6.0	7.5	8.8	10.0	11.1	12.0	12.9	13.6	14.3	15.0	15.0
75	4.4	6.3	7.9	9.4	10.7	11.9	13.0	14.1	15.0	15.9	16.7	16.7
80	4.4	6.3	8.0	9.5	10.9	12.2	13.3	14.4	15.4	16.3	17.1	17.1

TABLE 16-9 PERMISSIBLE/OPTIMIZED SUPPORTED AND UNSUPPORTED STOPPING DIMENSIONS - LOWER QUALITY ROCKMASS CONDITIONS

Hydraulic Radius	Unsupported		Supported	
	Q'=3	Q'=10	Q'=3	Q'=10
BACK	2.5	3.5	6.0	8.0
HW	5.0	8.0	9.0	13.0

	Unsupported design for rock mass in lower end of design range.
	Unsupported design for rock mass in mid of design range.
	Supported design for rock mass in lower end of design range.
	Supported design for rock mass in mid of design range.

BACK		W (metre)											
L (metre)		HYD RAD	10	15	20	25	30	35	40	45	50	55	60
L (metre)	5		1.7	1.9	2.0	2.1	2.1	2.2	2.2	2.3	2.3	2.3	2.3
	10		2.5	3.0	3.3	3.6	3.8	3.9	4.0	4.1	4.2	4.2	4.3
	15		3.0	3.8	4.3	4.7	5.0	5.3	5.5	5.6	5.8	5.9	6.0
	20		3.3	4.3	5.0	5.6	6.0	6.4	6.7	6.9	7.1	7.3	7.5
	25		3.6	4.7	5.6	6.3	6.8	7.3	7.7	8.0	8.3	8.6	8.8
	30		3.8	5.0	6.0	6.8	7.5	8.1	8.6	9.0	9.4	9.7	10.0
	35		3.9	5.3	6.4	7.3	8.1	8.8	9.3	9.8	10.3	10.7	11.1
	40		4.0	5.5	6.7	7.7	8.6	9.3	10.0	10.6	11.1	11.6	12.0
	45		4.1	5.6	6.9	8.0	9.0	9.8	10.6	11.3	11.8	12.4	12.9
	50		4.2	5.8	7.1	8.3	9.4	10.3	11.1	11.8	12.5	13.1	13.6
	55		4.2	5.9	7.3	8.6	9.7	10.7	11.6	12.4	13.1	13.8	14.3
60		4.3	6.0	7.5	8.8	10.0	11.1	12.0	12.9	13.6	14.3	15.0	

HW		W (metre)											
H (metre)		HYD RAD	10	15	20	25	30	35	40	45	50	55	60
H (metre)	20		3.3	4.3	5.0	5.6	6.0	6.4	6.7	6.9	7.1	7.3	7.5
	25		3.6	4.7	5.6	6.3	6.8	7.3	7.7	8.0	8.3	8.6	8.8
	30		3.8	5.0	6.0	6.8	7.5	8.1	8.6	9.0	9.4	9.7	10.0
	35		3.9	5.3	6.4	7.3	8.1	8.8	9.3	9.8	10.3	10.7	11.1
	40		4.0	5.5	6.7	7.7	8.6	9.3	10.0	10.6	11.1	11.6	12.0
	45		4.1	5.6	6.9	8.0	9.0	9.8	10.6	11.3	11.8	12.4	12.9
	50		4.2	5.8	7.1	8.3	9.4	10.3	11.1	11.8	12.5	13.1	13.6
	55		4.2	5.9	7.3	8.6	9.7	10.7	11.6	12.4	13.1	13.8	14.3
	60		4.3	6.0	7.5	8.8	10.0	11.1	12.0	12.9	13.6	14.3	15.0
	75		4.4	6.3	7.9	9.4	10.7	11.9	13.0	14.1	15.0	15.9	16.7
	80		4.4	6.3	8.0	9.5	10.9	12.2	13.3	14.4	15.4	16.3	17.1

Rib Pillars have been incorporated within long hole open stoping zones to break exposed excavation spans into “permissible” and “stable” dimensions. A nominal pillar width of eight metres has been used, based on average ore thickness values and empirical and numerical pillar assessments. While this width is deemed to be suitable for the majority of stopes, these pillar dimensions can and should be reviewed, and modified if appropriate, as local mining knowledge is acquired. Optimized pillar designs may result in lower grade ore being selectively and preferentially left behind in favour of higher grade material.

2D and 3D linear-elastic and elasto-plastic finite element stress has been completed by AMEC and NRMS to review the proposed sequencing and stress evolution around the planned development, and to estimate the levels of ground support required. The models are considered suitable for studying the evolution of mining-induced stresses and displacements; however, they are not considered suitably calibrated to estimate precise magnitudes of stress, strain, or infer the relative degrees of rock mass damage resulting from failing/yielding rock (i.e. micro-cracking, spalling, on-going plastic deformation, and, in the extreme case, rock bursting).

The model results have been reviewed by NRMS with the aim of highlighting areas where: potential damage is concentrating; stress shadows (loss of confinement, or relaxation) are occurring; and standoff distances or pillar thicknesses are not sufficient to isolate some critical openings from significant mining-induced stress changes.

The modelling results are used as a guide only, to inform the design process. The models’ limited capacity arises from the current level of knowledge regarding geotechnical and hydrogeological conditions, in-situ stress conditions, local and intermediate-scale structural features, and geotechnical ‘zoning’, etc. All of these factors can significantly constrain the extent to which the model can be used to accurately predict local and global rock mass behaviour during the mine life.

The modelling results support the updated mine plan and the empirical analysis techniques used for stope designs. As expected, mining induced stresses tend to concentrate at various stages of the mining cycle, particularly within pillars and zone abutments, once significant spans have been developed.

These, and other locations of geotechnical interest, should be monitored closely as mining progresses, with step-wise increases to ground surveillance and monitoring programs as required, as determined by pre-defined event-triggers and responsive actions specified in the project's Ground Control Management Plan (GCMP).

As some stress induced damage may be anticipated a 32 channel microseismic system is recommended for the deeper region of stopes and the region below the pit. Additionally, standard displacement monitoring and cable bolt monitoring instrumentation is recommended to be used.

GROUND SUPPORT DESIGNS

Ground support designs have been determined as a function of anticipated ground conditions and provide an allowance for both 'Primary' and 'Secondary' classes of ground support for general mine development, representing increasing complexity depending on the location and rock conditions of an opening.

Primary ground support consists of:

- 1.8 m to 2.4 m long #6 (3/4") resin rebar or rock bolts (or equivalent 'permanent full-column' support), installed on 1.5 m centres in the back (roof) of all standard development, with the pattern extending to the excavation spring-line / shoulder.
- Bolt length is dependent on the drift width, with 1.8 m bolts for spans less than 6 m, and 2.4 m bolts required for spans greater than 6 m.
- Mid-gauge welded wire mesh or screen (6 ga. (0.192") to 9 ga. (0.144")) is to be installed in the back (roof), to the spring-line / shoulder.
- Spot bolting / localized mesh is to be installed on the walls, using 1.8 m long resin bolts and / or split sets, as required.
- Initial excavation designs require cable bolts for all spans greater than 8 m spans, particularly at depths greater than 500 m, where stress effects due to mining are indicated to become more apparent and impactful.

Lower bound geotechnical data suggests 'Secondary' support may be required in localized areas of 'fair' to 'poor' ground. Secondary support includes the following elements, progressively:

- Steel / mesh straps.
- Five centimetres to 10 cm of plain shotcrete applied to the back, walls, or other specified target areas.

- Five metre long bulbed double-strand cable bolts on 2.5 m square pattern. These may be required to support intersections and larger spans, potential wedges and blocks formed by persistent structure, fault zones, etc.

Initial review suggests a reasonable allowance of approximately 10% of development requiring secondary support.

BACKFILL

Backfill testing indicates that a 3% binder consisting of 10% cement (NPC) and 90% slag has acceptable strength development at 28 days of 1 MPa and no degeneration in strength was noticed for testing at 180 days (BBA, 2014). Cut and fill stopes will be partially backfilled with cemented aggregate fill at an average binder dosage of 4%. The cemented backfill will be placed at an angle of repose until level with the top cut, and the remainder of the stope will be filled with uncemented rock fill. The average binder dosage of 4% has been used as a conservative stand point. Continued testing to verify field performance of binder content is recommended.

DILUTION

Discussed in Section 15

OPEN PIT – UNDERGROUND INTERACTION

Achievement of the mine's full production rate is planned for 2022. Development and sequencing of stopes in the ODME long hole open stoping zone, and the 433 and Upper ODMU zones (all with stopes located under/beside the pit) have been scheduled for Q3 2026 and 2030 to 2031, respectively, after the open pit mining has been completed.

The updated plan results in several geotechnical advantages compared to the previous plan in the Feasibility Study, including a thicker crown pillar beneath the pit existing for longer, thereby minimizing the potential for water ingress from possible breakthroughs. In addition, delayed development of these zones provides long lead-in time to adequately assess local ground conditions and develop evidenced-based designs. It is noteworthy that stopes in these areas are relatively narrow and may be backfilled with cemented rock fill as required and, as such, are not anticipated to cause significant operational challenges.

MINING METHOD AND STOPE DESIGN

The selected mining method is long hole open stoping with longitudinal stopes. Stopes will be the width of the ore zone, with a minimum three metre width and sublevels will be located at 20 m to 30 m vertical intervals (floor to floor). Stopes will be mined in a longitudinal retreat sequence with open stopes up to 48 m long and with stopes separated by an eight-metre wide pillar. Sill pillars will be used as work platforms in top down mining and then recovered with the subsequent lift.

The typical stope long section is shown in Figure 16-5 and cross sections are shown in Figure 16-6.

The deposit has areas where the dip is less than 60° and refinements to the plan will be required to address potential ore loss in shallow areas versus the dilution to develop a steepened footwall in the stope.

Section A-A'
Longitudinal Down-hole
Long Section

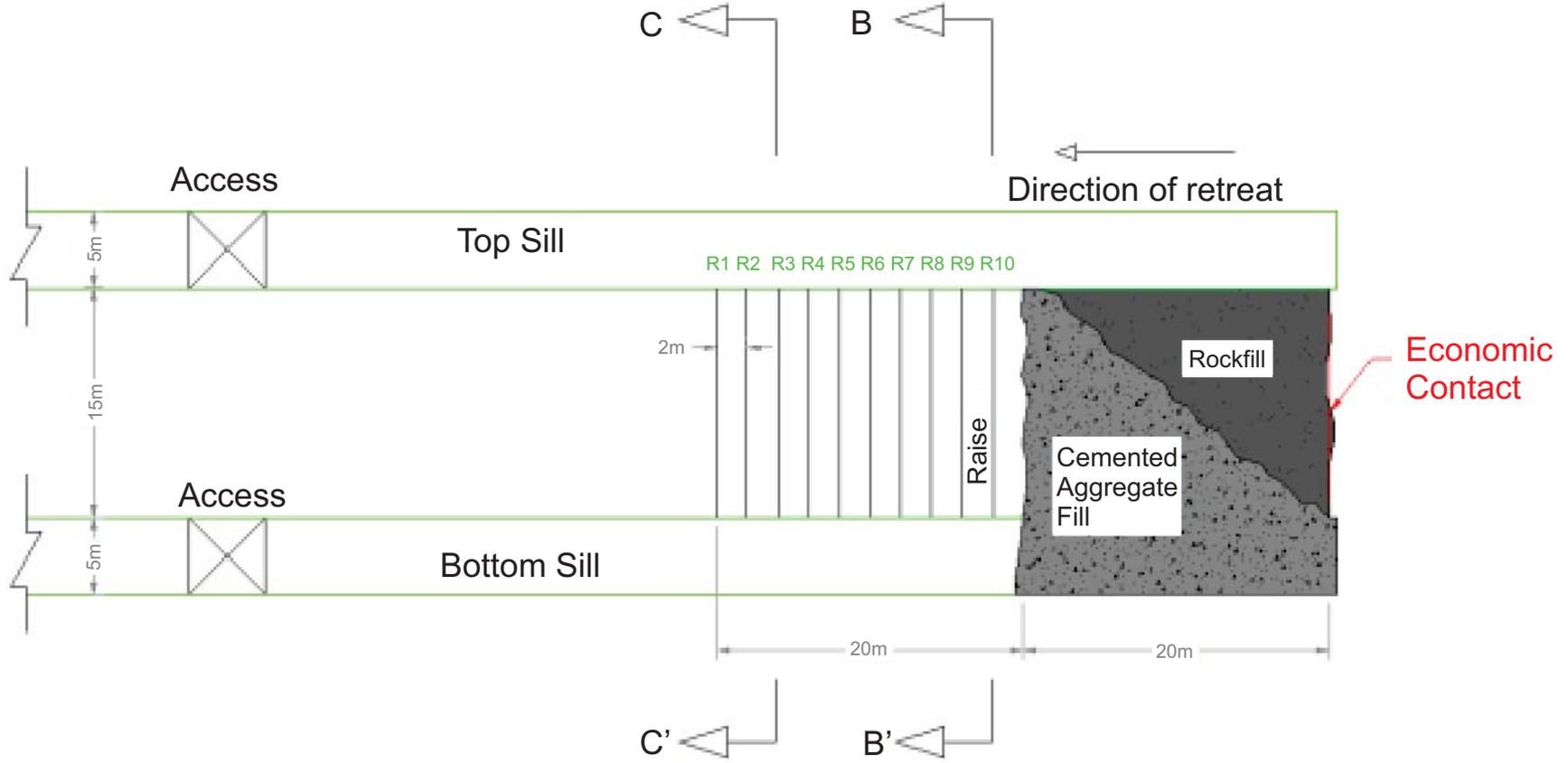


Figure 16-5

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Stope Long Section (Schematic)

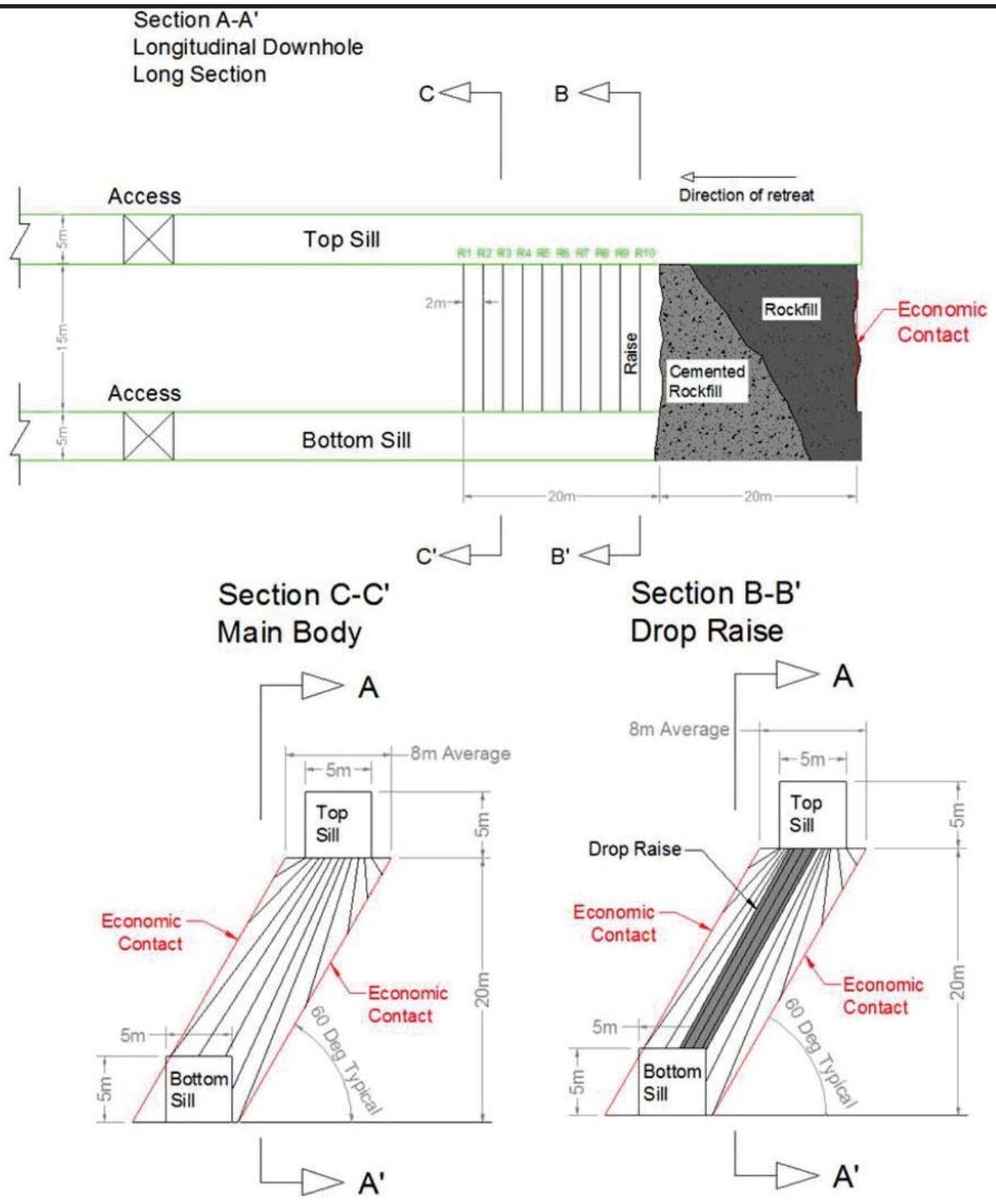


Figure 16-6

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Stope Cross Sections (Schematic)

DEVELOPMENT ACCESS

A four-kilometre long main 5.0 m x 5.0 m decline driven at a -15% gradient from the surface portal will provide access for personnel and equipment. The main decline will connect to independent internal production ramps that will service all levels in each mining area. Internal production ramps are of spiral configuration and will be generally driven at a -10% gradient to provide level access on each revolution. The ramps and ore development are planned to be typically 5.0 m x 5.0 m headings.

The mine design includes raise development to establish and extend the primary air circuit. Three raises to surface (two exhaust raises and one fresh air raise) will be required. As the mining levels are developed from the internal ramps, raising from level to level will be necessary to provide an exhaust airway. A backfill raise from surface will also be required for the delivery of aggregate to an underground cemented aggregate fill station. The ventilation raises are planned to be three metre diameter raises and the backfill raise is planned to be a two metre diameter raise.

UNDERGROUND INFRASTRUCTURE**MINE VENTILATION**

The mine ventilation system will expand over the LOM as the workings are extended. The basic ventilation system will use exhaust fans on the Intrepid and Main exhaust raises with fresh air entering the mine via the main fresh air raise and the main portal. Fresh air will be heated in winter using direct-fired heaters to keep the services from freezing and to limit ice build-up on the decline.

The mine air flow will be based upon the planned mobile equipment fleet plus service area needs and will be approximately 335 m³/s (710,000 cfm).

Two means of egress are provided for each production area of the mine (Figure 16-7). The primary means of egress is via the ramp system. Secondary emergency egress is provided in all internal raises and the Intrepid exhaust raise by means of installed ladderways. In the event of compromised egress to the surface through the main ramp, the Primary Intake Raise is fitted with an Alimak elevator.

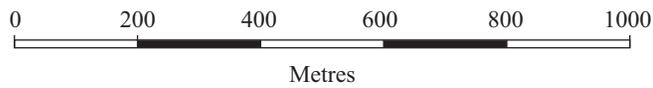
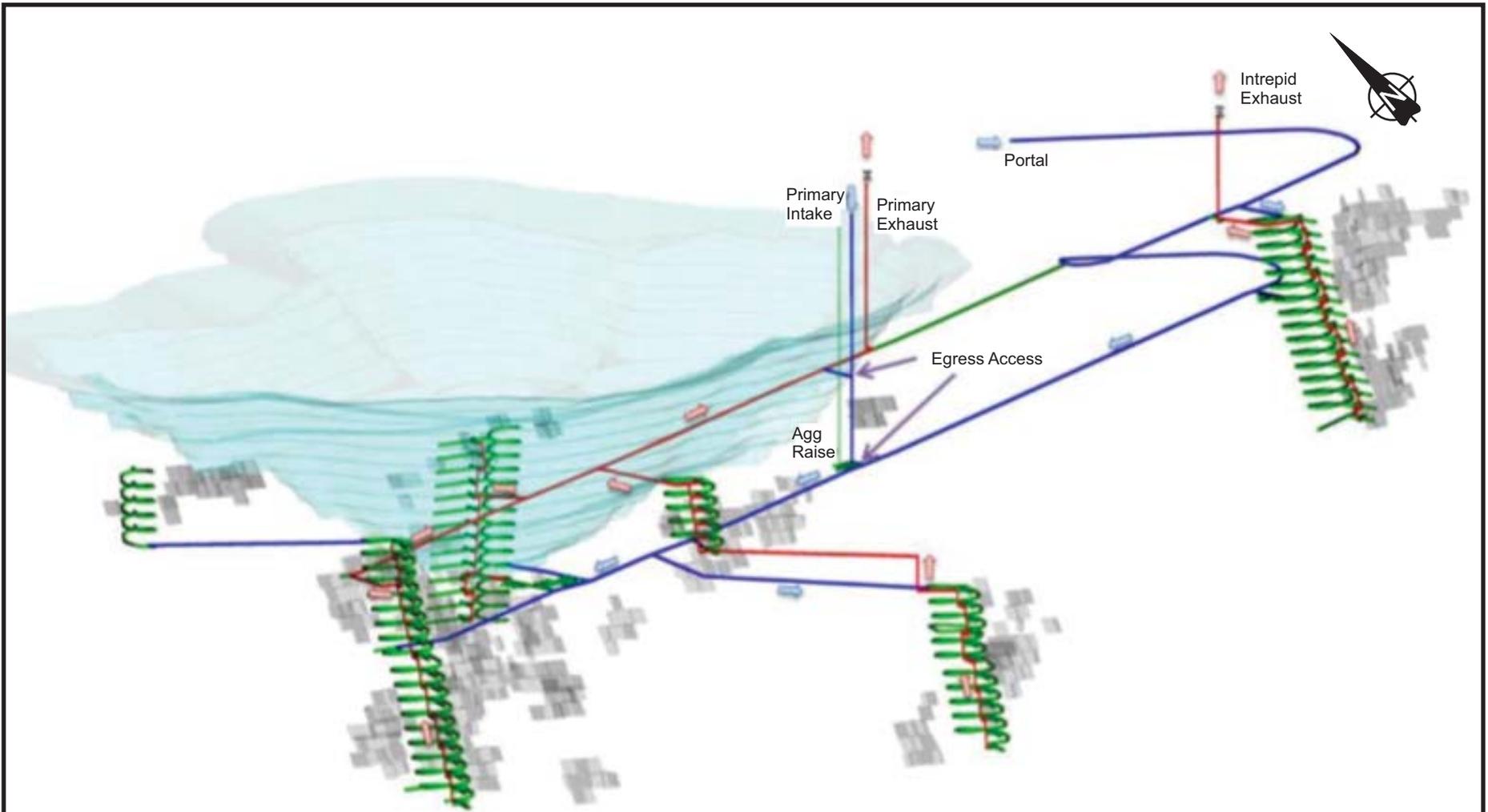


Figure 16-7

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Rainy River Mine,
Rainy River, ON, Canada newgold™

Schematic of Primary
and Secondary Egress

MINE DEWATERING

Previous hydraulic conductivity analyses generated an estimated groundwater inflow of 18.7 L/s. In the dewatering design, this was factored up to 28.6 L/s for sizing of sumps and pumps at each station. Mine dewatering will be handled by a combination of submersible and horizontal centrifugal pumps located throughout the Intrepid Zone, 17 East Zone, 433 Zone, and ODM Zone working levels. The pumps will handle ground inflow and spent drill water via multiple lifts throughout the mine to minimize pump size and power. The level sumps are designed to handle 36.6 L/s. A total of nine lift stations are in the current design. The breakdown is as follows:

- ODM Zone: 4 pump stations
- East Zone: 3 pump stations
- Intrepid Zone: 2 pump stations

A typical sump will allow for solids settling with an overflow weir to a clear water side before pumping to a holding tank. The sumps in all areas with the exception of Intrepid will report to a pumping station near the workshops. The Intrepid mining zone sumps and the workshop area water will ultimately report to a main sump located in the Intrepid Zone, from where the water is finally pumped out via the portal to surface.

BACKFILL

The mine design calls for backfill in approximately half of the stope voids in the mine. The backfill will be a combination of rock fill, with the fill coming from waste development, and cemented aggregate fill for areas requiring a stronger fill. The cemented aggregate fill system will require a surface slurry plant, aggregate production and stockpiles and a loading station underground. The underground loading station will have concrete floors, a sump for spilled slurry and a signalling system to control the delivery of slurry from surface. This will be a simple level sensing indicator in the underground tank which will request the delivery of the next batch of slurry from surface. The planned cement addition is 4% by weight.

Haulage trucks underground will load aggregate from the chute and cement will be sprayed in batches to manage the cement dosing requirement

The total backfill required and the split between cemented fill and rock fill is shown in Table 16-10 and the annual backfill schedule is shown in Table 16-11.

TABLE 16-10 BACKFILL REQUIRED BY AREA

Area	Cement Tonnes	CRF Volume m ³	RF Volume m ³
433	10,951	129,141	86,094
17E East Lower	10,658	125,679	83,786
17E East Upper	-	-	-
Intrepid Lower	4,978	58,707	39,138
Intrepid Upper	-	-	-
ODM East	5,540	65,334	43,556
ODM Main Lower	29,888	352,454	234,969
ODM Main Upper	20,473	241,427	160,951
ODM West	734	8,660	5,774
Total	83,223	981,402	654,268

TABLE 16-11 ANNUAL BACKFILL SCHEDULE

Year	Cement Tonnes	CRF Volume m ³	RF Volume m ³
2018	-	-	-
2019	-	-	-
2020	-	-	-
2021	2,433	28,694	19,130
2022	2,859	33,718	22,479
2023	5,234	61,716	41,144
2024	10,839	127,816	85,210
2025	9,912	116,892	77,928
2026	11,696	137,920	91,947
2027	12,457	146,894	97,930
2028	7,716	90,986	60,657
2029	6,266	73,897	49,265
2030	6,860	80,898	53,932
2031	5,509	64,962	43,308
2032	1,442	17,009	11,339
2033	-	-	-
Total	83,223	981,402	654,268

ELECTRICAL DISTRIBUTION SYSTEM

Electrical power will be supplied from the main surface substation at 27.6 kV to the surface substation for the underground mine located at the portal. From the substation the mine will be supplied using redundant feeders at 13.8 kV to feed the 500 kVA 13.8 kV/600V load centres in the mine. There will be additional power supply lines to the ventilation raises.

