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Technical Report

NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada

New Gold Inc.

Rainy River, North-Western Ontario, Canada

In accordance with the requirements of National Instrument 43-101 "Standards of Disclosure for Mineral Projects" of the Canadian Securities Administrators

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AMC Project 919006
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1 Summary

1.1 Introduction

This Technical Report (Report) on the Rainy River Property (Property) located in north-western Ontario (ON) in Canada has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) headquartered in Vancouver, Canada on behalf of New Gold Inc. (New Gold) headquartered in Toronto, Canada. It has been prepared to a standard which is in accordance with the requirements of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101), of the Canadian Securities Administrators (CSA) for lodgment on CSA's System for Electronic Document Analysis and Retrieval (SEDAR). This Report is an update to the report dated 25 July 2018 and filed on SEDAR on 7 August 2018, titled "New Gold Inc. Technical Report on the Rainy River Mine, Ontario, Canada".

New Gold is an international mid-tier gold mining company with Canadian operations in ON and British Columbia (BC) in Canada and a mine under reclamation in Mexico. New Gold owns 100% of the two Canadian operations, Rainy River and New Afton and the Blackwater project in BC, which is in the development phase. The Cerro San Pedro Mine in Mexico is in reclamation and is also 100% owned by New Gold.

New Gold is listed on both the TSX as "NGD" and NYSE as "NGD".

1.2 Location and ownership

The Property is located approximately 50 kilometres (km) to the north-west of Fort Frances, the nearest large town, in north-western ON. The property is centred in Richardson Township which is part of Chapple Township. Access from Thunder Bay through Fort Frances is approximately 415 km along Highway 11 to Emo, and then north on Highway 71, turning west on Korpi Road. Alternative access from Winnipeg is by driving east to Kenora via Hwy 1 / Hwy 17 and then south on Highway 71 and turning west on Korpi Road, a distance of 369 km. These access roads are sealed allowing year-round access.

1.3 Property description

The Property comprises a portfolio of 210 patented mining rights, surface rights (SR), and Crown Lease properties. The Project Lands covering the mine area comprise 119 separate properties of which New Gold has the rights to the Surface and Minerals, with this area covering approximately 6,077 hectares (ha). There are also 1,156 unpatented claims and the total area covers approximately 36,762 ha. All unpatented claims are in good standing. The mine is located in the townships of Fleming, Mather, Menary, Patullo, Potts, Richardson, Senn, Sifton, and Tait.

1.4 History

Exploration in the general area of the Property began in 1967. Various companies and government organizations were active on and around the project area from 1967 to 1989. In 1990 the Property was acquired by Nuinsco Resources Ltd. (Nuinsco), and they held the claims until 2004, with Rainy River Resources Ltd. (RRR) continuing exploration from 2005 to 2013, when New Gold completed a takeover of RRR on 15 October 2013.

Nuinsco actively explored the ground from 1993 to 2004. Exploration successes of note include the discovery of 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.

Numerous Mineral Resource estimates, now considered historical Mineral Resources, were prepared for the Rainy River Mine from 2003 to 2015. 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, and BBA Inc. (BBA) and collaborators in 2014 (Feasibility Study). 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.

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 and includes the Intrepid deposit.

1.5 Geology and mineralization

The Property is located within the 2.7 billion years (Ga) old Neoarchean Rainy River Greenstone Belt (RRGB). The RRGB forms part of the Wabigoon sub-province within the larger Superior Province. The Wabigoon sub-province is a 900 km long, east-west trending composite volcanic and plutonic terrane comprising distinct eastern and western domains separated by rocks of Mesoarchean 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 and 2.66 Ga.

The Rainy River deposit occurs within a sequence of felsic to intermediate, calc-alkaline metavolcanic rocks which is bounded to both the north and south by a lower mafic volcanic sequence. This mafic sequence is intruded by the trondhjemitic Sabaskong batholith to the north. Felsic to intermediate rocks are intruded to the east of the deposit by the Black Hawk monzonitic stock.

The Property encompasses an approximately 30 km long, north-east trending portion of the RRGB. In this area, the RRGB is bounded to the north-west by the Sabaskong Batholith, to the east by the Rainy Lake Batholithic Complex and to the south by the Quetico fault. In the north-east portion of the Property the RRGB is contiguous with the Kakagi-Rowan Lakes Greenstone Belt. The intermediate dacitic rocks host most of the Rainy River gold mineralization.

Structural analysis 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.

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 Zone, 433 Zone HS Zone, Western Zone).
- Deformed quartz-ankerite-pyrite shear veins in mafic volcanic rocks (CAP Zone).

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

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

1.6 Exploration

New Gold has completed several exploration programs at the Property since the announcement of the takeover of RRR in May 2013. In 2013, a mobile metal ion (MMI) sampling program, relogging of 56,000 metres (m) of core and a MSc thesis were completed. From 2014 to 2016, an additional 862 MMI samples were taken, 102,380 m of re-logging was completed, and two alteration surveys were completed: a 5,000 m Corescan hyperspectral alteration study and a 1,992-sample short-wave infra-red spectral survey. More recently, a drone airborne unmanned aerial vehicle (UAV) magnetic (MAG) survey and rock chip sampling programs were done.

1.7 Drilling

This summary outlines diamond drilling programs completed by RRR and New Gold from 2005 to present. Drill procedures used by Nuinsco between 1994 and 2004 and Bayfield between 2010 and 2014 are not well documented and are not described in this report.

Diamond drill programs completed at the Rainy River deposit and the Intrepid Zone were performed by Bradley Bros. Ltd. in partnership with Naicatchewenin Development Corporation. 97% of drilling used NQ core tools from surface collars. HQ (2.75%) and PQ (0.25%) size comprise the remaining 3% of drillholes. 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. 2,191 diamond core exploration holes, for a total of 924,688 m, were drilled on the Property.

A hand-held global positioning system (GPS) was used to locate and prepare drilling pads in the field. At the completion of each drillhole a Differential GPS (DGPS) was used to survey the casing collar. Drillhole deviation surveys were completed using a Reflex EZ-SHOT™ instrument. Downhole surveys were collected on 50 m intervals. At the Intrepid Zone, approximately 25% of the drillholes have been resurveyed with a Reflex Gyro at 5 m intervals.

Core logging data is captured directly onto laptop computers previously using Datamine's DHLogger™ and more recently, Maxwell LogChief™. Validation protocols are built into the software to ensure data consistency and minimize data collection errors. LogChief™ logging data is merged into a central Maxwell Datashed™ database where further validation is completed.

Sampling intervals have changed throughout the years. RRR initially selectively sampled parts of the drillholes based on visible observations. Core was marked for sampling at regular 1.5 m intervals. In 2012, RRR adjusted sampling procedures so that the entire drillhole was sampled with predominantly 1.5 m samples. Under New Gold in 2016 and 2017, sampling was performed at regular 1.0 m intervals. Shorter samples were collected at the contacts between geological domains.

Diamond core sample recovery data has been collected since New Gold acquired the Property in 2013. Core recoveries from New Gold drill programs vary between 2.33% and 100%, averaging 99.9%. A total of 219 of the 16,746 intervals in the database have recoveries less than 90%.

All diamond drill core is processed and stored at New Gold's secure onsite core logging facilities, which are security monitored 24 hours per day, seven days per week.

In AMC's opinion, there are no drilling, sampling, or recovery factors that could materially impact the accuracy and reliability of drill results.

1.8 Sample preparation and analysis

Sample preparation and analysis varied by year and operator. Between 1994 and 2004 during Nuinsco's ownership, samples were prepared and analyzed by ALS Chemex (ALS) in ON, Canada. Between 2005 and 2013 during RRR's ownership, samples were prepared and analyzed by ALS, Accurassay Laboratories Ltd. (Accurassay) and Activation Laboratories Ltd. (Actlabs). With the exception of ALS, which used the Thunder Bay prep laboratory (lab) and the North Vancouver Analytical Laboratory, all work was done in Thunder Bay. Between 2014 and 2017 during New Gold's ownership, samples were prepared and analyzed by ALS which used the Thunder Bay preparation lab and the North Vancouver Analytical Laboratory. Bayfield used both Actlabs in Thunder Bay and TSL Laboratories Inc. (TSL) in Saskatoon, Saskatchewan (SK). All laboratories were International Standards Organization accredited and are independent of New Gold.

No Quality Assurance / Quality Control (QA/QC) data is available for the period of 1994 to 2004 when Nuinsco was carrying out its exploration. Drilling programs completed on the Property between 2005 and 2017 included QA/QC monitoring programs which varied by operator but, in total comprised insertion of certified reference materials (CRMs), blanks, and field duplicates into the sample streams on a batch by batch basis. New Gold also included umpire checks, coarse duplicates and pulp duplicates.

In general, the QA/QC sample insertion rates have improved during New Gold's ownership but still fall somewhat below the general accepted industry standards. Despite the issues highlighted for improvement, the Qualified Person (QP) considers the assay database to be acceptable and does not consider these issues to be material to the global, long term Mineral Resource estimate.

1.9 Data verification

Data verification was carried out under the supervision of the QP, with 5.6% of the samples being verified in the database. This verification included comparing 1,360 of the 24,227 assays for the drilling conducted from 2015 to 2017. No errors were identified.

Reconciliation of the resource block model to grade control and ex-mine material is carried out monthly and has been reviewed for 2019. There is difficulty reconciling to the mill figures due to large moving stockpiles, but the results appear satisfactory.

In the opinion of the QP, 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 industry best standards as defined in the CIM Exploration Best Practice Guidelines and CIM Mineral Resource and Mineral Reserve Best Practice Guidelines. As such, the data are suitable for use in the estimation of Mineral Resources.

1.10 Metallurgical testwork

The original metallurgical testwork programs on Rainy River samples were used to support the design and engineering of the Rainy River process plant.

Post plant start-up metallurgical testwork has been conducted since the start-up of the Rainy River process plant, including:

- Acid wash testwork – Carbon activity tests were completed on carbon samples that had been acid washed and carbon samples that were not. The testwork demonstrated that there was no significant difference in carbon activity between the two sample types.
- Flocculant screening testwork - Settling rates in the pre-leach thickener are a plant bottleneck. Several flocculant screening testwork programs have been completed to attempt to rectify these issues. These programs identified that, during winter periods, the cold solution reduces flocculant dissolution rates.

Predictive formulas were developed for estimating plant gold recovery and silver recovery.

The formulas for the Non-CAP Zone gold recovery were updated in 2019. The CAP Zone and Intrepid Zone gold recovery formulas have not been modified.

Silver recovery predictive formulas were updated from metallurgical programs (Kenny 2016).

The resultant average orebody predicted metal recoveries are 89% for gold and 57% for silver. Note that the process plant has regularly been able to achieve gold recoveries that have exceeded the original design criteria.

It is AMC's opinion that the metallurgical test programs for the Rainy River deposit were comprehensive and have included the major ore types and taken the mine plan into consideration when developing the composite samples. The types of tests performed were appropriate and provided sufficient information for preparing the designs for the process plant.

1.11 Mineral Resources

The Mineral Resource estimates for the Rainy River Mine are based on two block models. These are for the Main and Intrepid Zones. The Main Zone was modelled and estimated by Mr Mauro Bassotti (formerly of New Gold), and the estimate for the Intrepid 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.

A summary of Mineral Resources at the Property as of 31 December 2019 is presented in Table 1.1. Mineral Resources stated here are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and do not have demonstrated economic viability. Definitions for Mineral Resource categories used in this report are consistent with those defined by CIM Definition Standards for Mineral Resources and Mineral Reserves (2014).

The parameters and modifying factors that apply are listed in the footnotes.

Table 1.1 Mineral Resources effective 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000's)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Direct processing Mineral Resources					
Open pit					
Measured	695	1.46	2.9	33	64
Indicated	4,813	1.18	3.4	182	531
Sub-total open pit M + I	**5,508**	**1.21**	**3.4**	**214**	**596**
Inferred	2,015	0.61	1.8	39	114
Underground					
Measured	-	-	-	-	-
Indicated	14,866	3.49	9.1	1,669	4,331
Sub-total underground M + I	**14,866**	**3.49**	**9.1**	**1,669**	**4,331**
Inferred	1,297	3.76	3.5	157	146
Low grade Mineral Resources					
Open pit					
Measured	293	0.34	1.9	3	18
Indicated	2,460	0.34	2.2	27	175
Sub-total open pit M + I	**2,753**	**0.34**	**2.2**	**30**	**193**
Inferred	167	0.35	1.4	2	8
Total Mineral Resources					
Measured	989	1.13	2.6	36	82
Indicated	22,139	2.64	7.1	1,878	5,037
Total M + I Mineral Resources	**23,127**	**2.57**	**6.9**	**1,914**	**5,120**
Total Inferred Mineral Resources	3,479	1.77	2.4	198	268

Notes:

- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Resources.
- The Mineral Resources are stated exclusive of Mineral Reserves.
- Mineral Resources are estimated using a long-term gold price of US$1,375 per troy oz and a long-term silver price of US$19 per troy oz. The exchange rate used was 1:1.30 US$/C$.
- Direct processing open pit Mineral Resources are estimated at a gold equivalent (AuEq) cut-off grade (COG) of 0.45 g/t for the CAP Zone and 0.44 g/t for the Non-CAP Zone. Low grade open pit Mineral Resources were estimated at an AuEq cut-off of 0.30 g/t. Gold equivalency was estimated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 90)].
- Open pit assumptions include:
  - Metal recoveries are variable dependent on metal head grade. At COG, the gold recoveries are as follows:
    - Direct Processing Ore
      CAP zone gold = 73.8%
      Non-CAP zone gold = 77.0%
    - Low Grade Ore
      CAP zone gold = 73.1%
      Non-CAP zone gold = 68.9%
  - Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
  - Open pit Mineral Resources are constrained by a conceptual pit shell.
  - Inferred open pit Mineral Resources include inferred material from within the Mineral Reserve open pit.
- Underground Mineral Resources are estimated at an AuEq COG of 2.00 g/t. Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 95)].
- Underground assumptions include:
  - Average gold and silver recoveries of 95% and 60%, respectively.
  - Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
- Effective date of Mineral Resources is 31 December 2019.
- The QP for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC.
- Totals may not compute exactly due to rounding.

The Main Zone estimate is based on a block model completed in 2017 using Maptek's Vulcan software, and the estimate of the Intrepid Zone is based on a block model completed in 2015 using GEMS software. Interpolation of gold and silver grades for all models was completed using ordinary kriging (OK). Bulk density values were interpolated in the Main Zone using inverse distance squared (ID^2) and were assigned to the Intrepid Zone based on rock type.

The 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.

1.12 Mineral Reserves

The Mineral Reserve estimate conforms to CIM Definition Standards for Mineral Resources and Mineral Reserves (2014). The Mineral Reserves are effective 31 December 2019 and only Measured and Indicated Resources have been used for Mineral Reserve estimation.

Mr F. McCann and Mr H.A. Smith both of AMC, take responsibility for the open pit (OP) and underground (UG) Mineral Reserve estimates, respectively.

Proven and Probable Mineral Reserves for the Rainy River deposit are shown in Table 1.2. The parameters and modifying factors that apply are listed in the footnotes to the table.

The Mineral Reserves reported herein supersede the Mineral Reserves reported previously at year-end 2018 by New Gold for the Rainy River Mine.

Table 1.2 Mineral Reserves – effective 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	**Gold (g/t)**	**Silver (g/t)**	**Gold (koz)**	**Silver (koz)**
Total Mineral Reserves					
Open pit (including stockpile)					
Proven	27,331	0.88	2.0	779	1,740
Probable	46,145	0.88	2.4	1,308	3,492
Sub-total open pit	**73,476**	**0.88**	**2.2**	**2,087**	**5,231**
Underground					
Proven	-	-	-	-	-
Probable	4,096	4.17	7.8	549	1,034
Sub-total underground	**4,096**	**4.17**	**7.8**	**549**	**1,034**
Total					
Proven	27,331	0.88	2.0	779	1,740
Probable	50,241	1.15	2.8	1,857	4,526
Total Mineral Reserves	**77,572**	**1.06**	**2.5**	**2,636**	**6,266**

Notes:
- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Reserves.
- Mineral Reserves are estimated using a long-term gold price of US$1,275 per troy oz and a long-term silver price of US$17 per troy oz. The exchange rate used was 1:1.30 US$/C$.
- Direct processing open pit Mineral Reserves are estimated at an AuEq COG of 0.49 g/t for the CAP Zone and 0.46 g/t for Non-CAP Zones. Low grade open pit Mineral Reserves were estimated at an AuEq cut-off of 0.30 g/t. Gold equivalency was estimated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 90)].
- Open pit assumptions include:
  - COGs applied to a regularized 10 m x 10 m x 10 mine planning block model, which was generated from re-blocking the original resource model. Modifying factors representing a planned dilution of 1.3 m below 290 m and 4.0 m above 290 m were applied, the latter factor being higher as it reflects the uncertainties in the geometry of the rock / overburden contact. Ore blocks surrounded by waste blocks were accounted as ore loss, while waste blocks surrounded by ore blocks were included as additional dilution.
  - Metal recoveries are variable dependent on metal head grade. At COG, the gold recoveries are as follows:
    - Direct processing ore (DPO)
      CAP Zone gold = 73.9%
      Non-CAP Zone gold = 78.2%
    - Low-grade ore (LGO)
      CAP Zone gold = 73.1%
      Non-CAP Zone gold = 68.9%
  - Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
- Underground Mineral Reserves are estimated at an AuEq COG of 2.20 g/t for stoping and 0.80 g/t for development. Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 95)].
- Underground assumptions include:
  - Planned hangingwall (HW) and footwall (FW) dilution of 0.6 m and 0.3 m, respectively, with total unplanned dilution of approximately 12%.
  - Average mining recovery estimated as 95%.
  - Average gold and silver recoveries of 95% and 60%, respectively.
  - Cut-off value of $83.12/t, inclusive of costs for mining, processing, General and Administrative (G&A), refining & transport, royalties, and sustaining capital allowance.
- Effective date of Mineral Reserves is 31 December 2019.
- The QP for the OP estimate is Mr F. McCann, P.Eng., and for the UG estimate is Mr H.A. Smith, P.Eng., both of AMC.
- Totals may not compute exactly due to rounding.

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

1.13 Mining

Mining at Rainy River is currently conducted using open pit mining methods and will transition into a combined open pit and underground operation over the next two years, with underground production commencing in 2022. An average processing rate of approximately 25,800 tonnes per day (tpd) is scheduled over the life-of-mine (LOM).

1.13.1 Open pit mining

The open pit mine is a conventional truck and shovel mining operation, with a fleet of 220 t payload haul trucks combined with diesel-powered hydraulic excavators and large front-end loaders (FELs) as primary loading units. The open pit operates at a peak mining rate of 151,000 tpd of ore and waste and has an overall strip ratio of 2.53:1 (waste:ore).

The open pit design is based on overburden slope recommendations from Golder Associates Ltd. (Golder), and hard rock slope recommendations from SRK. The overburden slope ranges between 3:1 and 8:1 (horizontal:vertical) while the hard rock slope is designed at inter-ramp angles ranging from 37° to 54° with 25 m wide geotechnical berms left every 120 m in height unless the slope was otherwise interrupted by a similar acting feature (i.e. haulage ramp).

The mine plan is executed to take advantage of the installed mine fleet productive capacity, allowing an elevated cut-off grade (COG) policy to be adapted, whereby higher grade, direct processing ores (DPOs) are preferentially sent to the mill for processing while lower grade ores (LGOs) are sent to stockpile for deferred processing. This results in an open pit mine life extending to Q1-2025 with stockpile rehandling occurring in parallel with the underground operations through to Q1-2028 to fulfill available process plant capacity.

Waste from the open pit is identified as either overburden (including glacial tills and clays), non-acid generating waste (NAG) or potentially acid generating waste (PAG). Waste is stored at two locations; the East Mine rock stockpile (EMRS) and the West Mine rock stockpile (WMRS).

Commencing in 2020, NAG requirements for the tailings management area (TMA) construction are capable of being fulfilled from in-pit mine production. No future mining of the east outcrop (EOC) for NAG construction rock is included in the mining schedule.

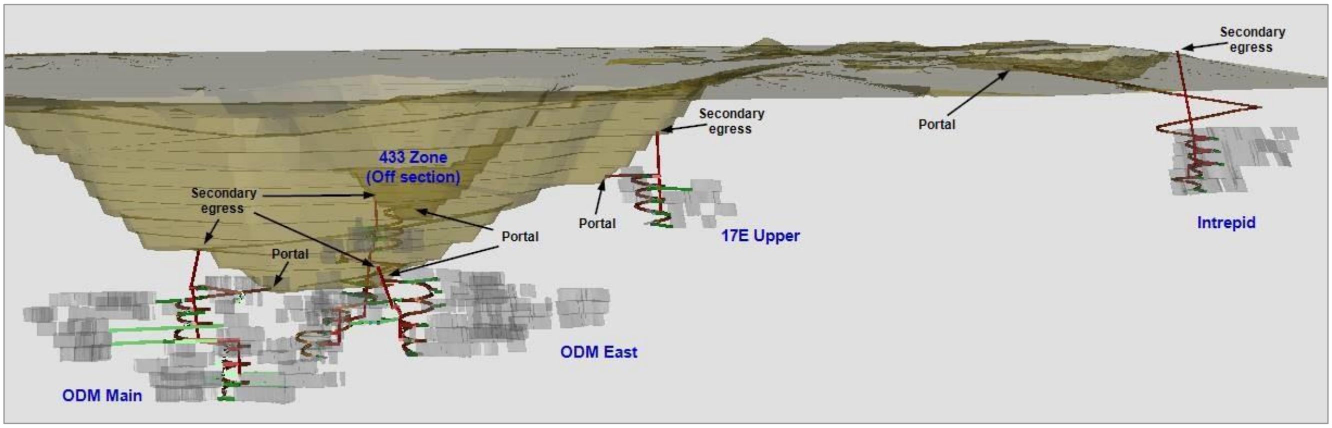
Except for the addition of a leased CAT 994HL to the fleet in 2020 to assist principally with stockpile rehandle, peak open pit mining equipment requirements correspond to the current fleet size.

1.13.2 Underground mining

The underground is designed as a mechanized ramp access mine that will use longitudinal longhole open stoping (LLHOS) techniques to exploit the underground Mineral Reserves. Underground ore production rates will be variable but are planned to reach a maximum of approximately 3,200 tpd in year 2026.

The underground mine is planned to be accessed from five portals targeting separate ore zones: ODM Main Zone, ODE East Zone, 17 East Upper Zone, 433 Zone, and Intrepid Zone. These are shown in Figure 1.1.

Figure 1.1 Underground isometric view (looking north)



Source: AMC 2019.

Underground mining commences in the 17 East Upper Zone in July 2022, followed by: Intrepid Zone in June 2023, 433 Zone in July 2024, and finally ODM Main Zone and ODM East Zone concurrently in April of 2025. The underground Mineral Reserve is mined in its entirety by Q1-2028.