The mine electrical load is projected to increase to approximately 4.4 MW by 2021 and will average 6.6 MW from 2023 to 2029.

MAINTENANCE FACILITIES

There will be an underground maintenance shop and service area in the mine. Refueling of vehicles that routinely come to surface will be done on surface and the remainder of the underground fleet will be refueled from small satellite fueling stations.

The underground mine infrastructure also includes:

- Potable and permanent refuge stations
- Explosives magazines
- Communications systems
- Monitoring and control systems

There is no mine wide compressed air distribution system; compressed air will be supplied where required by local compressors.

MINE EQUIPMENT

The underground mobile equipment requirements were built up from the detailed activities in the mine plan and the projected equipment performance, availability and utilization. The mobile fleet size was estimated based upon production estimates and the availability and utilization values shown in Table 16-12. The maximum fleet size for the operating period of the mine life is shown in Table 16-13.

TABLE 16-12 AVAILABILITY AND UTILIZATION ASSUMPTIONS

Units	Value (%)
Truck/Jumbo/Bolter availability	85
Scoop availability	85
Jumbo / LH Drill utilization	50
Truck/Bolter/LHD utilization	80

TABLE 16-13 UNDERGROUND MOBILE EQUIPMENT

Unit	#	Maximum
Drill Jumbo, 2-Boom Sandvik Axera 7-260 DD421-60	units	3
Drill Longhole, Sandvik DL431-7C, Production Drill	units	3
Slot Rig, Atlas Copco Easer L	units	1
Pneumatic Cartridge Loader, CB32R, UH stope charging	units	1
Pneumatic ANFO Loader, CBAC125R, DH stope charging	units	1
Haulage Truck, 45 Tonne, CAT AD45B	units	7
LHD, Diesel 17t, CAT 2900G, incl. remote costs	units	4
Bolter, Sandvik DS411	units	2
Personnel Carriers, Light Utility Vehicles	units	16
Service vehicles (scissor lift, service trucks, face charger, etc.	units	12
Total	units	50

While equipment brands are included for some units in the table there is no certainty that those brands will be used at the mine. The brands and models shown indicate the type and size of equipment planned for use in the mine.

In light of the ventilation requirements for the diesel-powered equipment, consideration of the use of electric drive units (either trolley or battery) is recommended.

MINE DEVELOPMENT SCHEDULE

Underground development is forecast to commence in 2018, with the first ore from development obtained in Q3 2019. Work in the mine will be undertaken by contract crews through the pre-production period and for the first one and one-half years of the operating phase. The contractor personnel will be replaced by company personnel after that time.

The mine development schedule is summarized in Table 16-14. The main ramp development provides access to ore in the Upper Intrepid zone and development continues downwards with simultaneous production.

The mine development schedule is based upon a single heading advance of 150 m per month. This is an aggressive development rate but it is considered to be attainable with use of dedicated crews and equipment, but with some risk of shortfalls. Close monitoring of the development progress is recommended together with quick response and remedial action in the event of development advance rates not being achieved.

The mine development includes infill drill platforms and it is recommended that the schedule be reviewed to include time for drilling and data analysis before stope development and production commence.

TABLE 16-14 MINE DEVELOPMENT SCHEDULE

Year	Alimak metres	Development metres	Ore Drifting metres	Raise Bore metres	Stope Development metres
2018	-	1,721	-	-	-
2019	-	6,477	936	407	-
2020	436	13,024	2,170	1,103	106
2021	413	10,308	4,351	44	1,836
2022	537	6,855	7,465	-	1,935
2023	191	4,517	8,870	-	2,341
2024	227	9,031	1,258	-	533
2025	466	6,051	3,857	-	885
2026	200	2,233	4,788	-	3,143
2027	94	2,533	2,134	-	730
2028	196	1,982	2,748	-	627
2029	-	-	4	-	-
2030	-	-	-	-	-
2031	-	-	-	-	-
Total	2,760	64,732	38,580	1,553	12,136

PRODUCTION SCHEDULE

Underground Mine production is expected to commence in Q2 2019 and continue to 2031, beginning in the Upper Intrepid zone and carrying on with annual ore production from as many as nine different zones in any given year over the life of mine plan. Beginning in 2019 planned mine production is expected to increase over a four-year period from approximately 2,200 tpd to approximately 2,400 tpd and to remain at that level for the remainder of the underground mine life.

The production build-up is relatively rapid and will be dependent on attaining the development rate advance described above.

The underground mine production by zone is shown in Table 16-15 and the annual underground production schedule is summarized in Table 16-16.

TABLE 16-15 ANNUAL PRODUCTION BY AREA

Year	433 t (000)	17 E L t (000)	17 E U t (000)	INTRE t (000)	LINTRE t (000)	U ODM E t (000)	ODM L t (000)	ODM U t (000)	ODM W t (000)	Total t (000)
2018	-	-	-	-	-	-	-	-	-	-
2019	-	-	-	-	60	-	-	-	-	60
2020	10	-	-	-	150	6	-	1	-	167
2021	57	26	-	-	122	229	19	43	-	497
2022	39	113	-	-	109	157	257	25	-	700
2023	-	122	-	-	54	118	395	152	-	840
2024	3	123	-	4	86	148	371	122	-	857
2025	2	112	-	0	11	28	361	320	4	838
2026	164	228	-	2	-	22	146	264	12	837
2027	221	79	6	39	12	71	198	180	42	848
2028	151	44	8	205	49	36	257	96	-	845
2029	141	125	-	135	69	65	198	146	15	894
2030	63	65	-	137	56	40	104	302	113	881
2031	42	34	-	122	-	-	25	263	93	579
2032				45			31	35		112
Total	892	1,071	14	6,894	778	919	2,362	1,948	280	8,954

Notes:

1. Totals may not add due to rounding.

TABLE 16-16 ANNUAL PRODUCTION SCHEDULE

Year	Ore Tonnage Tonnes	Au g/t	Ag g/t	Au oz	Ag oz
2019	59,855	2.96	26.63	5,689	51,240
2020	166,711	3.29	23.75	17,633	127,304
2021	496,597	3.15	11.43	50,339	182,564
2022	699,602	3.46	12.74	77,808	286,665
2023	840,103	2.90	7.94	78,218	214,381
2024	856,817	3.47	12.41	95,563	341,958
2025	837,901	3.66	7.21	98,542	194,191
2026	836,951	3.81	9.33	102,457	251,176
2027	848,485	3.69	4.68	100,584	127,762
2028	845,497	3.76	8.19	102,126	222,724
2029	894,092	3.59	11.46	103,202	329,404
2030	880,729	3.83	7.97	108,407	225,618
2031	578,808	3.67	7.69	68,238	143,013
2032	111,524	3.42	8.39	12,255	30,089
Total	8,953,674	3.55	9.48	1,021,060	2,728,088

Notes:

1. Totals may not add due to rounding.

17 RECOVERY METHODS

PROCESS DESCRIPTION

The Rainy River process facilities were designed to process 21,000 tpd or 7.67 Mtpa from the open pit and underground mines. The target production was originally 19,500 tpd from the open pit mine and 1,500 tpd from the underground mine. The present expansion will increase the plant throughput to 24,000 tpd in 2019 and 2020, followed by a ramp-up to 27,000 tpd in 2021 for the remainder of the LOM. The operating availability of the primary crusher is 65% and the operating availability of process plant is 92%.

Figure 17-1 illustrates the simplified flowsheet of the Rainy River processing facilities.

The process flowsheet consists of the following unit processes:

- Primary gyratory crushing;
- Coarse ore stockpile, discharged through draw pockets by apron feeders;
- SAG mill feed conveyor;
- Primary SAG mill grinding;
- Pebble crushing;
- Secondary ball mill grinding;
- Gravity concentration of primary cyclone feed slurry using centrifugal concentrators;
- Intensive cyanide leaching of the gravity concentrate using an Acacia reactor;
- Screening of cyclone overflow prior to thickening to remove trash;
- Grinding circuit thickening;
- Cyanide leaching using eight tanks in series;
- Carbon in pulp gold recovery using seven tanks in series;
- Cyanide destruction using the SO₂ (sulphur dioxide) air process in two tanks in series;
- Carbon stripping using the Zadra process;
- Acid washing of the carbon using hydrochloric acid;
- Electrowinning of the carbon eluent and gravity concentrate leach solution; and
- Casting of gold and silver bars in an induction furnace.

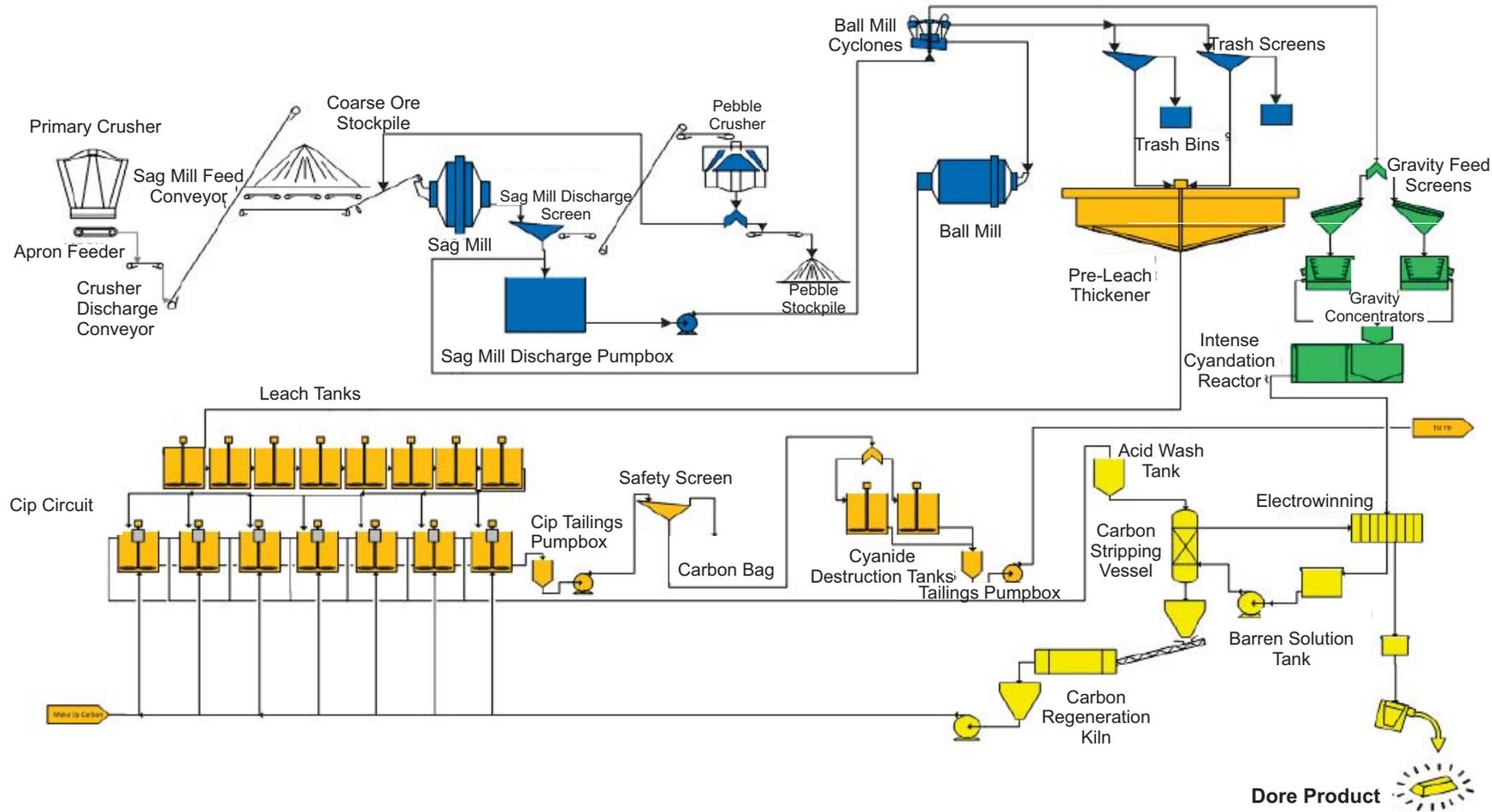


Figure 17-1

newgold New Gold Rainy River Mine, Rainy River, ON, Canada newgold

Site Flowsheet

ORE DELIVERY FROM THE MINE

Ore is delivered from the Rainy River open pit mine using 220 t haul trucks. Low grade ore is trucked to stockpiles for future production.

PRIMARY CRUSHING

The primary crusher consists of a 1,400 mm x 2,100 mm, 600 kW gyratory crusher. The crusher is designed for 220 t mine haul trucks to dump directly into the crusher feed pocket. Two dump positions on opposite sides of the crusher allow for simultaneous dumping. The capacity of the dump pocket is 330 t or approximately 1.5 trucks. The crusher is equipped with a hydraulic rock breaker for reducing oversized material and a mobile crane is available for removal of unbreakable oversize, tramp steel, and maintenance of the crusher.

The crusher is designed to process 1,346 tph of ore with an F_{100} feed size of 1,050 mm and an F_{80} of 550 mm and an operating availability of 65%. The crusher will operate with an open side setting of 165 mm to produce an F_{80} product size of 162 mm. The maximum, or F_{100} , particle size is projected to be 300 mm to 350 mm. The crusher discharge surge pocket live capacity is 418 t or approximately 1.9 trucks. Ore is removed from the discharge surge pocket by a single 2,134 mm wide apron feeder, FE01, which discharges onto the 1,372 mm wide crusher discharge conveyor, CV10, which then transfers ore to the 1,372 mm wide coarse ore stockpile feed conveyor, CV 11. CV11 transports the ore to the coarse ore stockpile. CV10 is equipped with a weightometer to measure the primary crusher production rate and total ore processed.

COARSE ORE STOCKPILE AND RECLAIM SYSTEM

The coarse ore stockpile has a total capacity of 85,700 t and a live capacity of 19,000 t. Ore is drawn from the coarse ore stockpile by three apron feeders, two operating and one standby. The apron feeders discharge onto the 1,372 mm wide SAG mill feed conveyor, which is installed in a single reclaim tunnel beneath the stockpile. The SAG mill feed conveyor has a variable frequency drive (VFD), and delivers ore to the SAG mill feed chute. The SAG mill feed conveyor is equipped with a weightometer to monitor and control the SAG mill feed rate.

PRIMARY GRINDING – SAG MILL

The SAG mill is an 11.0 m diameter by 6.1 m long grate discharge mill with a dual pinion drive consisting of two 7,500 kW motors with VFDs. The mill feed is projected to have an F_{80} of 162 mm and the discharge transfer size is estimated to have a P_{80} of 2,800 μm . The mill operates at 76% of critical speed to achieve a design production rate of 951 tph. The design operating power at the pinions is 12,580 kW, which is approximately 84% of the installed power. The mill has a grate discharge with 75 mm pebble ports.

The mill discharge is fitted with a single deck horizontal vibrating screen with 10.5 mm openings to remove oversized pebble, ball chips and tramp steel. The mill operates with a 13% ball charge made up with 125 mm balls and a total charge volume of 25%. The maximum design ball charge is 16% with a maximum design mill fill volume of 30%.

The oversized pebble is conveyed from the SAG mill discharge screen to a Raptor L500, 3.5 m x 4.0 m x 3.6 m, cone crusher, with a 447 kW drive via three conveyors, CV31, CV32, and CV33. Two belt magnets followed by a metal detector are installed on CV32. If metal is detected, a two-way gate will be opened and the metal containing ore is bypassed to a reject bin. The nominal operating rate of the crusher is 238 tph, 25% of nominal mill feed, with an average operating power draw of 235 kW. The crusher is operated at a closed side setting (CSS) of 13 mm to reduce the pebble to a P_{80} of 13 mm. The crushed product is conveyed to the SAG mill feed conveyor transfer tower where it is either discharged onto the SAG mill feed conveyor and recycled to the mill or fed to a bypass conveyor, which feeds a pebble stockpile adjacent to the conveyor transfer tower.

The SAG mill is operated with a slurry density of approximately 70% solids and discharges into the primary cyclone feed pumpbox where it is diluted and pumped to a cluster of nineteen 508 mm hydrocyclones for classification. The cyclone distribution header has 25 ports. A total of 19 ports are fitted with hydrocyclones, four are piped to the gravity concentration circuit feed distributor, and two are spare and blind flanged. The cyclone feed slurry density is controlled to approximately 52.5% solids to obtain a cyclone overflow density of 28% solids and a target particle size distribution of P_{80} 74 μm . The design cyclone underflow slurry density is 74% solids and flows by gravity to the ball mill feed chute.

GRAVITY CONCENTRATION

Four ports of the cyclone feed distribution header are piped directly to the gravity concentration distributor. The distributor has two bottom outlet ports with dart valves to control the discharge flow. The slurry is directed to two 48 inch Knelson centrifugal concentrators for gravity gold recovery. The flow rate to each concentrator is approximate 300 tph for a total of 600 tph. This equates to approximately 23.4% of mill discharge. The slurry to each concentrator flows over a 2.15 m wide x 4.9 m long sizing screen. The sizing screen undersize flows to the centrifugal concentrator, while the screen oversize flows to the gravity circuit launder and continues by gravity to the cyclone feed pumpbox. The capacity of each centrifugal concentrator is 400 tph. The operating slurry density is 48% solids. Estimated concentrate projection is approximately 3,100 kg/d. Tailings from the centrifugal concentrators combine with the screen oversize in the gravity circuit launder and flow by gravity to the cyclone feed pumpbox. Concentrate from the centrifugal concentrator is pumped to the Acacia intensive cyanide leach circuit.

SECONDARY GRINDING – BALL MILL

The ball mill is a 7.9 m diameter by 12.3 m long overflow mill with a dual pinion drive consisting of two 7,500 kw motors with variable frequency drives. The mill feed is projected to have an F_{80} of 2,800 μm and the target product size is a P_{80} of 74 μm . The mill operates at 75% of critical speed to achieve a design production rate of 951 tph. The design operating power at the pinions is 12,360 kW, which is approximately 82.4% of the installed power of 15,000 kW. The slurry discharges from the mill through a perforated box for steel removal and into the cyclone feed pumpbox. The mill is also equipped with a trunnion magnet.

The ball mill is operated with a target slurry density of 72% solids and a circulating load of 300%. The maximum circulating load is projected to be 400%. The design ball charge is 32% with a design maximum design ball charge of 36%.

INTENSIVE CYANIDE LEACHING OF GRAVITY CONCENTRATE

The concentrate from the centrifugal concentrators is treated in an Acacia intensive cyanide leach reactor, located in a locked section directly beneath the concentrators. The Acacia reactor is an automated batch system providing security for the processing of gravity gold

concentrates. The concentrate is leached at 54°C in a 2.5% sodium cyanide and 1.5% sodium hydroxide solution and leach aid to recover the gold. The pregnant Acacia leach solution is then pumped to a heated storage tank. The solution is then pumped to the gold room in preparation for electrowinning. The tailings from the Acacia leach reactor is pumped to the cyclone feed pumpbox for re-processing.

THICKENING

The grinding circuit cyclone overflow flows through two parallel Delkor belt screens with 600 µm openings to remove oversize material, plastic and other debris, before the slurry flows to the pre-leach thickener. The screen underflow flows by gravity to a 3.2 m diameter by 2.7 m diameter pre-leach thickener feed tank from which it overflows into the centre well of a 45 m diameter by 3.3 m high leach feed thickener.

The thickener underflow density is controlled to approximately 50% solids using density measurement and variable speed underflow pumps. The underflow slurry is pumped to the cyanide leach tanks.

The thickener overflow solution is very low in solids and is pumped to the 17 m diameter by 9.1 m high process water tank. Water is pumped from the process water tank to all areas of the plant requiring water using two 406 mm x 356 mm low pressure centrifugal pumps and two 254 mm x 203 mm medium pressure centrifugal pumps, which draw from the process water tank. The medium press process water pumps also feed the high-pressure process water distribution pump. In addition to the pre-leach thickener overflow the process water tank receives water from the process recirculation heat exchangers, cooling water return, the mine rock pond and the tailing reclaim pumps. Tailings reclaim water also reports to the pre-leach thickener feed tank and the tailings pumpbox.

LEACHING AND CARBON IN PULP

The thickener underflow slurry is adjusted to 50% solids and pumped to the leaching and CIP circuit. The leaching circuit consists of eight 18.0 m diameter by 22.7 m high tanks in series for a total retention time of 30 hours. The elevation required for gravity flow is achieved by reducing the height of each tank by 0.5 m so tank No. 8 is 18 m in diameter by 19.2 m high. The first four tanks use oxygen to supply oxygen for the leach reaction. The last four tanks

have air injection to supply oxygen. The design gold and silver recoveries were projected to be 86.6% and 60.1%, respectively.

Leach tank No. 1 can be used for pre-aeration if required. The slurry overflows the pre-aeration tank to leach tank No. 2 where cyanide is added and leaching continues through leach tank No. 8.

The leach slurry flows from leach tank No. 8 by gravity through the leach discharge primary sampler to the CIP feed launder and into the carousel style CIP pump cell circuit where it is contacted with activated carbon for gold recovery. The CIP circuit consists of seven tanks that are 7 m diameter by 12 m high in series, each with an operating volume of 360 m³ for a total operating volume of 2,520 m³ and a total retention time of two hours. The CIP circuit is a carousel system where the feed and discharge to and from each CIP tank is operated separately to simulate countercurrent carbon transfer without advancing the carbon from tank to tank. There is no transfer of carbon between tanks. A specified amount of carbon is added to each tank and operated until fully loaded. The flow to a given tank is closed and the total volume of slurry is pumped to the loaded carbon screen. The loaded carbon, screen oversize, flows by gravity through a diverter gate to either the acid wash tank or the carbon stripping vessel. The screen undersize slurry flows by gravity to the CIP feed launder. The feed to the CIP tank is opened, the tank refilled, the specified amount of carbon is added and the cell put back on line. Each vessel is loaded with approximately 20 t of carbon. The CIP tanks are at the same elevation and use KEMIX inter-stage screens, which provide pumping to advance the slurry from tank to tank.

The target carbon concentration is 55.6 g carbon/L slurry. The average carbon transfer rate is once every two days and the design rate is once per day. The total transfer and refill time is approximately 3 hours. The average carbon loading for the two-day cycle is 3,030 g/t Au and 4,420 g/t Ag or a total of 7,450 g/t Au and Ag. The carbon loading would be half this amount for daily transfers. The washed loaded carbon is re-pulped in-line with water and flows by gravity through a three-way diverter valve to either the acid wash vessel or to one of the two carbon stripping vessels. The screen underflow slurry returns to the CIP tank No. 1. The slurry discharging the CIP circuit flows to the CIP tailings pumpbox, from which it is pumped to the carbon safety screen for removal of fine carbon. The screen undersize slurry flows by gravity to the cyanide destruction distributor. The screen oversize carbon fines are transferred to a

carbon safety screen dewatering screen. The water from the carbon safety screen dewatering screen flows to the CIP tailings pumpbox. The carbon fines recovered are loaded into bags.

Process controls in the leaching circuit include analysers for both pH and cyanide concentration. There are primary and secondary slurry samplers on the CIP discharge following the carbon safety screen for analysis of the CIP tailings.

CARBON DESORPTION AND REGENERATION

The gold is desorbed from the carbon using the high pressure and temperature Zadra process. One 10 t acid washing vessel and two 10 t carbon stripping vessels are installed. The CIP carbon transfer batch size is 20 t, so only 50% of a given batch can be acid washed. The procedure when acid washing each time is to fill the acid wash vessel and one of the stripping vessels and start the acid wash and strip together. Once complete the acid washed carbon will be transferred into the second stripping vessel and stripped along with the first vessel. During this overlap time, the two stripping vessels will operate in series. Acid washing cycle time is 60 minutes for transferring carbon into the vessel and 240 minutes of stripping. Stripping cycle will include 60 minutes to transfer carbon and 480 minutes to strip. The overlap time will be approximately 240 minutes. Cooling of the carbon following stripping will be 60 minutes and carbon unloading will be 60 minutes. The total stripping time for a batch including the acid washing and overlap in stripping will be 930 minutes. The design also allows for pumping the carbon from the stripping vessels to the acid wash tank, so it is possible to acid wash 100% of the carbon by acid washing the remaining 50% after stripping. Acid washing is achieved using 5% hydrochloric acid.

The carbon is added to a 10 t capacity acid wash column. The column is then filled with a 5% hydrochloric acid solution and allowed to soak for 60 minutes. Filtered process water is then added to displace the acid solution, wash, and neutralize the carbon prior to transferring it to one of two 10 t capacity carbon stripping vessels. In the Zadra process, gold and silver are eluted from the carbon and recovered by electrowinning continuously. Eluent containing 3% sodium cyanide and 2% sodium hydroxide is pumped from the barren solution tank through heat exchangers, which heat the solution to 150°C, then upflow through the carbon stripping vessels. The pregnant solution then flows back through the heat exchanger to reduce the temperature to below boiling, and then through the electrowinning cells to precipitate the gold

and silver as a sludge. The barren solution exiting electrowinning then flows to the barren solution tank and the cycle is complete. The eluent is circulated in this manner until the gold and silver are recovered from the carbon.

The stripped carbon is then washed with process water to remove any residual gold, cyanide, and caustic, and to cool the carbon. After washing, the carbon is discharged from the stripping vessel and pumped to the carbon dewatering screen. The dewatered carbon, screen oversize, flows into the 12 t carbon regeneration kiln feed bin. The water passing through the screen flows into the fine carbon collection tank.

CARBON REACTIVATION

The stripped carbon is reactivated in a horizontal electric rotary kiln operating at 650°C to 700°C. The reactivated carbon is discharged into a 4 t quench tank for cooling and then pumped to the fresh carbon sizing screen to remove any fine carbon. The screen oversize carbon flows into the 12 t carbon storage tank. The reactivated carbon is then pumped via the carbon storage tank transfer pump to the CIP tanks to be reloaded. The capacity of the carbon regeneration kiln is 500 kg/h for a total of 12 tpd. The target is to regenerate 60% of the carbon stripped.

ELECTROWINNING

The pregnant solutions from the Acacia intensive cyanide leach reactor and from the carbon stripping circuit are combined in the electrowinning cell distribution box and circulated through the electrowinning cells to recover the gold. There are three parallel trains of two 3.5 m³ cells with design flows of 44 m³/h and 15 minutes retention times. The gold and silver in solution is plated onto stainless steel cathodes. Once the cathodes are loaded and the circulating electrolyte is reduced to the target gold and silver concentration, the cathodes are removed from the cells and the gold and silver sludge is washed from the cathodes with high pressure water. The gold and silver sludge is filtered in a plate and frame filter press, dried in drying ovens, fluxes are added, and the mixture is melted in a 300 kW electric induction furnace to produce 31.1 kg gold and silver doré bars.

CYANIDE DESTRUCTION

The slurry leaving the last CIP tank passes through a carbon safety screen to recover fine carbon and then flows to the cyanide destruction circuit. The circuit consists of two, 11.5 m diameter by 13.5 m high mix tanks in series to provide a retention time of 1.5 hr. A mixture of copper sulphate, sodium meta-bisulphite, SO₂ gas, and lime are added to destroy the cyanide. The system is designed to reduce the CN_{WAD} from a feed concentration of 125 ppm CN_{WAD} to less than 5 ppm CN_{TOTAL}.

TAILINGS AND RECLAIM WATER SYSTEM

TAILINGS MANAGEMENT AREA

The detoxified slurry flows from cyanide destruction to the tailings pumpbox. The tailings slurry is then pumped by two 356 mm x 304 mm, 550 kW centrifugal pumps in series to the Tailings Management Area (TMA).

Reclaim water is pumped from the TMA to the process water tanks and tailings pumpbox by two 1,350 m³/h, 522 kW vertical turbine pumps, one operating and one spare. The reclaim water demand for the process facilities is 1,080 m³/h.

WATER MANAGEMENT POND

Fresh water is currently supplied to the water management pond from the Pinewood River with two 225 kW vertical turbine water intake pumps. After sufficient reclaim water is available from the tailings management pond, the pumps will be moved from the Pinewood River to the water management pond pump station and will pump through the same pipelines back to the Pinewood River. Once this transition is made, reclaim water from the TMA would supply water to the water management pond using the vertical turbine pumps.

Fresh water is supplied to the plant from the water management pond by two 74.6 kW vertical turbine pumps, one operating and one spare. Fresh water is pumped to the firewater tank, the truck filling station, the truck shop, and the truck wash facility.

WATER DISCHARGE POND

The water discharge pond receives water by gravity from the water management pond. Water then flows by gravity from the water discharge pond to the wetlands.

MINE ROCK POND

The mine rock pond has a capacity of approximately 2.93 Mm³ and receives water from the open pit and eventually from underground mine dewatering. Water from the mine rock pond is pumped to the reclaim water tank, the tailings pumpbox, the process water tank and the cyclone feed pumpbox. The mine rock pond is not available during the winter due to freezing. Water will be supplied from tailings management area reclaim water only during the winter months.

REAGENTS

Reagent systems were designed for each of the major reagents. The sizing of the reagent tanks is based on consumption rates except for the reagents with small consumptions, which are based on the supply truck size.

SODIUM CYANIDE

Sodium cyanide will be received as a dry solid pellet or briquette in ISO containers. A measured amount of reclaim water is added to the 4.5 m diameter by 6.5 m high cyanide mixing tank to achieve the required solution strength. The water is then circulated through the ISO container and back to the mix tank until all of the sodium cyanide is dissolved using the cyanide recirculation pumps. Air is then introduced to transfer all of the solution from the ISO container and piping into the mix tank. The mixed cyanide solution is then transferred from the mixing tank into the 4.85 m diameter by 6.9 m high cyanide holding tank. Cyanide solution will then be metered from the cyanide holding tank to the process using the cyanide feed pumps.

LIME

Quicklime (CaO) will be delivered as bulk dry pebble by hopper truck and transferred into a 3.6 m diameter by 21.7 m high storage silo using compressed air. Screw feeders at the bottom of the silo will convey the quick lime to the 1.2 m diameter by 2.3 m long ball mill, where water is added and the quicklime is slaked to hydrated lime. The slaked lime will be mixed to a density of 25% solids and transferred to a 5.0 m diameter by 7.0 m high slaked lime holding tank. The lime will then be recirculated from the holding tank, through the plant lime loop piping and back to the holding tank, by the lime recirculation pumps. The lime will be metered to the destination points, including the mill feed, pre-leach thickener, leach tanks, and cyanide destruction through lime delivery piping that tees off of the main lime loop.

CAUSTIC SODA – SODIUM HYDROXIDE

Caustic soda (NaOH) will be shipped by tanker truck as a 50% concentration solution. The solution will be diluted with reclaim water in a 4.2 m diameter by 6.2 m high mixing tank to lower the concentration to 30%. The mixing tank is sized to for one and a half 36 t tanker trucks.

HYDROCHLORIC ACID

Hydrochloric acid is delivered at a concentration of 31.5% by 30 t tanker trucks and transferred to the hydrochloric acid holding tank. The holding tank is sized for 1.5 tanker trucks. The hydrochloric acid will be diluted to 5% and for use in the stripping circuit for acid washing.

SULPHUR DIOXIDE

Sulphur dioxide (SO₂) is delivered in liquid form by 28 t tanker truck and stored in a pressured horizontal holding tank. The holding tank package is complete with a padding system (compressors, dryers, and receivers) with all required instrumentation for metered reagent delivery to the cyanide destruction tanks. This arrangement ensure that no lines connected to the SO₂ system enter the process plant.

COPPER SULPHATE

Copper sulphate (CuSO₄.5H₂O) as 95% dry crystal will be delivered in 1,000 kg supersacks and transported by truck. The bags will be mixed with warm reclaim water and dissolved to approximately 15%. The 2.4 m diameter by 3.1 m high mix tank will have the capacity to mix two bags to the required concentration of 15%. The mix tank will supply a small 1.5 m diameter by 1.0 m holding tank or day tank for pumping.

ACTIVATED CARBON

Natural coconut shell type activated carbon (typical dimensions 6 mesh x 12 mesh) will be used in the CIP adsorption circuit. The total estimated consumption is 34 g/t, based on operational standards, moving 20 t of carbon every two days and the utilization of the carbon-in-pulp pump cell circuit minimizing carbon losses as fines. The delivery will be by truck in 20 shipments of 500 kg bulk bags. The new carbon will be added to the attrition tank feed hopper and into the carbon attrition tank, when it is agitated to remove fine carbon. The carbon is then pumped to the fresh carbon sizing screen. The screen oversize flows to the carbon storage tank and the screen undersize reports to the fine carbon collection tank. The fresh carbon is

pumped from the carbon storage tank to the CIP circuit using the carbon storage tank transfer pump.

ANTISCALANT

Antiscalant will be used in the process water reservoir and in the stripping circuit to minimize scale build-up. Each area has its own tote and antiscalant metering pump. Antiscalant will be delivered in totes and stored inside the building.

FLOCCULANT

Flocculant will be delivered to the plant in 750 kg super sacks. The flocculant bags will be stored in an outdoor container and a small supply will be kept in the reagent mixing area. The bags will be lifted onto a platform over the hopper/feeder which feeds a wetting device which wets the powder, forming a solution. The solution is mixed in an agitated mixing tank and then transferred to a flocculant holding tank by a progressive cavity pump.

SODIUM METABISULPHITE

Sodium metabisulphite ($\text{Na}_2\text{S}_2\text{O}_5$) will be delivered as dry crystal in 1,000 kg super sacks by truck. The bags will be mixed to a 20% solution concentration in a 3 m diameter by 5 m high agitated tank. The solution will be pumped to the cyanide destruction circuit via a metering pump. Sodium metabisulphite is only used as a backup to SO_2 so the consumption is typically 0 g/t solution.

REAGENT CONSUMPTIONS

The budgeted versus actual unit reagent consumptions for the process facilities for the months of October 2017 through February 2018, are presented in Table 17-1. The budgeted values are based on metallurgical testwork, vendor information and operating data from similar plants. These results represent the first five full months of plant operation.

TABLE 17-1 PROCESS PLANT REAGENT CONSUMPTIONS

	Units	Reagent Consumptions					
		Budget	Oct 17	Nov 17	Dec 17	Jan18	Feb 18
Mill Production	t		545,601	395796	581,544	457,919	578,001
SAG Media	kg/t	0.55	0.613	0.260	0.429	0.756	0.634
Ball Mill Media	kg/t	0.665	1.138	0.766	0.209	0.464	0.724
Flocculant	g/t	70.0	22	194	19	34	23
Lime	g/t	907.0	1182	1015	959	922	506
Sodium Cyanide	g/t	360.0	199	173	283	306	301
Sulphur Dioxide	g/t	460.0	659	549	688	664	533
Copper Sulphate	g/t	80.0	94.8	134.1	87.3	48	60.6
Activated Carbon	g/t	60.0	38	90	65	95	87
Liquid Oxygen	kg/t	0.30	0.40	0.31	0.39	0.51	0.26
Sodium Hydroxide	g/t	120.0	12	17	17	5	13
Hydrochloric Acid	g/t	50.0	0	0	0	0	1
SMBS	g/t	1.0	0	0	0	0	0
Antiscalant	g/t	10.0	0	1	9	12	5
Leachaid	g/t	0.0	0.05	0.02	0.22	0.07	0.13
Dust Suppressant	g/t	0.0	3	25	4	11	12

AUXILIARY SYSTEMS

COMPRESSED AIR

Instrument and plant air compressors are provided for each area of the plant. Table 17-2 shows a list of the compressors and their capacities.

TABLE 17-2 AIR COMPRESSORS

Area	Type	Number	Operating Pressure, kPa	Maximum Pressure, kPa	Capacity, Nm ³ /h
Crusher and stockpile and reclaim	Rotary screw	1	861.8	1034.2	246
Leaching	Rotary screw	1	861.8	1034.2	1,498
Plant air	Rotary screw	2	861.8	1034.2	1434
Cyanide Destruction	Rotary screw	3	861.8	1,034.2	6,545

OXYGEN PLANT

Oxygen is supplied to the first four cyanide leach tanks. Oxygen will be supplied as a bulk liquid during the first two years of operation. A packaged vacuum, pressure, swing, adsorption (VPSA) oxygen plant will be utilized starting in year 3 to provide oxygen for leaching. The capacity of the plant will be 770 Nm³/h. The nominal requirement will be approximately 464 Nm³/h.

MINERAL PROCESSING PLANT PERFORMANCE AND PRODUCTION STATISTICS

The key operating parameters and performance indicators for the Rainy River processing plant for the months of December 2017, January 2018, and February 2018 are presented in Table 17-3. The Rainy River process plant was commissioned in September 2017 and has been working through initial operating and maintenance issues and plant optimization. The production rates are approaching the targeted 21,000 tpd and have been achieved on a daily basis, however, plant availability has been lower than projected and so the average production rates on a monthly basis have been below target.

TABLE 17-3 RAINY RIVER PROCESSING PLANT OPERATING PARAMETERS

Operating Parameter Description, units	AMEC Design Criteria	December 2017	January 2018	February 2018
Mill Production, tpa, tpm	7,670,000	581,544	458,016	578,001
Year 1 Open Pit Ore Milled, tpd	21,000	18,759	14,775	20,643
Years 2–8.5 Open Pit Ore Milled, tpd	19,500	N/A	N/A	N/A
Years 2-8.5 Underground Ore Milled, tpd	1,500	N/A	N/A	N/A
Crusher Utilization, %	65	N/A	45.0	57.1
Mill Utilization, %	92	N/A	68.2	90.5
Crusher Production, tph	1,346	N/A	1,337	1,456
Mill Production Rate, tph	951.3	850.0	902.6	950.4
Gold Grade, g/t Years 1 – 8.5	1.31	1.072	1.101	0.931
Silver Grade, g/t Years 1 – 8.5	2.79	3.19	2.79	1.72
Target Particle Size P ₈₀ , microns	74	N/A	N/A	N/A
Gravity Gold Recovery, %	30.0	N/A	N/A	N/A
Leach Gold Recovery, %	86.6	N/A	N/A	N/A
Overall Gold Recovery, %	90.6	88.6	83.0	82.2
Gravity Silver Recovery, %	10.0	N/A	N/A	N/A
Leach Silver Recovery, %	60.1	N/A	N/A	N/A
Overall Silver Recovery, %	64.1	52.7	59.8	46.2

Gold grades have been slightly lower than projected and both gold and silver recoveries have been lower than projected. The primary focus through the start-up period has been on gold recovery with silver being of secondary economic concern. Key parameters with respect to gold and silver recovery are grind size, 80% passing 75 microns, cyanide concentration, retention time in the leaching circuit, and carbon adsorption and elution in the CIP and elution circuits. Consistent operation is also a factor as stabilizing of the circuit is required following

start-ups and shut-downs resulting in reduced performance. Operating personnel are focusing on these areas in order to improve recoveries.

Some of the key issues encountered during plant commissioning and start-up included:

- Mill motor and VFD drive programming issues, which caused frequent shut-down of the mill.
- Material discharge from the coarse ore stockpile to the mill feed conveyor, including plugged apron feeder discharge chutes and some conveyor damage resulting in down-time.
- Carbon containment and carbon transfer from the CIP tanks to the carbon elution circuit.
- Carbon elution circuit control and performance, including eluent heating and temperature control.
- Cyanide destruction, SO₂-air, circuit operation and performance.
- Tailings pumping.
- Reagent mixing and distribution.

These types of challenges are common during plant start-ups and most will be remedied through adjustments during continuous operation of the plant and correction of issues identified during scheduled maintenance periods.

PLANT EXPANSION STUDY

New Gold has identified sections of the plant, including the carbon elution and cyanide destruction circuits that require modification to achieve required performance. Taking this into consideration the Company engaged Ausenco to perform a review of the elution circuit and to prepare the New Gold Rainy River 24 ktpd Debottlenecking Feasibility Study report, which was issued on March 15, 2018.

Site visits were performed by Ausenco in January and February 2018 to review the operation of the plant, determine solutions to existing performance issues and to identify opportunities for plant improvement. The projects identified are presented Table 17-4.

TABLE 17-4 IDENTIFIED PROJECTS FOR PLANT EXPANSION

Project Package	Project Description	Identified During	Recommendation
1	Cyanide Destruction Testwork	January Site Visit	Complete
2	Air and O ₂ supply	January Site Visit	Further review required
3 and 5	Copper sulphate mixing and dosing	January Site Visit	Proceed to execution
4	Cyanide destruction tank feed box modifications	January Site Visit	Proceed to execution
6	Frozen instruments on top of cyanide destruction tanks	January Site Visit	By New Gold
7	pH fluctuation correction, cyanide destruction tanks	January Site Visit	By New Gold
8	Elution carbon movement automation	January Site Visit	Proceed to execution
9	Elution direct heating conversion	January Site Visit	Proceed to execution
10	Carbon handling screens	January Site Visit	Proceed to execution
11	Telescopic Chute (Crushed ore stockpile)	February Site Visit	Further review
12	Stockpile cover	February Site Visit	Further review
13	Tailings pumping	February Site Visit	Further review

EQUIPMENT SIZING REVIEW

Ausenco performed a review of the major process plant equipment to determine whether they have sufficient capacity for operating at 24,000 tpd on a steady state basis. Ausenco contacted the vendors for the major process equipment and report that all of the equipment reviewed has the capacity for steady state 24,000 tpd operation. Some of the equipment will be very close to maximum capacity at 24,000 tpd and so the capability to handle significant surges in production rates or flow will be limited. It will be important to maintain the 92% equipment availability in order to consistently reach production targets. A table summarizing the details of the equipment capacity evaluations is included in the appendix of the New Gold Rainy River 24 ktpd Debottlenecking Feasibility Study.

Equipment that is close to capacity at 24,000 tpd and should be reviewed in more detail include:

- The SAG mill will be at 96% of motor capacity assuming an operating work index of 13.2 kWh/t and the ball mill will be at 94% capacity assuming a work index of 13.0 kWh/t. Both mills are close to maximum capacity and may be sensitive to variations in ore hardness. FL Smidth suggested a site audit and baseline mill performance review.

- The final tailings pumps will be reviewed due to issues with pumpbox overflow during surges in flow. This is on the current Ausenco project list for review discussed in this document. A systematic review of the tailings pumping system will be performed in conjunction with the tailing dam construction schedule.
- The water management pumps, Pinewood River water return pumps, the mine rock pond water pumps and fresh water pumps should be reviewed with respect to the water management plan and tailings dam construction schedule. Ausenco recommend that the site water balance be reviewed to confirm the projected pumping requirements.

PLANT DEBOTTLENECKING AND EXPANSION PROJECT DESCRIPTIONS

The following sections summarize the projects planned for the plant expansion.

CYANIDE DESTRUCTION TESTWORK

Cyanide destruction testwork was performed by Kemetco on Site from February 19 to February 23, 2018. The results of the testwork were issued on February 25, 2018.

The results of the tests were that:

- The cyanide destruction tanks have sufficient residence time for the reactions to occur.
- Copper sulphate should be used as zinc sulphate is not effective for the reaction
- The mix tanks do not have sufficient freeboard to accommodate the volume expansion from the quantity of air injected to support the reaction. Oxygen can be used instead of air to reduce the gas volume and resulting tank overflow.
- The existing SO₂ system has sufficient capacity for 24,000 tpd operation.

AIR AND OXYGEN SUPPLY

The capacity of the existing cyanide destruction circuit air compressors were reviewed. It was determined that the capacity was sufficient as they are currently using one of three available compressors. The tanks overflow when the second compressor is put on line. It was determined that sufficient oxygen is currently being supplied and the required less than 5 ppm CN_{WAD} content is being achieved in the tailings, however the dissolved oxygen is not being measured to confirm saturation. Ausenco's conclusion was to install an additional oxygen vaporizer to allow the use of oxygen instead of air if required.

COPPER SULPHATE PUMPS AND DOSING POINTS

The existing copper sulphate mixing system and metering pump are not currently being used due to blockage with fibrous trash. Instead, the sodium metabisulphite mixing tanks and

dosing pump is being used for copper sulphate. Sodium metabisulphite is not being used at this time.

Ausenco's recommendation is to install a new copper sulphate day tank and replace the positive displacement pumps with centrifugal pumps and a flow control metering system.

CYANIDE DESTRUCTION FEED BOX MODIFICATIONS

Recommendations have been made to modify the feed box to better distribute the slurry between the two mix tanks. This will be accomplished by:

- Installing an underflow baffle to provide back pressure and reduce slurry turbulence over the weir.
- Installing vortex breakers on the dart valves to eliminate vortex formation.

FROZEN INSTRUMENT DISPLAYS

During the January site visit the LED displays for the pH and ORP instruments were not functional due to the cold temperatures. It is recommended to replace the displays or to move them indoors. The instruments operating through the DCS are suitable for the -40°C temperatures and are functioning.

PH FLUCTUATION IN THE CYANIDE DESTRUCTION TANKS

It was found that the pneumatic actuators for the lime addition valves were freezing up during winter. It was recommended to install moisture traps for all pneumatic actuators located outdoors.

ELUTION AUTOMATION

The current acid wash and elution system is operated using manual valves. It is difficult to operate the acid wash and elution columns manually due to the heights of the columns and the location of valves on both the ground level and tops of the columns. The manual valves are currently being replaced with automatic valves.

ELUTION HEATING

The existing elution heating circuit is not achieving the design temperature of 135°C to 140°C. The eluent is heated by a plate and frame heat exchanger, which is part of a closed loop solution heating system comprised of three in-line immersion heaters, a circulation pump, an expansion tank and the plate and frame heat exchanger. The heat transfer liquid is de-ionized water which is heated to 150°C.

The operators report that several failures of the immersion heating elements have occurred and that scaling of the heat exchanger has been severe. The result is a reduction in the efficiency of the heat exchanger requiring longer contact times to achieve the required solution temperatures for elution.

Ausenco recommends converting the circuit to direct heating by eliminating the indirect heating loop. The eluant would pass directly through the inline immersion heaters on the way to the stripping column. The down side of this system is that scale formation will be in the immersion heaters rather than on the plates of the heat exchangers. This will require regular cleaning and replacement of the immersion heaters to maintain the heating system. KSN is not in favour of this conversion due to the severe scale reported in the existing heat exchanger. A different type of heat exchanger such as a shell and tube may be more appropriate as it would be easier to clean by hydro-blasting than the plate and frame.

CARBON SCREENS

The loaded, fresh and stripped carbon screens are not performing. Operations has changed the screen panels in an effort to solve the problem, with limited success. Ausenco have identified the source of the problem as the circular motion of the downward inclined screens, which prevents the carbon from discharging from the screens. The recommendation is to replace the screens with flat or upward inclined screens with a liner vibration motion.

The performance of the Kemix inter-tank carbon screens are also under review.

CRUSHED ORE STOCKPILE

Dust from the crushed ore stockpile has been identified as an environmental and health concern that should be improved. Dust suppression chemicals are being used in the process facilities. To address the dust from the crushed ore stockpile feed conveyor discharge and the stockpile as a whole, a telescopic conveyor discharge chute, and a stockpile cover are being considered. Suppliers are being contacted to investigate the options.

TAILINGS PUMPING

The capacities of the tailings pumps are being reviewed due to overflow of the tailings pumpbox during surges in flow. A systematic review of the tailings pumping system will be performed in conjunction with the tailing dam construction schedule.

PROJECT SCHEDULE

The projects listed are now in the detailed engineering and procurement stages. Engineering procurement and construction is scheduled to begin in March and be completed in 4.5 months. Commissioning and start-up is scheduled for August 2018.

REVIEW OF RAINY RIVER COMMINUTION AND GOLD PLANT EQUIPMENT FOR MAXIMUM CAPACITY

Following the plant expansion program, which is designed to increase the mill production rate from 21,000 tpd to 24,000 tpd, Rainy River commissioned Ausenco to perform a more extensive review of the processes and facilities to determine the maximum operating rate that the plant can achieve without the addition of new grinding equipment. Additional grinding equipment is not being considered at this time, as it would significantly increase capital costs and the associated down time would decrease production.

Moderate capital cost additions, such as the replacement of pump motors or installing additional screens, are being considered. These types of modifications can be installed without significantly affecting production.

The study is being performed in three phases: a Concept Study phase, a Feasibility Study Phase and an Execution Phase. The schedule for the project will begin with the current Conceptual Study reports and the final implementation phase is to be completed and commissioned by 2019. The production forecast projects total production of 7.5 Mt milled in 2018, 8.5 Mt milled in 2019 and 2020, and 9.9 Mt milled in 2021, which equates to 27,000 tpd.

The two study reports include:

- Ausenco, 2018, New Gold Rainy River Review of the Comminution Circuit Design July 2018.
- Ausenco, 2018, New Gold Rainy River Gold Plant Assessment, July 10, 2018.

The scope of work for the studies includes:

- Reviewing all the available plant information, including circuit design and control philosophy, daily, weekly, and monthly metallurgy reports and supervisory system data (PI system).
- Analyzing the performance of the plant, including the comminution circuit and the gold leaching and CIP circuits.

- Identifying improvements or methods to optimize the current operation and determine their effects on throughput rates.
- Recommending options to modify equipment, process procedures and operating parameters and equipment to improve performance or mitigate constraints.
- The scope of these two reports includes the entire plant from the grinding circuit through the tailings pumpbox.

The methodology will include:

- Reviewing plant data;
- Assessing current operating parameters and practices;
- Reviewing and assessing efficiency of unit processes; and
- Identifying measures for throughput and efficiency improvements

The historical information was reviewed to determine equipment limitations, circuit performance, and risks and opportunities to support the development of the study.

The current study assumes that all of the equipment and system modifications recommended in the 24,000 tpd Debottlenecking and Expansion study have been implemented. The work is scheduled to be completed in August 2018. The focus areas for increasing the production to the 2021 target of 27,000 tpd include:

- Primary crusher in relation to SAG mill grinding performance
- Grinding mills and discharge screens
- Pebble crusher
- Ball mill cyclone feed pump
- Cyclones and feed to gravity concentrators
- Gravity concentrators
- Trash screens
- Grinding circuit performance with respect to circulating load
- CIP circuit including carbon loadings
- CIP interstage screens

SUMMARY OF COMMINUTION CIRCUIT ASSESSMENT

The comminution circuit evaluation includes an assessment of the current operation and an evaluation of the operational methods to increase plant throughput without adding new grinding equipment. The main observations from the review of the historical information and control philosophy are that:

- The circuit was designed for a harder ore than is currently being processed.
- The primary crusher is producing a very fine SAG mill feed.
- Under current operating conditions, the grinding mill's installed power is underutilized (66% for the SAG mill and 74% for the ball mill). Operating data indicate that the ball mill is consistently operating in the 11 kWh/t range and the grind size is coarser than design at 89 µm.
- The fine SAG mill feed results in a coarse transfer size to the ball mill.
- The grinding load is being pushed to the ball mill, which is producing a product particle size with a P₈₀ of approximately 90 µm rather than the design product size of P₈₀ 75 µm.
- An analysis of the ball mill cyclones indicated that the grinding circuit is operating with a circulating load between 500% and 600% instead of the projected 300% with a 400% maximum given in the process design criteria.
- The maximum power draw for the SAG mill is calculated to be 14,770 kW and the ball mill is 13,967 kW. The ball charge could be increased to improve the grind size and the circulating load. A decrease in the circulating load would have a positive impact on the capacities of the downstream equipment.