Note that although the Intrepid Zone is scheduled for production mining in January 2024, the decline will start advancing in Q2-2020 for a distance of 600 m as part of an orebody investigation project for the Intrepid Zone.

Ore grade gold / silver mineralization occurs in sub-vertical horizons ranging from 3 m to 20 m thick, and the weighted average thickness is 8 m. A 3 m 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 methods rely upon gravity ore flow along the footwall.

The underground mine will use two mining methods:

- Blind up-hole LLHOS without backfill (uphole).
- Downhole LLHOS with backfill (downhole).

Stopes less than 15 m wide will be mined as uphole with 8 m thick rib pillars generally spaced 48 m along strike. Stopes greater than 15 m wide will be mined as downhole, which utilizes backfill, but do not require rib pillars between stopes.

Ore from the UG will be mined at a rate of approximately 3,100 tpd from 2025 onwards and will be blended with open pit stockpiles to maintain the total (UG and OP) mill feed rate at approximately 25,800 tpd over the LOM.

1.13.3 Mine-to-mill schedule - all sources

Over the LOM, the open pit (including stockpile rehandle) and underground operations will feed to the mill a total of 77.6 million tonnes (Mt) of ore grading 1.06 grams per tonne (g/t) gold and 2.5 g/t silver, totaling 2,636 thousand ounces (koz) of contained gold and 6,266 koz of contained silver. The mine-to-mill schedule from all sources is presented in Table 1.3.

Table 1.3 Mine-to-mill schedule

Year	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
2020	9,260	0.95	2.5	284	759
2021	9,605	1.02	2.6	315	788
2022	9,549	1.17	2.3	360	714
2023	9,521	1.19	2.6	364	795
2024	9,563	1.26	3.4	387	1,047
2025	9,474	1.25	2.4	380	726
2026	9,421	0.85	2.2	257	652
2027	9,439	0.84	2.2	255	662
2028	1,739	0.61	2.2	34	121
Total	**77,572**	**1.06**	**2.5**	**2,636**	**6,266**

Note: Totals may not compute exactly due to rounding.

1.14 Processing

The Rainy River process plant consists of a flowsheet that includes crushing, grinding, gravity separation, cyanide leaching, carbon-in-pulp (CIP), carbon stripping and regeneration, electrowinning, and doré production. Plant tailings are deposited in the TMA, and TMA water is reclaimed and returned to the process plant.

Rainy River's current focus is to operate at higher throughputs than the original plant design throughput. This produces a coarser grind size P_{80} than the design criteria grind P_{80} of 75 µm, which reduces gold recovery. Rainy River has determined that an increase in throughput at the expense of gold recovery is the most economically viable option.

The LOM process plant throughput expansion will increase the plant overall throughput beyond the 22,000 tpd - 24,000 tpd range achieved in 2019 to an average of approximately 25,300 tpd in 2020. The LOM throughput averages approximately 25,800 tpd.

Rainy River has been permitted to operate at a daily plant throughput of 27,000 tpd, averaged over a calendar quarter. The daily peak throughput limit has been permitted at 32,400 tpd.

Some recent modifications have been made to the process plant, including:

- The pebble crusher circuit has been commissioned to assist in achieving the planned throughputs when the ore becomes harder.
- The gravity concentration circuit finished re-commissioning in January 2020.
- The acid wash circuit has been de-commissioned, based on the carbon activity testwork program results
- Dust from the crushed ore stockpile has been identified as an environmental and health concern. Solutions that are being trialled include adding spray water in summer and adding a calcium chloride solution in winter.

Bottlenecks were identified in the process plant that are planned and budgeted to be resolved in 2020:

- Rainy River will trial a polymer slicing unit and an alternative flocculant supply system with a goal of improving settling rates in the pre-leach thickener, particularly during winter months when the solution temperature is cold.

- The tailings pumps capacity is currently insufficient when depositing along the North Dam of the TMA at the planned throughput rates. In 2020, a booster pump station will be installed that will sustain an average throughput of 27,000 tpd when depositing at the North Dam.

1.15 Markets and contracts

Gold and silver markets are mature global markets with reputable refiners located throughout the world. Mineral Reserves have been assessed at $1,275/oz gold and $17/oz silver.

According to the London Bullion Market Association (LBMA), the average daily PM Fix gold price for 2019 was $1,393 per troy ounce. The three-year and five-year rolling average prices through the end of December 2019 are $1,306 and $1,266 per troy ounce, respectively.

According to the LBMA, the average daily silver price for 2019 was $16.21 per troy ounce. The three-year and five-year rolling average prices through the end of December 2019 are $16.32 and $16.35 per troy ounce, respectively.

Based on the undertaken review, AMC considers the metal prices selected by New Gold to be reasonable, particularly given the recent appreciation in the gold metal price.

Gold output from the Rainy River Mine operation is in the form of doré bars (doré) containing approximately 40% gold and 60% silver on average. Silver credits are received from the Refiner. The doré is shipped to either Asahi Refining Canada Ltd. in Brampton, ON or to the Royal Canadian Mint in Ottawa, ON. Transportation of the doré to either refinery is contracted out by the respective refineries. Responsibility for the doré changes hands at the gold room gate upon signed acceptance by the Refiner or its Transport Provider.

In the second quarter of 2019, New Gold entered into gold price option contracts to cover approximately 50% of the company's consolidated gold production by purchasing put options at an average strike price of $1,300 per ounce and selling call options at an average strike price of $1,355 per ounce for 72,000 ounces of gold production between January 2020 and June 2020. Further, New Gold entered into gold price option contracts by purchasing put options at an average strike price of $1,300 per ounce and selling call options at an average strike price of $1,415 per ounce for 96,000 ounces of gold production between July 2020 and December 2020.

In 2015, New Gold entered into a $175 million (M) streaming transaction with RGLD Gold AG (Royal), a wholly-owned subsidiary of Royal Gold Inc. (Royal Gold). Under the terms of the agreement, New Gold will deliver to Royal Gold 6.5% of gold production from Rainy River up to a total of 230,000 oz of gold and then 3.25% of the mine's gold production thereafter. New Gold will also deliver to Royal Gold 60% of the mine's silver production to a maximum of 3.1 million ounces and then 30% of silver production thereafter.

In addition to the upfront deposit, Royal Gold will pay 25% of the average spot gold or silver price at the time each ounce of gold or silver is delivered under the stream. The difference between the spot price of metal and the cash received from Royal Gold will reduce the $175M deposit over the life of the mine. Upon expiry of the 40-year term of the agreement (which may be extended in certain circumstances), any balance of the $175M upfront deposit remaining unpaid will be refunded to Royal Gold.

1.16 Infrastructure

Principal site infrastructure at the Rainy River Mine has generally been constructed, with only a few outstanding buildings scheduled for completion in 2020. Major infrastructure currently installed (or in the process of being installed) includes:

- Primary access and mine haul roads.
- A 27,000 tpd CIP gold process plant with electrowinning and refining capability.
- Mine maintenance facilities including truck shops, a truck wash and a fuel bay.
- Warehousing facilities including a general warehouse and separate lubricant and hydrocarbon storage buildings.
- An explosive magazine and emulsion plant provided and operated by a specialized contractor.
- A 5 m x 5 m decline developed for a distance of 166 m towards the Intrepid Zone.
- A 230 kilovolts (kV) power line connected to the Hydro One power grid, feeding two main 230 kV to 13.8 kV, 42/56/70 mega volt amperes (MVA) transformers for a combined power of 100 MVA.
- An integrated water treatment train.
- A camp facility capable of providing concurrent accommodation for 376 personnel.
- Misc. security, administration, and general offices and buildings located throughout the property.

In addition, principal infrastructure includes a TMA and two mine rock stockpile storage facilities. These are both described in more detail in the following.

1.16.1 Tailings management area

1.16.1.1 Tailings management area and dams

The TMA at the Rainy River Mine has an approximate footprint area of 550 ha. Tailings containment is provided by three perimeter dams: TMA North Dam, TMA West Dam (comprising Dam 4 and Dam 5), TMA South Dam, and elevated topography along the north-east side of the impoundment. Currently, the TMA is divided into three independent tailings deposition areas: TMA Cell 1, TMA Cell 2, and TMA Cell 3. Active tailings deposition is occurring in TMA Cell 2 and TMA Cell 3, whereas TMA Cell 1 is currently inactive. In Q4-2019 TMA Cell 2 and TMA Cell 3 merged into a contiguous cell.

1.16.1.2 Tailings dam design and construction

The tailings dams are designed and constructed following the centerline method. The dam cross-sections generally consist of a low permeability clay core and downstream granular chimney and blanket filters, which are surrounded by a rockfill shell. Clay and rockfill materials are sourced from the mine property, whereas the filter materials are produced by crushing or are obtained from offsite locations. The maximum ultimate dam heights vary from 16 m to 27 m, and the downstream side of the embankment is supported by rockfill buttresses with an approximate overall slope of 14 – 18H:1V (horizonatal:vertical).

Stability criteria are based on the Lakes and Rivers Improvement Act (LRIA) guidelines for dams constructed in ON and the Canadian Dam Association (CDA) guidelines for mining dams. The applicable stability factors of safety (FOS) are:

- Short term, end of construction, FOS = 1.5
- Pseudo-static, FOS = 1.0
- Post-earthquake, FOS = 1.1

The seismic risk in the area of the Rainy River Mine is low. Dam stability design considered a Site Class D response spectrum for a seismic event with a return period of 10,000 years. The emergency spillway of the TMA is designed to pass the probable maximum flood. The TMA will be operated using the 50^{th} percentile pond volume (based on probabilistic modelling) and is intended to retain the 99^{th} percentile pond volume, corresponding to a minimum annual probability of 1% or less likelihood of discharge from the emergency spillway.

Construction of portions of the dams is weather dependent. Clay core and filter materials need to be placed and compacted in an unfrozen condition, limiting construction to the late spring, summer, and early fall. Rockfill shell materials can be placed and compacted year-round.

1.16.1.3 Tailings deposition plan

The revised dam raise schedule, based on updated tailings deposition modelling results, guides the yearly engineering design and construction requirements of the TMA dams. Tailings are currently end spilled and spigoted from the TMA South Dam and TMA Cell 1 Dam and will eventually move to the TMA North Dam. Once Stage 3 construction is complete, tailings discharge will occur from the perimeter dams to create an upstream beach along these structures and will move the supernatant pond adjacent to the elevated topography along the north-east section of the TMA. Following the beginning of tailings deposition along the TMA North Dam, the current fixed reclaim pump station will be abandoned and a barge reclaim system will be implemented. After Stage 3 construction, TMA Cell 1 will merge into a single unit with TMA Cell 2 and TMA Cell 3 which have already merged into a contiguous cell during Q4-2019.

1.16.1.4 Ultimate dam design and construction

The main factors governing the design and construction of the TMA dams are the relatively low shear strength and development of excess pore pressures (resulting from dam construction) in the cohesive soils present beneath the dam foundations. As a result, relatively large downstream buttresses, which are required to maintain dam stability, will impact existing downstream infrastructure. These impacts will need to be mitigated with ground improvement methods (i.e. preloading, wick drains, or shear key construction), infrastructure relocation, or revising the dam design section. Future dam designs will need to refine assumptions regarding excess pore pressure dissipation resulting from yearly raising of the TMA dams.

Future TMA dam design and construction will need to consider material availability and cost. In particular, the availability and location of clay and granular filter material based on LOM quantity estimates.

Opportunities exist to implement ground improvement strategies ahead of subsequent TMA dam raises, such as preloading downstream buttress footprints or preloading combined with wick drain installation where poorer subsurface conditions exist. Furthermore, such an approach will provide operational flexibility for placing buttress material when available from the open pit, thereby reducing material double handling and a greater utilization of the mine fleet for dam construction.

The main risk with respect to dam stability is managing the increase and dissipation of excess pore pressures within the cohesive soils present in the dam foundations caused by raising the dams. Although excess pore pressure increases are minimized by lower, more frequent, dam raises, lower than estimated pore pressure dissipation rates may require the construction of additional buttressing.

1.16.2 Mine rock and overburden stockpiles

Storage of mine waste rock and overburden waste is provided at two locations, the EMRS and the WMRS. The west area of the EMRS is also being used for the temporary storage of stockpiled LGOs.

The EMRS provides storage for a combination of PAG waste rock, overburden waste and LGOs. Ground improvement measures have been and / or are currently being implemented around the perimeter of the EMRS to improve the shearing resistance of the foundation. EMRS ground improvement measures include constructing waste rock shear keys (where foundation clay thicknesses are between 3 m to 8 m) and installing wick drains at 2 m spacing. No ground improvement measures are required where the clay thickness is less than 3 m. A controlled rate of stockpile raising is required within the perimeter area to allow time for the dissipation of excess porewater pressures (PWPs) due to loading, and the associated consolidation strength gain. In general, a maximum rate of rise of 9 m/year within the EMRS perimeter has been considered in the design; however, a greater rate of placement may be allowed in some areas in the initial year upon approval from the engineer-of-record.

A preliminary design of the WMRS has been completed to assess the maximum stockpile height and geometry which may be obtained without the requirement for foundation improvement measures, such as wick drain installation or construction of shear keys, (which have been implemented at the EMRS). The WMRS provides storage for a combination of NAG waste rock and overburden waste. The design provides storage for overburden waste internally in an overburden waste dump, with waste rock to be stockpiled around the perimeter, where it will function to buttress the overburden waste. Optimization is expected to somewhat increase the nominal capacity of the WMRS.

Under the current mine plan, the mine rock and overburden stockpile designs have sufficient capacity to accommodate the quantities of waste materials and LGO expected to be produced.

1.17 Environmental

New Gold remains committed to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against the Health, Safety, Environment and Corporate Social Responsibility Policy approved by New Gold’s Board of Directors on 25 July 2018.

At the time of this review, the Mine Environmental Department was adequately staffed, and had increased accountabilities, with the addition of water resource management. New Gold conducts ambient air quality, surface and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

From 2018 to 2019, the Rainy River Mine has recorded eleven non-compliance related issues. All non-compliances were reported to regulatory agencies. No charges or fines were levied.

The environmental budget for 2020 was reviewed and appears to be reasonable to accommodate the number of regulatory reporting and study commitments, and flexible enough to respond to changes in operating conditions.

New Gold continues to develop the Environmental Management System (EMS). Phase 1 was completed during 2019. Phase 2 is scheduled for implementation in 2020.

1.18 Capital and operating costs

Total LOM capital costs are estimated to total $642M as summarized in Table 1.4. This excludes $107M in funds identified for progressive, final and post-closure activities identified in the last line of Table 1.4.

Table 1.4 Capital costs summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Open pit	$(000)	60,909	62,758	94,536	53,945	22,246	506	5,584	606	-	301,090
Underground	$(000)	3,846	-	16,599	16,881	19,579	52,185	10,857	333	-	120,281
Process and TMA	$(000)	35,483	37,152	29,991	31,159	30,260	21,721	-	-	-	185,765
Infrastructure & Other	$(000)	22,708	6,637	1,923	1,923	1,923	-	-	-	-	35,115
Grand total	**$(000)**	**122,946**	**106,547**	**143,050**	**103,908**	**74,008**	**74,411**	**16,441**	**938**	**-**	**642,250**
Project capital	$(000)	3,846	-	10,072	3,390	11,149	27,695	-	-	-	56,152
Sustaining capital	$(000)	119,100	106,547	132,978	100,518	62,859	46,716	16,441	938	-	586,098
Grand total	**$(000)**	**122,946**	**106,547**	**143,050**	**103,908**	**74,008**	**74,411**	**16,441**	**938**	**-**	**642,250**
Reclamation / closure [1]	$(000)	1,178	3,051	3,036	3,044	3,028	2,974	3,013	4,655	82,953	106,932

Notes:
- [1] The 2028 amount includes $66M of final and post-closure costs to be expended after 2028.
- Totals may not compute exactly due to rounding.

A summary of the estimated LOM operating costs is shown by year in Table 1.5. Estimated unit operating costs, plus the LOM average, are shown in Table 1.6.

Table 1.5 Operating cost summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Open pit	$(000)	113,445	108,811	66,554	95,214	88,850	29,725	23,448	21,286	3,589	550,922
Underground	$(000)	-	-	4,145	12,061	27,094	55,861	62,162	40,816	5,188	207,327
Process	$(000)	69,435	71,322	70,139	70,316	69,680	68,612	68,306	68,173	9,028	565,010
G&A	$(000)	32,430	33,273	29,037	28,547	25,920	21,290	21,076	19,547	2,222	213,342
Grand total	**$(000)**	**215,310**	**213,406**	**169,875**	**206,137**	**211,544**	**175,489**	**174,992**	**149,822**	**20,026**	**1,536,601**

Note: Totals may not compute exactly due to rounding.

Table 1.6 Unit operating cost summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg.
Open pit	$/t moved	2.36	2.27	2.24	2.54	2.73	3.21	2.83	2.55	2.33	2.47
Open pit	$/t mined	2.74	2.57	2.84	2.90	3.14	15.40	-	-	-	3.23
Underground	$/t ore mined	-	-	80.74	84.30	50.45	57.11	54.08	36.42	44.72	50.62
Mining	$/t milled	12.25	11.33	7.40	11.27	12.12	9.03	9.09	6.58	5.05	9.77
Process	$/t milled	7.50	7.43	7.34	7.39	7.29	7.24	7.25	7.22	5.19	7.28
G&A	$/t milled	3.50	3.46	3.04	3.00	2.71	2.25	2.24	2.07	1.28	2.75
Grand total	**$/t milled**	**23.25**	**22.22**	**17.79**	**21.65**	**22.12**	**18.52**	**18.57**	**15.87**	**11.52**	**19.81**

Note: Totals may not compute exactly due to rounding.

1.19 Economics

A financial analysis for Rainy River was developed by New Gold using a discounted cash flow approach on a pre-tax and after-tax basis. The net present value (NPV) was calculated from the cash flow generated by the project based on a discount rate of 5% utilizing metal selling prices of $1,300 per ounce gold and $16 per ounce silver. Annual cash flows were assumed to occur end-of-period.

The pre-tax base case financial model resulted in an undiscounted cash flow of $559M with an NPV of $426M at a discount rate of 5%. On an after-tax basis, the base case financial model resulted in an undiscounted cash flow of $557M with an NPV of $424M with a discount rate of 5%. The minimal change in value between pre-tax and after-tax cash flows is a result of the utilization of current and projected tax attributes.

The cash cost of production (net of silver sales) is $655 per ounce gold produced, with an all-in-sustaining cost of $964 per ounce gold produced.

AMC has reviewed New Gold's LOM financial model and has performed an economic analysis of the Rainy River Mine using this model adjusted for Mineral Reserve reporting metal prices declared in this report of $1,275 per ounce gold and $17 per ounce silver. The pre-tax Mineral Reserve price financial model resulted in an undiscounted cash flow of $505M with an NPV of $382M at a discount rate of 5%. On an after-tax basis, the Mineral Reserve financial model resulted in an undiscounted cash flow of $504M with an NPV of $381M at a discount rate of 5%. The minimal change in value between pre-tax and after-tax cash flows is a result of the utilization of current and projected tax attributes.

Due to the complexity of New Gold's financial model, only the key drivers are varied in the preceding analysis, with other costs in the model maintained constant per the base case (i.e. royalty payments, refining and freight, etc.). The impact of this does not change the overall interpretation of the analysis

AMC confirms that the outcome is a positive cash flow that supports the statement of Mineral Reserves.

1.20 Conclusions

The QPs offer the following conclusions:

1.20.1 Geology and Mineral Resources

1.20.1.1 Geology

The assay database to be acceptable for Mineral Resource estimation as part of a global, long term Mineral estimate. However, there is no guarantee that there are no material impacts on the local scale.

1.20.1.2 Mineral Resources

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

Measured and Indicated Mineral Resources are estimated to total 23.1 Mt at grades of 2.57 g/t Au and 6.9 g/t Ag, containing 1,914 koz of gold and 5,120 koz of silver. Inferred Mineral Resources are estimated to total 3.5 Mt at grades of 1.77 g/t Au and 2.4 g/t Ag, containing 198 koz of gold and 268 koz of silver. The Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

The block model has been performing adequately as indicated by reconciliation results and presents low risk to the project. The opportunity for growth of Mineral Resources on the deposit are mainly price and cost driven.

The geology block model has not been updated for some years and the interpretation should be revisited to include any new information gained though mining of the deposit. There is also some new data that should be included.

1.20.2 Mining and Mineral Reserves

1.20.2.1 Open pit

Open pit Proven and Probable Mineral Reserves, including stockpile, total 73.5 Mt grading 0.88 g/t gold and 2.2 g/t silver, containing 2,087 koz and 5,231 koz of gold and silver, respectively.

The reconciliation of the mine planning model to 2019 in-pit production indicates that the mine planning model is performing well within the normal range of variability (±10%).

The current pit design and resulting Mineral Reserve is robust over a range of metal prices under the current design criteria.

The open pit overburden design slopes satisfy the required FOS. The design requires placement of a toe berm / slope buttress shortly following the completion of excavation. Timely placement of the rockfill toe berm is critical; delays in rockfill placement may result in slope instability. Control of the surface water is important.

The open pit hard rock pit slope stability and resulting design is defined by:

- The orientation of the regional south-southwest dipping foliation structures (North Wall).
- The kinematic stability related to the major joint sets (all pit walls).

With consideration to the above, the revised design recommendations were predominantly driven by the bench-berm stability assessments with shallower bench face angles (BFAs) adopted to reduce the undercutting potential from planar sliding and wedge intersections.

Approximately 238 Mt of material is scheduled to be extracted from the open pit using conventional truck and shovel mining methods. Open pit mining is executed by a fleet of 220 t payload haul trucks combined with diesel powered hydraulic excavators and large front-end loaders (FELs) as primary loading units.

Low grade ore mined and stockpiled is critical to the success of the underground mine and needs to be managed strategically to ensure viability of the underground operation.

No additional or replacement capex is required for the principal mining fleet within the current mine plan with the exception of a leased CAT 994HL FEL. Fleet size and age are suitable to execute the proposed mine plan.

The principal fleet is capable of the productivity requirements to execute the LOM plan. The vertical advance of mine development is within industry norms for large scale gold mines, albeit at the upper range. Rainy River has not mined at these rates in the past and will need to adjust their short-range planning to manage this quantity of bench turn-over, which is achievable with best-in-class planning practices.

NAG quantities for TMA construction are available from in-pit mining. No mining of the east out-crop is included in the current mine plan.

1.20.2.2 Underground

The underground Probable Mineral Reserves are estimated to be 4.1 Mt grading 4.17 g/t gold and 7.85 g/t silver, using a cut-off grade of 2.2 g/t gold equivalent. Checking of the COG against final estimated costs for this NI 43-101 report indicates a full-cost value of 2.4 g/t gold equivalent, however, less than 1% of the designed stopes tonnage lies between 2.2 g/t gold equivalent and 2.5 g/t gold equivalent. The Mineral Reserve is not sensitive to moderate changes in COG.