Throughput calculations indicate that production rates can be increased to 26,500 tpd for the current ore being process and with cyclone overflow of 75 µm. The current operating strategy for treating the less competent ores shall focus on:

- Maximizing the SAG and ball mill power draw by increasing the ball charge and adjusting the mill speed.
- Reducing the particle transfer size of the SAG mill, which feeds the ball mill.
- Allowing a coarser ball mill product size in order to achieve higher grinding circuit production rates.

COMMINUTION CIRCUIT CONCLUSIONS AND RECOMMENDATIONS

The following conclusions were drawn by Ausenco from the review of historical information:

- The primary crusher is producing a finer particle size than is typical with the current open side setting of 165 mm. Either the feed to the crusher is finer than expected or the crusher open side setting is set to a narrower setting than the specified 165 mm. To increase the production rate and increase the product size from the crusher, the setting can be increased. The SAG mill would benefit from a coarser feed size.
- The SAG mill is currently drawing less than the designed power. The SAG mill can potentially draw approximately 14.2 MW with optimal ball charge and mill speed. The lower SAG and ball mill power consumption may be due to a lower than designed ball charge and reduced mill speed. The ball charges in the SAG mill and ball mill can be increased to 16% and 32%, respectively.
- The SAG mill discharge density should be maintained at 70% as per the design criteria.
- The speed of the SAG mill can be increased from the current 76.5% and the ball mill from the current 75% of critical speed to maximize power draw.
- The volumetric ball load in the ball mill can be increased from the normal operating load of 32% to a maximum of 36%.
- The ball loads in both the SAG and ball mills should be monitored to maintain a maximum ball charge and associated maximum power draw to each of the mills.
- Recent survey data were used to model the ball mill circuit and analyse the circuit performance:
 - The grinding efficiency was similar in both surveys, indicating an operating BWi of 12.1 kWh/t and 11.8kWh/t, both lower than the measured BWi of 15.0 kWh/t from the testwork.
 - Morrell's revised C Power Model was used to check the estimated ball mill power. Results indicate that the mill is operating under slurry pooling conditions as the slurry filling level was at 32% and the ball charge was estimated to be 26%.

COMMINUTION CIRCUIT MODELING BY AUSENCO

A range of options were reviewed to increase the comminution circuit throughput, which were selected according to the circuit bottleneck and in a way that installed motor power was fully exploited with SAG mill at 90% and ball mill at 98% utilization.

For the scenarios where relatively less competent and softer ODM ores are considered, the following options were identified:

- Coarser SAG mill feed was considered by operating the primary crusher at a larger opening would allow higher feed rates to the primary crusher while maximizing crusher power.

- A finer transfer size to the ball mill achieved through modification of the SAG mill discharge screen aperture.
- Use of up to 60% of the SAG mill spare power achieved through modification of the operating conditions, such as returning some of the cyclone underflow recycle to SAG mill.

For the scenarios where relatively more competent and harder ores are considered, the following options were identified:

- Utilization of the existing pebble crusher, which is currently not in operation.
- A finer SAG mill feed (F_{80} of 110 μm) may be achieved via intensive blasting as the design ore is relatively more competent than ODM ore.

The comminution circuit analysis indicates a maximum throughput of:

- 28,400 tpd for the ODM ore with cyclone overflow P_{80} of 90 μm (target).
- 26,500 tpd for the ODM ore with cyclone overflow P_{80} of 75 μm (target).
- 25,200 tpd for the design ore with cyclone overflow P_{80} of 75 μm (target).

The comminution circuit analysis at maximum SAG mill capacity and coarser grind size indicates a maximum throughput of:

- 35,400 tpd for the ODM ore with cyclone overflow P_{80} of 171 μm .
- 25,300 tpd for the design ore with cyclone overflow P_{80} of 85 μm .

The maximum capacity of the main equipment in the comminution circuit was evaluated to identify potential limitations with increased throughput rates. The analysis indicated that:

- As higher throughput rates are targeted, larger apertures for the primary crusher can be used. The only limit downstream would be the capacity of the SAG mill.
- The SAG mill grate discharge would only be able to handle a maximum of 1,419 m^3/h with the current SAG mill liner/lifter design, which is sufficient for SAG mill feed rates up to 1,800 tph.
- Ball mill superficial velocity should not be an issue for increased throughput. However, for the 30 ktpd scenario, circulating loads should be maintained at a maximum of 500%.
- Ausenco recommends that when operating in SAG mill-ball mill crushing (SABC) mode, the open area of the SAG mill discharge screen should be investigated.
- To achieve a throughput rate of 30,000 tpd, 21 cyclones will have to be operated and additional cyclones would have to be installed.

PRIMARY CYCLONES

The cyclone cluster contains a total of 19 Cavex 500CVX cyclones, 15 operating and four standby. The analysis of PI data indicated that the feed slurry density varies between 55% and 62% solids, the cyclone overflow density is 28% to 36% solids and the number of operating cyclones varies between 12 and 15 cyclones. Each 500 mm Cavex cyclone has a capacity of 288 m³/h to 360 m³/h at a feed pressure of 100 kPa. In addition, there are six additional outlets, four of which are used to feed the gravity circuit.

Table 17-5 presents the results of a throughput assessment of the primary cyclones, showing the number of cyclones required at each of varying production rates.

TABLE 17-5 THROUGHPUT ASSESSMENT OF THE PRIMARY CYCLONES

Criteria	Units	24.0 ktpd	26.0 ktpd	28.0 ktpd	30.0 ktpd
Slurry Flowrate	m ³ /h	5,367	5,814	6,261	6,709
Duty Cyclones Required	Number	15.3	16.6	17.9	21.0
Installed Cyclones	Number	19.0	19.0	19.0	19.0

The results of the assessment of the primary cyclones are;

- The maximum capacity for the existing primary cyclone cluster producing a cyclone overflow product size P₈₀ of 90 µm was 28,000 tpd with a circulating load of 300%. The number of operating cyclones would be 18 and the number of spares would be one.
- With 18 cyclones operating at the target grind size of P₈₀ of 75 µm, as per the AMEC design criteria, different vortex finders and/or apexes may be required along with a higher operating pressure of 120 kPa.
- At 30,000 tpd and a circulating load of 300%, a total of 21 operating cyclones would be operating, requiring the installation of larger or new cyclones.

Ausenco recommends conducting a review of the cyclone operation and design to determine the best cyclone geometry that provides sufficient capacity for the required production rates, while increasing ball milling efficiency, reducing circulating loads, and improving cyclone overflow particle size distribution by increasing classification efficiency.

SUMMARY OF GOLD PLANT ASSESSMENT

The results from the throughput assessment are presented in Table 17-6. The table includes the major equipment reviewed, the remedial action recommended and order of magnitude capital cost and installation labor cost estimates. The capital costs were sourced from Ausenco internal references for mechanical equipment, typically referencing North American projects.

It is assumed that each of the existing operational deficiencies as outlined in the recent plant expansion study report will be completed and Rainy River will achieve the sustained 24,000 tpd mill throughput and gold production prior to the execution of the expansion modifications. The operating costs were based on power unit rates of C\$0.07/kWh and typical annualized maintenance factors for North American gold plants. The costs in Table 17-6, however, only represent additional costs associated with the new equipment and not incremental operating costs that will be incurred as a function of increased mill throughput, such as power steel, consumables, reagents, and mining.

TABLE 17-6 SUMMARY TABLE OF GOLD PLANT EQUIPMENT AT 26.0 KTPD MILL THROUGHPUT

Equipment	26.0 ktpd	28.0 ktpd	30.0 ktpd	Remedy/Action	Projected Capital Costs (C\$000)	Projected Operating Costs (C\$ '000)
Gyratory Crusher Discharge Conveyor	No	No	No	Power limited above 24 ktpd so decrease belt speed, review pulleys and reducer design	900	Negligible change
Fine Ore Stockpile Feed Conveyor	No	No	No	Class limited above 24 ktpd so increase belt speed, review pulleys and reducer design	1,200	Negligible change
Cyclone Feed Pump	Yes	No	No	Install larger pump motor/VFD	200	Power – 250 Maint – 6.0
Primary Cyclones	Yes	No	No	Install additional cyclones, install dedicated gravity circuit feed pump	178	Maint – 2.5
Trash Screens	No	No	No	Install third screen, cloth aperture to 840 um, same opening as CIP screens	1,350	Power – 3.0 Maint – 45
Pre-leach Thickener Underflow Pump	No	No	No	Install larger pump motor/VFD	100	Power – 50 Maint – 5.0
CIP Tails Pumps	Yes	No	No	Install larger pump motor/VFD	100	Power – 40 Maint – 5.0

Equipment	26.0 ktpd	28.0 ktpd	30.0 ktpd	Remedy/Action	Projected Capital Costs (C\$000)	Projected Operating Costs (C\$ '000)
Tailings Pump	No	No	No	Option 1 – install larger pump motor/VFD plus booster station	1,800	Power – 850 Maint – 10
Tailings Pump	No	No	No	Option 2 – install tailings thickener	12,500	Power – 5.0 Maint - 25
Lime Circuit	No	No	No	Review/upgrade screw feeder	10	Negligible change
Gravity Circuit	No	No	No	Change cone size to G7 to maximize GRG in short term	100	Negligible change
Intensive Leach Reactor	No	No	No	Piping changes to enable dedicated EW bank coupled with pregnant tank	50	Negligible change
Elution Circuit	No	No	No	Install dedicated fresh water tank and pump	200	Power – 40 Maint – 5.0
Regen Kiln	No	No	No	Install second regen kiln	2,800	Power – 400 Maint - 75
				Total Cost (C\$)	\$21.6 M	\$1.82 M/a

IN PLANT ELECTRICAL REVIEW

In conjunction with the gold plant mill throughput findings, a high level electrical review was conducted to determine if existing electrical capacity and space for installation are available to support the additional equipment and larger motor drives.

Based on a high level review of existing electrical room layouts and motor control centres, it is assumed that space is available for adding/upgrading VFD panel enclosures and replacing or adding new feeder breakers in existing MCC enclosures.

Table 17-7 presents the results of the electrical review and conclusions made with respect to electrical capacity and availability of space for installation of new electrical equipment.

TABLE 17-7 HIGH LEVEL ELECTRICAL REVIEW FOR INSTALLATION OF NEW GOLD PLANT EQUIPMENT

Equipment	Remedy/Action	Comment
Cyclone Feed Pump	Install larger pump motor/VFD	Available capacity and space in 3210-ER-0051 for larger VFDs and breakers VFDS, feeder, breaker and cabling will have to be replaced, upsized.
Trash Screens	Install 3 rd screen, cloth aperture to be 840 µm	Available capacity and space to install and operate new trash screen.
Pre-leach thickener underflow pumps	Install larger pump motor/VFD	Available capacity and space in 3210-ER-0021B for larger VFDs and breakers VFDS, feeder, breaker and cabling will have to be replaced, upsized.
CIP Tailings Pump	Install larger pump motor/VFD	Available capacity and space in 3260-ER-0012B for larger VFDs and breakers VFDS, feeder, breaker and cabling will have to be replaced, upsized.
Tailings Pumps	Install larger pump motor/VFD	Available capacity and space in 3260-ER-0012A for larger VFDs and breakers VFDS, feeder, breaker and cabling will have to be replaced, upsized.
Elution Circuit	Install dedicated fresh water tank and pump	Available capacity and space to install and operate new pump
Carbon Regen Kiln	Install 2 nd regen kiln	Available capacity and space to install and operate new regen kiln. Power reading showed multiple options to power the new regen kiln

INFRASTRUCTURE SUPPORTING THE PLANT EXPANSION

In support of the plant throughput assessments, the infrastructure associated with the increased production rates should be addressed.

MINE PRODUCTION

The mine production capacity should be reviewed along with the plant expansion. A revised mine plan may be required to support the new production rates.

The mill metallurgical team should review the breakage characteristics of the ores to be delivered in the current mine plan. Mill production may be significantly impacted by increases in ore hardness and other changes in ore characteristics.

FRESH WATER PERMITS

Fresh water supply and associated permits should be reviewed and if necessary, new ones obtained. Preliminary mass and water balances were reviewed by Ausenco for the study.

The required fresh water make-up for the expansion stages are 73.3 m³/h for the 24,000 tpd case, 75.2 m³/h for the 26,000 tpd case, 77.1 m³/h for the 28,000 tpd case and 79.0 m³/h for the 30,000 tpd case.

Other areas that should be reviewed include;

- Operating permits
- Tailings reclaim water capacity
- Tailings deposition and storage capacity schedules
- Site power transmission
- Reagent supply chain

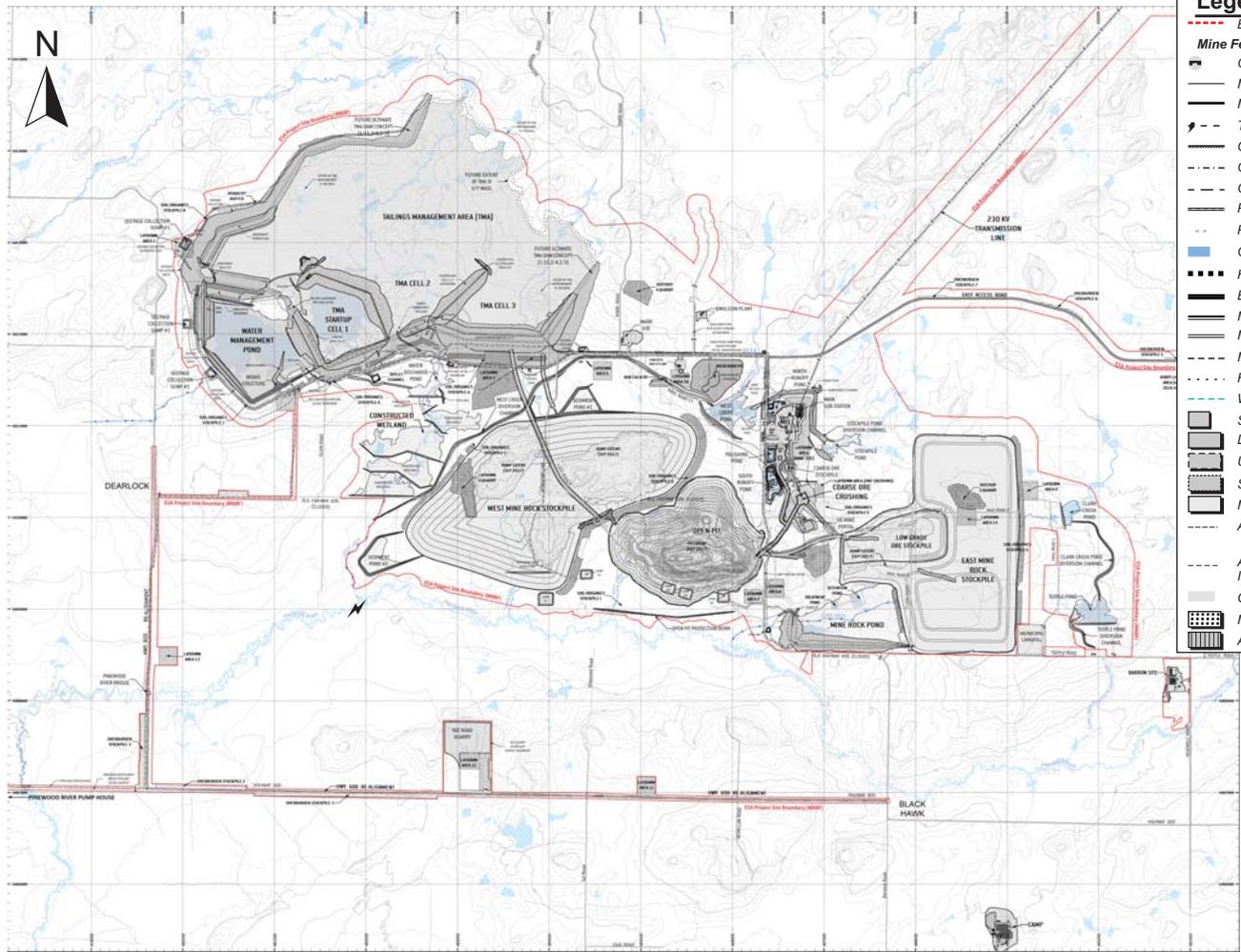
18 PROJECT INFRASTRUCTURE

PRIMARY SITE ACCESS ROADS

The mine site access roads and onsite roads make use of existing roads and easements, upgrading and extending them as required. The main entrance to the site is the east access road, which connects the Korpi Road from Finland (Highway 71) with the Roen Road. Branches of the Roen Road connect the main access road to the plant site to the south and the TMA via the Haul Road 13. A branch to the north provides access to the explosive magazine and the emulsion plant.

The access roads to the TMA are wide enough to accommodate tailings and reclaim water pipelines and light vehicle and truck traffic. The roads are surfaced with crushed stone.

Plant site roads connect the process plant area to the coarse ore stockpile at the primary crusher, the low-grade stockpile, the underground portal, and the open pit. Highway 600 was rerouted around the development area. The Rainy River site plan is presented in Figure 18-1.



Legend:

- ESA Project Site Boundary (MNR)
- Mine Features**
 - Gate
 - Minor Feature Outline
 - Major Feature Outline
 - Transmission Line 230 KV
 - Conveyor
 - On-Site Powerline
 - Generator Power
 - Pipeline
 - Pipe
 - Culvert
 - Hwy 600 Realignment
 - East Access Road
 - Mine Road, Haul Road
 - Mine Service Road
 - Mine Road Centreline
 - Fence
 - Water Elevation
 - Structure
 - Dam
 - Ultimate TMA Dam Concept
 - Starter Pit
 - Mine Feature Footprint
 - Anticipated Extent of TMA Dam Impoundment @377 & 378 M ASL
 - Anticipated Extent of TMA Starter Dam Impoundment @365 & 366 M ASL
 - Channel Diversion Spillway
 - Mine Pond/ Constructed Wetland
 - Aggregates Pit
- Temporary Features**
 - Aggregate, Quarry, Outcrop
 - Top Soil Stockpile
 - Laydown Area
 - Overburden Stockpile
 - Open Pit Limit- First Cut
 - Temporary Pond
 - Quarry Boundary
- Base Features**
 - Existing Hydro Line
 - Road Closure
 - Expressway / Highway
 - Arterial, Collector
 - Local
 - Municipal Drain
 - Preexisting Culvert
 - Intermittent Stream
 - Visible Watercourse
 - Waterbody
 - Major Contour 10M Int
 - Minor Contour 2M Int
 - Municipal Landfill
 - Aggregates Site



Figure 18-1

newgold New Gold
 Rainy River Mine,
 Rainy River, ON, Canada

Rainy River Mine Site Plan

MINE HAUL ROADS

The mine haul roads provide the following:

- Connect the open pit to the overburden and waste rock stockpiles.
- Connect the open pit to the crusher pad, mine facilities, including the truck shop and truck wash buildings and tailings dam.
- Connect the open pit to the tailings dam to haul mine rock for dam construction.

MINE SERVICES FACILITIES

The mine services facilities include the truck shop and truck wash facilities. The facility dimensions were based on Komatsu 830E (220-tonne) class haul truck.

The mine truck shop has a total of six maintenance bays, including two bays for auxiliary vehicles and one bay dedicated to welding. The bays are aligned in a row with one 50 t overhead crane servicing all bays. The building includes a 1,400 m² warehouse and a mechanical workshop that serves as a maintenance area for small vehicles. The workshop is equipped with a 10 t crane. The truck shop has a centralized lubricant distribution system for oils, greases, and other fluids, which feed the lube stations at each bay. Every lube station has compressed air and service water outlets.

The truck wash facility is located approximately 80 m south of the mine shop. The truck wash system has mud settling basins for oil and grease removal and a water filtration system for continuous recycling of wash water.

GENERAL OFFICES AND ASSAY LABORATORY

The office facilities are prefabricated type buildings made up of 12' x 60' modules. There are three main buildings; the administration, the mine office and dry, and the plant office. The office requirements for each building were designed to accommodate the projected staffing requirements. The staffing requirements for the peak year (Year 8, 2025) are presented in Table 18-1.

TABLE 18-1 STAFF REQUIREMENTS

Area	Total Personnel
Site Management	3
Site G&A Human Resources	7
Site G&A Accounting	15
Site G&A IT Services	3
Site G&A Employee Transportation	6
Site G&A Purchasing and Warehousing	20
Site G&A Safety and Security	8
Site G&A Environmental and Regulatory	11
Site G&A Community Relations	6
Mining – Open Pit Administration	33
Mining – Open Pit Engineering	16
Mining – Open Pit Geology	15
Mining – Open Pit Mobile Equipment	232
Mining – Site Services	20
Mining – Underground Administration	29
Mining – Underground Operations	130
Mining - Underground Maintenance	43
Processing – Mill Operations	48
Processing – Assay Lab	17
Maintenance - Management	4
Maintenance – Planning and Engineering	8
Maintenance – Field and Shop	116
Maintenance - Mill	43
Project	2
Total Personnel Including Project	835

MAIN ADMINISTRATION BUILDING

The main administration is located by the mine entrance and houses administration and safety personnel and consists of 10 offices and space for open plan work stations. There is a small kitchen and three meeting rooms.

MINE OFFICE AND DRY

The mine office is located next to the truck shop and houses the mine, maintenance and engineering office staff. The building has dry facilities with lockers, one room for the men and one for the women.

PLANT OFFICE

The plant office is located on the west side of the process building between the leach tanks and the pre-leach thickener and connected to the main mill building via a corridor. The building houses process operations and maintenance staff and has a dry facility as well.

PARKING AREA

Parking is located adjacent to mill building room for 150 vehicles and two buses. During the site visit it was noted that all personnel were being transported to the mine site by bus. Personal vehicles were to be allowed in the weeks following the visit.

ASSAY LAB

The assay laboratory is located adjacent to the administration building and was commissioned in December 2016 through February 2017. The laboratory is designed to process 200 mine blast hole and mill solids samples per day. The assay laboratory has facilities for:

- Sample preparation including weighing, drying, crushing and splitting;
- Fire assaying, including a balance room for weighing final gold and silver buttons;
- Atomic absorption (AA) spectrophotometers for analysis of the gold and silver following fire assay;
- LECO Analyzers for carbon and sulphur analyses;
- Wet chemical lab for solution samples;
- Environmental lab;
- Two offices, a lunch room, and two washrooms.

FUEL STORAGE AND DISPENSING

The fuel island is located near the crusher on the main haul road to the plant. The tank farm is located outside the blast radius of the pit and consists of seven 80,000 L double walled storage tanks for a total capacity of 560,000 L. This volume provides sufficient fuel for six days of operation based on daily fuel consumption of 95,000 L.

The light vehicle fuel station consists of horizontal double walled tanks with the appropriate distribution equipment for light vehicles. Diesel and gasoline are available for light vehicles, which are primarily diesel pick-up trucks.

EXPLOSIVE MAGAZINE STORAGE AND EMULSION PLANT

The explosive magazine storage area and emulsion plant are located on a dedicated road to the north of the east access road. The facilities were constructed and are being operated by the explosive supplier. The explosive magazine is located midway up the road and the emulsion plant is located at the end of the road in an isolated area.

ELECTRICAL POWER AND COMMUNICATIONS

The total power connected for the project is estimated to be 57 MW. The estimated power demand by area is provided in Table 18-2.

TABLE 18-2 POWER CONNECTED

Area	Installed Power (MW)	Standby Power (MW)	Connected Power (MW)
Process Plant and Site Infrastructure	48.6	3.7	44.9
Underground Mine	5.6		5.6
Network Loss (2%)	1.1		1.1
Power Demand Subtotal	55.3		51.6
Security Factor (10%)	5.5		5.5
Total Power	60.8		57.1

Electricity is supplied by a 16.7 km long 230 kV power line and connected to the regions existing 230 kV Hydro One power line currently connecting Fort Frances and Kenora.

The main 230 kV to 13.8 kV substation is located to the northeast of the concentrator building where the large loads are installed. Both dual-pinion SAG and ball mills are equipped with one active front end, VFD with an installed power of 15,000 kW, 7,500 kW per motor. The mills are fed directly from the 13.8 kV switchgear. Two main 230 kV to 13.8 kV, 42/56/70 MVA transformers are used for combined power of 100 MVA. This provides room for future expansion and mitigates the risk of downtime due to transformer failure. A 15 kV Gas Insulated Switchgear, complete with electrical protection devices is included.

Electricity for the underground mine is to be provided by a 13.8 kV line routed down the decline ramps in the mine. Power will be delivered from the main substation by an overhead power line to the mine portal.

EMERGENCY POWER

There are two emergency generators, one supplying 4.16 kV and a second supplying 600 V power. During a power outage, a programmable logic controller (PLC) will manage the critical loads. The loads are grouped into fixed loads such as lighting, heating, sequential loads such as leach tank agitators, cyanide destruction tank, and manually operated loads, such as sump pumps, rake mechanisms, and reactive heating.

The critical 4.16 kV loads include tailings pumps, heat exchanger feed pumps, leach tank agitators and cyanide destruction tank agitators.

The critical 600 V loads include process water pumps, service air compressor, tailings pumps, thickener underflow pumps, sump pumps, and gland seal water pumps.

The two generator systems are connected to the main 4.16 kV and 600 V switchgear.

COMMUNICATION

A fibre optic loop connects all areas of the operation. The fibre optic lines are run on the 13.8 kV overhead distribution lines and transmit voice, video, and data on the following systems:

- Telemetry, data acquisition and control between the process plant and exterior process equipment;
- Computer network between all departments;
- Local telephone lines;
- Computer network for maintenance on all electrical equipment;
- Fire detection;
- Video surveillance and access control systems;
- Electrical tele-protection equipment.

TAILINGS MANAGEMENT AREA

The TMA covers an area of approximately 765 ha and its design is based on an ultimate settled density of 1.4 t/m³, providing storage capacity for the estimated 74 Mm³ of tailings anticipated to be produced over the LOM. The facility is bounded by natural topography in the northeast and by impoundment dams along the remaining perimeter and has the capacity for further expansion.

TAILINGS DEPOSITION PLAN

Tailings will be deposited from spigots located inside the perimeter of the TMA dams in order to develop beaches to protect each of the dams. This will separate the ponds from the dams and allow reclaim of water using barge pumps. After 2024, the spigots will require extension inwards to displace the water in the ponds and minimize the volume of water required to cover the tailings for mine closure.

The starter dam has a crest elevation of 371.5 MASL. The dams will be raised sequentially over the LOM to the final dam height of 23.5 m to contain the design total tonnage of 94.6 Mm³. Table 18-3 presents the design capacity of the TMA by construction stage.

TABLE 18-3 TAILINGS DAM CAPACITY BY STAGE

Parameter	Stage 1, Start-up Dam	Stage 2	Stage 3	Stage 4	Stage 5
Construction Period	Pre-Production	End of Year 2	End of Year 5	End of Year 8	End of Year 11
Dam Crest Elevation, MASL	371.5	371.5	374.5	377.5	379.5
Maximum Height, m	10.5	15.5	18.5	21.5	23.5
Storage Capacity, Mm ³	10.4	33.2	50.5	69.5	94.6

TAILINGS DAM DESIGN

Tailings management dams are designed for the most severe flood and earthquake criteria due to their potential for failure. The site has low to moderate seismic risk with a 0.096 g horizontal peak ground acceleration (PGA) for a 10,000-year return period earthquake.

The design criteria used in the construction of the dams include:

- Short term, end of construction with induced pore pressures: 1.3
- Long term, when excess pore pressures have fully dissipated: 1.5
- Rapid draw down of the Water Management Pond slope: 1.2
- Worst case, with potentially slicken-sided upper varved clay: 1.0
- Pseudo-static loading with a seismic coefficient of 50% of the PGA: 1.0

The design slopes for the impoundment dams were initially 4H:1V for heights up to 15 m for approximately 70% of the dam length. After a review of the design of the Mine Rock Pond (MRP), the design has been changed to require 11H:1V slopes for all of the dams.

The primary dam construction materials are mine waste rock and selected clay from the open pit development, which are available during the preproduction and early periods of operation.

- Zone 1 is select overburden from open pit development that is placed in lifts with controlled compaction to act as the water retaining element or core of the dam;
- Zone 2 is random rock fill (non-potentially acid generating (NPAG) or potentially acid generating (PAG)) or overburden placed and compacted by the mine fleet to for the upstream shell of the dam;
- Zone 3 is NPAG mine rock that forms the downstream shell of the dam and toe berm for stability, South Dam only;
- Zone 4 is select or processed sand filter/drain downstream of the core to protect against cracks or construction defects and as a downstream shell foundation filler;
- Zone 5 is a select sand and gravel or processed rock transition between the Zone 4 filters and the Zone 3 rock fill.

Descriptions of the construction fill materials by zone are presented in Table 18-4.

TABLE 18-4 CONSTRUCTION FILL MATERIALS BY ZONE

Material	Description
Zone 1	Core – select clay
Zone 1A	Core – random clay
Zone 2	Upstream shell – random granular rock fill, (minus 600 mm)
Zone 3	Downstream shell – clean mine rock (minus 900 mm)
Zone 3A	Downstream shell – clean mine rock (minus 450 mm)
Zone 4	Filter sand
Zone 4A	Fine filter sand
Zone 5	Transition/drain – processed rock (minus 75 mm)
Zone 6	Road surfacing – sand and gravel
Zone 7	Upstream erosion protection - cobbles and boulders
Zone 8	Downstream erosion protection – cobbles and boulders
Zone 9	Erosion protection – bedding gravel
Zone 10	Armour Stone
Zone 11	Rip Rap

CONSTRUCTION AND OPERATIONAL CONSIDERATIONS

Overburden within the pit pre-strip area that is suitable for dam core construction, is delineated in the block model. The dam construction schedule is incorporated into the mine plan to make sure that the required construction materials are available when needed.

The design of the dam allows for placement of a significant amount of the construction material using mine haul trucks. The core, filter, drain, and erosion protection zones are constructed by qualified earthworks contractors.

TAILINGS MANAGEMENT AREA DAM ENGINEERING AND CONSTRUCTION SCHEDULE

TMA dam engineering and construction is a continuous process that is scheduled to provide capacity for tailings placement through the LOM. The TMA is divided into a water management pond, which was constructed first to be able to manage water containment and distribution. The water required for start-up of the process facilities was provided by pumping water from the water management pond, which was filled by pumping water from the Pine Wood River. Tailings containment is divided into three cells. Cell 1 is the start-up cell, which provided capacity for mill tailings through March 2018, followed by Cells 2 and 3, the first lifts of which will be constructed by October 2019. Completion of the construction of the first lift of the three cells is considered Stage 1. Subsequent stages will consist of increasing the heights of the dams in all three cells. General constraints on the dam construction schedule are the availability of clay material for the core and placement time required for that material.

The TMA construction schedule is summarized as follows:

Stage 1 Engineering and Construction – Construction of Initial Dams for Cells 1, 2, and 3

- Cell 1 is currently in operation.
- Cell 2 is currently in operation
- Cell 3 is in the construction stage and scheduled to be completed by October 2018. Cell 3 will begin receiving tailings in April 2019. Cell 3 is relatively small and will be filled by September 2019.

Stage 2 Engineering and Construction – Raise Dams to 371.5 m

- Stage 2 consists of raising the West Dam, North Dam, and South Dam from the Stage 1 elevation of 366.5 m to 371.5 m. The start-up cell was constructed to 371.5 m.
- Stage 2 must be ready for operation by October 2019. In order to accomplish this, the engineering must start in November 2017.
- The Stage 2 construction schedule will be constrained by the availability and placement of clay. The clay can be placed at a rate of 100,000 m³/month. The material schedule indicates the need for 596,063 m³ of Zone 1 and 1A.
- The clay must also be placed during the months of June through November. The construction will need to start in July 2018 and finish in August 2019, a 14 month period.

- Work is planned to start on the North Dam followed by the West Dam and South Dam.

Stage 3 Engineering and Construction – Raise Dams to 374.0 m

- Stage 3 engineering is scheduled to begin in November of 2020 followed by 5 months of permitting.
- Stage 3 construction is scheduled to begin in July 2021 and finish in August 2022, a 14-month period. There will be no construction during the winter months of December through March.

Stage 4 Engineering and Construction - Raise Dams to 377.5 m

- Stage 4 engineering will begin in May 2023 to support a construction start date of April 2024.
- Stage 4 construction will be completed by September 2025

Stage 5 Engineering and Construction – Raise Dams to 379.5 m

- Stage 5 engineering will begin in November 2027 to support a construction start date of July 2028.
- Stage 5 construction will be completed by August 2029.
- Stage 5 will provide capacity through December 2031.

MINE ROCK AND OVERBURDEN STOCKPILES – STAGE 1 STUDY

A Stage 1 preliminary scoping level evaluation was performed to determine the most effective method for improvement of the foundation materials beneath the proposed waste dump areas. There are many areas of instability due to soil types and moisture. The study consisted of identifying methods of foundation improvement, preparing preliminary designs for each alternative, preparing cost estimates for each alternative, and then evaluating the results.

The ground improvement methods were evaluated using the following considerations:

- Geology and subsurface conditions;
- Design and performance requirements;
- Site constraints;
- Availability of technology, materials and installation contractors;
- Cost;
- Schedule.

Figure 18-2 is a site layout drawing and a satellite photograph showing the locations of the waste rock and overburden stockpiles.

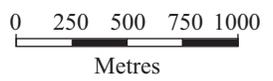
Figure 18-3 is a preliminary layout, plan and section, of the existing wick drain design for the west mine rock/overburden stockpile. The base design consists of a foundation layer or pad, which provides a working platform for installing the prefabricated wick drains. The drains are driven completely through the compressible non-draining overburden soils to the incompressible WST granular till. The stockpile is placed in 3 m lifts alternating between mine rock and overburden. Construction of the external slope of each 3 m lift will be 1H:1V, with an overall slope of 6H:1V. Upon completion of the stockpile, the stockpile will be regarded to a uniform 6H:1V slope to provide positive drainage. Wick drain spacing will be either 2 m or 4 m depending on the material types.



Figure 18-2

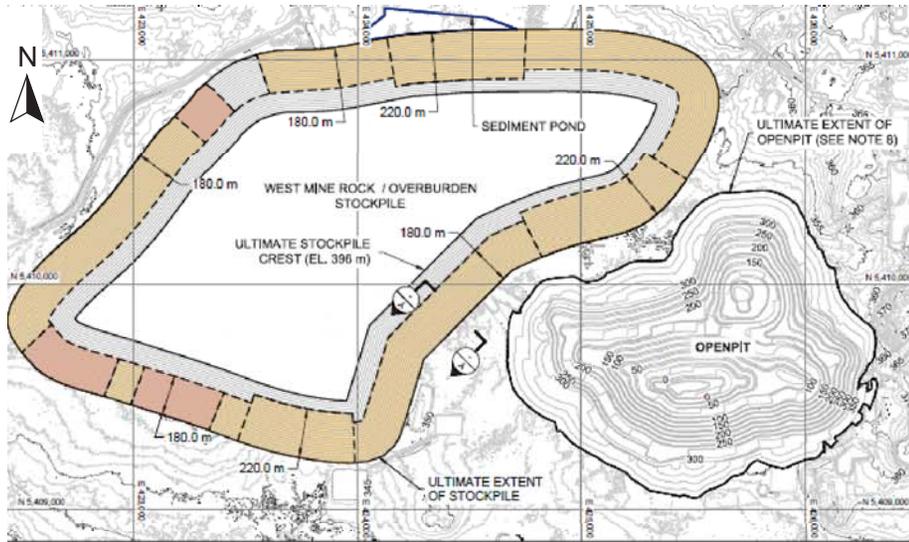
newgold[™] New Gold Rainy River Mine, Rainy River, ON, Canada newgold[™]

Satellite Photograph of Site Layout Showing Stockpile Locations

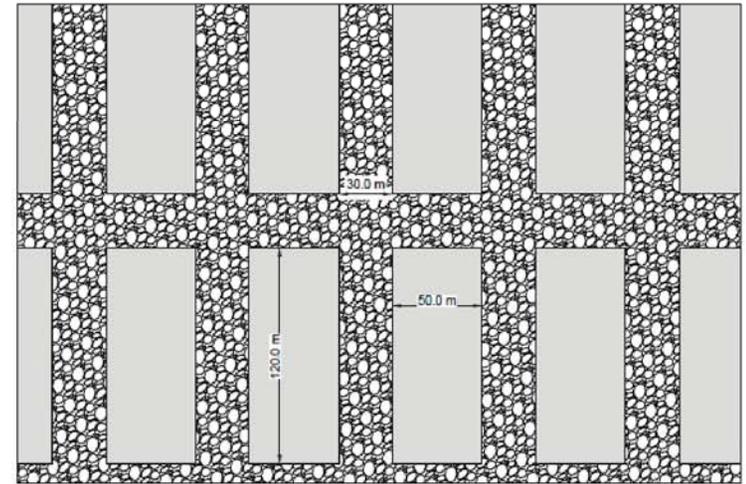


July 2018

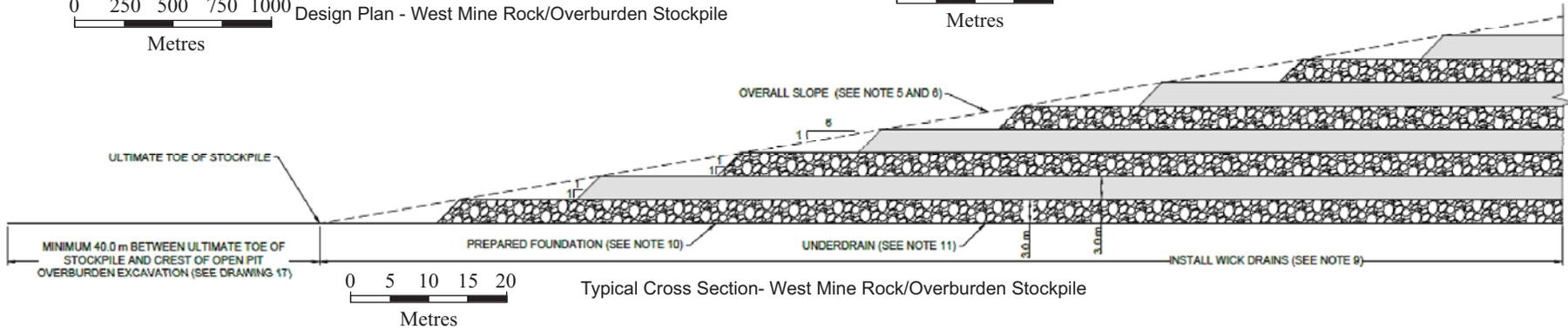
Source: New Gold, 2018.



0 250 500 750 1000 Metres
Design Plan - West Mine Rock/Overburden Stockpile



0 25 50 75 100 Metres
Typical Lift Plan - West Mine Rock/Overburden Stockpile



Typical Cross Section - West Mine Rock/Overburden Stockpile

Legend:

-  Overburden Waste
-  Wick Drains at 4 m Spacing
-  Wick Drains at 2 m Spacing
-  NAG / PAG Waste Rock

July 2018

Figure 18-3

newgold New Gold
Rainy River Mine,
Rainy River, ON, Canada **newgold**

Existing Wick Drain
Design - Base Case

Source: New Gold, 2018.

A variety of ground improvement alternatives were reviewed for suitability at the Rainy River mine. The alternatives that were considered applicable for the clay soils at Rainy River included:

- Wick drains
- Stone columns
- Removal and replacement (shear keys)
- Deep soil mixing
- Electro-osmosis
- Vacuum consolidation
- Flattening the slope

Each of these alternatives was evaluated and the most favourable was selected for further analysis. The pros and cons for each of these alternatives are presented in the following sections.

WICK DRAINS

Advantages:

- Opportunities to optimize the design. The existing waste dump design can be modified to become more cost effective.
- Maximum effective depth of 65 m. Applicable to the waste dump areas.
- Low risk and proven technology.
- Simple. Not difficult to find an experienced contractor.

Disadvantages

- Unsuitable if obstructions exist above the compressible layer. Not a concern in the waste dump areas.
- Requires a construction pad.
- Requires special equipment.

STONE COLUMNS

Advantages:

- Provides drainage similar to wick drains and also provides ground reinforcement.
- Provides uniformity with depth in a given soil.
- Bottom feed dry process puts fill where needed.

Disadvantages

- Unsuitable in soils with cobbles and boulders.
- Fines may intermix and clog columns. A possible concern for the clay foundations.
- Backfill may be costly. Maximum grain size is approximately 75 mm and minimal fines. This will require screening of materials before placement.
- Maximum depth of 30 m.
- Difficult QA/QC.
- Requires a construction pad.
- Requires special equipment.

FLATTENING THE SLOPES

Advantages

- Increases stability.
- Eliminates the need for ground improvement.
- Easy QA/QC.
- Defers capital (trucks) and operating costs until after 2020.

Disadvantages

- Land acquisition.
- Permitting; time required for permitting.
- Additional areas for reclamation and closure.
- Additional trucks are required after 2020.

REMOVAL AND REPLACEMENT OF MATERIALS (SHEAR KEY)

Advantages

- Low cost for shallow depths. Shear key is only practical for areas with clay not exceeding 8 m to 10 m depth.
- Design to desired improvement and stability level.
- Easy QA/QC.

Disadvantages

- Potentially high cost, depending on the volume and depth that requires improvement. The cost increases as the depth of clay increases.
- Increases the volume of waste overburden. Materials excavated from the shear keys must be placed in the waste dumps.

- May require dewatering. Dewatering will likely be required for the project.
- Excavations may temporarily destabilize adjacent ground.
- Requires a contractor to carry out the excavation.

DEEP SOIL MIXING

Advantages:

- Positive ground reinforcement.
- Can be used to depths greater than 30 m (typically 20 m to 30 m). Applicable to the waste dump areas.

Disadvantages

- High cost.
- Requires special equipment. It may be difficult to acquire enough equipment to complete the work in the required time.
- Brittle elements. Failure could be sudden.
- Requires a construction pad.

Electro-osmosis and vacuum consolidation methods were evaluated but considered unacceptable for the Rainy River application and were eliminated from further consideration.

The alternatives that were considered acceptable were further evaluated by preparing preliminary scoping level designs and cost estimates. Figure 18-4 presents the Base Case wick drain design and includes the stratigraphically defined sections. The stratigraphy of each section is defined in Table 18-5.

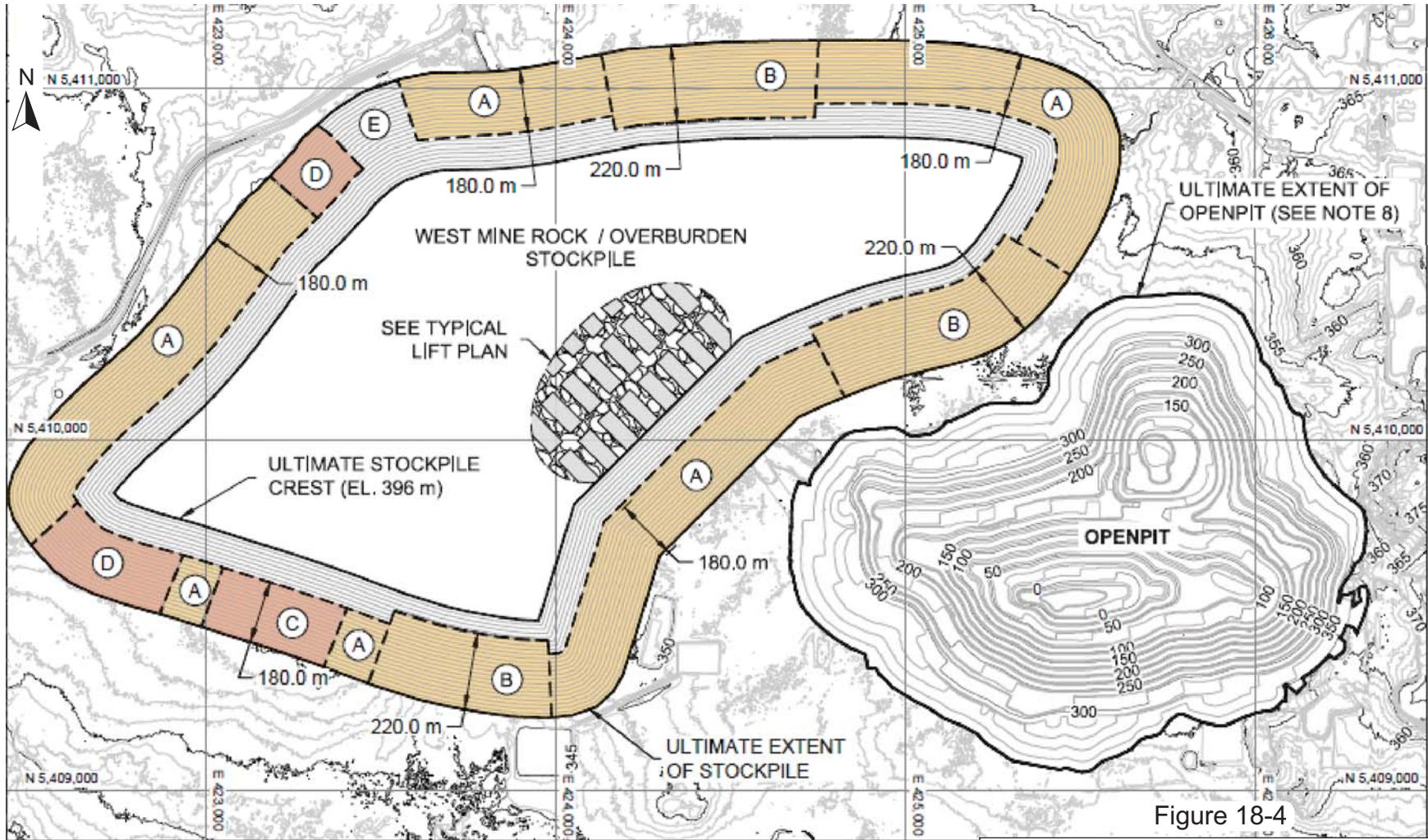


Figure 18-4

Legend:

-  Overburden Waste
-  Wick Drains at 4 m Spacing
-  Wick Drains at 2 m Spacing

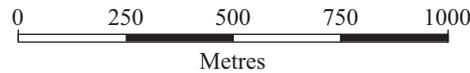


TABLE 18-5 BASE CASE WICK DRAIN DESIGN - STRATIGRAPHY OF STABILITY SECTIONS

Foundation Unit (from top to bottom)	Stratigraphic Thickness (m)				
	Stability Section A (typical)	Stability Section B	Stability Section C	Stability Section D	Stability Section E
BRE Silt	4	3	2	2	2
BRE Clay	1	2	1	1	1
Glaciofluvial/Delta Sand	-	-	-	5	12
WML Clay Till	18	28	3	9	9
WYL Clay	1	1	2	1	1
WST Granular Till	5	5	5	5	5
Bedrock	-	-	-	-	-

Table 18-6 presents the results of Association for the Advancement of Cost Engineering International (AACEI) Class 5 capital cost estimates for each of the ground improvement alternatives. The estimates are based on conceptual designs with an accuracy of -50% to +100% of the price for new construction. The information is suitable for early stage design development.

TABLE 18-6 GROUND IMPROVEMENT ALTERNATIVE STUDY – CLASS 5 COST ESTIMATE

Option	Description	Estimated Cost (\$)
Base Case	Wick Drains	80,841,548
Alternative 1	Removal and Replacement, Section C areas only, with Wick Drains for the remaining sections	76,820,604
Alternative 2	Stone Columns	295,606,884
Alternative 3	Deep Soil Mixing	928,940,015
Alternative 4	Flattening the Slope	80,951,504

FURTHER DEVELOPMENT WORK - STAGE 2 STUDY

Based on the capital cost estimates and schedules for installation, the following alternatives were selected for further investigation in a Stage 2 study.

- The Base Case – Wick drains
- Alternative 1 – Removal and replacement or shear key, only in areas with shallow clay deposits. Wick drains or slope flattening will still be required for zones with deeper clay.
- Alternative 4 – Flattening the slope by expanding the east mine rock stockpile.

Some additional optimization work was performed on the base case wick drain alternative. The ground improvement widths in Section A areas were reduced from 180 m to 125 m by including a 50 m width without wick drains and a 50 m to 75 m width with wick drains. This configuration provides significant cost improvement in the wick drain option. This optimized base case will be investigated in the Stage 2 study.

19 MARKET STUDIES AND CONTRACTS

MARKETS

Gold and silver markets are mature global markets with reputable and refiners located throughout the world.

Gold is a principal metal traded at spot price for immediate delivery. The market for gold is trading 24 hours per day with a location somewhere in the world that is usually open. Gold trading activity takes place in many markets including New York, London, Zurich, Sydney, Tokyo, Hong Kong, and Dubai. Daily prices are quoted on the New York spot market and can be found on www.kitco.com. The average gold price for 2017 was \$1,257 per troy ounce. The three-year and five-year rolling average prices through the end of December 2017 are \$1,222 and \$1,269 per troy ounce, respectively. This Technical Report uses the current bank consensus price for gold of \$1,300.00 per troy ounce for reserve reporting.

Silver trading is similar to gold. The market for silver is trading 24 hours per day. Daily prices are quoted on the New York spot market and can be found on www.kitcosilver.com. The average silver price for 2017 was \$17.05 per troy ounce. The three-year and five-year rolling average prices through the end of December 2017 are \$16.62 and \$18.55 per troy ounce, respectively. This Technical Report uses the current bank consensus price for silver of \$17.00 per troy ounce for reserve reporting.

The mill at Rainy River is expected to produce an annual average over LOM of 283,000 ounces of gold and 498,000 ounces of silver per year in the form of doré.

CONTRACTS

The doré will be shipped for refining by one of the internationally-established refiners. Local Canadian refineries include Asahi Refining in Brampton, Ontario or the Royal Canadian Mint in Ottawa, Ontario or Winnipeg, Manitoba.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

New Gold has made a significant commitment to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against their Health, Safety, Environment and Corporate Social Responsibility Policy last reviewed and approved by their Board of Directors on July 25, 2018. Key Elements include:

Health and Safety – This includes promoting and protecting the well-being of employees through safety-first work practices and a culture of safety excellence. Commitment to leading industry practices and systems in health and safety that focus on prevention of accidents and incidents are a priority.

Environment – This includes a commitment to preserving the long-term health and viability of the natural environments affected by the project and operation.

Labour Practices – This includes a commitment to upholding fair employment practices and encouraging a diverse workforce, where people are treated with respect and are supported to realize their full potential.

Community Engagement and Development – This requires a commitment to establishing relationships based on mutual benefit and active engagement with host communities to contribute to healthy communities and development.

At the time of this review, the Mine Environmental Department was adequately staffed, but resources appeared to be stretched. Subsequent to the site visit on October 2 to 4, 2017, the environmental lead for the Mine retired. This role and responsibilities have been assumed by internal staff. New Gold contracts AMEC to do inspections and third-party groundwater monitoring.

At the time of the site visit, the Mine reported two non-compliance related issues associated with an effluent discharge in excess of permit limits and failing to report an effluent discharge above permit limits. New Gold pleaded guilty to the charges and was fined \$187,500.

The Mine environmental budget for 2018 was reviewed and appears adequate/reasonable for the size of operation, issued permits, and conditions imposed.

ENVIRONMENTAL STUDIES

New Gold has collected and documented pre-project baseline environmental conditions to better design the Mine and obtain the various permits/authorizations required to avoid and/or mitigate environmental impacts. This information will be used to assess compliance with issued permits and to determine reclamation and closure success.

The Feasibility Study and permitting did not reveal any environmental aspects that are considered limiting to permitted development.