At $1,275 per ounce gold price, only 25% of current underground Mineral Resources are converted to Minerals Reserves. This low ratio highlights current economic drivers and the dependence of the underground mine operation on the concurrent delivery rate of open pit material to the mill. In a higher gold price environment and subject to realized costs, the proportion of underground Mineral Resources converted to Mineral Reserves could substantially increase.

Access to the underground is via five portals. Mining will be a mechanized operation utilizing uphole stoping for areas less than 15 m wide and downhole stopes (ODM Main Zone only) with cemented or uncemented rockfill for areas wider than 15 m.

The overall underground rock mass quality in terms of rock quality designation (RQD) is classified as "Fair" to predominantly "Good", with RQD typically ranging from 90% to 100%.

There is an opportunity to start mining the Intrepid Zone at an earlier date. Stoping is planned to begin in December 2023. Start dates could potentially be earlier, depending on results of a 2020 decline advance in the Intrepid Zone for orebody investigation and further economic assessment.

To achieve the underground mining schedule, some key activities will require close monitoring. 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. However, there are multiple stoping areas that will be operated concurrently, which will provide significant flexibility.

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. In some non-entry areas, large hangingwall exposures will exist, supported by relatively small pillars. Any failures of the hangingwall or pillars in these locations could generate additional dilution and / or loss or ore.

1.20.3 Metallurgy and Process

Rainy River's current focus is to operate at higher throughputs than the original plant design throughput. This produces a coarser grind size, which reduces gold recovery. Rainy River has determined that an increase in throughput at the expense of gold recovery is the most economically viable option.

Bottlenecks in the pre-leach thickener and the tailings pumps have been identified. Corrective solutions have been included in the 2020 Budget to resolve these issues.

New Gold has been able to reduce reagent consumptions compared to the forecast and LOM average values. This has subsequently reduced operating costs.

1.20.4 Tailings management area

To contain the proposed LOM tailings and water storage volumes, the Rainy River TMA perimeter dams will need to be raised to an ultimate elevation of 379.0 m. Preliminary stability modelling showed that some structures and constraints downstream of the Rainy River TMA would be impacted by the proposed buttress extents. New Gold indicated that impacts to BCR1 and the West Creek Diversion Channel would need to be avoided, but that other downstream impacts were acceptable. Acceptable impacts require the purchase of minimal additional lands and an amendment to the permit, granted under the Endangered Species Act, which currently limits the extension of the TMA to the north-west

1.20.5 Mine rock and overburden stockpiles

The design of the slope on the south side of the LGO stockpile will be updated shortly. This is expected to slightly reduce the nominal capacity of the EMRS, but also to slightly reduce the number of wick drains required in the area. Some of the mine rock that is produced in the open pit will be directly placed as buttresses against the open pit overburden slopes. This will reduce the storage capacity required for mine rock in the EMRS and WMRS.

The design of the WMRS slopes was conceptual and based on consideration of only two cross sections. More detailed design could optimize the slope design and potentially increase the storage capacity of the WMRS should this be required in the future.

A geotechnical instrumentation system is being installed in the foundation of the EMRS and WMRS. Piezometer readings supplemented by engineering analysis may indicate that the placement of mine rock can be accelerated in certain areas, particularly in the initial lifts.

1.20.6 Environmental, social, community, and reclamation / closure

New Gold remains committed to environment, social and community resources and relations in and around the Rainy River Mine.

The Mine Environmental Department is adequately staffed and New Gold conducts ambient air quality, surface and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

The environmental budget for 2020 appears to be reasonable to accommodate the number of regulatory reporting and study commitments, and flexible enough to respond to changes in operating conditions.

Phase 2 of the EMS is scheduled for implementation in 2020.

1.21 Recommendations

The main recommendations offered by the QP's are as follows:

1.21.1 Geology and Mineral Resources

1.21.1.1 Geology

- Ensure that the New Gold internal protocol which dictates a 5% insertion rate for CRMs, blanks and umpire samples, is achieved.
- Acquire an additional CRM that covers the COG of the open pit.
- Address issues with low-grade Geostats CRMs.
- Consider adding the HoleID to the QA/QC sample database as a cross check.
- Send any potential new blank material to an analytical lab to ensure the material is below analytical detection with respect to any minerals of economic interest.
- Lower the blank failure limit to 3X detection limit.
- Further investigative work to be completed to assess pulp duplicate performance.
- Assaying for any new diamond drilling to be included in the block model to be done in a certified lab.

1.21.1.2 Mineral Resource

- Reconcile the resource model short term controls and production on a rolling 3-month basis.
- Revisit classification of the Mineral Resource to improve resulting interpretations.
- Consider updating the geological model with new information and geological interpretations as well as any drilling carried out since production commenced.
- Investigate the high grades of domain ODM 114 located to the west of the open pit and assess impact of the new results. Domain ODM 114 to be re-modelled and re-estimated.

1.21.2 Mining and Mineral Reserves

1.21.2.1 Open pit

- Modifying factors in the development of the mine planning resource model to be reviewed periodically to continue validation of the model performance.
- Monitor the growth of low-grade ore stockpiles (LGOSs) during the LOM to ensure correct strategic decisions are made regarding its utilization.
- Consider developing a very low-grade ore stockpile.
- Implement results of reverse circulation grade control results in the development of a short- to mid-term grade control model. Consider implementation of reverse circulation grade control for NAG identification during Phase 3 and Phase 4 stripping should negative NAG reconciliation be encountered.
- Minimize mining pit bottoms during spring and fall when water inflows to the pit are at their peak.
- Develop a root-cause evaluation behind the elevated stockpile rehandle rate.
- Maintain continued focus on improving equipment productivities, utilizations and availabilities.
- Arrange a visit by Technical Services planning engineers to similar operations with higher vertical advance rates to learn best practices.
- Investigate the implementation of a two-ramp approach through Phase 2/3 to reduce consequences of an instability location above or below critical accesses.
- Implement / continue geotechnical practices regarding the identification, measurement and compilation of geotechnical information / structures as they are exposed by open pit mining. Utilize a three-dimensional modelling software package to compile and review the data and use in ongoing and future geotechnical investigations.
- Implement a monitoring system that includes a combination of field inspections, prism, and radar systems.
- Conduct a pit water management review.
- Install at least two new vibrating wire piezometers in 2020 to evaluate near-slope groundwater conditions.
- Conduct an inventory of inactive and active monitoring well locations and carry out rehabilitation where required. Incorporate all active monitoring installations into a single database.
- Carry out regular review of groundwater monitoring data. Evaluate the data against the pit slope stability assumptions.
- Investigate and trial different blasting approaches and designs to achieve slope design criteria and minimize blast-induced damage to the pit walls.
- Develop new blast designs based on recommended slope design criteria.

1.21.2.2 Underground

- Ensure allowance of time for infill drilling and analysis prior to the commencement of underground production stoping.
- Close monitoring of mine development and rapid implementation of remedial actions in the event of development advance shortfalls.
- Conduct cavity monitoring surveys (CMSs) as part of the production records and reconciliation of production to the Mineral Reserve estimates.
- Install ground control monitoring systems for analysis of the HW and pillar stability in the open stope areas.
- Further consider and review the stability of the overall stope HW for the open stopes, including review of rib pillar dimensions as local mining knowledge increases.
- Further calibrate numerical models for the stability of stopes and pillars based on geotechnical modelling and CMS data. Additional modelling to be undertaken to investigate the stope and stability as per mine design and sequence using the calibrated model.
- Introduce a micro seismic monitoring system for underground.
- Continue testing to verify field performance of backfill binder content.
- Should the development schedule allow, consider relocating and reusing pumping, ventilation and electrical equipment such as pumps, fans, and switchgear from one portal / ramp area to the next.
- Investigate all options, including that of process plant sizing, for an economically viable operation handling underground ore only.

1.21.3 Metallurgy and processing

Orway Mineral Consultants Canada Ltd. (OMC) completed an audit and survey of the process plan comminution circuit. OMC used the survey data for creating a JKSimMet model for the purpose of forecasting process plan throughput. The following comminution circuit recommendations are from OMC's audit report and supported by AMC.

- Increase the semi-autogenous grinding (SAG) mill speed from ~56% critical speed (Nc) to a range of 72.5% Nc to 75% Nc.
- Aim to increase the SAG motor power utilization from 69% to a range of 80% - 95%.
- Increase the ball mill steel charge to 30% - 32% to allow better power utilization.
- Conduct a thorough review to identify the root causes of grinding circuit downtime causes.
- Perform routine mill grind-outs for steel ball charge measurement or crash stops for total load measurement in the SAG mill and ball mill. Create a database which includes details of SAG mill crash stops and ball mill grind-out measurements.
- Investigate equipment and sensor types which can provide real-time data for advanced process control purposes to help improve circuit stability and metallurgical performance.
- Use real-time control of the SAG mill load, using a combination of bearing pressure control and vibration or acoustic sensor logic for measuring the SAG mill load instead of the SAG mill bearing pressure.
- Improve interaction between the metallurgy group and the mining group to log daily / weekly SAG feed ore blends and major ore types.
- Obtain MolyCop Tools© software, for forecasting SAG mill media makeup.
- Program the distributed control system with equipment sensor operating hour totalizers to assist with maintenance programs.
- Perform routine monthly checks of plant feed conveyors.

1.21.4 Infrastructure and other

- Monitor metal price fluctuation and trends and adapt the LOM plan as required to maximize value.

1.21.5 Environmental

- Provide training of employees to reduce reliance on consultants for routine work.
- Continue with development and implementation of the EMS.
- Continue consultation and coordination with regulators for permit changes and amendments.
- Plan and start interim reclamation where possible to reduce post mining obligations.
- Continue with revegetation test plots to determine what species and composition works best for stockpiles and disturbed areas.
- Develop a water resources team to provide technical support in order to mitigate risk to operations from water resource uncertainty.

Contents

Tables

Figures

Abbreviations & acronyms

Abbreviations & acronyms	Description
$	United States dollar
%	Percentage
°	Degree
°C	Degrees Celsius
μ	Poisson's ratio
µm	Micrometre
σ_t	Tensile strength
φ	Friction angle
2D	Two-dimensional
3D	Three-dimensional
A	Rock stress factor; Amps
AA	Atomic absorption
AAS	Atomic absorption spectroscopy
Accurassay	Accurassay Laboratories Ltd.
Actlabs	Activation Laboratories Ltd.
AEM	Airborne electromagnetics
AES	Atomic emission spectroscopy
Ag	Silver
ALS	ALS Chemex
AMC	AMC Mining Consultants (Canada) Ltd.
ANFO	Ammonium nitrate fuel oil
AR	Aqua regia
Au	Gold
AuEq	Gold equivalent
Avg	Average
B	Joint orientation factor
BAW	Beach above water
Bayfield	Bayfield Ventures Corp.
BBA	BBA Inc.
BBMWi	Bond ball mill work index
BBW	Beach below water
BC	British Columbia
BCR	Biochemical Reactor
BFA	Bench face angle
BGC	BGC Engineering Inc.
BRE	Brenna Formation
BWi	Bond work index
C	Gravity adjustment factor
c	Cohesion
C$	Canadian dollar
Capex	Capital expenditure
CaO	Calcium oxide
CCIC	Caracle Creek International Consulting Inc.
CDA	Canadian Dam Association

Abbreviations & acronyms	Description
CFM	Cubic feet per minute
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIP	Carbon-in-pulp
cm	Centimetre
CMS	Cavity Monitoring Survey
CN_{TOTAL}	Total cyanide concentration
CN_{WAD}	Weak acid dissociable cyanide
COG	Cut-off grade
Contango	Contango Strategies Ltd.
CRF	Cemented rockfill
CRM	Certified reference material
CSA	Canadian Securities Administrators
CSD	Critical solids density
CV	Coefficient of variation
Cu	Copper
$CuSO_4$	Copper sulphate
CWi	Crusher work index
d	Day
DCS	Distributed control system
DDH	Diamond drillhole
DGPS	Differential global positioning system
DOM	Declared ore mined
doré	Doré bar
DPO	Direct processing ore
DWT	Drop weight tests
E	Young’s modulus; East
EA	Environmental assessment
EGL	Effective grinding length
ELOS	Equivalent linear overbreak / sloughing
EM	Electromagnetics
EMRS	East mine rock stockpile
EMS	Environmental management system
EOC	East outcrop; end of construction
EOM	End-of-mine
ESA	Endangered Species Act; effective stress analysis
ESE	East-south-east
F_{80}	80% passing feed size
FE	Finite element
Fe	Iron
FEL	Front-end loader
FLS	Felsic metasediments
FLSmidth	FLSmidth Minerals Ltd.
FOS	Factor of safety
FOS_{min}	Minimum factor of safety
FTE	Full-time equivalent
FW	Footwall

Abbreviations & acronyms	Description
g	Gram
G Mining	G Mining Services Inc.
G&A	General and Administrative
g/cm^3	Gram per cubic metre
g/L	Gram per litre
g/t	Grams per tonne
Ga	Billion year; gauge (with respect to wire diameter)
GC model	Grade control model
Golder	Golder Associates Ltd.
GPa	Gigapascal
GPS	Global positioning system
GRG	Gravity recoverable gold
GSI	Geological strength index
GU	General usage
H	High
h	Hour
H_2O	Water
ha	Hectare
HDPE	High-density polyethylene
Hg	Mercury
HGO	High-grade ore
hp	Horsepower
HPGR	High-pressure grinding rolls
HR	Hydraulic radius
Hudbay	Hudson’s Bay Exploration and Development Co Ltd
HW	Hangingwall
ICP	Inductively coupled plasma
ID^2	Inverse distance squared
ID^3	Inverse distance cubed
IMV	Intermediate Metavolcanics
INCO	International Nickel Corporation of Canada Ltd.
IP	Induced polarization
IRR	Internal rate of return
ISO	International Organization for Standardization
JK DW	JK drop weight tests
Kk value	Hydraulic conductivity
kg	Kilogram
kg/h	Kilogram per hour
kg/t	Kilogram per tonne
km	Kilometre
koz	Thousand ounces
kPa	Kilopascal
kt	Thousand tonnes
kV	Kilovolt
kVA	Kilovolt-ampere
kW	Kilowatt

Abbreviations & acronyms	Description
kWh	Kilowatt-hour
kWh/t	Kilowatt-hour per tonne
L	Litre; Level
L/s	Litre per second
lab	Laboratory
LBMA	London Bullion Market Association
LDL	Lower detection limit
LE	Limit equilibrium
LGO	Low-grade ore
LGOS	Low-grade ore stockpile
LHD	Load-haul-dump
LHOS	Longhole open stoping
LiDAR	Light detection and ranging
LLHOS	Longitudinal longhole open stoping
LOM	Life-of-mine
LRIA	Lakes and Rivers Improvement Act
M	Million
m	Metre
m/h	Metre per hour
m/d	Metre per day
m^2	Metre squared
m^3	Cubic metre
m^3/h	Cubic metre per hour
m^3/min	Cubic metre per minute
m^3/t	Cubic metre per tonne
MAG	Magnetic
masl	Metre above sea level
Max	Maximum
MECP	Ministry of Environment, Conservation and Parks
MENDM	Ministry of Energy, Northern Development and Mines
Metso	Metso Minerals Canada Ltd.
Mg	Magnesium
mg/L	Milligram per litre
MGO	Medium-grade ore
$\boldsymbol{m_i}$	Material constant for the intact rock in Hoek Brown criterion
Min	Minimum
Mingold Resources	Mingold Resources Inc.
Minnow	Minnow Environmental Inc.
MLAS	Mining Lands Administration System
MLC	Mine load centre
mm	Millimetre
Mm^3	Million cubic metres
MMI	Mobile metal ion
MMV	Mafic metavolcanics
MNDM	Ministry of Northern Development and Mines
MNRF	Ministry of Natural Resources and Forestry

Abbreviations & acronyms	**Description**
ModBWi	Modified bond work index
MPa	Megapascal
MR	Mineral rights
MSO	Mineable Shape Optimizer
Mt	Million tonnes
MVA	Mega volt amperes
MW	Megawatt
N	North
N'	Modified stability number
NaCN	Sodium cyanide
NAD	North American Datum
NAG	Non-acid generating
NaOH	Sodium hydroxide
Nc	Critical speed
NE	North-east
New Gold	New Gold Inc.
NI 43-101	National Instrument 43-101
NN	Nearest neighbour
NNE	North-north-east
NPI	Net profit interest
NPV	Net present value
NRMS	North Rock Mining Solutions Inc.
NSR	Net smelter return
Nuinsco	Nuinsco Resources Ltd.
NW	North-west
OA	Open area
OGS	Ontario Geological Survey
OK	Ordinary kriging
OKC	M.A. Okane Consultants Inc.
OMC	Orway Mineral Consultants Canada Ltd
ON	Ontario
OP	Open pit
OREAS	Ore Research and Exploration
oz	Troy ounce
OZ	Ore zone
P&P	Proven and probable
P_{80}	80% passing product size
PAG	Potentially acid generating
Pb	Lead
PEA	Preliminary Economic Assessment
pH	pH is a measure of hydrogen ion concentration; a measure of the acidity or alkalinity of a solution
PIN	Property identification number
pop.	Population
ppb	Parts per billion
PPM	Pore pressure model

Abbreviations & acronyms	Description
ppm	Parts per million
Property	Rainy River Property
PRV	Pressure reducing valve
psi	Pound per square inch
PWP	Porewater pressure
Q	Tunneling quality index
Q’	Modified Q with stress reduction factor = 1
QA/QC	Quality assurance / quality control
QP	Qualified Person as defined by NI 43-101
Quadra	Quadra Chemicals Ltd.
RC	Reverse circulation
Report	Technical Report
RF	Rockfill
RL	Reduced level or relative level
RLOM	Remaining life-of-mine
RMR	Rock mass rating
Royal	RGLD Gold AG
Royal Gold	Royal Gold Inc.
RPD	Relative paired difference
rpm	Revolutions per minute
RQD	Rock quality designation
RRGB	Rainy River Greenstone Belt
RRR	Rainy River Resources Ltd.
S	Sulfur; South
SAG	Semi-autogenous grinding
SC	Support class
SEDAR	System for Electronic Document Analysis and Retrieval
SGS	SGS Canada Minerals Services Lakefield Laboratory in Lakefield, Ontario
SK	Saskatchewan
SNF	SNF Canada Ltd.
SO_2	Sulfur dioxide
SR	Surface rights
SRF	Stress reduction factor
SRK	SRK Consulting (Canada) Inc.
Stdv	Standard deviation
S_u	Undrained strength
SWIR	Short-wavelength infrared
t	Tonne
T_{80}	80% transfer size of ore as it passes from the SAG mill to the ball mill
t/m^3	Tonne per cubic metre
t_a	Value describing particle size distribution of the product in the JK drop weight test
Te	Tellurium
TK	Traditional knowledge
TLU	Traditional land use
TMA	Tailings management area
tonne	Tonne = 1,000 kg

Abbreviations & acronyms	Description
tpd	Tonnes per day
tph	Tonnes per hour
tpoh	Tonnes per operating hour
TSL	TSL Laboratories Inc.
UAV	Unmanned aerial vehicle
UCS (σ_c)	Unconfined; uniaxial compressive strength
UG	Underground
URF	Uncemented rockfill
US$	United States dollar
USA	Undrained strength analysis
UTEM	University of Toronto electromagnetic system
UTM	Universal Transverse Mercator
V	Volt
v/v	Volume of solute / volume of solution (L/L)
VF	Vortex finders
VFD	Variable frequency drive
VLGO	Very low-grade ore
VMS	Volcanogenic massive sulphide
VTEM	Versatile time domain electromagnetic
VWP	Vibrating wire piezometer
W	Wide; West
w/v	Weight in grams of solute / milliliters of solute (g/ml)
w/w	Weight of solute / weight of solution (gram/gram)
WML	Whitemouth Lake Formation
WMP	Water management pond
WMRS	West mine rock stockpile
WNW	West-north-west
WST	Whiteshell Till Formation
WTP	Water treatment plant
Wood	Wood PLC
WYL	Wylie Formation
Zn	Zinc

Distribution list

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2 Introduction

2.1 General and terms of reference

This Technical Report (Report) on the Rainy River Property (Property) located in north-western Ontario (ON) in Canada has been prepared by AMC Mining Consultants (Canada) Ltd. (AMC) headquartered in Vancouver, Canada on behalf of New Gold Inc. (New Gold) headquartered in Toronto, Canada. It has been prepared to a standard which is in accordance with the requirements of National Instrument 43-101, Standards of Disclosure for Mineral Projects (NI 43-101), of the Canadian Securities Administrators (CSA) for lodgment on CSA's "System for Electronic Document Analysis and Retrieval" (SEDAR).

The Property is located some 50 kilometres (km) north-west of Fort Frances in the southern half of Richardson Township in north-western ON. This Report is an update to the report filed in July 2018 titled "New Gold Inc. Technical Report on the Rainy River Mine, Ontario, Canada".

2.2 The Issuer

New Gold is an international mid-tier gold mining company with Canadian operations in ON and British Columbia (BC) in Canada and a mine under reclamation in Mexico. New Gold owns 100% of the two Canadian operations, Rainy River and New Afton and the Blackwater project in BC, which is in the development phase. The Cerro San Pedro Mine in Mexico is in reclamation and is also 100% owned by New Gold.

New Gold is listed on the TSX as "NGD" and also on the NYSE as "NGD".

2.3 Report authors

The names and details of persons who prepared, or who have assisted the Qualified Persons (QPs) in the preparation of this Report, are listed in Table 2.1.

Table 2.1 Persons who prepared or assisted in preparation of this Technical Report

Qualified Persons responsible for the preparation and signing of this Technical Report*						
Qualified Person	**Position**	**Employer**	**Independent of New Gold**	**Date of site visit**	**Professional designation**	**Sections of report**
Mr F. McCann	General Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	Various, last visit 13-15 Jan 2020	P.Eng. (ON)	Sections 2 - 5, 19, 22, 24 and parts of Sections 1, 15, 16, 18, 21, 25 - 27
Mr H. Smith	Senior Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (ON)	Parts of Sections 1, 15, 16, 21 and 25 - 27
Mr M. Molavi	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Eng. (BC)	Parts of Sections 1 and 16
Dr A. Ross	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	P.Geo. (BC)	Section 6 -11, 23 and parts of Sections 1 and 25 - 27
Ms D. Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	11 Apr 2018	P.Geo. (BC)	Sections 12, 14 and parts of Sections 1 and 25 - 27
Mr A. Millar	Principal Metallurgist	AMC Consultants Pty Ltd.	Yes	13–15 Aug 2019	MAusIMM, CP	Sections 13, 17 and parts of Sections 1, 21, and 25 - 27
Mr K. Bocking	Principal	Golder Associates Ltd.	Yes	Various, last visit 11 Feb 2020	P.Eng.(ON)	Part of Sections 1, 16, 18, and 25 - 26
Mr E. Saunders	Senior Consultant, Mining Rock Mechanics	SRK Consulting (Canada) Inc.	Yes	Various, last visit 3-5 Feb 2020	P.Eng. (ON)	Part of Sections 1, 16, and 25 - 27
Mr A. Zerwer	Principal Geotechnical Engineer	BGC Engineering Inc.	Yes	Various, last visit 25-28 Nov 2019	P.Eng. (ON)	Part of Sections 1, 18 and 25 - 27
Ms T. Griffith	Senior Environmental Specialist	New Gold Inc., Rainy River Mine	No	Site employee	P.Geo. (ON)	Section 20 and parts of Sections 1 and 25 - 27

Note: * QP responsibility for 'part' sections are governed by their respective areas of expertise: Mr F. McCann – open pit mining aspects; Mr H. Smith - underground mining aspects; Mr M. Molavi – underground infrastructure aspects; Dr A. Ross – geological aspects; Ms D. Nussipakynova – data verification and Mineral Resource aspects; Mr A. Millar – metallurgical and mineral processing aspects; Mr K Bocking – overburden and waste / stockpile storage aspects; Mr E Saunders - open pit rock slope aspects; Mr A Zerwer - tailings aspects; Ms T Griffith - environmental aspects.

Other Experts who have assisted the Qualified Persons					
Expert	**Position**	**Employer**	**Independent of New Gold**	**Visited site**	**Sections of Report**
Mr M. Shannon	General Manager / Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	Sections 2-5 and review work on all
Mr S. Robinson	Senior Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	Sections 7, 9, and 10
Dr A. Rainbow	Director / Geologist	Independent Consultant	Yes	No visit	Section 11
Mr P. Salmenmaki	Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	11-12 Sep 2019	Sections 15 and 16
Mr S. Chan	Senior Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	No visit	Section 21
Mr A. Croal	Director of Technical Services	New Gold Inc.	No	Various, last visit 13-15 Jan 2020	All Sections
Mr M. Della Libera	Director, Exploration	New Gold Inc.	No	Various, last visit 25-27 Jun 2019	Sections 6 - 12
Ms B. Muir	Manager Financial Planning and Analysis	New Gold Inc.	No	Various, last visit 16-19 Sep 2019	Sections 21 and 22
Mr E. Koiengu	Chief Engineer	New Gold Inc.	No	Site employee	Sections 15 and 16
Mr P. Buchanan	Sr. Planning Engineer	New Gold Inc.	No	Site employee	Sections 15 and 16
Mr B. Gagne	Capital Projects Manager	New Gold Inc.	No	Site employee	Section 18
Mr T. Buckingham	Mill Manager	New Gold Inc.	No	Site employee	Sections 13 and 17

2.4 Sources of information

Key sources of information supplied include the diamond drill-hole database, block models metallurgical test work reports, and other information provided by New Gold. A full reference list is included at the end of the report. A further source of information is the report titled "New Gold Inc. Technical Report on the Rainy River Mine, Ontario, Canada", dated 25 July 2018, (2018 New Gold Technical Report).

Other reference material has been the Feasibility Study of the Rainy River Mine, prepared in 2014 for New Gold by BBA Inc. (BBA) and collaborators, (2014 BBA Technical Report).

Parties who have supplied some information that was used for the development of this report include BGC Engineering Inc. (BGC) Golder Associates Ltd. (Golder) SRK Consulting (Canada) Inc. (SRK), and Orway Mineral Consultants Canada Limited (OMC).

2.5 Other

An inspection of the property was carried out by Ms D. Nussipakynova, Mr F. McCann and Mr A. Millar all of AMC at dates shown in Table 2.1. These inspections included review of representative drill core, data collection facilities, mine site, including open pit area, waste rock storage stockpiles, processing plant, general plant site area and tailings management area (TMA).

In addition, inspections were carried out by Mr K. Bocking, Golder, Mr E. Saunders, SRK and Mr A. Zerwer BGC specifically on soft rock geotechnical issues, hard rock geotechnical issues, and tailings, respectively. Ms T. Griffith of New Gold, QP for Section 20 works at site and hence makes frequent inspections.

Units of measurement used throughout this report are metric, unless otherwise stated.

Currency used throughout this report is US$, unless otherwise stated. Where applicable, conversion factors used are as shown in Table 2.2.

Table 2.2 Exchange rates

Currency code	Currency name	Exchange rate
US$	United States Dollar	US$1.00 = C$1.30

This report has an effective date of 12 March 2020.

3 Reliance on other experts

The QPs have relied, in respect of legal and tenure aspects, upon the source listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Report.

Portion of Report to which disclaimer applies:

The following disclosure is made in respect of this Expert:

- Ontario Ministry of Energy, Northern Development and Mines (MENDM) – Mining Lands Administration System (MLAS).

Report, opinion, or statement relied upon:

- MLAS database, data retrieved on 5 January 2020.

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Section 4.2 Land Tenure.

The QPs have relied, in respect of taxation and royalty aspects, upon the work of the issuer's Expert listed below. To the extent permitted under NI 43-101, the QPs disclaim responsibility for the relevant section of the Report:

The following disclosure is made in respect of this Expert:

- New Gold.

Report, opinion, or statement relied upon:

- Information on taxation and royalty aspects.

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Section 22.3 and 22.6 Royalties and Taxation, respectively.

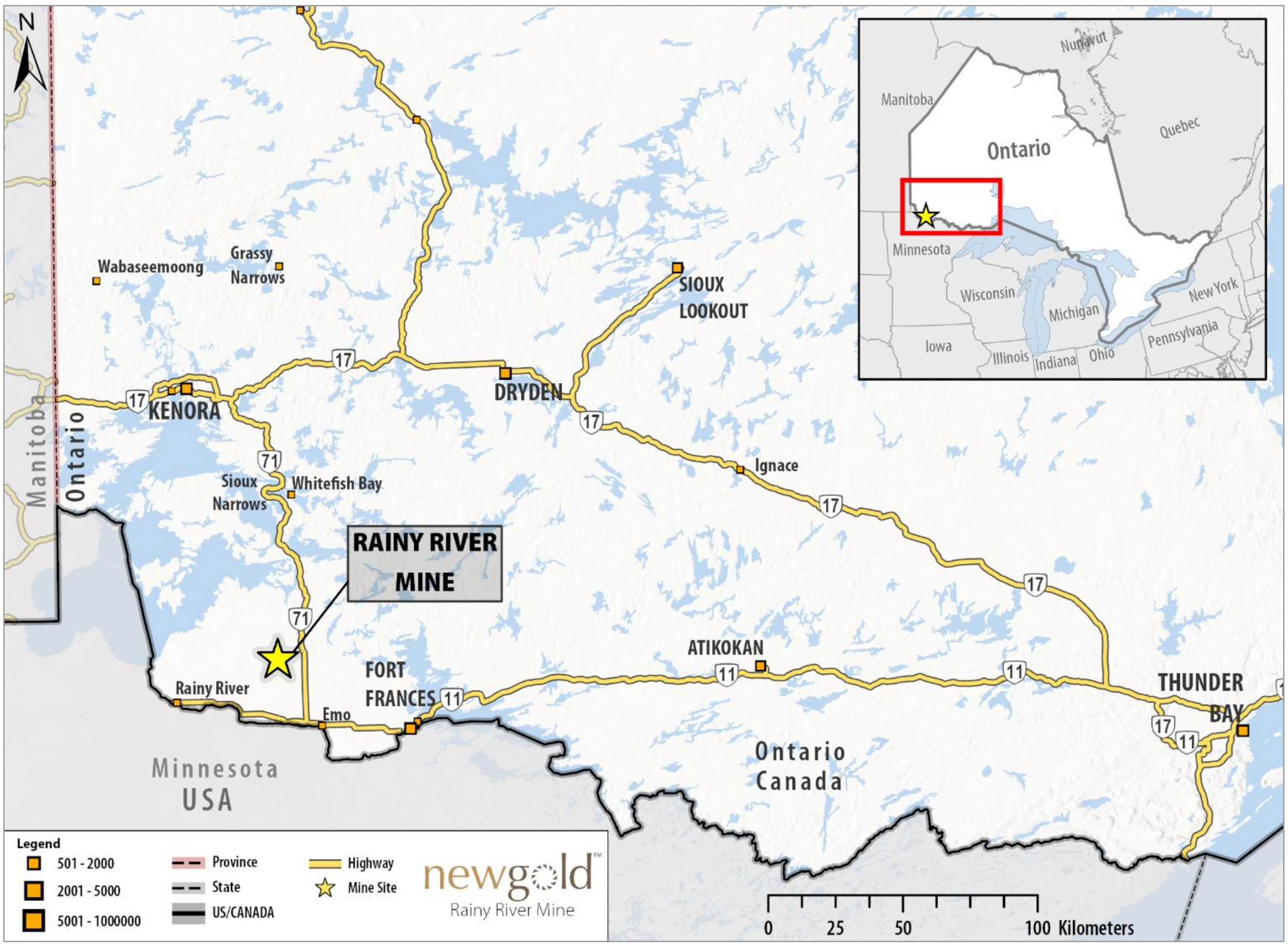
4 Property description and location

4.1 Property location

The Property is located at Latitude 48° 50’ North and Longitude 94° 01’ West in ON, Canada. The Property is located in the Township of Chapple, District of Rainy River, in north-western ON, approximately 50 km north-west of Fort Frances, and 415 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 metres above sea level (masl).

Figure 4.1 Site location



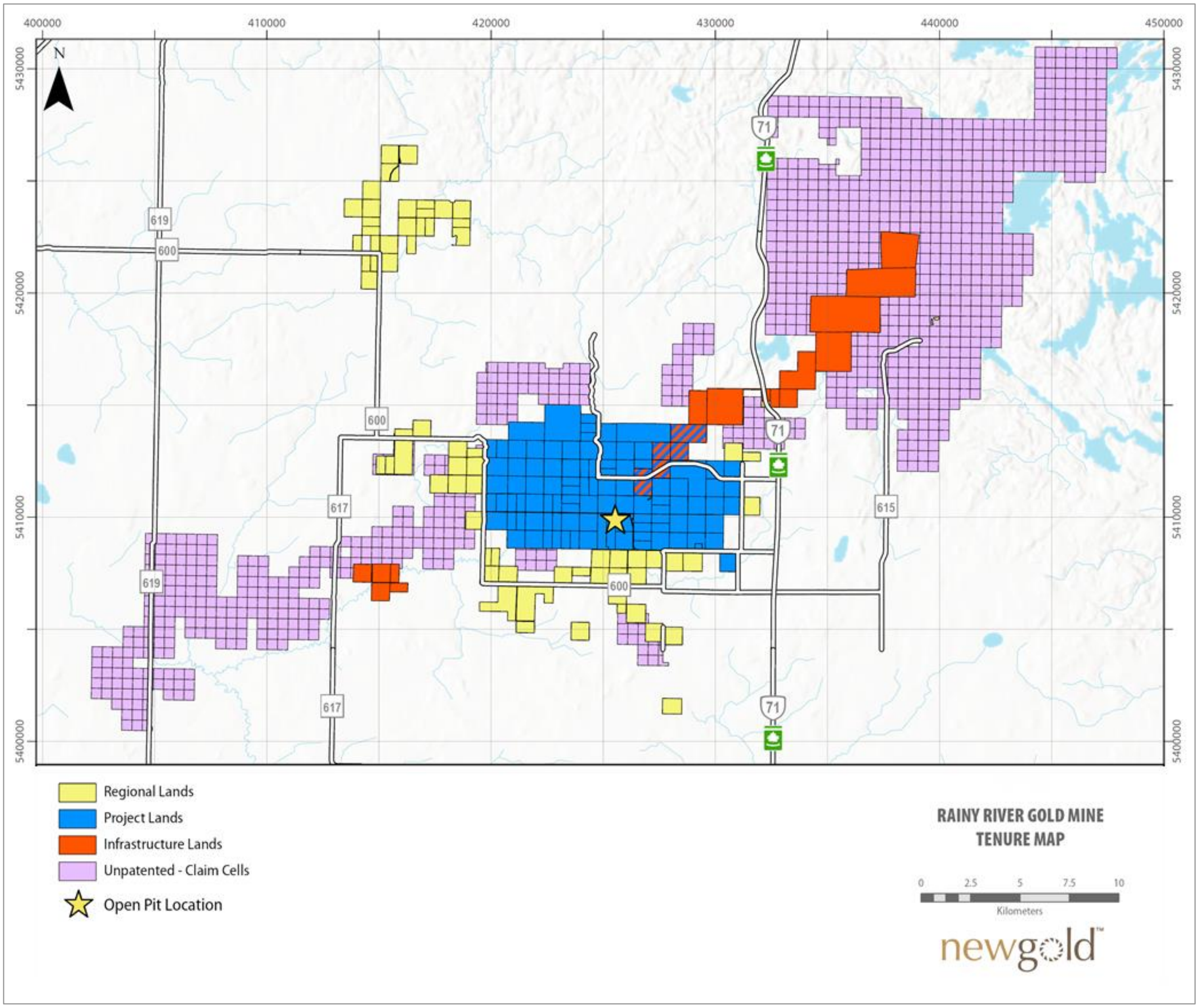
Source: New Gold January 2020.

4.2 Land tenure

4.2.1 General

The Property comprises a, portfolio of 210 patented mining rights, surface rights (SR), and Crown Lease properties. The Project Lands (mine area) comprise 119 separate properties of which New Gold has the rights to the Surface and Minerals, and this area covers approximately 6,077 hectares (ha). Infrastructure Lands cover a further area of 2,800 ha, six of which overlap Project Lands, and Regional Lands cover 3,888 ha. New Gold's total land package covers an aggregate area of approximately 36,762 ha. The Property is located in the Townships of Fleming, Mather, Menary, Patullo, Potts, Richardson, Senn, Sifton, and Tait. A land tenure map is shown in Figure 4.2. A list of the patented claims is presented in Table 4.1 (Project Lands), Table 4.2 (Infrastructure Lands), and Table 4.3 (Regional Lands). A list of unpatented claims cells and their expiry dates is presented in Table 4.4.

Figure 4.2 Tenure map



Source: New Gold January 2020.

4.2.2 Patented Lands

All Patented Lands for surface and mineral rights (MR) are held in the name of New Gold. Patented lands do not have assessment work obligations but require both municipal realty and provincial mining taxes being maintained.

The patented lands consist of patented mining rights, SR, and Crown Lease properties. Crown Lease properties are unpatented mining claims which have been brought to lease for which the Patents have been issued and registered at the Land Registry office. These properties are now identified by Property Identification Numbers (PINs) which are shown in the following tables: Table 4.1, Table 4.2, and Table 4.3. Patented lands consist of Project, Infrastructure and Regional lands as shown in Figure 4.2.

The Project Lands as shown in Figure 4.2 are listed in Table 4.1. Under tenure type, SR stands for surface rights and MR for mineral rights. These are owned unless they are shown as leased. A lease number in the Tenure type column indicates that the PIN is a Crown Lease.

Table 4.1 Summary of Patented Lands – Project Lands only

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
56042-0112	01: NG SR and MR	64.46	56042-0065	01: NG SR and MR	32.45
56042-0037	01: NG SR and MR	32.38	56042-0157/0156	01: NG SR and MR	64.42
56042-0119	01: NG SR and MR	82.69	56042-0051	01: NG SR and MR	31.81
56042-0055	01: NG SR and MR	64.48	56042-0145	01: NG SR and MR	32.08
56042-0123	01: NG SR and MR	63.39	56042-0151/0150	01: NG SR and MR	63.33
56042-0113/0102	01: NG SR and MR	32.28	56042-0033	01: NG SR and MR	64.17
56042-0134	01: NG SR and MR	31.71	56042-0058	01: NG SR and MR	32.26
56042-0215	01: NG SR and MR	0.09	56042-0005	01: NG SR and MR	63.11
56042-0011/0098	01: NG SR and MR	63.00	56042-0006	01: NG SR and MR	1.17
56042-0208	01: NG SR and MR	42.48	56042-0002	01: NG SR and MR	64.31
56035-0066	01: NG SR and MR	65.99	56035-0178	01: NG SR and MR	64.36
56042-0034	01: NG SR and MR	62.63	56042-0061/0100	01: NG SR and MR	62.87
56042-0024	01: NG SR and MR	31.87	56042-0027	01: NG SR and MR	63.92
56042-0109	01: NG SR and MR	63.83	56042-0114	01: NG SR and MR	63.24
56042-0012	01: NG SR and MR	64.92	56042-0043	01: NG SR and MR	32.41
56035-0098	01: NG SR and MR	64.12	56042-0056	01: NG SR and MR	31.89
56042-0069	01: NG SR and MR	32.17	56042-0101/0128	01: NG SR and MR	64.25
56042-0078	01: NG SR and MR	33.47	56042-0153/0152	01: NG SR and MR	32.24
56042-0104/0139	01: NG SR and MR	32.65	56042-0124	01: NG SR and MR	63.27
56042-0107	01: NG SR and MR	32.72	56042-0063	01: NG SR and MR	33.29
56042-0206	01: NG SR 0206	63.12	56042-0212	01: NG SR and MR	81.00
56042-0117	01: NG SR and MR	63.39	56042-0133	01: NG SR and MR	64.39
56042-0220	01: NG SR and MR	0.47	56042-0009	01: NG SR and MR	63.09
56042-0064	01: NG SR and MR	65.91	56042-0050	01: NG SR and MR	64.05
56042-0154/0155	01: NG SR and MR	32.86	56042-0059	01: NG SR and MR	31.27
56035-0176	01: NG SR and MR	64.95	56042-0048	01: NG SR and MR	31.88
56042-0026	01: NG SR and MR	40.49	56042-0028	01: NG SR and MR	63.49
56042-0067	01: NG SR and MR	32.23	56042-0022	01: NG SR and MR	64.53
56042-0042	01: NG SR and MR	32.35	56042-0090	01: NG SR and MR	0.18
56042-0129	01: NG SR and MR	33.74	56042-0091	01: NG SR and MR	0.33
56042-0081	01: NG SR and MR	0.00	56042-0224	01: NG SR and MR	6.61

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
56042-0044	01: NG SR and MR	31.42	56042-0089	01: NG SR and MR	3.27
56042-0116	01: NG SR and MR	59.96	56042-0092	01: NG SR and MR	0.36
56042-0029	01: NG SR and MR	82.90	56042-0086	01: NG SR and MR	0.33
56042-0007	01: NG SR and MR	0.95	56042-0085	01: NG SR and MR	0.27
56042-0036	01: NG SR and MR	64.72	56042-0221	01: NG SR and MR	3.16
56042-0053	01: NG SR and MR	32.38	56041-0240	01: NG SR and MR	2.73
56042-0025	01: NG SR and MR	31.83	56041-0268	01: NG SR and MR	0.05
56042-0052	01: NG SR and MR	32.44	56042-0222	01: NG SR and MR	2.69
56042-0206	01: NG SR and MR	63.96	56042-0217	01: NG SR and MR	2.56
56042-0016	01: NG SR and MR	64.97	56042-0214	01: NG SR and MR	1.28
56035-0194	01: NG SR and MR	64.93	56042-0084	01: NG SR and MR	0.07
56042-0219	01: NG SR and MR	0.02	56042-0093	02: SR only, MR is PIN 56042-0233	10.24
56042-0088	01: NG SR and MR	1.11	56035-0074	12: NG SR (No MR Option)	64.44
56036-0084	01: NG SR and MR	72.59	56042-0082/0141	15: NG SR, MR Leased	32.32
56042-0077	01: NG SR and MR	31.30	56042-0141	15: NG SR, MR Leased	62.81
56042-0213	01: NG SR and MR	0.14	56042-0140	15: NG SR, MR Leased	63.29
56042-0062	01: NG SR and MR	32.42	56042-0140	15: NG SR, MR Leased	31.64
56042-0131	01: NG SR and MR	65.44	56042-0142	15: NG SR, MR Leased	63.60
56042-0038	01: NG SR and MR	31.94	56042-0140	15: NG SR, MR Leased	64.10
56042-0047	01: NG SR and MR	65.49	56042-0141	15: NG SR, MR Leased	31.90
56035-0090	01: NG SR and MR	63.57	56035-0255	21: NG SR and MR Lease #109578	63.95
56042-0060	01: NG SR and MR	64.02	56042-0194	21: NG SR and MR Lease # 109426	129.81
56042-0121	01: NG SR and MR	63.91	56042-0195	21: NG SR and MR Lease #109427	198.77
56042-0076	01: NG SR and MR	40.98	56042-0203	21: NG SR and MR Lease # 109587	454.05
56042-0081	01: NG SR and MR	64.67	56042-0204	21: NG SR and MR Lease # 109587	193.78
56042-0106	01: NG SR and MR	30.34	56042-0223	21: NG SR and MR Lease # 109626	54.88
56042-0008	01: NG SR and MR	64.33	56042-0192	21: NG SR and MR Lease #109424	236.01
56042-0021	01: NG SR and MR	64.91	56042-0202	21: NG SR and MR Lease #109588	97.39
56042-0018	01: NG SR and MR	64.64			
Total hectares: 6,076.88					

The Infrastructure Lands as shown in Figure 4.2 are listed in Table 4.2. Under tenure type, SR stands for surface rights and MR for mineral rights. These are owned unless they are shown as leased. Note there is an overlap of Infrastructure and Project lands. Table 4.2 excludes those Infrastructure lands on Project Lands.

Table 4.2 Summary of Patented Lands – Infrastructure Lands only

PIN	Tenure type	Area (ha)
56035-0256	21: NG SR and MR Lease	260.52
56035-0036/0249/0248	01: NG SR and MR	33.42
56035-0168/0247/0246	01: NG SR and MR	18.39
56035-0015	13: NG Easement	3.23
56035-0195	01: NG SR and MR	64.92
56042-0205	21: NG SR and MR Lease	121.63
56034-0003	21: NG SR and MR Lease	389.10
56032-0285	21: NG SR and MR Lease	252.26
56035-0253	21: NG SR and MR Lease	199.93
56035-0254	21: NG SR and MR Lease	277.21
56034-0002	21: NG SR and MR Lease	498.77
56046-0159	01: NG SR and MR	66.51
56046-0175	01: NG SR and MR	31.71
56046-0135	01: NG SR and MR	65.82
56046-0128/0028	12: NG SR (No MR Option)	32.55
56046-0178	01: NG SR and MR	65.00
Total hectares: 2,380.99 (note including the overlaps the total is 2,800.22 ha)		

The Regional Lands as shown in Figure 4.2 are listed in Table 4.3, and consist of buffer zones, purchased properties, or others such as habitat protection. Under tenure type, SR stands for surface rights and MR for mineral rights. These are owned unless they are shown as leased. A lease number indicates that the PIN is a Crown Lease.