METEOROLOGY AND AIR QUALITY

The closest Environment Canada climate station to the Rainy River Mine for which long term records are available, is located at Barwick, Ontario. This station is located 23 km southwest from the site, at UTM coordinates 428,807E and 5,387,043N; and has climate records dating back to 1978. Mean monthly temperatures range from a low of -15.9°C in January to a high of 18.8°C in July. The mean annual precipitation for Barwick is 695.7 mm, with 79.5% of this average value occurring as rain. June is typically the wettest month. This information has been supplemented and correlated with an on-site weather station established in 2009.

There are no large urban centres and industrial sources near the Rainy River Mine site. Background air quality and sound levels are typical of rural, low-density agricultural regions. Air quality in the Rainy River Mine area is influenced by long-range transport of air emissions from the south and by volatile organic emissions from vegetation and natural fires. The greatest impact on air quality is increased particulate matter generated from traffic, logging/cattle ranching operations and borehole drilling projects. Background air quality data was compiled for the Rainy River Mine site from regional monitoring stations that provide multi-year data for a variety of parameters. All parameters assessed during the permitting process were within

the acceptable levels for the National Ambient Air Quality Objectives. Background particulate matter data is collected from two on site monitoring stations.

SOILS

Soils in the Mine area are generally comprised of gray luvisols, gleysols, humisols, and rockland soils, with lesser expressions of podzolic and brunisolic soils. Gray luvisols are typically clay or clay/silt rich and imperfectly drained. Gleysols are poorly drained/frequently saturated, and in the Mine area generally consist of silt loams to coarser textured soils. Humisols (organic soils) are associated with wetland systems. From a textural perspective, the majority of project soils on the project site consist of clay and clay loam soils, with lesser quantities of sandy clay loam, sandy loam, loam, silt loam and silty clay.

Soils in the project area are overwhelmingly calcareous, due to the nature of the parent material. Organic soils, where present, are acidic in composition. Cation exchange capacity tends to be relatively high because of the elevated organic and clay content of most soils present. Soil metal contents were typical of expected background soil conditions for the soil types present.

GEOCHEMISTRY

AMEC conducted environmental geochemical characterization testwork of selected bulk rock samples representative of the mine rock, overburden open pit and tailings. Testing was carried out on three simulated tailings materials and a total of 659 deposit-wide mine rock samples, of which 366 represent in-pit non-ore mine rock.

Geochemical studies on mine rock indicate that approximately half the samples have the potential to produce acid rock drainage. A block model was developed to refine the estimated tonnage of PAG rock. Study results indicated that the contents of metals found in mine rock samples were typical for their rock types and the risk for metals leaching, under neutral conditions, was projected to be low. Humidity cell analysis was performed on the mine rock samples to evaluate the long-term metal leaching characteristics of these materials. Results identified a potential risk of short term neutral metal leaching (cadmium and zinc) from the tailings. These results suggest that a simple lime treatment system may be required for the discharge of effluent from the tailings management to the water management pond during operations phase.

Results of the mine rock and tailings testwork indicate a future risk for acid rock drainage from a portion of the mineralized waste rock, if not appropriately managed. The mine design has taken this into account during the operation and closure of the east mine rock stockpile and TMA.

HYDROLOGY

Much of the Mine area has been cleared over the years for timber harvesting and cattle ranching. Some of the natural drainage systems have been altered near the project site by the development of agricultural drains and ongoing beaver activities. The Pinewood River is a low gradient system with a watershed of 575 km². Local creek catchments draining to the Pinewood River range in size from less than 10 km² to approximately 25 km². The creeks generally originate in rocky uplands, but also frequently originate from or pass through headwater wetland systems. Peak stream flows occur in the spring, with a secondary smaller peak flow in the late fall. Low flows occur in the late winter, and more variably during the late summer and early fall. The average annual runoff for the Rainy River Mine site area is approximately 195 mm.

Hydrological systems to the northeast (upstream) of the project site show an abrupt transition to larger lake systems in bedrock-dominated terrain. This terrain is remote from project development areas with the exception of the transmission line corridor that passes through this northeast area, but will not directly impact any lakes.

Groundwater base flow to area creeks is limited due to the prevailing clay substrates. In certain years, zero or near zero flows are experienced in both local creeks and the Pinewood River during late winter and late summer periods.

HYDROGEOLOGY

The groundwater regime is governed by the overall structure and hydraulic properties of the overburden and bedrock sequences, and by the local topography and associated surface watercourses. A network of over 100 installations has been used to assess site area groundwater conditions, consisting of monitoring wells, test wells, drill holes, auger holes, mini-piezometers and cone penetration test holes.

Based on model applications and sensitivity analyses performed by AMEC, the predicted groundwater seepage rates into the open pit and underground workings are expected to be in the order of 2,900 m³/d to 3,900 m³/d at full open pit development. The predicted drawdown cone from dewatering of the open pit is predicted to extend approximately three kilometres to four kilometres in all directions from the pit by the end of mining. No private wells are located within the current estimated drawdown cone. Parts of the Pinewood River and several of its tributaries also lie within the projected drawdown cone. The impact of a reduction in groundwater discharge to the Pinewood River is expected to be minor and difficult to measure, given that flow in these sections of the river and its tributaries is dominated by surface water runoff contributions and these features can often be dry. A network of monitoring wells and surface water level stations have been constructed, to monitor and confirm these projections.

Based on the high clay content of the Keewatin Till and the associated glacial Lake Agassiz sediments, the local creek and wetland systems are expected to be perched, and not connected or affected by open pit dewatering.

SURFACE WATER

Numerous surface water sampling stations were established for the Rainy River Mine during baseline data collection. Surface stations are located both upstream and downstream of current plant and mine facilities, positioned according to evaluated impact of the local drainage systems.

Surface water quality in the project area has been characterized as quite good, with all parameters typically meeting applicable water quality objectives/guidelines for the protection of aquatic life, except for:

- iron, aluminum and phosphorus, which commonly exceeded their respective objectives;
- cadmium, copper and cobalt, which occasionally to commonly exceeded either Federal or Provincial objectives; and
- occasional to rare exceedances for arsenic, lead, nickel and zinc.

Increased coliform levels were also noted at some monitoring sites that may be related to area cattle operations and cattle foraging activities. It is not unusual for baseline water quality to exceed objectives/guidelines for various metals due to: high suspended solids loadings in

some samples, naturally elevated metal content in the local soil and rock, and seasonal ion concentration processes involving ice formation in winter and evaporative process in summer. Erodible clay/silt soils, cattle activity, and low creek base flow conditions (making them prone to ion concentration effects), are all contributing factors to observed water quality conditions.

SEDIMENTS

Sediment quality samples were, collected from 2008 to 2013 from various upstream and downstream stations and, analyzed for a suite of parameters. Sediment quality is generally good with parameters found below Federal guideline values, and below Provincial sediment quality guideline lowest effect levels.

GROUNDWATER

Groundwater quality samples are collected from monitoring wells on the project site. Samples are analyzed for a complete suite of parameters and compared with Federal and Provincial objectives and guidelines for the protection of aquatic life, and permit standards. Groundwater baseline water quality reflects the naturally elevated metal content in the local soil and rock. Results from Municipal and private wells showed generally good water quality, with occasional exceedances of for some parameters.

AQUATIC RESOURCES

The mine site area is somewhat unique from an environmental perspective, in that there are no lakes located within, or adjacent to the site. While limited bait fishing does occur within certain area creeks, the area does not support a significant commercial or recreational fishery. In addition, the creeks present within the site often encounter zero flow during dry periods.

Multi-season studies of fisheries and aquatic resources are carried out for the project site area. In the general vicinity, the Pinewood River shows typical widths of 10 m to 15 m, with wider sections associated with beaver impoundments and drowned oxbows. Summer water depths are typically 0.9 m to 1.7 m, with maximum summer water depths in the order of 2 m. Substrates consist of clays and silts, with some detritus. Gravel, rock or cobble substrates are sparse and contribute little to in-stream habitat/cover for fish. Turbidity is high because of erosion of the clay and silt substrates, and agricultural drainage inputs. Beaver dams are frequent and present periodic obstacles to fish passageways in the local study area.

The smaller creeks/Municipal drains that flow into the Pinewood River (Loslo Creek/Cowser Drain, Marr Creek, West Creek, Clark Creek/Teepie Drain, Tait Creek and Blackhawk Creek) typically exhibit summer widths of 0.5 m to 3 m, except where they are impounded by beaver dams, with upper creek reaches being smaller, generally from less than 0.5 m to 1.5 m and frequently exhibiting intermittent flow. Headwater areas of many of these tributary creek systems are associated with wetland systems. Beaver impoundments are frequent.

Large-bodied fish species (Northern Pike, Brown Bullhead and White Sucker) were found only in the Pinewood River and not in the smaller tributaries, with the exception of White Sucker (also found in Loslo Creek and Clark Creek). Walleye and Yellow Perch occur further downstream in the Pinewood River, but not in the general area of the project area site. Lake Sturgeon, classified provincially as threatened (Endangered Species Act) in the area, were not present within the project site area during the baseline surveys. However, three Lake Sturgeon were caught during the spring of 2013, approximately 27 km downstream of the current open pit. As this was the result of a focused effort at regulatory request, it potentially indicates a lack of suitable habitat and/or a small spawning population specific to the lower Pinewood River.

Small-bodied fish of several species are abundant within the Pinewood River mainstem, as well as in its tributaries. These tributaries likely provide seasonal refuge from predators and contribute to the overall productivity of the Pinewood River system. All fish species present in the system are spring/early summer spawners.

Area benthic communities exhibit a low-to-moderate abundance, with a relatively poor representation of taxa used as indices for characterization and comparison of benthic invertebrate communities, due to the lack of larger substrate particles and dominant clay-silt conditions.

VEGETATION COMMUNITIES

The Rainy River Mine site area occurs within the Agassiz Clay Plain Ecoregion that extends from Lake of the Woods in the west to Fort Frances in the east, and from the United States border northward. The Pinewood River watershed is dominated by mixed Poplar and Black Spruce forests, and by non-forested areas (mainly agricultural lands), together with wetlands. The local area shows an even greater preponderance of mixed poplar forests that occupy more than 50% of the landscape, together with wetlands and agricultural lands.

Wetlands are comprised mainly of treed and open fens, together with wetland thickets and marsh areas. Agricultural lands are mainly pasture and hay fields. Poplar forests, comprised principally of Trembling Aspen, are indicative of disturbed lands as Trembling Aspen is a successional species in Ontario.

Only two provincially rare species have been identified in the local area: New England Violet and Field Sedge. Muskrout, another rare species, have been identified as being present historically. There are no specific approvals related to these species.

WILDLIFE

Wildlife surveys were carried out for the Mine site area. These surveys were focused principally on birds due to regulatory requirements; and to a lesser extent on mammals, amphibians and dragonflies/damselflies. A proactive focus was placed on Species at Risk assessment and permitting planning by Rainy River Resources and the Ministry of Natural Resources and Forestry.

Focused surveys have been conducted on forest birds, breeding birds, owls, marsh birds and waterfowl species, Sharp-tailed Grouse, nocturnal avian species (Whip-poor-will, Common Nighthawk and owls), raptors and amphibians, using established survey protocols. The relative diversity of high avian species present in the area reflects the mosaic of principal habitats in the areas (forest, wetlands, fields and shrublands), and the transitional (or near transitional) position of the study area relative to the Great Lakes, Boreal and Prairie regions. The Mississippi Flyway passes over the regional study area. However, the Mine site/area is not considered an important migratory stopover location, as very low numbers of migrating waterfowl, raptors, shorebirds and songbirds were recorded.

Twenty-two mammal species have been identified in the Mine site area through direct observation, trapping records or sign. Species of cultural significance protected under the Fish and Wildlife Conservation Act, including game animals (Black Bear, Snowshoe Hare, Moose, Elk and White-tailed Deer), furbearers (Red Squirrel, Beaver, Muskrat, weasels, American Mink, American Marten, Fisher, River Otter, Bobcat, Lynx, wolf and Red Fox) and specially protected mammalian guilds (bats, shrews and chipmunks), have been recorded in the local study area. Three commercial trap lines overlap with the local area. Fur returns for these trap

lines for the period of 1993 through 2008 indicated that Beaver, American Marten, Red Fox, Otter, Fisher and Mink are the most frequent and valued furbearers taken.

Amphibian and reptile surveys identified eight frog species and three reptile species. No salamander species were observed. Twelve species of dragonflies/damselflies were observed, or are known to occur, in or adjacent to the project area site, three of which are provincially rare (Horned Clubtail, Arrowhead Spiketail and Green-faced Clubtail) but do not require special permit or authorization considerations.

SPECIES AT RISK AND CRITICAL HABITAT

The Species at Risk known to occur in the Mine site area listed in Table 20-1. The company worked closely with the Ministry of Natural Resources in support of meeting permitting requirements of the Ontario Endangered Species Act.

TABLE 20-1 SPECIES AT RISK

Species Common Name	Endangered Species Act	Species at Risk Act
Birds		
Barn Swallow	Threatened	-
Bobolink	Threatened	-
Whip-poor-will	Threatened	Threatened
American White Pelican	Threatened	Not at Risk
Bald Eagle	Special Concern	Not at Risk
Canada Warbler	Special Concern	Threatened
Common Nighthawk	Special Concern	Threatened
Golden-winged Warbler	Special Concern	Threatened
Olive-sided Flycatcher	Special Concern	Threatened
Peregrine Falcon (migrant)	Special Concern	Special Concern
Red-headed Woodpecker	Special Concern	Threatened
Short-eared Owl	Special Concern	Special Concern
Mammals		
Little Brown Myotis (bat)	Endangered	-
Northern Myotis (bat)	Endangered	-
Reptiles		
Snapping Turtle	Special Concern	Special Concern

TRADITIONAL KNOWLEDGE AND TRADITIONAL LAND USE (SOCIAL LICENSE)

Traditional Knowledge (TK) and Traditional Land Use (TLU) sessions were held with several of the consultation and notification Indigenous groups, including: Rainy River First Nations,

Naicatchewenin First Nation, Big Grassy First Nation, Couchiching First Nation, Mitaanjigamiing (Stanjikoming) First Nation, Seine River First Nation and the Métis Nation of Ontario.

The TK/TLU studies and relations are led by Ms. Stacey Jack, the Project Human Resources and Community Manager. Ms. Jack is a licensed Archaeologist, who, as a resident of the District and member of the Couchiching First Nation, has extensive knowledge of regional history. She has worked extensively with area First Nations over the past 20 years, including a leadership role in the development of the Manitou Mounds National Historic Site that is located approximately 35 km south of the Mine site. In support of the TK/TLU studies, data sharing agreements were signed with these First Nations to ensure that sensitive information is protected and held strictly between the First Nations and RRR.

Through these consultations, no traditional activities were identified within the project area. The company has committed to undertaking a joint water quality monitoring and reporting program with the area First Nations as part of the existing monthly water quality monitoring program carried out on the project site. The program is funded by the company and forms an integral part of the overall environmental management program as it relates to First Nations TK, and assurances of maintaining water quality and by extension aquatic life protection.

CULTURAL HERITAGE

The cultural pre-European contact history of the project area is similar to that in eastern Manitoba and northern Minnesota and can be divided into the following generalized temporal and cultural sequences: Late Paleo (circa 9000 to 6000 BC), Shield Archaic (circa 6000 to 500 BC) and Middle/Late Woodland (circa 500 BC to AD 1600), and Historic (circa AD 1600 to present).

Inventoried and registered sites have been scheduled for mitigation according to provincial archaeological assessment requirements for developing sites.

OVERALL ENVIRONMENTAL SENSITIVITIES

There are a number of Species at Risk known or expected to occur within the project footprint. Avoidance of critical Species at Risk habitat is one of the mitigation measures that has already been incorporated into the Project design to a practical level. Based on information currently

available, Provincial Species at Risk Permits, pursuant to requirements of the Endangered Species Act, are required for Whip-poor-will and Bobolink.

There are no Federal lands within the Mine footprint and Federal Species at Risk Permits will not be required. There are no Areas of Natural Scientific and Interest, Environmentally Sensitive Areas, Provincially Significant Wetlands, or Federal or Provincial parks within the Mine area.

All of this information is tracked through the Mine's GIS and Environmental Management System(s).

Environmental staffing and operating budget was reviewed. Level(s), amounts and allowances are considered reasonable, with no obvious categorical omissions noted. Contingency planning for loss of staff and cross-training of job responsibilities needs to be tracked and maintained.

There are no known or listed Legacy Mining Environmental Issues. The only historical site issues of note are considered to be minor and associated with old homesteads (i.e., lead based paint and asbestos containing materials). These have been documented and cleaned up in advance of construction of planned mine facilities.

PROJECT PERMITTING

The Mine has all of the permits and authorizations to construct and operate. Specific permits and authorizations are provided in Table 20-2.

TABLE 20-2 PERMIT LIST

Title	Permit Type
Tailings Management Area Construction	Permit to Take Water
Aggregate Dewatering OC3_4 Roen Pit	Permit to Take Water
Construction Phase Minor Takings	Permit to Take Water
Aggregate Dewatering - Tait Quarry	Permit to Take Water
Mine Dewatering	Permit to Take Water
Pinewood River	Permit to Take Water
Endangered Species Act Permit	Endangered Species Act
Air and Noise	Environmental Compliance Approval
Fisheries Act Authorization 35(2)(b)	Authorization
Clark Creek Diversion	Work Permit – Letter of Authority
West Creek Diversion	Work Permit – Letter of Authority
Culverts C2, C6, C8	Work Permit – Letter of Authority
Culvert Crossing CPL5	Work Permit – Letter of Authority
Pinewood River	Work Permit – Letter of Authority
Culvert Crossing C-15	Work Permit – Letter of Authority
Mine Rock Pond	Work Permit – Letter of Authority
Tailings Management Area	Work Permit – Letter of Authority
West Creek Natural Channel Temporary Bypass	Work Permit – Letter of Authority
Aggregate Resources – Tait Quarry	Aggregate Resources Licence
Fish Collection Permits	Authorization
Wildlife Scientific Collectors Authorization	Authorization
Authorization for Wildlife Interference	Authorization
Nuclear Substance and Radiation Device Licence	Nuclear Radiation Licence
Electricity Wholesaler Licence	Electricity Wholesaler Licence
Overlapping Agreement with Resolute FP	Forestry Resource Licence
Licence to Harvest	Forestry Resource Licence
Authority to Haul	Forestry Resource Licence
Transmission Line - Linear Structure	Aeronautic Obstruction Clearance
Land Use Permits	Land Use Permit
Provincial EA Commitments	Environmental Assessment
Federal EA Commitments	Environmental Assessment
Follow Up Monitoring EA Commitments	Environmental Assessment
Final EA Commitments	Environmental Assessment
Closure Plan Commitments	Environmental Assessment

New Gold is developing an environmental management system (EMS) and has an established GIS to manage permit and environmental conditions at the Mine.

New Gold reports generally good compliance with issued permits. The only exception referenced was an ammonia, total suspended solids, Notice of Violation for water quality in the Mine Rock Pond.

SOCIAL OR COMMUNITY REQUIREMENTS

The Mine tracks and reports good standing with the local community, including local First Nations bands and the Métis Nation of Ontario. They have signed Participation Agreement(s) with the impacted local Indigenous groups. The Participation Agreements covers benefits to impacted party(s) through:

- Environmental Matters
- Human Resources, Employment and Training
- Education
- Business/Contract Opportunities
- Financial Considerations.

As of December 2017, the mine work force was 26% Indigenous.

MINE CLOSURE

New Gold has developed a reclamation and closure plan that satisfies corporate and regulatory requirements. Reclamation and closure planning requires regular review and update to accommodate change(s) in planned and constructed facilities/operations.

New Gold submitted the closure plan for review in January 2015 as the Rainy River Mine entered Phase 2 (Construction) of its development. Public comments and responses followed and the closure plan (the 2015 Closure Plan) was filed by the Ministry of Natural Development and Mines (MNDM) on February 23, 2015. Closure Plan Amendment #1 was prepared and submitted in February 2016 to document responses to outstanding comments. The Closure

Plan has included consultation with agencies, the Aboriginal Community(s) and the public. These consultations will continue through (and beyond) closure.

Major Components of the Closure Plan include:

- Open Pit
- Mine Rock Stockpiles
- TMA
- Plant Site
- Water Management Ponds and Creek Diversions

For closure planning and financial assurance calculation purposes, New Gold has divided the Rainy River Mine into six distinct phases:

- Phase 1 – Construction
- Phase 2 – Construction and Pre-production
- Phase 3 – Start of Production (Open Pit Mining and Processing)
- Phase 4 – Start of Production Underground Mining and Expansion of Existing Facilities
- Phase 5 – Processing of Low Grade Ore
- Phase 6 – End of Operation/Post Closure

Listed below are specific planned closure objectives:

- Prevent, reduce or mitigate residual adverse environmental effects associated with each phase of the Rainy River Mine, including closure and post-closure phases;
- Provide for the reclamation of all affected sites and landscapes to a stable and safe condition;
- Reduce the need for long term monitoring and maintenance by designing for closure and instituting progressive reclamation, as possible;
- Provide for long term monitoring and maintenance of areas affected by the Rainy River Mine where appropriate;
- Provide for mine closure using current available proven technologies in a manner consistent with sustainable development; and
- Meet applicable regulatory requirements, including the Mine Reclamation Code of Ontario (the Code) under Ontario Regulation 240/00 of the Ontario Mining Act.

Post Closure activities will occur over a 75 + year period and include the following:

- Active reclamation for two years post operations;

-
- More intensive monitoring and maintenance for the first ten years post operations;
 - Diversion of stockpile runoff / excess TMA water to the enhance flooding of the open pit 10 years after flooding commences;
 - Monitoring, inspection and ongoing site maintenance until open pit flooding is completed at approximately 75 years post operations; followed by
 - Final close out of the site, with ongoing inspection and maintenance as needed.

The cost estimate for implementing project closure in the Environmental Assessment was estimated to be C\$175 million, and assumed third party implementation costs, no resale or scrap values, and that all materials will be treated as waste. Certain items, such as mobile equipment may have residual resale value. Financial assurance will be phased in over the life of the Mine. New Gold will prepare and submit an update(s) to the reclamation costs and if needed, a Closure Plan amendment. The cost update(s) will reflect new knowledge, progressive reclamation, project changes if any, as well as any changes to costing due to inflation. The financial assurance provided to MNDM will also be increased as needed at that time, although there is the potential that a request may be made for reduction to reflect completed progressive/concurrent reclamation activities. The current obligated financial assurance obligation/commitment is C\$97 million.

Financial assurance (Instrument) for closure of the Rainy River Mine has been provided to the Crown, as represented by MNDM that recognizes a phased approach consistent with operation, development, and disturbance with the Closure Plan submitted for filing with the Province.

New Gold recognizes that closure costs must be reassessed to determine present-time rates within the terms of the Mining Act Regulations. Any addition to, or refunding of, financial assurances would be undertaken following acceptance of the assessment report and discussions with the Director of the MNDM. Financial and commercial information required to support the financial assurance for closure has been provided by New Gold to the responsible Agency(s).

21 CAPITAL AND OPERATING COSTS

CAPITAL COSTS

SUMMARY

Total LOM capital costs are estimated to total C\$1,344 million as summarized in Table 21-1.

Details for each category follow in this section and include data source.

TABLE 21-1 CAPITAL COSTS SUMMARY

Description	Unit	2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Open Pit	C\$(000)	76,734	133,329	71,510	42,442	41,670	31,903	67,888	465,476
Underground	C\$(000)	24,516	58,380	95,454	100,472	20,597	16,397	109,508	425,324
Process/Tailings	C\$(000)	129,145	54,436	7,240	20,341	26,850	1,576	108,277	347,864
Infrastructure	C\$(000)	3,197	680	1,290	150	440	190	2,040	7,987
Reclamation/Closure	C\$(000)	126	126	-	-	-	-	73,521	96,956 ¹
Grand Total	C\$(000)	233,717	246,950	175,494	163,404	89,557	50,067	361,234	1,343,607
UG Project Capital	C\$(000)	24,516	48,833	-	-	-	-	-	73,350
Sustaining	C\$(000)	209,201	198,117	175,494	163,404	89,557	50,067	361,234	1,270,258
Grand Total	C\$(000)	233,717	246,950	175,494	163,404	89,557	50,067	361,234	1,343,607

Note:

¹Contains \$23 million of post mining final closure costs in 2032.

OPEN PIT CAPITAL COST ESTIMATE

The open pit capital cost is estimated to total C\$465 million as summarized in Table 21-2 based on 2017 cost estimates (New Gold, 2017a).

TABLE 21-2 OPEN PIT 2017 CAPITAL COSTS

Description	Unit	2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Mining Equipment	C\$(000)	4,363	36,183	2,933	20,383	19,213	9,742	18,800	111,616
Mining Equipment Rebuilds	C\$(000)	18,684	14,605	16,853	20,784	19,741	21,911	46,720	159,299
Other Equipment (Dispatch System)	C\$(000)	754	-	-	-	-	-	-	754
Mine Infrastructure	C\$(000)	16,661	82,541	2,000	1,000	-	-	-	102,202
Mobile Maintenance	C\$(000)	289	-	417	275	2,717	250	2,367	6,315
Capitalized Mining Costs	C\$(000)	35,982	-	49,307	-	-	-	-	85,289
Grand Total	C\$(000)	76,734	133,329	71,510	42,442	41,670	31,903	67,888	465,476

The capital cost estimate is considered to be appropriate for the open pit operation.

UNDERGROUND CAPITAL COST ESTIMATE

The underground capital cost is estimated to total C\$425 million with C\$73 million in project capital and C\$352 million in sustaining capital as summarized in Table 21-3 based on 2017 estimates (New Gold, 2018e).