Table 4.3 Summary of Patented Lands – Regional Lands only

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
56044-0077	18: SAR Habitat Compensation Land	31.59	56036-0207	01: NG SR and MR	67.56
56041-0152/0257	01: NG SR and MR	55.89	56041-0159	01: NG SR and MR	64.73
56044-0041	18: SAR Habitat Compensation Land	63.21	56041-0160	01: NG SR and MR	62.70
56045-0039	22: NG MR, SR Option	65.48	56041-0152	01: NG SR and MR	6.45
56044-0067	18: SAR Habitat Compensation Land	61.57	56045-0027	01: NG SR and MR	65.60
56035-0089	02: NG MR (No SR)	9.04	56044-0118	18: SAR Habitat Compensation Land	64.05
56044-0037	18: SAR Habitat Compensation Land	31.75	56035-0009	01: NG SR and MR	64.69
56045-0134	01: NG SR and MR	64.35	56045-0103	18: SAR Habitat Compensation Land	33.29
56041-0138	22: NG MR, SR Option	64.22	56044-0007	18: SAR Habitat Compensation Land	32.62
56045-0086	18: SAR Habitat Compensation Land	31.77	56032-0281	22: NG MR, SR Option	4.18
56036-0035	12: NG SR (No MR Option)	59.55	56041-0215	01: NG SR and MR	10.09
56044-0071	18: SAR Habitat Compensation Land	65.03	56041-0002	01: NG SR and MR	3.31
56045-0023	01: NG SR and MR	0.05	56044-0068	18: SAR Habitat Compensation Land	63.28
56044-0059	18: SAR Habitat Compensation Land	32.12	56045-0003	01: NG SR and MR	65.68
56041-0233	21: NG SR and MR Lease #109555	63.20	56044-0016	18: SAR Habitat Compensation Land	32.70
56041-0164	01: NG SR and MR	59.59	56044-0003	18: SAR Habitat Compensation Land	64.77
56044-0063	18: SAR Habitat Compensation Land	32.72	56044-0105	18: SAR Habitat Compensation Land	56.57
56041-0023	22: NG MR, SR Option	16.53	56041-0030	01: NG SR and MR	65.52
56041-0002	01: NG SR and MR	28.27	56041-0234	21: NG SR and MR Lease # 109564	214.77
56045-0024	01: NG SR and MR	64.22	56041-0220 MR only	02: NG MR (No SR)	53.79
56041-0140	22: NG MR, SR Option	70.29	56045-0138	01: NG SR and MR	65.16

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
56044-0054	18: SAR Habitat Compensation Land	31.19	56044-0008	18: SAR Habitat Compensation Land	64.01
56044-0055	18: SAR Habitat Compensation Land	31.82	56041-0239	21: NG SR and MR Lease # 109608	222.58
56041-0279	01: NG SR and MR	0.23	56041-0162	01: NG SR and MR	64.09
56041-0158	22: NG MR, SR Option	31.16	56045-0099	18: SAR Habitat Compensation Land	129.35
56045-0022	01: NG SR and MR	0.56	56044-0111	18: SAR Habitat Compensation Land	32.61
56041-0117	22: NG MR, SR Option	64.76	56045-0188 MR only	02: NG MR (No SR)	63.64
56044-0020	18: SAR Habitat Compensation Land	63.98	56044-0078	18: SAR Habitat Compensation Land	32.50
56044-0017	18: SAR Habitat Compensation Land	63.05	56044-0030	18: SAR Habitat Compensation Land	31.81
56044-0052	18: SAR Habitat Compensation Land	32.97	56045-0014	18: SAR Habitat Compensation Land	63.72
56044-0047	18: SAR Habitat Compensation Land	64.27	56036-0077	01: NG SR and MR	76.02
56044-0014	18: SAR Habitat Compensation Land	64.44	56041-0126	01: NG SR and MR	63.99
56035-0187	01: NG SR and MR	32.03	56045-0012	01: NG SR and MR	30.47
56041-0278	01: NG SR and MR	0.59	56041-0163 MR only	22: NG MR (No SR)	68.35
56041-0281	01: NG SR and MR	0.28	56044-0006	18: SAR Habitat Compensation Land	65.69
56036-0118 SR only	12: NG SR (No MR Option)	78.42	56044-0103	18: SAR Habitat Compensation Land	62.13
56041-0235	21: NG SR and MR Lease # 109589	29.04	56045-0052	18: SAR Habitat Compensation Land	31.95
56035-0042	01: NG SR and MR	64.80			
Total hectares: 3,888.47					

4.2.3 Unpatented claims

These claims, which are a mix of staked and paper staked claims, are valid for either a one or two year period, and these are shown in Figure 4.2 in purple. There is a total of 1,156 such claims and these are all owned 100% by New Gold. They cover a total area of 24,416 ha. The individual claims are termed Single Cell Mining Claims or Boundary Cell Mining Claims and are all recorded as active. These have been retrieved from the MLAS of the MENDM. All unpatented land claims are in good standing and assessment work credits are sufficient to maintain that standing for several years. These are listed in Table 4.4.

Table 4.4 Summary of unpatented land claims

Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary
222989	10 Apr 2018	16 Jul 2020	218488	10 Apr 2018	8 May 2021	213491	10 Apr 2018	15 Oct 2021
100496	10 Apr 2018	27 Oct 2020	225840	10 Apr 2018	8 May 2021	224179	10 Apr 2018	15 Oct 2021
115791	10 Apr 2018	27 Oct 2020	225841	10 Apr 2018	8 May 2021	224909	10 Apr 2018	15 Oct 2021
115792	10 Apr 2018	27 Oct 2020	226439	10 Apr 2018	8 May 2021	232297	10 Apr 2018	15 Oct 2021
125635	10 Apr 2018	27 Oct 2020	265628	10 Apr 2018	8 May 2021	230883	10 Apr 2018	15 Oct 2021
171464	10 Apr 2018	27 Oct 2020	273575	10 Apr 2018	8 May 2021	230884	10 Apr 2018	15 Oct 2021
227625	10 Apr 2018	27 Oct 2020	285662	10 Apr 2018	8 May 2021	232195	10 Apr 2018	15 Oct 2021
227626	10 Apr 2018	27 Oct 2020	321704	10 Apr 2018	8 May 2021	278093	10 Apr 2018	15 Oct 2021
227627	10 Apr 2018	27 Oct 2020	322309	10 Apr 2018	8 May 2021	280893	10 Apr 2018	15 Oct 2021
274788	10 Apr 2018	27 Oct 2020	344589	10 Apr 2018	8 May 2021	281017	10 Apr 2018	15 Oct 2021
274789	10 Apr 2018	27 Oct 2020	344590	10 Apr 2018	8 May 2021	281019	10 Apr 2018	15 Oct 2021
323478	10 Apr 2018	27 Oct 2020	121761	10 Apr 2018	15 May 2021	282248	10 Apr 2018	15 Oct 2021
323479	10 Apr 2018	27 Oct 2020	179729	10 Apr 2018	15 May 2021	282249	10 Apr 2018	15 Oct 2021
158847	10 Apr 2018	22 Nov 2020	204138	10 Apr 2018	15 May 2021	289621	10 Apr 2018	15 Oct 2021
262844	10 Apr 2018	22 Nov 2020	270246	10 Apr 2018	15 May 2021	298203	10 Apr 2018	15 Oct 2021
268221	10 Apr 2018	22 Nov 2020	280269	10 Apr 2018	15 May 2021	298224	10 Apr 2018	15 Oct 2021
314676	10 Apr 2018	22 Nov 2020	329434	10 Apr 2018	15 May 2021	314797	10 Apr 2018	15 Oct 2021

Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary
314677	10 Apr 2018	22 Nov 2020	101958	10 Apr 2018	16 May 2021	314798	10 Apr 2018	15 Oct 2021
326881	10 Apr 2018	22 Nov 2020	128932	10 Apr 2018	16 May 2021	314799	10 Apr 2018	15 Oct 2021
115963	10 Apr 2018	27 Nov 2020	145463	10 Apr 2018	16 May 2021	327520	10 Apr 2018	15 Oct 2021
115964	10 Apr 2018	27 Nov 2020	158210	10 Apr 2018	16 May 2021	341350	10 Apr 2018	26 Oct 2021
128961	10 Apr 2018	27 Nov 2020	164234	10 Apr 2018	16 May 2021	341351	10 Apr 2018	26 Oct 2021
128962	10 Apr 2018	27 Nov 2020	203524	10 Apr 2018	16 May 2021	341224	10 Apr 2018	26 Oct 2021
164259	10 Apr 2018	27 Nov 2020	212121	10 Apr 2018	16 May 2021	539565	10 Apr 2018	26 Oct 2021
178957	10 Apr 2018	27 Nov 2020	230923	10 Apr 2018	16 May 2021	101087	10 Apr 2018	26 Oct 2021
212147	10 Apr 2018	27 Nov 2020	279658	10 Apr 2018	16 May 2021	101520	10 Apr 2018	26 Oct 2021
223675	10 Apr 2018	27 Nov 2020	278142	10 Apr 2018	16 May 2021	101521	10 Apr 2018	26 Oct 2021
278171	10 Apr 2018	27 Nov 2020	312710	10 Apr 2018	16 May 2021	101522	10 Apr 2018	26 Oct 2021
281565	10 Apr 2018	27 Nov 2020	180481	10 Apr 2018	17 May 2021	101678	10 Apr 2018	26 Oct 2021
279679	10 Apr 2018	27 Nov 2020	180430	10 Apr 2018	17 May 2021	101680	10 Apr 2018	26 Oct 2021
296979	10 Apr 2018	27 Nov 2020	197525	10 Apr 2018	17 May 2021	101681	10 Apr 2018	26 Oct 2021
312743	10 Apr 2018	27 Nov 2020	235003	10 Apr 2018	17 May 2021	101682	10 Apr 2018	26 Oct 2021
312744	10 Apr 2018	27 Nov 2020	270962	10 Apr 2018	17 May 2021	101550	10 Apr 2018	26 Oct 2021
326764	10 Apr 2018	27 Nov 2020	283586	10 Apr 2018	17 May 2021	100995	10 Apr 2018	26 Oct 2021
535473	10 Apr 2018	28 Nov 2020	284268	10 Apr 2018	17 May 2021	102588	10 Apr 2018	26 Oct 2021
535472	10 Apr 2018	28 Nov 2020	102920	10 Apr 2018	25 May 2021	116218	10 Apr 2018	26 Oct 2021
286903	10 Apr 2018	2 Dec 2020	117295	10 Apr 2018	25 May 2021	116219	10 Apr 2018	26 Oct 2021
310722	10 Apr 2018	2 Dec 2020	118151	10 Apr 2018	25 May 2021	116852	10 Apr 2018	26 Oct 2021
115763	10 Apr 2018	2 Dec 2020	125184	10 Apr 2018	25 May 2021	116853	10 Apr 2018	26 Oct 2021
125604	10 Apr 2018	2 Dec 2020	127048	10 Apr 2018	25 May 2021	117903	10 Apr 2018	26 Oct 2021
125605	10 Apr 2018	2 Dec 2020	127049	10 Apr 2018	25 May 2021	117907	10 Apr 2018	26 Oct 2021
154885	10 Apr 2018	2 Dec 2020	125782	10 Apr 2018	25 May 2021	116873	10 Apr 2018	26 Oct 2021
171439	10 Apr 2018	2 Dec 2020	125783	10 Apr 2018	25 May 2021	122358	10 Apr 2018	26 Oct 2021
274757	10 Apr 2018	2 Dec 2020	126365	10 Apr 2018	25 May 2021	122359	10 Apr 2018	26 Oct 2021
274758	10 Apr 2018	2 Dec 2020	126525	10 Apr 2018	25 May 2021	121684	10 Apr 2018	26 Oct 2021
340573	10 Apr 2018	19 Dec 2020	126526	10 Apr 2018	25 May 2021	121685	10 Apr 2018	26 Oct 2021
116058	10 Apr 2018	19 Dec 2020	153747	10 Apr 2018	25 May 2021	124451	10 Apr 2018	26 Oct 2021
121027	10 Apr 2018	19 Dec 2020	153666	10 Apr 2018	25 May 2021	151631	10 Apr 2018	26 Oct 2021
166206	10 Apr 2018	19 Dec 2020	170973	10 Apr 2018	25 May 2021	151632	10 Apr 2018	26 Oct 2021
164832	10 Apr 2018	19 Dec 2020	173073	10 Apr 2018	25 May 2021	151686	10 Apr 2018	26 Oct 2021
164854	10 Apr 2018	19 Dec 2020	173074	10 Apr 2018	25 May 2021	160828	10 Apr 2018	26 Oct 2021
177651	10 Apr 2018	19 Dec 2020	173075	10 Apr 2018	25 May 2021	161483	10 Apr 2018	26 Oct 2021
177652	10 Apr 2018	19 Dec 2020	173855	10 Apr 2018	25 May 2021	161484	10 Apr 2018	26 Oct 2021
177653	10 Apr 2018	19 Dec 2020	173856	10 Apr 2018	25 May 2021	161485	10 Apr 2018	26 Oct 2021
179030	10 Apr 2018	19 Dec 2020	183126	10 Apr 2018	25 May 2021	161502	10 Apr 2018	26 Oct 2021
194959	10 Apr 2018	19 Dec 2020	189210	10 Apr 2018	25 May 2021	160946	10 Apr 2018	26 Oct 2021
204136	10 Apr 2018	19 Dec 2020	206907	10 Apr 2018	25 May 2021	160947	10 Apr 2018	26 Oct 2021
268216	10 Apr 2018	19 Dec 2020	208924	10 Apr 2018	25 May 2021	160948	10 Apr 2018	26 Oct 2021
269556	10 Apr 2018	19 Dec 2020	208397	10 Apr 2018	25 May 2021	160949	10 Apr 2018	26 Oct 2021
281645	10 Apr 2018	19 Dec 2020	208264	10 Apr 2018	25 May 2021	160950	10 Apr 2018	26 Oct 2021
298930	10 Apr 2018	19 Dec 2020	219163	10 Apr 2018	25 May 2021	160805	10 Apr 2018	26 Oct 2021
314657	10 Apr 2018	19 Dec 2020	220352	10 Apr 2018	25 May 2021	160806	10 Apr 2018	26 Oct 2021
314674	10 Apr 2018	19 Dec 2020	227795	10 Apr 2018	25 May 2021	160807	10 Apr 2018	26 Oct 2021
538576	8 Jan 2019	8 Jan 2021	267551	10 Apr 2018	25 May 2021	166299	10 Apr 2018	26 Oct 2021
538577	8 Jan 2019	8 Jan 2021	267530	10 Apr 2018	25 May 2021	167638	10 Apr 2018	26 Oct 2021
538578	8 Jan 2019	8 Jan 2021	266844	10 Apr 2018	25 May 2021	168222	10 Apr 2018	26 Oct 2021
538579	8 Jan 2019	8 Jan 2021	266845	10 Apr 2018	25 May 2021	166941	10 Apr 2018	26 Oct 2021
538580	8 Jan 2019	8 Jan 2021	266295	10 Apr 2018	25 May 2021	166942	10 Apr 2018	26 Oct 2021

Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary
538581	8 Jan 2019	8 Jan 2021	274262	10 Apr 2018	25 May 2021	166967	10 Apr 2018	26 Oct 2021
538582	8 Jan 2019	8 Jan 2021	275006	10 Apr 2018	25 May 2021	166968	10 Apr 2018	26 Oct 2021
538583	8 Jan 2019	8 Jan 2021	286409	10 Apr 2018	25 May 2021	169578	10 Apr 2018	26 Oct 2021
538584	8 Jan 2019	8 Jan 2021	286348	10 Apr 2018	25 May 2021	169579	10 Apr 2018	26 Oct 2021
538590	8 Jan 2019	8 Jan 2021	287087	10 Apr 2018	25 May 2021	169580	10 Apr 2018	26 Oct 2021
538591	8 Jan 2019	8 Jan 2021	293725	10 Apr 2018	25 May 2021	180310	10 Apr 2018	26 Oct 2021
538592	8 Jan 2019	8 Jan 2021	293738	10 Apr 2018	25 May 2021	180311	10 Apr 2018	26 Oct 2021
538593	8 Jan 2019	8 Jan 2021	322973	10 Apr 2018	25 May 2021	180312	10 Apr 2018	26 Oct 2021
538594	8 Jan 2019	8 Jan 2021	322974	10 Apr 2018	25 May 2021	180331	10 Apr 2018	26 Oct 2021
538585	8 Jan 2019	8 Jan 2021	323602	10 Apr 2018	25 May 2021	180332	10 Apr 2018	26 Oct 2021
538586	8 Jan 2019	8 Jan 2021	345341	10 Apr 2018	25 May 2021	180333	10 Apr 2018	26 Oct 2021
538587	8 Jan 2019	8 Jan 2021	345358	10 Apr 2018	25 May 2021	179795	10 Apr 2018	26 Oct 2021
538588	8 Jan 2019	8 Jan 2021	345359	10 Apr 2018	25 May 2021	179652	10 Apr 2018	26 Oct 2021
538589	8 Jan 2019	8 Jan 2021	345289	10 Apr 2018	25 May 2021	179653	10 Apr 2018	26 Oct 2021
145402	10 Apr 2018	9 Jan 2021	102777	10 Apr 2018	2 Jun 2021	179674	10 Apr 2018	26 Oct 2021
100489	10 Apr 2018	11 Jan 2021	102900	10 Apr 2018	2 Jun 2021	180367	10 Apr 2018	26 Oct 2021
100490	10 Apr 2018	11 Jan 2021	103211	10 Apr 2018	2 Jun 2021	180368	10 Apr 2018	26 Oct 2021
102758	10 Apr 2018	11 Jan 2021	118006	10 Apr 2018	2 Jun 2021	182368	10 Apr 2018	26 Oct 2021
117133	10 Apr 2018	11 Jan 2021	118153	10 Apr 2018	2 Jun 2021	196213	10 Apr 2018	26 Oct 2021
157580	10 Apr 2018	11 Jan 2021	118154	10 Apr 2018	2 Jun 2021	196234	10 Apr 2018	26 Oct 2021
188483	10 Apr 2018	11 Jan 2021	125128	10 Apr 2018	2 Jun 2021	196253	10 Apr 2018	26 Oct 2021
188484	10 Apr 2018	11 Jan 2021	125129	10 Apr 2018	2 Jun 2021	197630	10 Apr 2018	26 Oct 2021
207698	10 Apr 2018	11 Jan 2021	125747	10 Apr 2018	2 Jun 2021	197549	10 Apr 2018	26 Oct 2021
207699	10 Apr 2018	11 Jan 2021	125748	10 Apr 2018	2 Jun 2021	205583	10 Apr 2018	26 Oct 2021
226386	10 Apr 2018	11 Jan 2021	127081	10 Apr 2018	2 Jun 2021	205627	10 Apr 2018	26 Oct 2021
230301	10 Apr 2018	11 Jan 2021	127082	10 Apr 2018	2 Jun 2021	205628	10 Apr 2018	26 Oct 2021
230302	10 Apr 2018	11 Jan 2021	154331	10 Apr 2018	2 Jun 2021	214998	10 Apr 2018	26 Oct 2021
279029	10 Apr 2018	11 Jan 2021	153044	10 Apr 2018	2 Jun 2021	214999	10 Apr 2018	26 Oct 2021
277502	10 Apr 2018	11 Jan 2021	153045	10 Apr 2018	2 Jun 2021	215015	10 Apr 2018	26 Oct 2021
285689	10 Apr 2018	11 Jan 2021	153046	10 Apr 2018	2 Jun 2021	214257	10 Apr 2018	26 Oct 2021
296857	10 Apr 2018	11 Jan 2021	153047	10 Apr 2018	2 Jun 2021	214258	10 Apr 2018	26 Oct 2021
125113	10 Apr 2018	11 Jan 2021	153071	10 Apr 2018	2 Jun 2021	214259	10 Apr 2018	26 Oct 2021
337117	10 Apr 2018	28 Jan 2021	153668	10 Apr 2018	2 Jun 2021	214904	10 Apr 2018	26 Oct 2021
337118	10 Apr 2018	28 Jan 2021	155023	10 Apr 2018	2 Jun 2021	214932	10 Apr 2018	26 Oct 2021
101019	10 Apr 2018	28 Jan 2021	169682	10 Apr 2018	2 Jun 2021	215657	10 Apr 2018	26 Oct 2021
102013	10 Apr 2018	28 Jan 2021	169683	10 Apr 2018	2 Jun 2021	215658	10 Apr 2018	26 Oct 2021
117464	10 Apr 2018	28 Jan 2021	169686	10 Apr 2018	2 Jun 2021	215659	10 Apr 2018	26 Oct 2021
117465	10 Apr 2018	28 Jan 2021	169687	10 Apr 2018	2 Jun 2021	216309	10 Apr 2018	26 Oct 2021
117466	10 Apr 2018	28 Jan 2021	169688	10 Apr 2018	2 Jun 2021	215632	10 Apr 2018	26 Oct 2021
121758	10 Apr 2018	28 Jan 2021	170311	10 Apr 2018	2 Jun 2021	215633	10 Apr 2018	26 Oct 2021
121759	10 Apr 2018	28 Jan 2021	171658	10 Apr 2018	2 Jun 2021	217764	10 Apr 2018	26 Oct 2021
123755	10 Apr 2018	28 Jan 2021	182478	10 Apr 2018	2 Jun 2021	225726	10 Apr 2018	26 Oct 2021
123756	10 Apr 2018	28 Jan 2021	182482	10 Apr 2018	2 Jun 2021	233655	10 Apr 2018	26 Oct 2021
123757	10 Apr 2018	28 Jan 2021	183125	10 Apr 2018	2 Jun 2021	233656	10 Apr 2018	26 Oct 2021
130236	10 Apr 2018	28 Jan 2021	189140	10 Apr 2018	2 Jun 2021	233659	10 Apr 2018	26 Oct 2021
130237	10 Apr 2018	28 Jan 2021	188504	10 Apr 2018	2 Jun 2021	233660	10 Apr 2018	26 Oct 2021
146947	10 Apr 2018	28 Jan 2021	188505	10 Apr 2018	2 Jun 2021	234244	10 Apr 2018	26 Oct 2021
152272	10 Apr 2018	28 Jan 2021	189905	10 Apr 2018	2 Jun 2021	234246	10 Apr 2018	26 Oct 2021
158782	10 Apr 2018	28 Jan 2021	207723	10 Apr 2018	2 Jun 2021	233681	10 Apr 2018	26 Oct 2021
158783	10 Apr 2018	28 Jan 2021	207724	10 Apr 2018	2 Jun 2021	233682	10 Apr 2018	26 Oct 2021
164297	10 Apr 2018	28 Jan 2021	207725	10 Apr 2018	2 Jun 2021	262891	10 Apr 2018	26 Oct 2021

Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary
167653	10 Apr 2018	28 Jan 2021	207726	10 Apr 2018	2 Jun 2021	262194	10 Apr 2018	26 Oct 2021
168873	10 Apr 2018	28 Jan 2021	207632	10 Apr 2018	2 Jun 2021	262195	10 Apr 2018	26 Oct 2021
172298	10 Apr 2018	28 Jan 2021	207635	10 Apr 2018	2 Jun 2021	262196	10 Apr 2018	26 Oct 2021
180470	10 Apr 2018	28 Jan 2021	208940	10 Apr 2018	2 Jun 2021	262220	10 Apr 2018	26 Oct 2021
180471	10 Apr 2018	28 Jan 2021	208263	10 Apr 2018	2 Jun 2021	262864	10 Apr 2018	26 Oct 2021
180479	10 Apr 2018	28 Jan 2021	208265	10 Apr 2018	2 Jun 2021	262865	10 Apr 2018	26 Oct 2021
190572	10 Apr 2018	28 Jan 2021	208266	10 Apr 2018	2 Jun 2021	262866	10 Apr 2018	26 Oct 2021
205006	10 Apr 2018	28 Jan 2021	218490	10 Apr 2018	2 Jun 2021	270343	10 Apr 2018	26 Oct 2021
205007	10 Apr 2018	28 Jan 2021	218374	10 Apr 2018	2 Jun 2021	269648	10 Apr 2018	26 Oct 2021
205008	10 Apr 2018	28 Jan 2021	218376	10 Apr 2018	2 Jun 2021	270315	10 Apr 2018	26 Oct 2021
204092	10 Apr 2018	28 Jan 2021	218377	10 Apr 2018	2 Jun 2021	270319	10 Apr 2018	26 Oct 2021
209063	10 Apr 2018	28 Jan 2021	218378	10 Apr 2018	2 Jun 2021	271578	10 Apr 2018	26 Oct 2021
212191	10 Apr 2018	28 Jan 2021	220428	10 Apr 2018	2 Jun 2021	270894	10 Apr 2018	26 Oct 2021
212192	10 Apr 2018	28 Jan 2021	225813	10 Apr 2018	2 Jun 2021	271658	10 Apr 2018	26 Oct 2021
212246	10 Apr 2018	28 Jan 2021	225814	10 Apr 2018	2 Jun 2021	271659	10 Apr 2018	26 Oct 2021
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129578	10 Apr 2018	6 May 2021	204064	10 Apr 2018	25 Sep 2021	145358	10 Apr 2018	26 Jan 2022
158851	10 Apr 2018	6 May 2021	230963	10 Apr 2018	25 Sep 2021	157596	10 Apr 2018	26 Jan 2022
158852	10 Apr 2018	6 May 2021	296992	10 Apr 2018	25 Sep 2021	163627	10 Apr 2018	26 Jan 2022
158853	10 Apr 2018	6 May 2021	312755	10 Apr 2018	25 Sep 2021	203419	10 Apr 2018	26 Jan 2022
166290	10 Apr 2018	6 May 2021	312756	10 Apr 2018	25 Sep 2021	203420	10 Apr 2018	26 Jan 2022
177676	10 Apr 2018	6 May 2021	326783	10 Apr 2018	25 Sep 2021	206230	10 Apr 2018	26 Jan 2022
177677	10 Apr 2018	6 May 2021	103071	10 Apr 2018	13 Oct 2021	211499	10 Apr 2018	26 Jan 2022
177678	10 Apr 2018	6 May 2021	103072	10 Apr 2018	13 Oct 2021	217093	10 Apr 2018	26 Jan 2022
212762	10 Apr 2018	6 May 2021	173749	10 Apr 2018	13 Oct 2021	217094	10 Apr 2018	26 Jan 2022
224260	10 Apr 2018	6 May 2021	206991	10 Apr 2018	13 Oct 2021	218105	10 Apr 2018	26 Jan 2022
232992	10 Apr 2018	6 May 2021	101917	10 Apr 2018	15 Oct 2021	223559	10 Apr 2018	26 Jan 2022
231544	10 Apr 2018	6 May 2021	100839	10 Apr 2018	15 Oct 2021	227400	10 Apr 2018	26 Jan 2022
269637	10 Apr 2018	6 May 2021	101425	10 Apr 2018	15 Oct 2021	259488	10 Apr 2018	26 Jan 2022
280270	10 Apr 2018	6 May 2021	116551	10 Apr 2018	15 Oct 2021	270335	10 Apr 2018	26 Jan 2022
282246	10 Apr 2018	6 May 2021	120316	10 Apr 2018	15 Oct 2021	270336	10 Apr 2018	26 Jan 2022
314683	10 Apr 2018	6 May 2021	120317	10 Apr 2018	15 Oct 2021	279552	10 Apr 2018	26 Jan 2022
314682	10 Apr 2018	6 May 2021	159581	10 Apr 2018	15 Oct 2021	277522	10 Apr 2018	26 Jan 2022
326883	10 Apr 2018	6 May 2021	159471	10 Apr 2018	15 Oct 2021	277523	10 Apr 2018	26 Jan 2022
102832	10 Apr 2018	8 May 2021	159487	10 Apr 2018	15 Oct 2021	282940	10 Apr 2018	26 Jan 2022
102833	10 Apr 2018	8 May 2021	164191	10 Apr 2018	15 Oct 2021	286030	10 Apr 2018	26 Jan 2022

Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary	Tenure number	Issue date	Anniversary
118152	10 Apr 2018	8 May 2021	165574	10 Apr 2018	15 Oct 2021	284378	10 Apr 2018	26 Jan 2022
118036	10 Apr 2018	8 May 2021	165575	10 Apr 2018	15 Oct 2021	294051	10 Apr 2018	26 Jan 2022
118037	10 Apr 2018	8 May 2021	165576	10 Apr 2018	15 Oct 2021	296866	10 Apr 2018	26 Jan 2022
118038	10 Apr 2018	8 May 2021	178324	10 Apr 2018	15 Oct 2021	306216	10 Apr 2018	26 Jan 2022
125080	10 Apr 2018	8 May 2021	178349	10 Apr 2018	15 Oct 2021	314100	10 Apr 2018	26 Jan 2022
153667	10 Apr 2018	8 May 2021	194225	10 Apr 2018	15 Oct 2021	314101	10 Apr 2018	26 Jan 2022
170310	10 Apr 2018	8 May 2021	194849	10 Apr 2018	15 Oct 2021	320899	10 Apr 2018	26 Jan 2022
188555	10 Apr 2018	8 May 2021	194851	10 Apr 2018	15 Oct 2021	326142	10 Apr 2018	26 Jan 2022
188556	10 Apr 2018	8 May 2021	213495	10 Apr 2018	15 Oct 2021	343305	10 Apr 2018	26 Jan 2022
207661	10 Apr 2018	8 May 2021	214228	10 Apr 2018	15 Oct 2021	259586	10 Apr 2018	16 May 2023
218486	10 Apr 2018	8 May 2021	214229	10 Apr 2018	15 Oct 2021			
218487	10 Apr 2018	8 May 2021	214127	10 Apr 2018	15 Oct 2021			
Total hectares: 24,416								

4.2.4 Surface rights

The SR are covered by the Patented Lands as listed in Table 4.1 to Table 4.3 and labeled by SR in the Tenure Type column. These constitute sufficient area for current operations, however, some additional small properties are currently contemplated for purchase as required for the future expansion of the tailings dam footprint.

4.3 Royalty and streaming agreements

Royal Gold Inc. (Royal Gold) through its wholly owned subsidiary RGLD Gold AG (Royal) entered into a $175 million (M) 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.

Further details of the streaming agreement are discussed in Section 19.3.2.

A portion of the Rainy River mineral lands are covered by either a 2% Net Smelter Return (NSR) royalty or a 10% net profits interest royalty. In addition, New Gold has agreed to financial participation in the Mine in the form of royalties in favour of certain First Nations.

4.4 Environmental, permits, and other factors

The QP is not aware of any environmental liabilities on the Property and New Gold has obtained all required permits to conduct the proposed work on the Property. The QP 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.

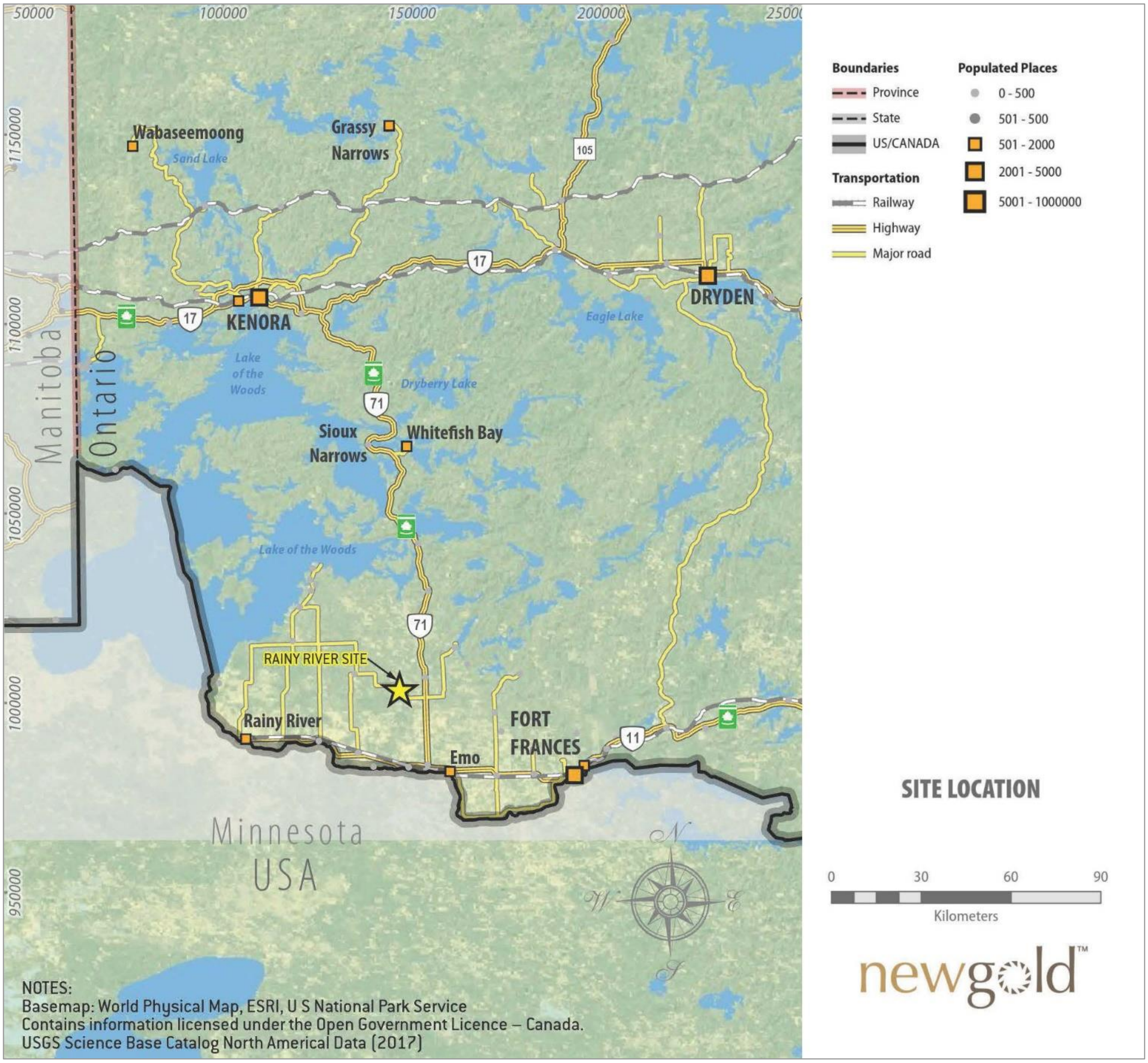
This item is more fully covered in Section 20.

5 Accessibility, climate, local resources, infrastructure, and physiography

5.1 Location and accessibility

The Rainy River Mine which is in the centre of the Property is located approximately 50 km to the north-west of Fort Frances, the nearest large town in north-western ON. The Property is centred in Richardson Township which is part of Chapple Township. Air access by regular scheduled flights is either through Thunder Bay or if coming from the west through Winnipeg. Figure 5.1 is an inset of Figure 4.1 and shows the location and access in more detail.

Figure 5.1 Location and access to the Rainy River Mine site



Source: New Gold 2019.

Access from Thunder Bay through Fort Frances is approximately 415 km along Highway 11 to Emo, and then north on Highway 71, turning west on Korpi Road. Alternative access from Winnipeg is by driving east to Kenora via Hwy 1 / Hwy 17 and then south on Highway 71 and turning west on Korpi Road, a distance of 369 km. These access roads are sealed allowing year-round access. See Figure 4.1 for all these routes.

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.

5.2 Infrastructure and local resources

There are three small towns within immediate driving distance of the Rainy River Mine: Emo (population (pop.) 1,333, 34 km by road), Rainy River (pop. 807, 79 km by road), and Fort Frances (pop. 7,420, 68 km by road). Note population figures are from the 2016 census, and data sourced from www12.statcan.gc.ca.

Hydroelectricity is produced north of Kenora at various locations, as well as west and east of Thunder Bay.

There is a ready supply of water in the area from lakes and rivers. Ground water is also likely to be in plenteous 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 south-east 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.

Infrastructure is more fully addressed in Section 18.

5.3 Climate and physiography

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 centimetres (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.

The Property ranges over an elevation from 340 masl to 400 masl and is divided into two physiographical regions. These regions are separated by a distinct north-west to south-east 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 metres (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 Labradorean ice centre to the north-east.

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 20 m to 40 m 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 north-east and west.

Where covered, the bedrock is immediately overlain by Labradorean 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 north-eastern hardwood region immediately adjacent to the southern margin of the boreal forest.

5.4 Surface rights

New Gold owns the land containing the entire current surface infrastructure associated with the Rainy River Mine. With the exception of some additional small properties which have to be purchased due to the expansion of the tailings dam footprint this is sufficient to allow the future operation of the mine without further land acquisition, see Section 4.

6 History

The following section has been modified from the 2014 BBA Technical Report which in turn references documentation of exploration in north-western ON that is archived in the Ministry of Northern Development and Mines (MNDM) offices at Kenora.

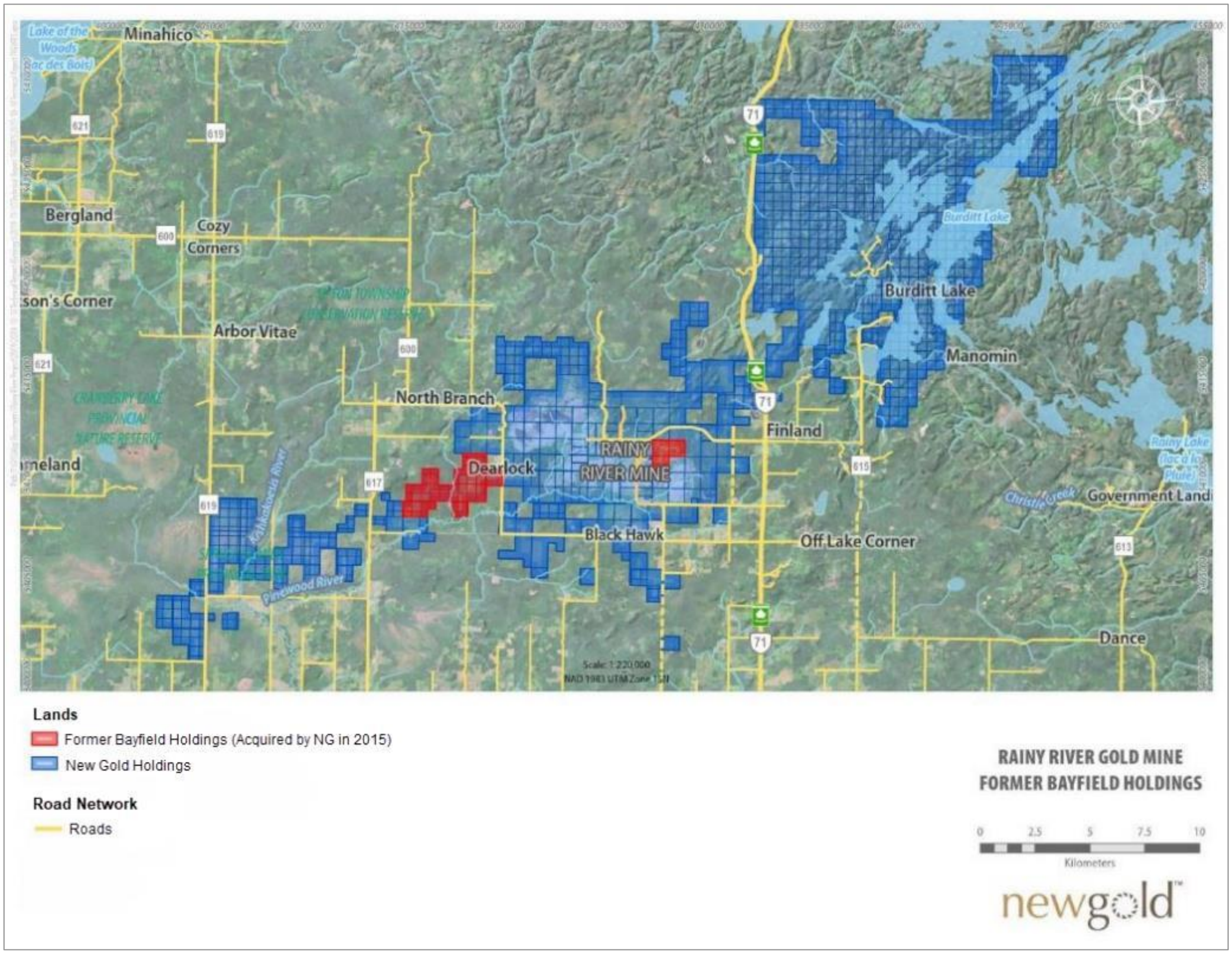
6.1 Prior owners

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

Nuinsco Resources Ltd. (Nuinsco) held the claims to the Rainy River area from 1990 to 2004. Nuinsco was acquired by Rainy River Resources Ltd. (RRR) who continued exploration from 2005 to 2013 when New Gold completed a takeover of RRR on 15 October 2013.

In addition, 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 include the Burns Block claim located immediately east of the current open pit and in which the Intrepid deposit is located. To facilitate the understanding in other sections, Figure 6.1 shows the land acquired during the Bayfield acquisition. Bayfield explored this ground from 2010 to 2014.

Figure 6.1 Claim map showing location of acquired Bayfield ground



Source: New Gold 2019.

6.2 Exploration 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 (RRGB). 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 drillholes in the vicinity of the current Rainy River Mine. 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 in 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 Township and some neighbouring townships. Mingold Resources' use of various sampling methodologies on the till, including reverse circulation (RC) drilling, gave inconclusive results.

The Property was acquired by Nuinsco in 1990 and it began exploring in 1993. 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
	IP and magnetometer survey	Val d'Or Géophysique
	Landsat linear study	DOZ Consulting Group
	Reconnaissance mapping and sampling	Nuinsco Resources
1994	Rota-sonic drilling	Midwest Drilling
	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
	Diamond drilling	Ultra Mobile Diamond Drilling
	Grid mapping and sampling	Nuinsco Resources
	Soil Sampling / Enzyme Leach	Nuinsco Resources
1995	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
	Diamond drilling	Ultra Mobile Diamond Drilling
	IP survey	JVX Geophysics
	Trenching and stripping, mapping	Nuinsco Resources
	Soil Sampling / Enzyme Leach	Nuinsco Resources
1996	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
	Diamond drilling	Ultra Mobile Diamond Drilling
	Diamond drilling	Bradley Brothers Diamond Drilling
	UTEM survey	Lamontagne Geophysics
	Surface pulse EM survey	Crone Geophysics and JVX Geophysics
	Borehole pulse EM survey	Crone Geophysics / JVX Geophysics
	IP and magnetic survey	JVX Geophysics
	Outcrop stripping	Nuinsco Resources
1997	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
	Diamond drilling	Ultra Mobile Diamond Drilling
	Diamond drilling	Bradley Brothers Diamond Drilling
	Airborne EM and Magnetic survey	Geoterrex-Dighem
	Surface and Borehole pulse EM survey	Crone Geophysics
	IP survey	Quantec IP
	Local detailed mapping and outcrop stripping	Nuinsco Resources
1998	Surface pulse EM survey	Crone Geophysics
	Diamond drilling	Ultra Mobile Diamond Drilling.
	Reverse circulation drilling	Bradley Bros. - Overburden Drilling
	Line cutting / magnetometer survey	Mtec Geophysics Inc.
	Diamond drilling	Ultra Mobile Diamond Drilling
1999	Diamond drilling	Ultra Mobile Diamond Drilling
	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	Unknown

Note: IP = induced polarization; EM = electromagnetics, UTEM = University of Toronto electromagnetic system.
Source: Modified after Mackie et al. 2003.

Upon acquisition of the Property from Nuinsco in June 2005, RRR relogged key sections of the drill core and input available data into an Excel 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. This survey was conducted by RRR. The test grid over the Intrepid Zone showed a weak to moderate gold anomaly which did not match with the surface projection of the Intrepid Zone mineralization.

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 21 DDH, structure & geology of Caldera Model	L.D. Ayres
	Summary of structural observations	G. Zhang
	Petrography and mineralogy	R.P. Taylor
	Structure and geology of Richardson Township	H. Paulsen
2006	Report of re-logging of Nuinsco DDH core	L.D. Ayres
	VTEM airborne geophysical survey	Geotech Limited
	U-Pb Zircon age dating	Geospec Consultants Limited
	Petrographic and mineralogical report	E. Schandl
	Structure and geology review	K. H. Paulsen
	3D borehole pulse EM survey	Crone Geophysics and Exploration
2007	IP Survey of 9 holes, 3D conductivity inversion	JVX Limited
	Models line cutting	Archer Exploration Inc.
	Ground gravity and EM survey	Abitibi Geophysics
2008	Titan 24 survey	Quantec Geoscience
	Airborne magnetic gradiometer survey	Fugro Airborne Surveys, Corp.
	Regional geophysical interpretation	J. Siddorn – SRK
	Socio-economic scoping study draft report	Klohn, Crippen and Berger Ltd.
	Preliminary pit slope design and waste management assessment	Klohn, Crippen and Berger Ltd.
2009	Age dating of lithologies	University of Toronto Geochronology Lab
	Surficial drainage project	K. Smart Associates Limited
	Socio-environmental baseline assessment, acid leach test	Klohn Crippen Berger Ltd.
	LiDAR survey	LiDAR Services International
2010	Preliminary metallurgical testing and metallurgical testwork	SGS Canada Inc.
	Environmental baseline studies, DD-4 geotechnical DDH (1,405 m)	Klohn Crippen Berger Ltd
	Review of pit slope design, structural study	SRK
	Memorandum of understanding with Fort Francis Chiefs Secretariat	Rainy River Resources Ltd.
	M.Sc Thesis on Richardson Deposit	J. Wartman - University of Minnesota
	Pre-Feasibility open pit slope design	Klohn Crippen Berger
	New core logging facility	C. Hercun, True-line Construction
	Line cutting geophysical Grid 33 km	Archer Exploration Inc.
	Titan survey 33 km	Quantec Geoscience
	Application for Advanced Exploration Permit	G. Macdonald, K. Stanfield

Year	Activity	Company
2011	88 km high-sensitivity potassium magnetometer ground survey	RDF Consulting
	Environmental baseline gap analysis	AMEC Earth and Environmental
	First quarter QA/QC report	Analytical Solutions Ltd.
	Fugro AEM survey	Fugro Airborne Surveys Corp.
	Report on ground gravity surveys, report on borehole surveys	Eastern Geophysics, Gerard Lambert
2012	Mobile metal ion soil surveys - various	Rainy River Resources Ltd.
	Report on 34 zone & Pinewood Ni, Cu & PGE mineralization	Revelation Geoscience Ltd.
	Intrepid specific gravity data	ALS Chemex Laboratory
2013	Soil gas hydrocarbon orientation survey	Rainy River Resources Ltd.
	Mobile metal ion soil survey – Intrepid	Rainy River Resources Ltd.