TABLE 21-3 UNDERGROUND 2017 CAPITAL COSTS

Description	Unit	2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Mobile Equipment	C\$(000)	-	-	-	31,107	-	2,063	4,713	37,888
-Rebuild/replace	C\$(000)	-	-	210	-	-	210	40,883	41,302
Stationary Equipment	C\$(000)	-	1,164	13,442	9,370	4,966	2,124	3,832	34,898
-Rebuild/replace	C\$(000)	-	-	478	1,176	1,551	2,187	16,708	22,101
Capital Development	C\$(000)	-	5,935	51,707	37,018	9,400	5,806	27,945	137,810
Capitalized Operating Cost	C\$(000)	1,833	8,021	-	-	-	-	-	9,854
Infrastructure	C\$(000)	-	22	4,353	2,071	-	-	-	6,446
Shop CAF and Services	C\$(000)	-	-	4,468	2,234	-	-	-	6,703
Portal	C\$(000)	22,683	40,813	-	-	-	-	-	63,491
Owners Costs	C\$(000)	-	855	4,995	3,897	1,496	1,737	4,127	17,106
Indirects	C\$(000)	-	116	1,789	4,551	497	429	2,964	10,347
Contingency	C\$(000)	-	1,455	14,011	9,048	2,687	1,842	8,336	37,379
Grand Total	C\$(000)	24,516	58,380	95,454	100,472	20,597	16,397	109,508	425,324
Project Capital	C\$(000)	24,516	48,833	-	-	-	-	-	73,350
Sustaining	C\$(000)	-	9,546	95,454	100,472	20,597	16,397	109,508	351,975
Grand Total	C\$(000)	24,516	58,380	95,454	100,472	20,597	16,397	109,508	425,324

The capital cost estimate has been prepared based upon quotations for mobile and stationary equipment and for contracted development in the first three years of the mine life. Capital development costs for the period after the contracted development were prepared based on first principle cost estimates for mine development.

The indirect costs include costs for freight, spares, and commissioning. The owner's costs include construction and contract labour, consulting engineering, field engineering and camp costs for capital projects. There is an allowance of C\$2.5 million for underground mine closure included in the owner's costs.

The contingency allowance was based upon

- 15% of the fixed equipment costs
- 15% of underground costs (excluding mobile equipment purchase)
- 15% of construction costs and indirect costs and
- 0% of Owner's costs, mobile equipment purchase, and mobile equipment replacement and rebuilds.

The overall contingency in the estimate is 12% of the capital costs. The capital cost estimate is considered to be appropriate for the underground project.

PROCESS AND TAILINGS CAPITAL COST ESTIMATE

Table 21-4 presents the process plant, process maintenance, and TMA engineering and construction sustaining capital costs from 2018 through 2025 and the total sustaining capital costs from 2018 to 2021 based on 2017 cost estimates (New Gold, 2017a).

TABLE 21-4 PROCESS AND TAILINGS 2017 SUSTAINING CAPITAL COSTS

Description		2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Processing									
HMI/PLC	C\$(000)	-	15	15	10	10	15	80	145
Assay Lab	C\$(000)	50	50	-	150	-	150	500	900
Refinery/Security	C\$(000)	60	-	-	-	-	250	250	560
Reclaim Water	C\$(000)	1,450	100	-	-	100	-	200	1,850
Assay Services	C\$(000)	-	-	-	50	-	75	250	375
VFD Upgrades/Replacement	C\$(000)	-	-	-	-	250	-	650	900
Elution Circuit	C\$(000)	864	-	-	-	-	300	550	1,714
Process Pumps	C\$(000)	-	-	125	-	125	125	425	800
Gravity Circuit	C\$(000)	-	-	-	450	-	-	250	700
Cyclone Feed Pump	C\$(000)	-	850	-	-	-	-	-	850
Carbon Fines Collection Line	C\$(000)	19	-	-	-	-	-	-	19
SRK Study - Inundation	C\$(000)	52	-	-	-	-	-	-	52
CND Circuit Improvements	C\$(000)	435	-	-	-	-	-	-	435
Engineering Upgrades to the Mill (Phase 2)	C\$(000)	1,041	-	-	-	-	-	-	1,041
Plant Infrastructure	C\$(000)	2,069	-	-	-	-	-	-	2,069
Subtotal Processing	C\$(000)	6,041	1,015	140	660	485	915	3,155	12,411
Operations									
Water Treatment Plant	C\$(000)	8,984	7,930	-	-	-	-	-	16,914
Constructed Wetlands	C\$(000)	-	-	3,530	-	-	-	-	3,530
WMP - Sustaining	C\$(000)	2,855	-	-	-	-	-	-	2,855
Drainage Sump Stations	C\$(000)	886	-	-	-	-	-	-	886
Subtotal Operations	C\$(000)	12,725	7,930	3,530	-	-	-	-	24,185
Plant Maintenance									
Maintenance shop	C\$(000)	150	-	75	175	75	75	525	1,075
Instrumentation	C\$(000)	6,339	5,000	500	-	250	-	1,250	13,339
Reline Equipment	C\$(000)	-	150	-	550	-	400	450	1,550
Sediment Ponds	C\$(000)	10,512	-	-	-	-	-	-	10,512
Subtotal Plant Maintenance	C\$(000)	17,001	5,150	575	725	325	475	3,375	27,626

Description		2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Tailings									
TMA Lifts - Filter Sand	C\$(000)	8,985	2,797	-	2,396	3,194	-	4,928	22,299
TMA Stage 1 - Cell 2 - Construction including Instrumentation	C\$(000)	17,899	-	-	-	-	-	-	17,899
TMA Stage 1 - Cell 3 - Construction including Instrumentation	C\$(000)	22,980	-	-	-	-	-	-	22,980
TMA Stage 2 - Construction including Instrumentation	C\$(000)	3,000	18,672	-	-	-	-	-	21,672
TMA Stage 3 - Engineering	C\$(000)	-	-	62	124	-	-	-	186
TMA Stage 3 - Construction including Instrumentation	C\$(000)	-	-	-	7,472	9,963	-	-	17,435
TMA Stage 4 - Engineering	C\$(000)	-	-	-	-	-	186	-	186
TMA Stage 4 - Construction including Instrumentation	C\$(000)	-	-	-	-	-	-	15,230	15,230
TMA Stage 5 - Engineering	C\$(000)	-	-	-	-	-	-	186	186
TMA Stage 5 - Construction including Instrumentation	C\$(000)	-	-	-	-	-	-	14,280	14,280
Final TMA completion incremental haulage and rehandle costs	C\$(000)	2,886	1,867	-	1,548	1,548	-	21,732	29,580
TMA Reclaim Pump house Raise	C\$(000)	297	-	-	-	932	-	932	2,162
Clearing of TMA	C\$(000)	1,573	-	-	-	-	-	-	1,573
TMA Fencing	C\$(000)	2,461	3,030	-	-	-	-	-	5,492
Tails Piping Lifts & Refurbishments	C\$(000)	1,772	932	-	-	932	-	1,865	5,501
Water Management	C\$(000)	2,083	15	-	-	-	-	-	2,098
Subtotal Tailings	C\$(000)	63,936	27,313	62	11,540	16,570	186	59,154	178,761
Tailings Construction Indirects									
Consultants (QA/QC/ EOR)	C\$(000)	16,134	5,843	-	3,896	5,843	-	23,374	55,090
Small Construction Contractors Allowance	C\$(000)	797	266	-	186	280	-	932	2,462
Camp Operating Costs	C\$(000)	12,512	6,917	2,932	3,334	3,347	-	18,287	47,329
Subtotal Tailings Construction Indirects	C\$(000)	29,443	13,027	2,932	7,416	9,470	-	42,593	104,881
Grand Total	C\$(000)	129,145	54,436	7,240	20,341	26,850	1,576	108,277	347,864

The capital cost estimate is considered to be appropriate for process and tailings functions.

INFRASTRUCTURE CAPITAL COST ESTIMATE

The infrastructure capital cost is estimated to total C\$8.0 million as summarized in Table 21-5 and are based on 2017 cost estimates (New Gold, 2017a).

TABLE 21-5 INFRASTRUCTURE 2017 CAPITAL COSTS

Description	Unit	2018	2019	2020	2021	2022	2023	2024 to 2031	LOM Total
Employee Busing	C\$(000)	-	-	1,000	-	90	-	-	1,090
IT	C\$(000)	74	180	40	-	-	40	40	374
Safety	C\$(000)	763	-	-	-	-	-	-	763
Environment	C\$(000)	930	500	50	150	150	150	1,200	3,130
Light Vehicle Replacement Projects	C\$(000)	350	-	200	0	200	-	800	1,550
Site Services	C\$(000)	182	-	-	-	-	-	-	182
Grand Total	C\$(000)	3,197	680	1,290	150	440	190	2,040	7,987

The capital cost estimate is considered to be appropriate for necessary infrastructure.

RECLAMATION/CLOSURE

The reclamation/final closure capital cost is estimated to total C\$97 million over the mine life with the current financial assurance at C\$98.2 million based on the current disturbance. The annual expenditures are presented in Table 21-6.

TABLE 21-6 RECLAMATION/CLOSURE CAPITAL COSTS

Description	Unit	2018	2019	2020	2021	2022	2023	2024 to 2031	2032	LOM Total
Concurrent Reclamation	C\$(000)	126	126	-	-	-	-	73,521	-	73,772
Final Closure	C\$(000)	-	-	-	-	-	-	-	23,184	23,184
Grand Total	C\$(000)	126	126	-	-	-	-	73,521	23,184	96,956

The capital cost estimate is considered to be appropriate for reclamation and closure.

OPERATING COSTS

SUMMARY

The LOM unit operating costs are summarized in Table 21-7. Details for each category follow in this section and include data source.

TABLE 21-7 UNIT OPERATING COSTS SUMMARY

Description	Unit	Value
Open Pit	\$/t mined	2.7
Underground	\$/t mined	72.5
Process	\$/t milled	9.5
G&A	\$/t milled	3.1
Royalties	\$/t milled	0.5

MINING OPERATING COSTS

OPEN PIT OPERATING COSTS

The open pit mining costs were estimated from first principles considering the planned activities and estimated productivities and costs. A breakdown of the costs is shown in Table 21-8 and is based on 2017 cost estimates (New Gold, 2018c).

TABLE 21-8 OPEN PIT UNIT OPERATING COSTS

Description	Unit	2018	2019	2020	2021	2022	2023	2024	2025	2018 - 2025 Average	2026 - 2031 Average
G&A	\$/t mined	0.50	0.38	0.30	0.30	0.30	0.31	0.31	0.31	0.33	2.03
Engineering	\$/t mined	0.09	0.08	0.04	0.04	0.04	0.04	0.04	0.04	0.05	0.18
Geology	\$/t mined	0.06	0.05	0.04	0.04	0.04	0.04	0.04	0.04	0.04	0.13
Drilling	\$/t mined	0.29	0.22	0.19	0.18	0.22	0.23	0.22	0.21	0.22	0.29
Blasting	\$/t mined	0.27	0.28	0.29	0.25	0.29	0.31	0.31	0.31	0.29	0.08
Loading	\$/t mined	0.38	0.25	0.20	0.21	0.20	0.20	0.20	0.21	0.23	0.67
Hauling	\$/t mined	0.90	0.98	0.85	0.88	0.82	0.82	0.91	0.93	0.88	2.72
Roads	\$/t mined	0.01	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.03
Dewatering	\$/t mined	0.04	0.04	0.03	0.03	0.03	0.03	0.03	0.03	0.03	0.10
Support Services	\$/t mined	0.49	0.24	0.18	0.18	0.11	0.13	0.12	0.11	0.13	0.89
Auxiliary Fleet	\$/t mined	0.37	0.38	0.31	0.30	0.29	0.29	0.30	0.29	0.31	1.63
Site Services	\$/t mined	0.15	0.14	0.12	0.12	0.12	0.11	0.12	0.12	0.12	0.67
Light Vehicles	\$/t mined	0.02	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.04
Grand Total¹	\$/t mined	3.58	3.05	2.57	2.55	2.48	2.53	2.63	2.62	2.66	9.47

Note:

¹Includes Maintenance costs

UNDERGROUND OPERATING COSTS

The underground mining costs were estimated from first principles considering the planned activities and estimated productivities and costs. The LOM underground cost is estimated to be C\$72.5 per tonne mined. The underground costs include rehandling of the ore on surface

to feed the ore to the mill. A breakdown of the costs for the period 2019 to 2025 plus the LOM average is shown in Table 21-9 and is based on 2017 cost estimates (New Gold, 2018e).

TABLE 21-9 UNDERGROUND UNIT OPERATING COSTS

Description	Units	2019	2020	2021	2022	2023	2024	2025	2026 to 2031 Average	LOM Total
Development	\$/t	16.0	43.8	33.4	5.7	5.6	0.9	2.4	1.2	4.8
LH Stoping	\$/t	10.4	15.4	17.5	2.1	2.3	3.7	3.4	3.7	4.4
Backfill	\$/t	0.0	0.0	1.7	1.5	2.2	4.6	4.3	3.8	3.3
Haulage/Rehandle	\$/t	0.7	1.3	1.3	1.3	1.3	1.3	1.3	1.3	1.3
General	\$/t	25.0	70.0	42.9	49.2	43.1	42.4	42.6	33.0	38.2
Maintenance	\$/t	0.7	1.6	10.2	25.1	23.2	21.1	22.2	20.8	20.4
Total UG	\$/t	52.8	132.0	106.9	84.9	77.7	74.0	76.2	63.7	72.5

The operating costs were generated from first principles and from the contractor quotations for the mine development and operation. The operating costs are considered to be appropriate for the project.

There is some risk that operating costs may be higher if the transition from contracted mining is delayed or if there are difficulties in recruiting experienced miners for the operation. There is also the potential for operating cost increases if the ratio of backfill for stopes must be increased.

PROCESS OPERATING COSTS

Table 21-10 presents the LOM process operating cost by element

TABLE 21-10 LOM PROCESS PLANT 2017 OPERATING COST

Description	LOM Operating Cost (C\$M)	Operating Cost (C\$/t milled)
Reagents	264,201	2.13
Spare Parts/Maintenance	122,349	0.99
Liners and Screen Components	59,370	0.48
Grinding Media	152,025	1.23
Personnel	233,335	1.88
Electric Power	238,649	1.93
Outside Services	20,759	0.17

Description	LOM Operating Cost (C\$M)	Operating Cost (C\$/t milled)
Transportation (reagents, media)	11,744	0.09
Miscellaneous	79,796	0.64
Grand Total	1,182,228	9.54

Table 21-11 presents the budgeted versus actual operating costs in Canadian dollars and the overall cost per tonne ore in C\$/tonne of ore processed for the months of December 2017 and January 2018 as reported in the December 2017 operations review. The production rates and operating costs are quite variable as the plant has only been operating a few months.

TABLE 21-11 PROCESS 2017 OPERATING COSTS RECONCILIATION

Item	Unit	Dec 17 Budget	Dec 17 Actual	Jan 18 Budget	Jan 18 Actual
Mill Production, t	C\$(000)	651,000	581,544	651,000	457,919
Personnel	C\$(000)	754	659	954	1,063
Chemicals and Reagents	C\$(000)	1,222	522	1,223	1,418
Energy	C\$(000)	1938	1,541	1,906	2,800
Supplies and Consumables	C\$(000)	719	343	1,328	882
Maintenance	C\$(000)	38	170	17	242
Other Operating Expenses	C\$(000)	359	1,420	105	1,362
G&A	C\$(000)	43	13	82	63
Grand Total	C\$(000)	5,073	4,648	5,615	7,830
	\$/t Processed	7.79	8.03	8.63	17.10

As a check of current operating costs, Table 21-12 presents the budgeted versus actual unit consumptions from monthly operational reports for grinding media and reagents for the months of October 2017 through February 2018 (New Gold, 2017d, 2018a, and 2018b). The reagent consumption rates are highly variable which is not uncommon during early stages of commercial production.

TABLE 21-12 PROCESS PLANT 2017 REAGENT CONSUMPTIONS

Category	Units	Budget	Oct 17	Nov 17	Dec 17	Jan 18	Feb 18
Mill Production	t		545,601	395,796	581,544	457,919	578,001
SAG Media	kg/t	0.55	0.613	0.260	0.429	0.756	0.634
Ball Mill Media	kg/t	0.665	1.138	0.766	0.209	0.464	0.724
Flocculant	g/t	70.0	22	194	19	34	23
Lime	g/t	907.0	1182	1015	959	922	506
Sodium Cyanide	g/t	360.0	199	173	283	306	301
Sulphur Dioxide	g/t	460.0	659	549	688	664	533
Copper Sulphate	g/t	80.0	94.8	134.1	87.3	48	60.6
Activated Carbon	g/t	60.0	38	90	65	95	87
Liquid Oxygen	kg/t	0.30	0.40	0.31	0.39	0.51	0.26
Sodium Hydroxide	g/t	120.0	12	17	17	5	13
Hydrochloric Acid	g/t	50.0	0	0	0	0	1
SMBS	g/t	1.0	0	0	0	0	0
Antiscalant	g/t	10.0	0	1	9	12	5
Leachaid	g/t	0.0	0.05	0.02	0.22	0.07	0.13
Dust Suppressant	g/t	0.0	3	25	4	11	12

For this report, a LOM average processing cost unit rate of \$10.86/t milled has been calculated in the 2017 cost estimates (New Gold, 2018e) based on the following formula:

- $(x \text{ tonnes ore} * \$9.25/\text{t base operating cost} + y \text{ oz} * \$15/\text{oz on site refining cost}) / x \text{ tonnes ore}$

Overall, the process operating costs appears reasonable and in-line with expectations for the LOM.

G&A OPERATING COSTS

Table 21-13 presents the LOM general and administrative function operating cost forecast by element as per the 2017 New Gold forecast.

TABLE 21-13 LOM G&A 2017 OPERATING COST SUMMARY

Description	LOM Operating Cost C\$(000)	Operating Cost (\$/t milled)
Site Admin., Insurance & Maintenance	189,695	\$0.37
Health and Safety	26,254	\$0.05
Environment	56,070	\$0.11

Description	LOM Operating Cost C\$(000)	Operating Cost (\$/t milled)
Human Resources	18,781	\$0.04
IT and Telecommunications	12,220	\$0.02
Warehousing	29,600	\$0.06
Projects	25,532	\$0.05
Personnel	118,626	\$0.23
Grand Total	476,779	\$0.94

Table 21-14 presents the results of the first year of production reported in December 2017 operations review. There is a discrepancy between the LoM average of C\$1.54/t compared to actuals of C\$24.65/t due to the low tonnage processed in 2017 versus a normal year of 8.6 Mtpa and higher overhead costs during start-up (New Gold, 2018a). It is recommended watching these costs going forward to confirm the budgeted values are appropriate.

TABLE 21-14 G&A 2017 OPERATING COSTS RECONCILIATION

Description	2017 Actuals		2017 Forecast		Variance
	C\$(000)	\$/t milled	C\$(000)	\$/t milled	%
Personnel	12,126	6.90	15,139	7.94	-13%
Energy	817	0.47	471	0.25	88%
Supplies and Consumables	428	0.24	1,033	0.54	-55%
Maintenance	180	0.10	153	0.08	28%
Contractors/Other Ops Expense	8,061	4.59	5,146	2.70	70%
G&A/Overhead/Undifferentiated	21,695	12.35	16,065	8.43	47%
Grand Total	43,307	24.65	38,007	19.94	24%
Tonnes Processed (kt)		1,757		1,906	

ROYALTIES

Royalties have been estimated at 1.12% of gross revenues for the various royalties payable on the Project which averages \$2.04/t milled over the LOM (New Gold, 2018e).

MANPOWER

All manpower rosters were sourced from 2017 cost estimates (New Gold, 2018d). Using the year 2024 as the baseline, the project is expected to employ approximately 819 employees as follows with the details for each function discussed afterward:

- Open pit mining: 421
- Underground mining: 208

- Processing: 130
- G&A: 60
- Total: 819

OPEN PIT MANPOWER

The projected manpower levels for the open pit operation for the peak period 2018 to 2026 are shown in Table 21-15.

TABLE 21-15 OPEN PIT MANPOWER 2018 TO 2031

Description	2018	2019	2020	2021	2022	2023	2024	2025	2026 to 2031
									Average
Engineering	13	13	13	13	13	13	13	13	6
Geology	11	9	9	9	9	9	9	9	6
Operations Admin	27	24	23	23	23	23	23	23	6
Operations Staff Sub-total	51	46	45	45	45	45	45	45	18
Mine Operations	382	379	360	360	364	364	376	376	154
Grand Total	433	425	405	405	409	409	421	421	189

The projected manpower levels for the open pit operation are considered to be appropriate for the project.

UNDERGROUND MANPOWER

During the first three years of underground activity, a contract labour force has been assumed for all underground work. During months 30 through 36, the labour force will transition from contractor to company personnel. The work schedule is based upon operating personnel working 12 hour shifts with two weeks on two weeks off. Technical and management personnel will work a five day on two day off schedule. The projected manpower levels for the underground mine for the period 2018 to 2026 are shown in Table 21-16.

TABLE 21-16 UNDERGROUND MANPOWER 2018 TO 2031

Description	2018	2019	2020	2021	2022	2023	2024	2025	2026 to 2031 Average
Staff									
Mine & Maintenance Supervision	2	3	3	12	18	18	18	18	15
Engineering and Geology	2	7	10	12	11	11	11	11	9
Staff Sub-total	4	10	13	24	29	29	29	29	24
Labour-Hourly									
Mine Operations	1	5	8	54	127	135	135	131	106
Mine Maintenance	1	0	0	22	44	44	44	44	42
Hourly Sub-total	2	5	8	76	171	179	179	175	148
Grand Total	6	15	21	100	200	208	208	204	172

The projected manpower levels for the underground operation are considered to be appropriate for the project.

PROCESS PLANT MANPOWER

The projected manpower levels for the process plant operation are shown in Table 21-17 and are expected to stay constant during the LOM operations.

TABLE 21-17 LOM MILL OPERATIONS AND MAINTENANCE MANPOWER

Description	LOM Avg Emp.
Mill Manager	1
Mill Superintendent	1
Mill Administration	2
Process Control	1
Process Metallurgy	6
Production Supervision	4
Mill Operators	39
Assay Laboratory	17
Maintenance Management	7
Maintenance Planning	3
Mill Maintenance	50
Grand Total	130

The projected manpower levels for the process plant operation are considered to be appropriate for the project.

G&A MANPOWER

The projected manpower levels for the general and administrative function for the peak period 2018 to 2026 are shown in Table 21-18.

TABLE 21-18 G&A MANPOWER 2018 TO 2031

Description	2018	2019	2020	2021	2022	2023	2024	2025	2026 to 2031 Average
Mine Manager	1	1	1	1	1	1	1	1	1
Ops Supervision	17	12	9	8	9	8	8	9	7
Training	5	4	4	4	4	4	4	4	4
Admin	41	34	36	37	36	37	37	36	28
Mid Professional	12	10	10	10	10	10	10	10	8
Grand Total	76	61	60	60	60	60	60	60	47

The projected manpower levels for the G&A function are considered to be appropriate for the project.

22 ECONOMIC ANALYSIS

Under NI 43-101 rules, producing issuers may exclude the information required for Section 22 – Economic Analysis, on properties currently in production, unless the Technical Report includes a material expansion of current production. New Gold is a producing issuer, the Rainy River Mine is currently in production, and the minor expansion from 21,000 tpd to 24,000 tpd is not considered a material expansion. New Gold has performed an economic analysis of the Rainy River Mine using the estimates presented in this report and confirms that the outcome is a positive cash flow that supports the statement of Mineral Reserves.

23 ADJACENT PROPERTIES

There are no adjacent properties to report in this section.

24 OTHER RELEVANT DATA AND INFORMATION

No additional information or explanation is necessary to make this Technical Report understandable and not misleading.

25 INTERPRETATION AND CONCLUSIONS

The QPs offer the following interpretations and conclusions:

GEOLOGY

The Rainy River deposit is an auriferous VMS system with a primary syn-volcanic source and possibly a secondary syn-tectonic mineralization event.

MINERAL RESOURCES

With regard to the QA/QC monitoring programs in place at Rainy River:

- Results of gold and silver CRMs indicate that the primary independent laboratory measures gold and silver in samples accurately.
- Results of the blank sample program indicate that there is little to no contamination of samples during sample preparation.
- Results of the duplicate sample program indicate no significant bias is present in the preparation or analysis of the samples.

The sample preparation, analysis, and security procedures at Rainy River are adequate for use in the estimation of Mineral Resources.

The Mineral Resource database is sufficiently reliable for grade modelling and Mineral Resource estimation.

Open pit Mineral Resources are reported at cut-off grades of 0.3 g/t AuEq to 0.5 g/t AuEq for Low Grade and 0.5 g/t AuEq for Direct Processing material. The underground Mineral Resources are reported at a cut-off grade of 2.0 g/t AuEq.

Measured and Indicated Mineral Resources are estimated to total 63.109 million tonnes at grades of 1.06 g/t Au and 3.8 g/t Ag, containing 2,142,000 ounces of gold and 7,651,000 ounces of silver. Inferred Mineral Resources are estimated to total 8.871 million tonnes at grades of 1.10 g/t Au and 2.4 g/t Ag, containing 313,000 ounces of gold and 672,000 ounces

of silver. The Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

MINERAL RESERVES

OPEN PIT MINERAL RESERVES

The open pit Mineral Reserves are estimated to be 71.88 million tonnes of direct processing ore grading 1.17 g/t Au and 2.9 g/t Ag, 34.3 million tonnes of low grade ore grading 0.36 g/t Au and 2.3 g/t Ag, and 5.3 million tonnes of above-ground stockpiled ore grading 0.58 g/t Au and 1.8 g/t Ag. The Mineral Reserves include factors for the cut-off grade, dilution, and minimum mining width.

The open pit Mineral Reserves are stated at a cut-off grade of 0.5 g/t Au for Direct Processing and 0.3 g/t Au for the Low Grade stock pile.

Open pit mining utilizes conventional truck and hydraulic shovel with 10 m high benches in the open pit.

The design parameters, development, and mining method are considered to be appropriate for the deposit.

UNDERGROUND MINERAL RESERVES

The underground Probable Mineral Reserves are estimated to be 8.95 million tonnes grading 3.55 g/t Au and 9.5 g/t Ag. The Mineral Reserves include factors for cut-off grade, dilution, minimum mining width, and mining extraction.

The underground Mineral Reserves are stated at a cut-off grade of 2.2 g/t Au, which is higher than the breakeven cut-off grade of 1.7 g/t Au. The higher cut-off grade was used to generate an underground production schedule that terminates when the open pit processing is complete.

The underground mining will utilize long hole open stoping for areas less than ten metres wide and cut and fill stoping with cemented aggregate fill for areas wider than ten metres.

The mine will be a decline access mechanized mine using medium scale mechanized equipment.

The design parameters, development, and mining method are considered to be appropriate for the deposit.

MINING

OPEN PIT MINING

The LOM plan was developed after evaluating multiple iterations of mining sequence including combinations of mining sequencing with different mining rates, mine sequencing alternatives and other economic parameters.

Mining at Rainy River is conducted using conventional open pit mining methods. The drill, blast, load and haul cycle uses large open pit equipment matched to the hydraulic shovel and diesel electric haul truck fleet. Waste material is categorized and stored in specific waste dump locations. The typical waste designations are PAG, NPAG, and overburden. Ore is designated in direct feed to the processing plant and low grade material is stored in a stock pile for end of mine life processing.

The planned equipment fleet is appropriate for the production rates, schedule, operating conditions and LOM plan.

UNDERGROUND MINING

The planned single heading advance rate and the rapid production build-up to the design production rate pose a moderate risk to the project schedule. There are multiple stoping areas that will be operated concurrently, which will provide significant flexibility in the event of delays in any one area.

Additional time may be required in the production schedule to allow for infill drilling and analysis prior to the commencement of production in an area.

Areas where the dip is less than 55° may suffer some additional ore loss and/or dilution, or higher costs to recover all the ore in the stope designs.

Ground conditions are generally expected to be “Fair” to predominantly “Good” based on Feasibility Study level geotechnical investigations. Geotechnical analyses and modelling support the planned open stope designs. In some non-entry areas, large hangingwall exposures will exist, supported by relatively small pillars (as designed). Controlled and uncontrolled failures of the hangingwall or pillars in these locations may generate additional dilution or loss of ore.

Where cemented backfill is used, an aggregate fill with an average 4% cement content has been assumed. Waste rock fill with no cement may also be used in stopes where there will be no subsequent mining against the fill.

The planned equipment fleet is appropriate for the mine as planned; ventilation benefits may be realized through the use of electric (battery and/or trolley) units in place of diesel units.

METALLURGICAL TESTING AND DEVELOPMENT OF DESIGN CRITERIA

The metallurgical testing programs supporting the engineering of the Rainy River process facilities were performed in support of the Feasibility Study prepared for New Gold in 2014, which is the primary source for this report. The testing programs were extensive and provided the design information required.

The design criteria developed for the process design engineering were sourced from the metallurgical testwork and from information supplied by equipment suppliers.

PROCESS

The Rainy River process plant was commissioned in September 2017 and has been working through initial operating and maintenance issues and plant optimization. The production rates are approaching the targeted 21,000 tpd and have been achieved on a daily basis, however, plant availability has been lower than projected and so the average production rates on a monthly basis have been below target.

Gold grades have been slightly lower than projected and both gold and silver recoveries have been lower than projected. Operations indicate that the primary focus through the start-up period has been on gold recovery with silver being of secondary economic concern. Key

parameters affecting gold and silver recovery are grind size, 80% passing 75 microns, cyanide concentration, retention time in the leaching circuit, and carbon adsorption and elution in the CIP and elution circuits. Consistent operation is also a factor as stabilizing of the circuit is required following start-ups and shut-downs resulting in reduced performance. Operating personnel are focusing on these areas in order to improve production rates and recoveries.

Some of the key issues encountered during plant commissioning and start-up included:

- Mill motor and VFD drive programming issues, which caused frequent shut-down of the mill.
- Material discharge from the coarse ore stockpile to the mill feed conveyor, including plugged apron feeder discharge chutes and some conveyor damage resulting in down-time.
- Carbon containment and carbon transfer from the CIP tanks to the carbon elution circuit.
- Carbon elution circuit control and performance, including eluent heating and temperature control.
- Cyanide destruction, SO₂-air, circuit operation and performance.
- Tailings pumping.
- Reagent mixing and distribution.

These types of challenges are common during plant start-ups and most will be remedied through adjustments during continuous operation of the plant and correction of issues identified during scheduled maintenance periods. Several unit operations have been targeted for review and modification and are the focus of a plant optimization and expansion study.

PLANT EXPANSION STUDY

New Gold has identified sections of the plant, including the carbon elution and cyanide destruction circuits that require modification to achieve required performance. Taking this into consideration, Ausenco was retained to perform a review of the elution circuit and to prepare the New Gold Rainy River 24 ktpd Debottlenecking Feasibility Study report, which was issued on March 15, 2018.

EQUIPMENT SIZING REVIEW

Ausenco performed a review of the major process plant equipment to determine whether they have sufficient capacity for operating at 24,000 tpd on a steady state basis. Ausenco contacted the vendors for the major process equipment and report that all of the equipment reviewed has the capacity for steady state 24,000 tpd operation.

Some of the equipment will be very close to maximum capacity at 24,000 tpd and so the capability to handle significant surges in production rates or flow will be limited. It will be important to maintain the 92% equipment availability in order to consistently reach production targets. Equipment that is close to capacity at 24,000 tpd and should be reviewed in more detail include:

- The SAG mill will be at 96% of motor capacity assuming an operating work index of 13.2 kWh/t and the ball mill will be at 94% capacity assuming a work index of 13.0 kWh/t. Both mills are close to maximum capacity and may be sensitive to variations in ore hardness. FL Smidth suggested a site audit and baseline mill performance review.
- The final tailings pumps will be reviewed due to issues with pumpbox overflow during surges in flow. This is on the current Ausenco project list for review discussed in this document. A systematic review of the tailings pumping system will be performed in conjunction with the tailing dam construction schedule.
- The water management pumps, Pinewood River water return pumps, the mine rock pond water pumps and fresh water pumps should be reviewed with respect to the water management plan and tailings dam construction schedule. Ausenco recommends that the site water balance be reviewed to confirm the projected pumping requirements.

REVIEW OF COMMINUTION CIRCUIT DESIGN AND GOLD PLANT ASSESSMENT

Following the plant expansion program, which is designed to increase the mill production rate from 21,000 tpd to 24,000 tpd, New Gold commissioned Ausenco to perform a more extensive review of the processes and facilities to determine the maximum operating rate that the plant can achieve without the addition of new grinding equipment. Adding grinding equipment is not being considered at this time, as it would significantly increase capital costs and the associated down time would decrease production.

Moderate capital cost additions such as the replacement of pump motors or installing additional screens is being considered. These types of modifications can be installed without significantly affecting production.

The study is being performed in three phases: a Concept Study phase, a Feasibility Study Phase and an Execution Phase. The schedule for the project will begin with the current Conceptual Study reports and the final implementation phase is to be completed and commissioned by 2019. The production forecast projects total production of 7.5 million tonnes milled in 2018, 8.5 million tonnes milled in 2019 and 2020, and 9.9 million tonnes milled in 2021, which equates to 27,000 tpd. The two study reports include:

- Ausenco, 2018, New Gold Rainy River Review of the Comminution Circuit Design July 2018.
- Ausenco, 2018, New Gold Rainy River Gold Plant Assessment, July 10, 2018.

The main observations from the review of the historical information and control philosophy are that:

- The circuit was designed for a harder ore than is currently being processed.
- The primary crusher is producing a very fine SAG mill feed.
- Under current operating conditions the grinding mill's installed power is underutilized (66% for the SAG mill and 74% for the ball mill). Operating data indicate that the ball mill is consistently operating in the 11 kWh/t range and the grind size is coarser than design at 89 µm.
- The fine SAG mill feed results in a coarse transfer size to the ball mill.
- The grinding load is being pushed to the ball mill, which is producing a product particle size with a P₈₀ of approximately 90 µm rather than the design product size of P₈₀ 75 µm.
- An analysis of the ball mill cyclones indicated that the grinding circuit is operating with a circulating load between 500% and 600% instead of the projected 300% with a 400% maximum given in the process design criteria.
- The ball charge could be increased to improve the grind size and the circulating load. A decrease in the circulating load would have a positive impact on the capacities of the downstream equipment.

MINE ROCK AND OVERBURDEN STOCKPILES – STAGE 1 STUDY

A Stage 1 preliminary scoping level evaluation was performed to determine the most effective method for improvement of the foundation materials beneath the proposed mine rock stockpile areas. There are many areas of instability due to soil types and moisture. The study consisted of identifying methods of foundation improvement, preparing preliminary designs for each alternative, preparing cost estimates for each alternative, and then evaluating the results.

The alternatives that were considered applicable for the clay soils at Rainy River included:

- Wick drains
- Stone columns
- Removal and replacement (shear keys)
- Deep soil mixing
- Electro-osmosis
- Vacuum consolidation
- Flattening the slope

FURTHER DEVELOPMENT WORK - STAGE 2 STUDY

Based on the capital cost estimates and schedules for installation, the following alternatives were selected for further investigation in a Stage 2 study.

- The Base Case – Wick drains
- Alternative 1 – Removal and replacement or shear key, only in areas with shallow clay deposits. Wick drains or slope flattening will still be required for zones with deeper clay.
- Alternative 4 – Flattening the slope by expanding the east mine rock stockpile

Some additional optimization work was performed on the base case wick drain alternative. The ground improvement widths in Section A areas were reduced from 180 m to 125 m by including a 50 m width without wick drains and a 50 m to 75 m width with wick drains. This configuration provides significant cost improvement in the wick drain option. This optimized base case will be investigated in the Stage 2 study.

ENVIRONMENTAL, SOCIAL, COMMUNITY AND RECLAMATION/CLOSURE

New Gold has made a significant commitment to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against their Health, Safety, Environment and Corporate Social Responsibility Policy last reviewed and approved by their Board of Directors on February 19, 2015 (See HSE-CSR Policy Attachment). Key elements include:

Health and Safety – This includes promoting and protecting the well-being of employees through safety-first work practices and a culture of safety excellence. Commitment to leading

industry practices and systems in health and safety that focus on prevention of accidents and incidents are a priority.

Environment – This includes a commitment to preserving the long-term health and viability of the natural environments affected by the project and operation.

Labour Practices – This includes a commitment to upholding fair employment practices and encouraging a diverse workforce, where people are treated with respect and are supported to realize their full potential.

Community Engagement and Development – This requires a commitment to establishing relationships based on mutual benefit and active engagement with host communities to contribute to healthy communities and development.

At the time of this review, the Mine Environmental Department was adequately staffed, but resources appeared to be stretched. Subsequent to the site visit, the environmental lead for the Mine retired. This role and responsibilities have been assumed by internal staff. New Gold contracts AMEC to carry out inspections and third-party groundwater monitoring.

At the time of the site visit, the Mine reported two non-compliance related issues associated with an effluent discharge in excess of permit limits and failing to report an effluent discharge above permit limits. New Gold pleaded guilty to the charges and was fined \$187,500.

The Mine environmental budget for 2018 was reviewed and appears adequate/reasonable for the size of operation, issued permits, and conditions imposed.

New Gold has developed a reclamation and closure plan that satisfies corporate and regulatory requirements. Reclamation and closure planning requires regular review and update to accommodate change(s) in planned and constructed facilities/operations.

26 RECOMMENDATIONS

The QP's offer the following recommendations:

MINERAL RESOURCES

- Where possible, identify mappable geologic features that act as controls to the distribution of gold and silver mineralization in the deposit to further refine the geologic constraints to future mineral resource estimates.
- As mining progresses, continually assess reconciliation performance of the mineral resource block model against mined production, and where appropriate, refine the geostatistical parameters and block model grade interpolation methods to further improve Mineral Resource estimation accuracy and precision.

MINERAL RESERVES

- Refine procedures for accurate grade control sampling, pit bench estimation and mine planning to ensure reconciliations between mine-to-mill production and reserves-to-mine production remain within acceptable levels of confidence.

MINING

Open Pit

- Further development of wall control and slope monitoring techniques and procedures in both rock and overburden as the pit progresses.
- Completion of the investigation of geotechnical waste dump foundation stabilization of overburden.
- Further operator and maintenance training, coaching, and upskilling to increase the production rate and reduce dilution to achieve the schedule defined in this report.
- The overall project economics are sensitive to total mining costs. A strategic review of efficiency, productivity, and costs is recommended in the near term.

Underground

- Close monitoring of the mine development and the rapid implementation of remedial actions in the event of development advance shortfalls.
- Review of the footwall geometry in areas where the dip is less than 55 degrees as part of the stope planning.
- Cavity monitoring surveys as part of the production records and reconciliation of production to the Mineral Reserve estimates.

- Ground control monitoring systems for analysis of the hangingwall and pillar stability in the open stope areas.
- Further consideration and review of the stability of the overall stope hangingwall for the open stopes, including review of rib pillar dimensions as local mining knowledge increases.
- Introduction of a microseismic monitoring system for deeper regions of the mine and for areas below the pit.
- Continued testing to verify field performance of backfill binder content.
- Review of the schedule to ensure that there is time allotted for infill drilling and data analysis between the initial development and the commencement of stoping.

PROCESS

The process facilities are still in the start-up and initial operation stage. It is recommended that the operations and maintenance personnel continue to develop their procedures and to identify opportunities that will improve plant availability. This process is well advanced and the studies and projects that Ausenco are currently pursuing will result in improvements.

Ausenco performed a review of the major process plant equipment to determine whether they have sufficient capacity for operating at 24,000 tpd on a steady state basis. Ausenco contacted the vendors for the major process equipment and report that all of the equipment reviewed has the capacity for steady state 24,000 tpd operation.

It appears that some of the equipment will be very close to maximum capacity at 24,000 tpd and so the capability to handle significant surges in production rates or flow will be limited. It will be important to maintain the 92% equipment availability in order to consistently reach production targets. There were suggestions by some of the equipment suppliers for additional design review as part of the expansion program. It is recommended that ore hardness be reviewed with respect to the mine plan to provide a consistent feed to the mills at the higher production rate. In addition, the water balance and pump capacities in the water management systems should be reviewed.

The question of converting the elution heating from indirect heating to direct should be reviewed with respect to scale formation and its effect on the in-line elution heaters. This conversion is not recommended due to the severe scale reported in the existing heat

exchanger. A different type of heat exchanger such as a shell and tube may be more appropriate as it would be easier to clean by hydro-blasting than the plate and frame.

ENVIRONMENTAL

- Promote cross training of employees to reduce reliance on key personnel (Environmental/Indigenous Relations, etc.)
- Set up/implement the EMS. Perform regular audits (at least annual). This will allow for tracking and assessment of success/failures and help with Cross Training recommendation above.
- Continue consultation and coordination with First Nations and public.
- Perform PAG confirmation testwork.
- Start interim reclamation to reduce post mining obligations
- Related to above, start revegetation test plots to determine what species and composition works best for the Mine area and environment.

CAPITAL AND OPERATING COSTS

The cost estimate for implementing project closure is estimated to be C\$97 million. Closure costs must be reassessed on a regular basis during the LOM to determine present-time rates within the terms of the Mining Act Regulations.

While the mine and mill production rates and operating costs are quite variable as the plant has only been operating a few months, the base operating costs appears reasonable and in-line with expectations for the LOM at C\$2.7/t in mining and C\$8/t to C\$10/t (excluding on site refining costs) range in processing. However, it is recommended watching these costs going forward to confirm the budgeted values are appropriate.

There is a discrepancy between the LOM average of C\$1.54/t compared to actuals of C\$24.65/t for G&A due to the low tonnage processed in 2017 versus a normal year of 8.6 Mtpa and higher overhead costs during start-up (New Gold 2018a). It is recommended watching these costs going forward to confirm the budgeted values are appropriate.

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28 DATE AND SIGNATURE PAGE

This report titled “Technical Report on the Rainy River Mine, Ontario, Canada” with an effective date of July 25, 2018 was prepared and signed by the following authors:

(Signed & Sealed) “Nicholas Kwong”

Dated at Toronto, ON
July 25, 2018

Nicholas Kwong, P.Eng.
Director Business Improvement, New Gold Inc.

(Signed & Sealed) “Michele Della Libera”

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Dated at Aberdeen, WA
July 25, 2018

Lee Patrick Gochnour, QP, MMSA
Principal, Gochnour & Associates, Inc

29 CERTIFICATE OF QUALIFIED PERSON

NICHOLAS KWONG

I, Nicholas Kwong, P.Eng., as an author of this report entitled ““Technical Report on the Rainy River Mine, Ontario, Canada” prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Director, Business Improvement for New Gold Inc., at;
181 Bay Street, Suite 3510
Toronto, Ontario, M5J 2T3
2. I am a graduate of University of British Columbia with a Bachelor of Applied Science Degree (B.A. Sc.) in 2005 and a Master of Business Administration (M.B.A.) Degree in 2010.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg.# 100523679). I have worked as a mining engineer for a total of 13 years since my graduation. My relevant experience for the purpose of the Technical Report is:
 - Corporate Director and Manager level roles for New Gold’s business improvement and technical services functions responsible for managing and directing technical, operational and cross functional activities across the company’s portfolio of mining operations and development projects in Canada, the United States, Australia and Mexico.
 - Open pit and underground Mineral Reserves Estimation, mine engineering design, development and operating experience at operations including the Rainy River Mine, Ontario, Canada; New Afton Mine, British Columbia, Canada; Mesquite Mine, Brawley, California, USA; Peak Mine, Cobar, NSW, Australia; and Cerro San Pedro Mine, San Luis Potosi, Mexico.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Rainy River Mine on numerous occasions since July 2013, the most recent being July 9, 2018. I also held the role of technical services manager onsite at Rainy River in 2017.
6. I am responsible for Sections 15, 16, 19, 21, 22, and 24, and have collaborated with my co-authors on Sections 1, 2, 3, 18, 20, 23, 25, 26, and 27 of the Technical Report.
7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) “Nicholas Kwong”

Nicholas Kwong, P. Eng.

MICHELE DELLA LIBERA

I, Michele Della Libera, P. Geo., as an author of this report entitled "Technical Report on the Rainy River Mine, Ontario, Canada" prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Director, Exploration for New Gold Inc., at;
181 Bay Street, Suite 3510
Toronto, Ontario, M5J 2T3
2. I am a graduate of University of Pisa, Italy in 1992 with a M.S. Degree in Geology.
3. I am registered as a Professional Geoscientist in the Province of Ontario Registration No. 2837. I have worked as a geologist for a total of 25 years since my graduation. My relevant experience for the purpose of this Technical Report is:
 - Practiced my profession as a geologist continuously for the last 23 years, with involvement in exploration projects from early stage to resource delineation phase.
 - Experienced in precious and base metals exploration in a variety of geological settings and ore deposit types on projects ranging from early to advanced stage as well as in active mining operations.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Rainy River Mine area on numerous occasions since July 2013, the most recent being July 9-13, 2018. In my role as Director of Exploration for New Gold I have had direct oversight of all exploration and resource delineation activities at Rainy River since July 2013.
6. I am responsible for Sections 4-12 and have collaborated with my co-authors on Sections 1-3, 25 and 26 of the Technical Report.
7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) “Michele Della Libera”

Michele Della Libera, P.Geol.

DINARA NUSSIPAKYNOVA

I, Dinara Nussipakynova, P.Geo., as an author of this report entitled “Technical Report on the Rainy River Mine, Ontario, Canada” prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Principal Geologist with AMC Mining Consultants (Canada) Ltd., at;
200 Granville Street, Suite 202
Vancouver, BC, V6C 1S4
2. I am a graduate of Kazakh National Polytechnic University with a B.Sc. and M.Sc. in Geology in 1987.
3. I am a member in good standing of the Engineers and Geoscientists of British Columbia (License #37412) and the Association of Professional Geoscientists of Ontario (License #1298). I have practiced my profession continuously since 1987 and have been involved in mineral exploration and mine geology for a total of 31 years since my graduation from university. My experience is principally in Mineral Resource estimation, database management, and geological interpretation.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Rainy River Mine on April 11, 2018.
6. I am responsible for Section 14, and have collaborated with my co-authors on Sections 1, 2, 12, and 25 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) “Dinara Nussipakynova”

Dinara Nussipakynova, P. Geo.

ANDREW P. HAMPTON

I, Andrew P Hampton, P.Eng., as an author of this report titled “Technical Report on the Rainy River Mine, Ontario, Canada” prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Principal with KSN Mineral Process Associates, LLC of Englewood, Colorado.
2. I am a graduate of Southern Illinois University in 1979 with a B.S. Degree in Geology, and a graduate of the University of Idaho in 1985, with an M.S. Degree in Metallurgical Engineering.
3. I am registered as a Professional Engineer in the Province of British Columbia, Licence No. 22046. I have worked as an extractive metallurgical engineer for a total of 30 years since my graduation. My relevant experience for the purpose of the Technical Report is:
 - Process plant engineering, operating and maintenance experience at mining and chemical operations, including the Sunshine Mine, Kellogg, Idaho, Beker Industries Corp, phosphate and DAP plants in Florida and Louisiana respectively, and the DeLamar Mine in Jordan Valley, Oregon.
 - Engineering and construction company experience on a wide range of related, precious metal projects and studies, requiring metallurgical testing, preliminary and detailed design, project management, and commissioning and start-up of process facilities and infrastructure. EPCM companies included Kilborn Engineering Pacific Ltd., SNC Lavalin Engineers and Constructors, Washington Group International Inc. and Outotec USA, Inc. I provide contract metallurgical services to operating and engineering companies as KSN Mineral Process Associates, LLC.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Rainy River Mine on October 2-4, 2017.
6. I am responsible for preparation of Sections 13, 17, and 18 and contributed to Sections 1 to 6, 21, 25, 26, and 27 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have not had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated this 25th day of July, 2018.

(Signed & Sealed) “Andrew Paul Hampton”

Andrew P. Hampton, P.Eng.

BINSAR SIRAIT

I, Binsar Sirait, as an author of this report entitled “Technical Report on the Rainy River Mine, Ontario, Canada” prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Engineering Project Manager with New Gold Inc., at;
181 Bay Street, Suite 3510
Toronto, Ontario, M5J 2T3
2. I am a graduate of Institut Teknologi Bandung (ITB), West Java, Indonesia with a Bachelor of Science Degree (B.Sc.) in 1994.
3. I am a Registered Member of the Society for Mining, Metallurgy and Exploration (SME), Englewood, CO, USA (Reg.# 04157141), and a member of the Australian Institute of Mining and Metallurgy (AusIMM) (Reg.#304280). I have worked as a mining engineer for a total of 23 [number of years] since my graduation. My relevant experience for the purpose of the Technical Report is:
 - Corporate Director and Manager level roles for New Gold’s technical services function responsible for managing and directing technical and operational planning activities at the company’s portfolio of open pit mining operations and development projects in Canada, and the United States. Mineral Reserves Estimation, mine engineering design, scheduling and operating experience at New Gold’s mining operations including the Rainy River Mine, Ontario, Canada; Mesquite Mine, Brawley, California, USA.
 - Manager Mine Performance Improvement and Mine Technical Services for Rio Tinto’s Oyu Tolgoi operation, Mongolia, responsible for acting as the company’s Competent Person for open pit Mineral Reserves estimates.
 - Senior Mining Engineer for Newmont Mining Corporation responsible for acting as the company’s Competent Person for open pit Mineral Resources and Reserves estimates at the company’s Twin Creeks Mine, Nevada.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have visited the Rainy River Mine on numerous occasions since January 2016, most recently on June 13, 2018.
6. I am responsible for the open pit Mineral Reserves estimate and open pit mining methods described in Sections 15 and 16 respectively, and have collaborated with my co-authors on Sections 1 and 2 of the Technical Report.
7. I am not independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) “Binsar Sirait”

Binsar Sirait, SME Registered Member No.04157141

HERBERT A. SMITH

I, Herbert A. Smith, P.Eng., as an author of the report entitled “Technical Report on the Rainy River Mine, Ontario, Canada” prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am Senior Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., at;
200 Granville Street, Suite 202
Vancouver, BC, V6C 1S4
2. I am a graduate of the University of Newcastle upon Tyne, England with a B.Sc. in Mining Engineering in 1972 and an M.Sc. in Rock Mechanics and Excavation Engineering in 1983.
3. I have worked as a Mining Engineer for a total of 40 years since my graduation and have relevant experience in underground mining, feasibility studies, and technical report preparation for mining projects.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I have not visited the Rainy River Mine.
6. I am responsible for the underground Mineral Reserve estimate in Section 15 and underground mining aspects of Sections 1, 16, 25, and 26.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) “Herbert A. Smith”

Herbert A. Smith, P. Eng.

LEE P. GOCHNOUR

I, Lee P. Gochmour, MMSA QP., as an author of this report entitled "Technical Report on the Rainy River Mine, Ontario, Canada" prepared for New Gold Inc. with an effective date of July 25, 2018, do hereby certify that:

1. I am the Principal with Gochmour & Associates, Inc.; at
915 Fairway Lane,
Aberdeen, WA, USA, 98520
2. I am a graduate of Eastern Washington University with a degree in Park Administration and Environmental Land Use Planning in 1981.
3. I am recognized as a Qualified Professional (QP) Member, with special expertise in Environmental, Permitting and Compliance with the Mining Metallurgical Society of America (Member# 01166QP), a Member of the Society of Mining, Metallurgy and Exploration, and Past President and Life Member of the Northwest Mining Association. I have worked as an environmental professional in the mining industry for a total of 37 years since my graduation from university.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfill the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the Rainy River Mine on October 2-4, 2017.
6. I am responsible for Section 20 of the Technical Report.
7. I am independent of the Issuer applying the test set out in Section 1.5 of NI 43-101.
8. I have had no prior involvement with the property that is the subject of the Technical Report.
9. I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.
10. At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated 25th day of July, 2018

(Signed & Sealed) "Lee Patrick Gochmour"

Lee P. Gochmour, QP