Note. VTEM = versatile time domain electromagnetic; LiDAR = light detection and ranging; AEM = airborne electromagnetics, DDH = diamond drillhole.

6.3 Historical Mineral Resource estimates

Numerous Mineral Resource estimates were prepared for the Rainy River Mine from 2003 to 2015. Authors of these reports include Mackie et al. in 2003, Caracle Creek International Consulting Inc. (CCIC) in 2008, SRK in 2009, 2010, 2011, and 2012, BBA and collaborators in 2014 (Feasibility Study). These Mineral Resource estimates are documented in previous technical reports prepared for the Property which are available on SEDAR.

The current Mineral Resource estimate contained in Section 14 of this Report supersedes all previous estimates.

6.4 Past production

There is no historical production from the Property.

7 Geological setting and mineralization

7.1 Regional geology

The Property is located within the 2.7 billion years (Ga) old Neoarchean Rainy RRGB. The RRGB forms part of the Wabigoon sub-province within the larger Superior Province – the core of the Canadian Shield of North America.

The Wabigoon sub-province is located in the western portion of the Superior Province as shown in 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 Mesoarchean 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, respectfully (Percival et al. 2006). This basal sequence is overlain by approximately 2.71 Ga to 2.70 Ga volcano-sedimentary sequences and by locally deposited, unconformable, immature clastic sedimentary sequences.

Volcanic rocks have been intruded by a wide variety of plutonic rocks including syn-volcanic 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 the town of Fort Frances, the Wabigoon sub-province is bounded to the south by the late Archean, dextral Seine River–Rainy Lake and Quetico faults. The Quetico Fault splays off the sub-province boundary and strikes west through the western Wabigoon domain just south of the Rainy River Mine.

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 Mine, and the Sturgeon Lake volcanogenic massive sulphide (VMS) deposits 250 km to the north-east of the Rainy River Mine. These deposits are shown in relationship to the Rainy River deposit in Figure 7.2.

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 derived from the Archean basement of the Canadian Shield to the north-east. 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.

Figure 7.1 Superior Province geological map

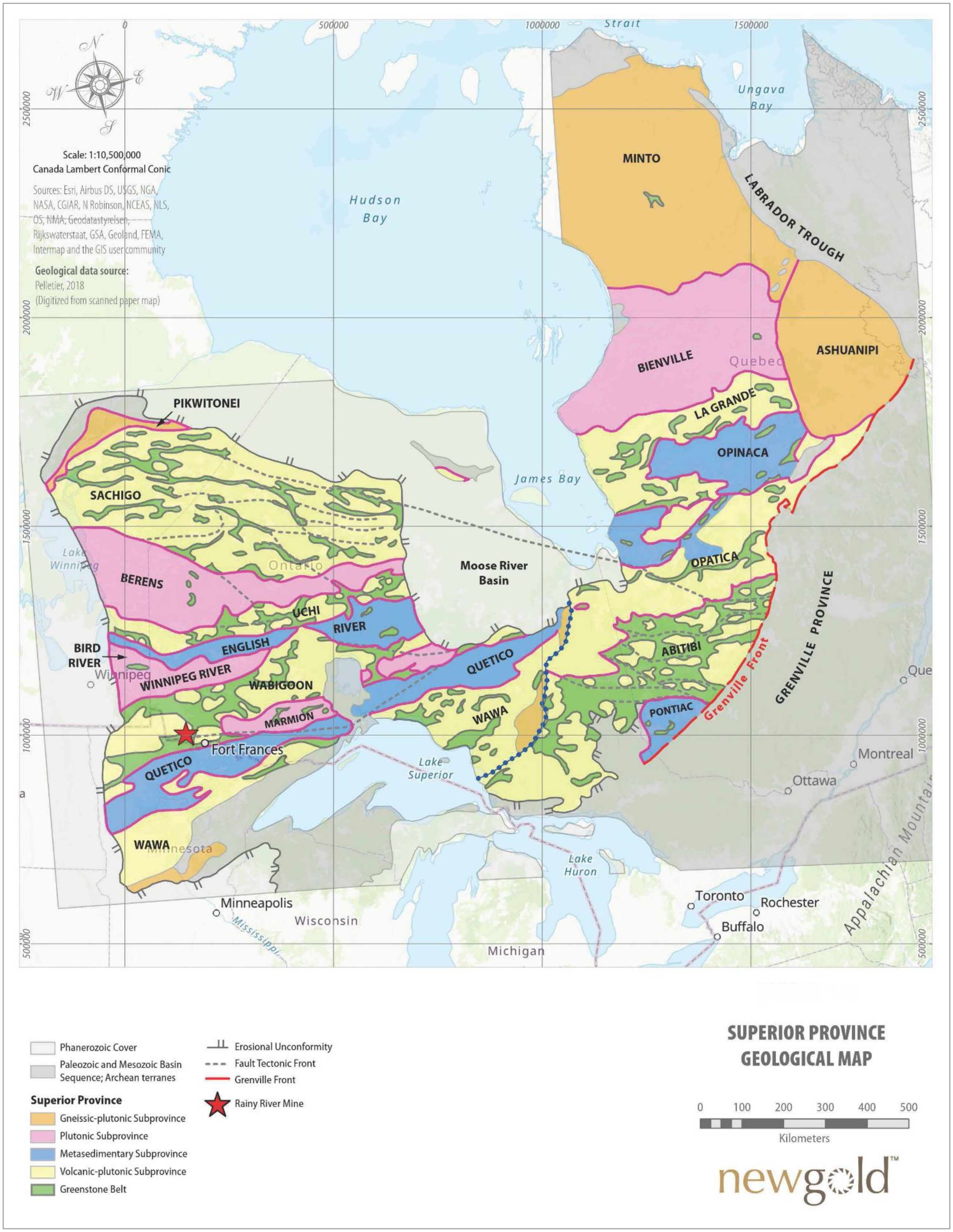
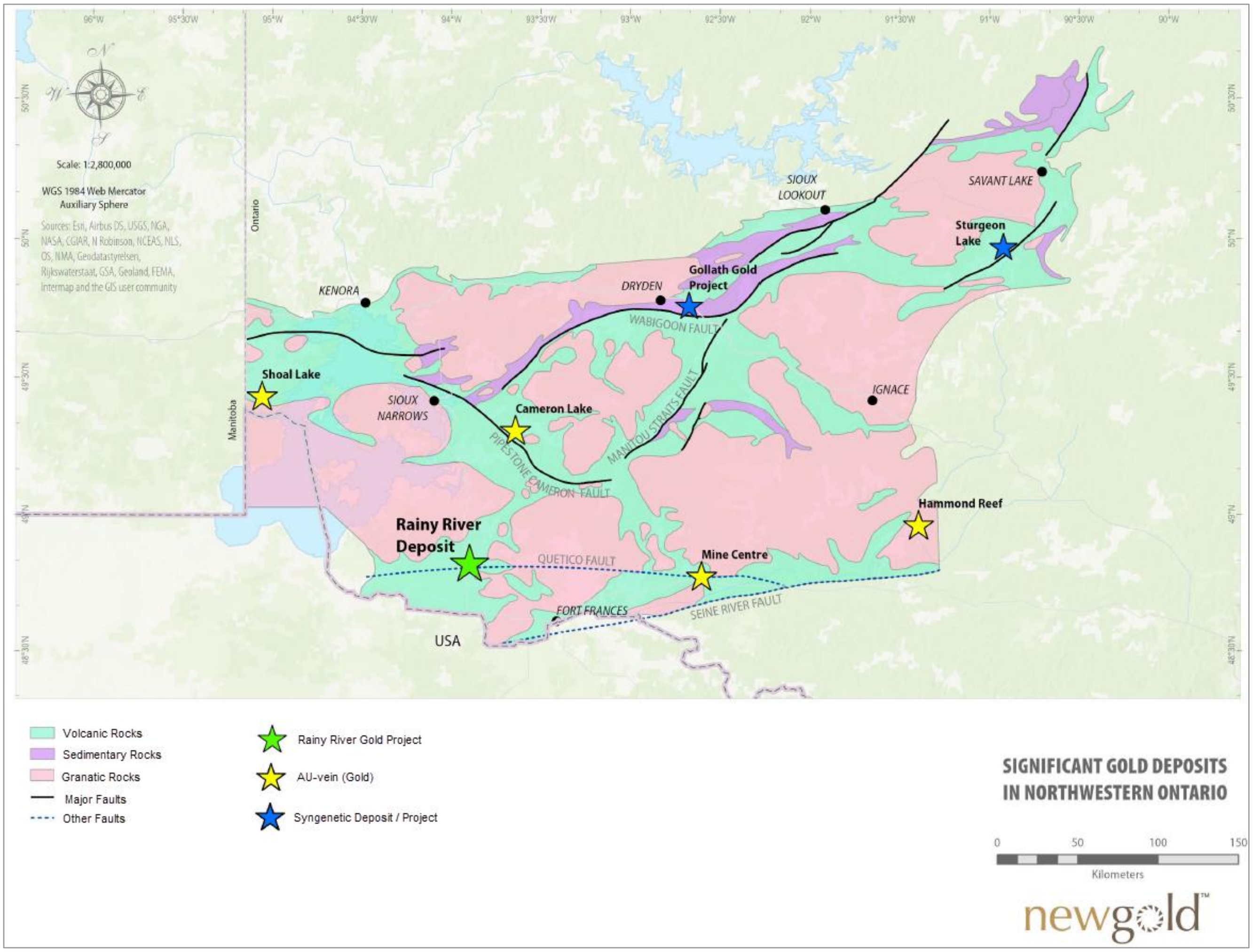


Figure 7.2 Significant gold deposits in north-western ON



Source: Pelletier 2016 (modified after Blackburn et. al. 1991).

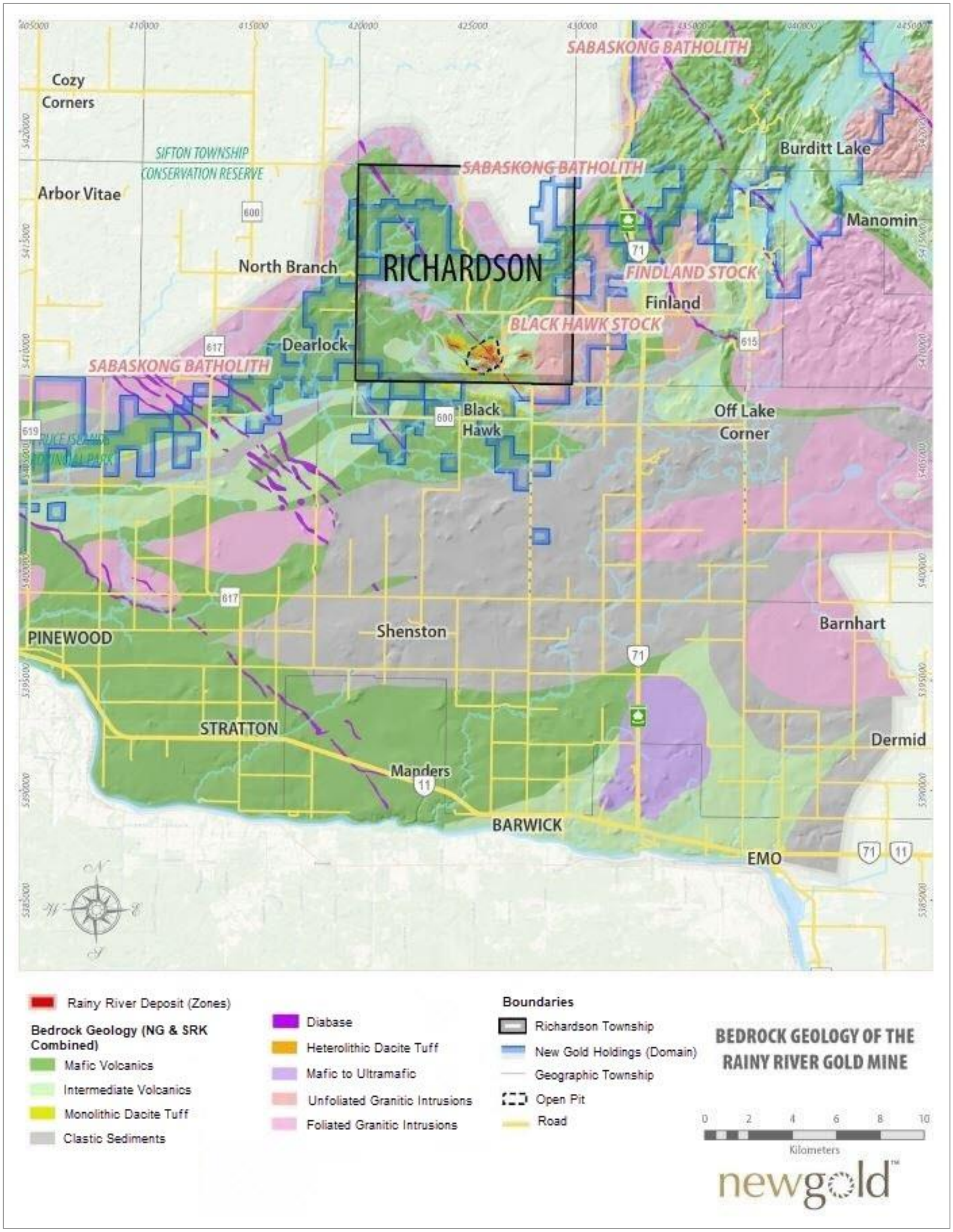
7.2 Property geology

The Property encompasses an approximately 30 km long, north-east trending portion of the RRGB. In this area, the RRGB is bounded to the north-west by the Sabaskong Batholith, to the east by the Rainy Lake Batholithic Complex and to the south by the Quetico fault. In the north-east portion of the Property the RRGB is contiguous with the Kakagi-Rowan Lakes Greenstone Belt.

The bedrock geology has been inferred from regional field mapping of limited rock exposures, extensive RC and diamond core drilling, OGS rota-sonic drilling, and airborne geophysics. Portions of the Property have been covered by the Labradorean and Keewatin ice sheets.

A bedrock geological interpretation produced by RRR for the area surrounding Rainy River is shown in Figure 7.3.

Figure 7.3 Bedrock geology of the Rainy River Mine



Source: New Gold 2019.

7.3 Local geology

The Rainy River deposit occurs within a sequence of felsic to intermediate, calc-alkaline metavolcanic rocks which is bounded to both the north and south by a lower mafic volcanic sequence. This mafic sequence is intruded by the trondhjemitic Sabaskong batholith to the north. Felsic to intermediate rocks are intruded to the east of the deposit by the Black Hawk monzonitic stock. In the deposit area all rock units strike approximately east-west and dip to the south, subparallel to the main foliation recognized in the area.

A summary of rock units in the area surrounding the Rainy River deposit are described below from oldest to youngest. Figure 7.4 shows a schematic stratigraphic column.

7.3.1 Lower mafic volcanic succession

The lower mafic volcanic succession comprises high-iron and high-magnesium basaltic rocks which occur as coarse-grained massive lava flows, massive and pillow flows, and flow breccias. Subordinate dacitic tuff and intrusive quartz-feldspar porphyry dikes and sills are commonly noted interbedded or intruding respectively throughout the mafic volcanic rock.

7.3.2 Pyritic sediment succession

Conformably overlying the lower mafic volcanic succession are a series of pyrite-bearing siliceous to chloritic wacke units, interpreted to be 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. These tuff horizons likely represent onset of the lateral equivalent of subsequent intermediate volcanism.

7.3.3 Intermediate fragmental volcanic succession

Overlying the pyritic sediment horizon is a complex succession of intermediate rocks. In the Richardson Township, these volcaniclastic 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 volcaniclastic 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 volcaniclastic rocks to the east. The rocks of the intermediate fragmental volcanic 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, Western, and HS Zones. These zones are discussed in Section 7.5.

7.3.4 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.

7.3.5 Upper diverse mafic volcanic succession

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. This zone is discussed in Section 7.5.

7.3.6 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 metal-bearing graphitic horizon. The upper contact of the succession is interbedded with the upper felsic succession.

7.3.7 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 4 km westwards from the Black Hawk Stock. It has been interpreted as a quartz-phyric rhyolite.

7.3.8 Intrusions

7.3.8.1 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.

7.3.8.2 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 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. The 34 Zone is hosted in a late-stage mafic-ultramafic intrusion which cross cuts the ODM/17 Zone. This zone is discussed in Section 7.5.

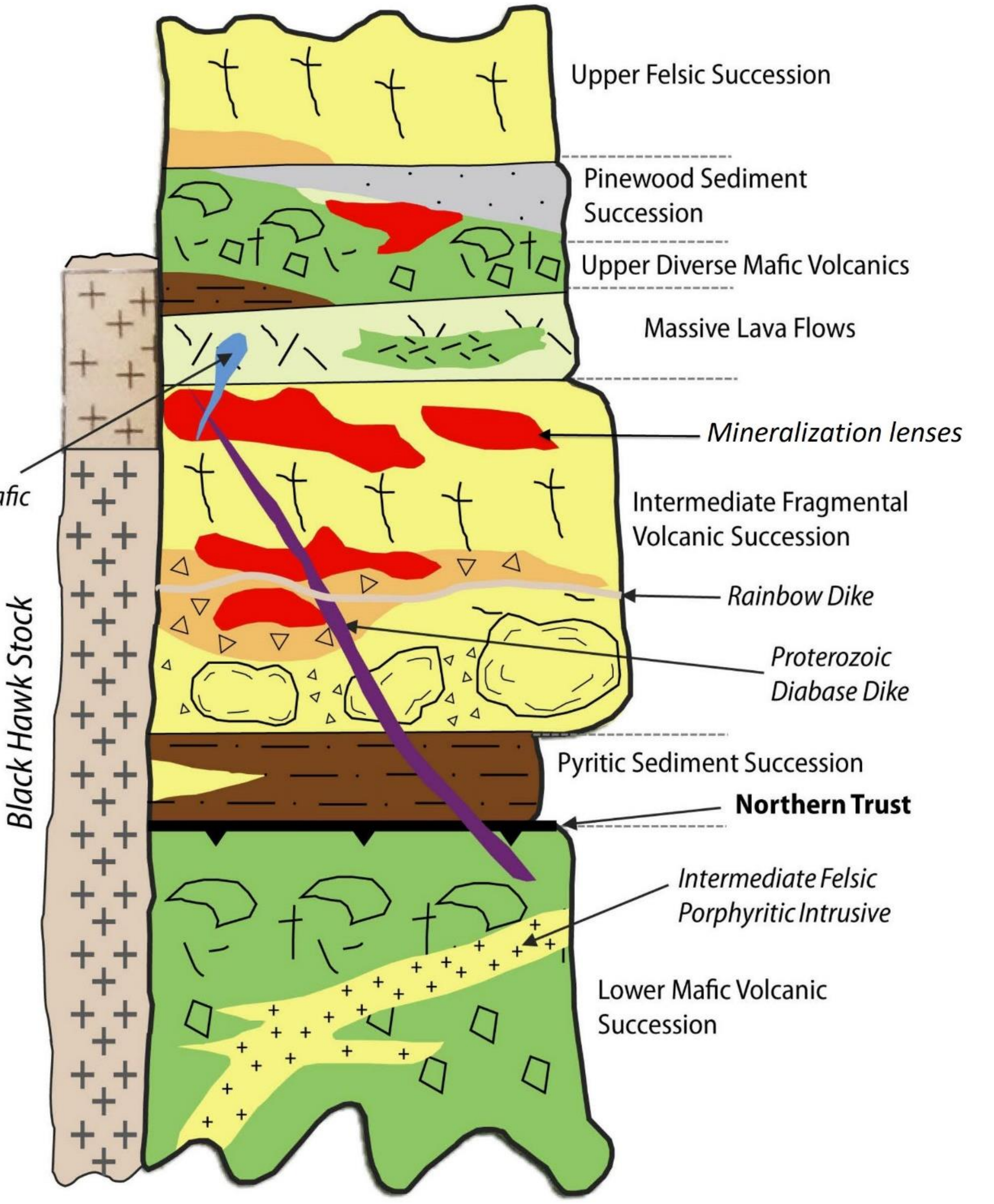
7.3.8.3 Black Hawk stock

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.

7.3.8.4 Proterozoic diabase dike

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

Figure 7.4 Stratigraphic column



Source: Modified from M. Pelletier, 2016

7.4 Structural geology

The volcano-sedimentary sequences of the Rainy River area and regional greenstone belt have been affected by at least five main deformation episodes (D1 to D5) as described in Rankin (2013).

7.4.1 D1 deformation – recumbent folding and thrusting

The earliest deformation event (D1) resulted in the development of large-scale recumbent folds (F1) with north-south trending, sub-horizontal fold axes, an associated, variable intensity, axial planar S1 foliation defined by sericite and chlorite and development of L1 mineral lineation. Folding was also accompanied by localized T1 thrusts. Pre-D1 mineralized veins are strongly folded and commonly transposed into the S1 foliation.

7.4.2 D2 deformation – ESE-WNW folding and thrusting

The second deformation event (D2) resulted in east-southeast trending upright to overturned shallow plunging folds (F2) of variable intensity, and refolded S1 fabrics and L1 lineations with variable dips and plunges across the belt. A weak S2 axial-planar foliation is locally visible in both drill core and outcrop. F2 fold axes are typically sub horizontal to shallowly plunging. F2 folding was possibly accompanied by T2 thrust to high-angle reverse faults, partitioning subdomains with varying D2 strain.

The Rainy River auriferous zones lie within a moderate to steeply dipping F2 limb with S0/S1 trending 110/55 (average) and L1 exhibiting a steep south-southwestern plunge (~down-dip). 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.

7.4.3 D3 deformation – NE and NW kink folding

The observed D3 deformation resulted in broad-scale kink folds in the greenstone belt. These trend north-northwest to north-east (with some conjugate kink geometry evident). F3 folds are associated with subvertical S3 spaced fracture cleavages and occasionally manifest as small-scale faults. 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, occurring where folds are steeply dipping (S0/S1 fabrics). Emplacement of north trending granitoid stocks east of the Rainy River Mine are interpreted to have occurred along F3 kink axes (possible reactivated basement faults).

7.4.4 D4 deformation – late stage faulting

D4 deformation is represented by a late-stage north-northwest to 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 (sub horizontal tension gash structures), and a weak east-southeast trending foliation in the Black Hawk granitoid stock.

7.4.5 D5 deformation – NW trending mafic dykes

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

7.4.6 Timing of mineralization

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:

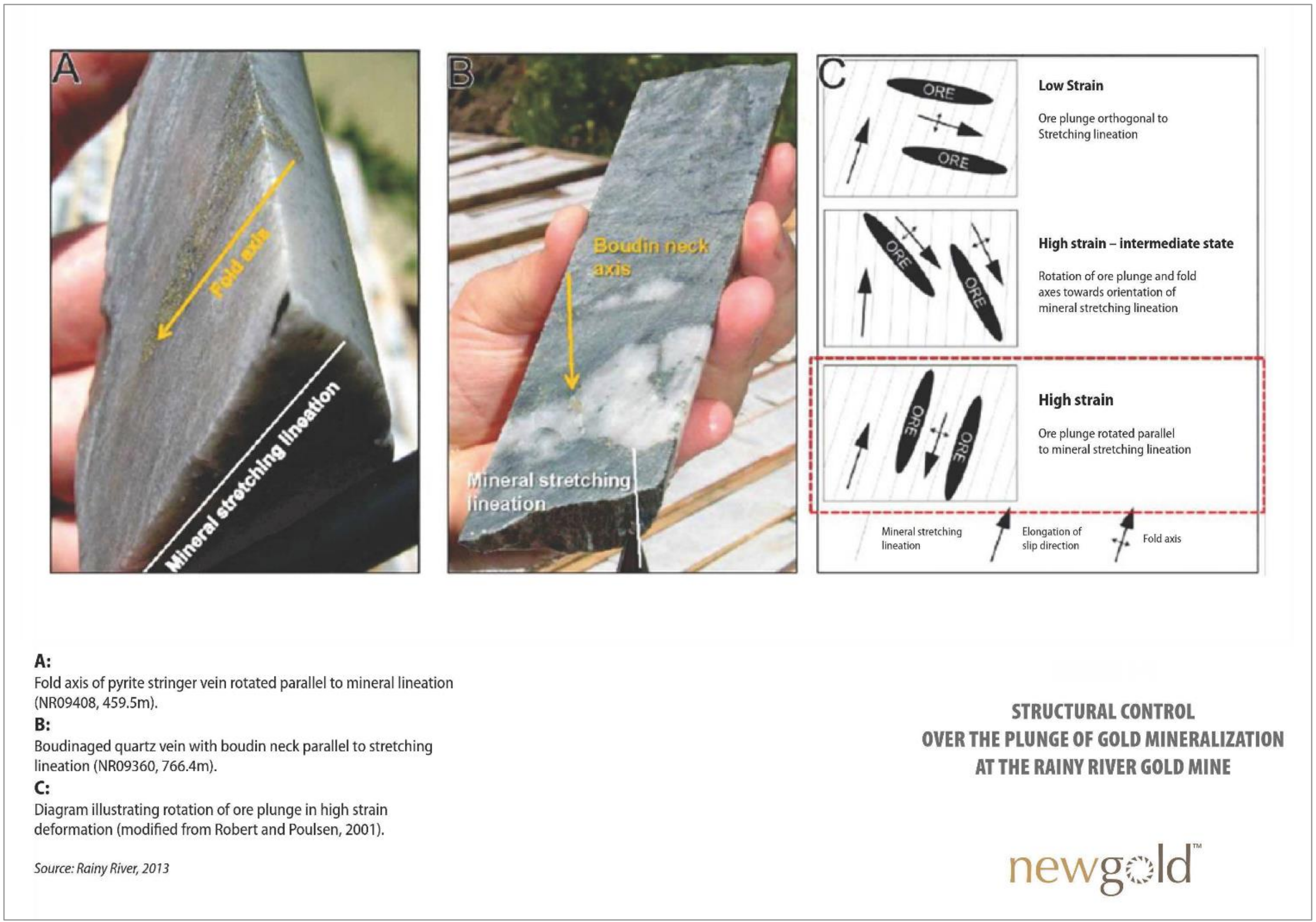
- Auriferous mineralization is aligned along the regional foliation.
- Fold axes of auriferous quartz veins and sulphide stringers are rotated subparallel to the stretching lineation.
- Fold axes, boudin necks, and stretching lineation are subparallel to the plunge of the gold mineralization.
- Early sulphide mineralization is deformed by folding (Figure 7.5).
- Later quartz-sulphide veins are variably deformed and overlap in time with the main regional deformation.

These observations strongly suggest 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 7.6 illustrates the structural controls on the plunge of mineralization.

Figure 7.5 Sulphide mineralization deformed by folding in drill core from Rainy River

Source: SRK 2011.

Figure 7.6 Structural control over the plunge of gold mineralization at Rainy River



Source: New Gold from SRK 2011.

7.5 Deposit geology and mineralization

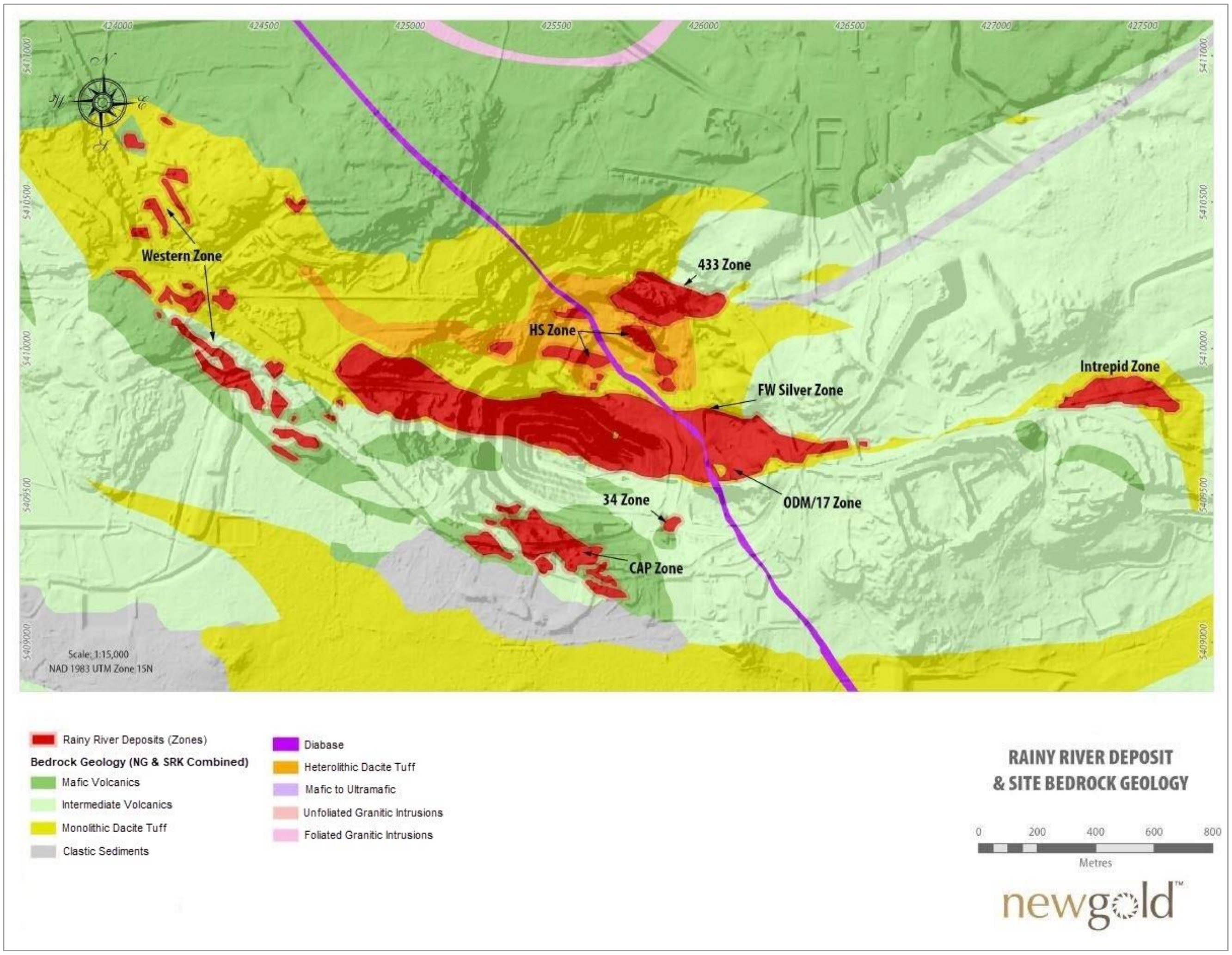
The Rainy River deposit comprises eight distinct zones of gold and silver mineralization as shown in Figure 7.7. These eight zones include four different styles of mineralization as shown in Table 7.1.

Table 7.1 Rainy River mineralization style

Zone	Mineralization style	Rock type
ODM/17 Zone, 433 Zone, HS Zone, Western Zone	Moderately to strongly deformed, sulphide and quartz-sulphide stringers and veins with Au mineralization	Felsic quartz-phyric rocks
CAP Zone	Deformed quartz-ankerite-pyrite shear veins with Au mineralization	Mafic volcanic rocks
Intrepid Zone, Footwall Silver Zone	Deformed sulphide-bearing quartz veinlets with high grade silver	Dacitic tuffs and breccias
34 Zone	Copper-nickel-platinum group mineralization	Mafic- ultramafic intrusion

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. Additional detail on mineralized zones that are part of the Mineral Resources is provided in Section 14.

Figure 7.7 Rainy River – mineralized zones



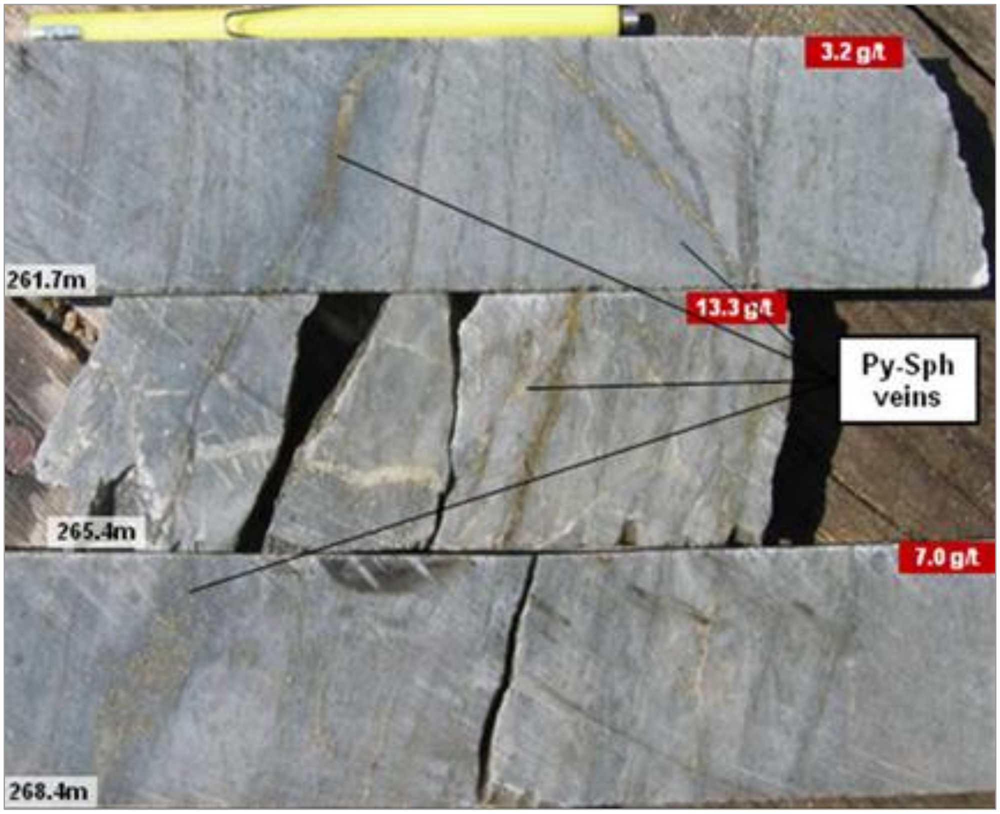
Source: New Gold 2019.

7.5.1 ODM/17 Zone

The ODM/17 Zone is a series of east-west trending, south dipping lenticular sheets hosted within calc-alkaline dacites of the intermediate fragmental volcanic succession. The zone is cut by numerous NNE trending faults. The ODM/17 Zone has presently been defined over a strike extent of 1,600 m and to depths of 975 m. The true width of the zone is approximately 200 m. High grade lenses plunge south-west (aligned with the L2 stretching lineation). Mineralization in the ODM/17 Zone is open below the modelled depth.

Three styles of gold mineralization are 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. Moderate-grade intervals are characterized by tightly folded and foliation parallel pyrite-sphalerite and pyrite stringer veins, commonly associated with stronger silica and weak garnet alteration. Examples are shown in Figure 7.8. High grade gold mineralization is associated with deformed quartz-pyrite-gold veinlets that overprint other mineralization styles. An example is shown in Figure 7.9.

Figure 7.8 ODM/17 Zone gold mineralization



Note: 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).
Source: New Gold 2018.

Figure 7.9 ODM/17 high grade gold mineralization

Note: 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).
Source: New Gold 2018.

7.5.2 433 Zone

The 433 Zone is located approximately 500 m north of the ODM/17 Zone and hosted within strongly sericitized calc alkaline dacite rocks and lesser tholeiitic basalts. The 433 Zone comprises a cigar-shaped lens which plunges steeply south-west. This zone has a strike length of 325 m, a vertical distance of approximately 820 m, and a true width of up to 125 m.

Gold mineralization is similar in style to the ODM/17 Zone but with a number of minor differences:

- The 433 Zone is dominated by chlorite alteration of quartz-phyric host rocks as opposed to sericite in the ODM/17 Zone.
- Chlorite-pyrite altered heterolithic conglomerates occur within the 433 Zone.
- Chalcopyrite and chlorite are associated with high-grade quartz-pyrite-gold veinlets as shown in Figure 7.10.

Figure 7.10 433 Zone high-grade gold mineralization

Note: 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.

7.5.3 Footwall Silver Zone

The Footwall Silver Zone occurs in altered dacitic tuffs and tuff breccias immediately adjacent to a high strain zone at the northern contact of the ODM/17 Zone. This zone plunges to the south-west in similar orientation to the ODM/17 Zone. It is hosted by centimetre scale sulphide-bearing quartz veinlets with common millimetre scale fracture filling to dendritic native silver inclusions. Sulphides contained within these veinlets, in order of frequency, comprise pyrite, sphalerite, chalcopyrite, and galena. Localized spessartine garnets have been noted. The presence of isoclinal folding of the veinlets suggest mineralization occurred prior to or synchronous with deformation. The zone is considered to genetically related to the ODM/17 Zone.

The zone is composed of numerous lenses that range from 5 to 30 m wide, have strike lengths between 5 to 50 m and plunge extents between 300 and 600 m.

7.5.4 HS Zones

Several subsidiary zones of gold mineralization occur between the ODM/17 Zone and 433 Zone.

The HS Zones comprise a series of small, discontinuous south-west plunging, flattened shoots of mineralization. Discontinuous, irregular low-grade gold mineralization is associated with chlorite-pyrite replacement of matrix in flattened, albitized, heterolithic pebble conglomerates. The zone has a strike length of 200 m and extends to a vertical distance of approximately 700 m. The full extent of the HS Zone has not been defined by drilling to date.

7.5.5 The Western Zone

The Western Zone occurs near surface approximately 500 m north-west of the western extent of the ODM/17 Zone. It is composed of stockwork of discrete centimetre scale anastomosing, folded to linear quartz and quartz-carbonate veinlets. The Western Zone is hosted predominantly within strongly deformed intermediate volcanic fragmental units (analogous to those that host the ODM/17 Zone) and mafic volcanic flows in the immediate footwall (FW) and hangingwall (HW). 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.

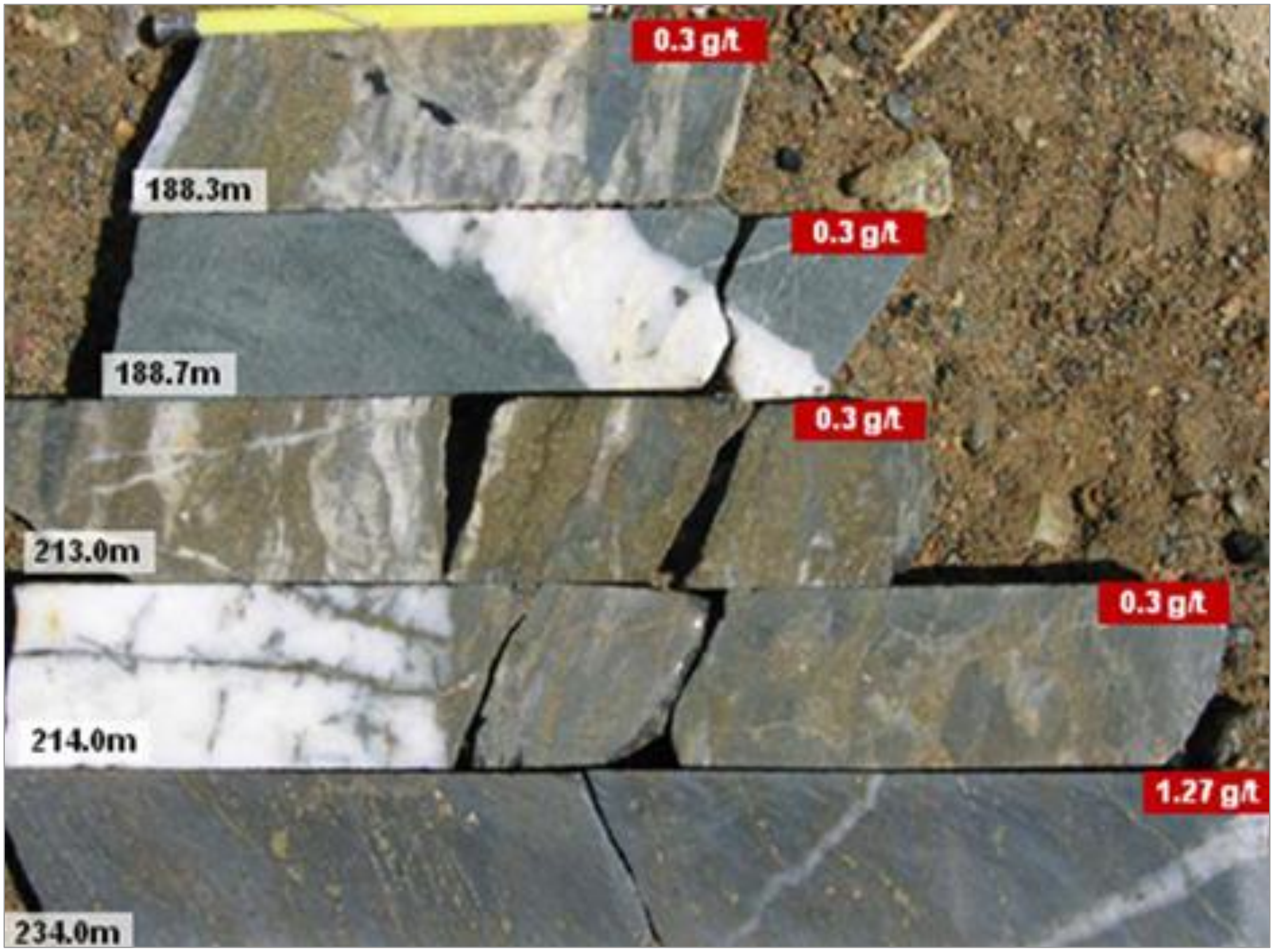
The Western Zone comprises a series of discontinuous 5 to 10 m wide zones of mineralization which strike approximately south-east and dip south-west at approximately 50°. Individual zones encompass a strike length of between 50 and 500 m. Collectively these zones occur over an area of approximately 500 x 1,200 m. They have been defined to down-dip depths of approximately 60 to 500 m.

7.5.6 The CAP Zone

The CAP Zone is located approximately 200 m to the south of the ODM/17 Zone in both tholeiitic basalts and calc-alkaline dacite of the upper diverse mafic volcanic succession. The CAP Zone has been defined over a strike length of 400 m, up to 120 m wide and with a down-dip extent of 750 m. Mineralization in the CAP Zone is open below the modelled depth.

Higher-grade gold mineralization is associated with deformed quartz-ankerite-pyrite shear and extensional veins hosted by quartz-ankerite-pyrite altered mafic volcanic rocks. Examples are shown in Figure 7.11. Relative to ODM/17 and 433 Zones, the CAP Zone has a higher pyrite-chalcopyrite content.

Figure 7.11 Higher-grade gold mineralization within the CAP Zone



Note: Borehole NR10-474 from 188.0 to 234.0 m.
Source: SRK 2011.

7.5.7 Intrepid Zone

The Intrepid Zone is located approximately 800 m east of the ODM/17 Zone within dacitic tuffs and breccias of the intermediate fragmental volcanic succession. The Intrepid Zone has been defined over a strike length of 410 m and to 450 m down-dip. The width of the zone is variable ranging between 10 m to 60 m.

High-grade gold and silver mineralization 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.

7.5.8 34 Zone

The 34 Zone comprises magmatic nickel copper sulphide mineralization associated with precious metals (gold, platinum group metals) within a tubular, ~100 m thick, late-stage pyroxenite gabbro intrusion which cross cuts the ODM/17 Zone and post-dates the main gold mineralization event. The host pyroxenite-gabbro intrusion is unmetamorphosed, but locally altered into serpentine and talc. Magmatic sulphides vary from massive to net-textured and disseminated. Gold and silver mineralization occur within 5 to 50 m thick dislocated (and therefor discontinuous) north-east oriented pods over a strike length of 500 m with a down-dip plunge of 100 m.

8 Deposit types

The following section has been summarized from Pelletier's M.Sc. thesis (2016) and the QP approves this information. The M.Sc. thesis represents the latest update on deposit style and formation of the Rainy River mineralization. Additional details of the deposit formation are given in the 2018 New Gold Technical Report. A schematic diagram of the potential formation of the Rainy River deposit is shown in Figure 8.1.

The Rainy River deposit is an auriferous VMS system (Pelletier 2016) with a primary syn-volcanic source and possibly a secondary syn-tectonic mineralization event (Mercier-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:

- Spatial correlation of gold with base metals at the deposit scale.
- Close spatial association between gold and zoned hydrothermal alteration.
- Stacking of auriferous bodies in a restrained volcanic pile.
- The presence of a gold-rich core and a barren rim of pyrite mineralization.
- Preferential association of alteration and auriferous zones with volcaniclastic 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 dacitic 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.

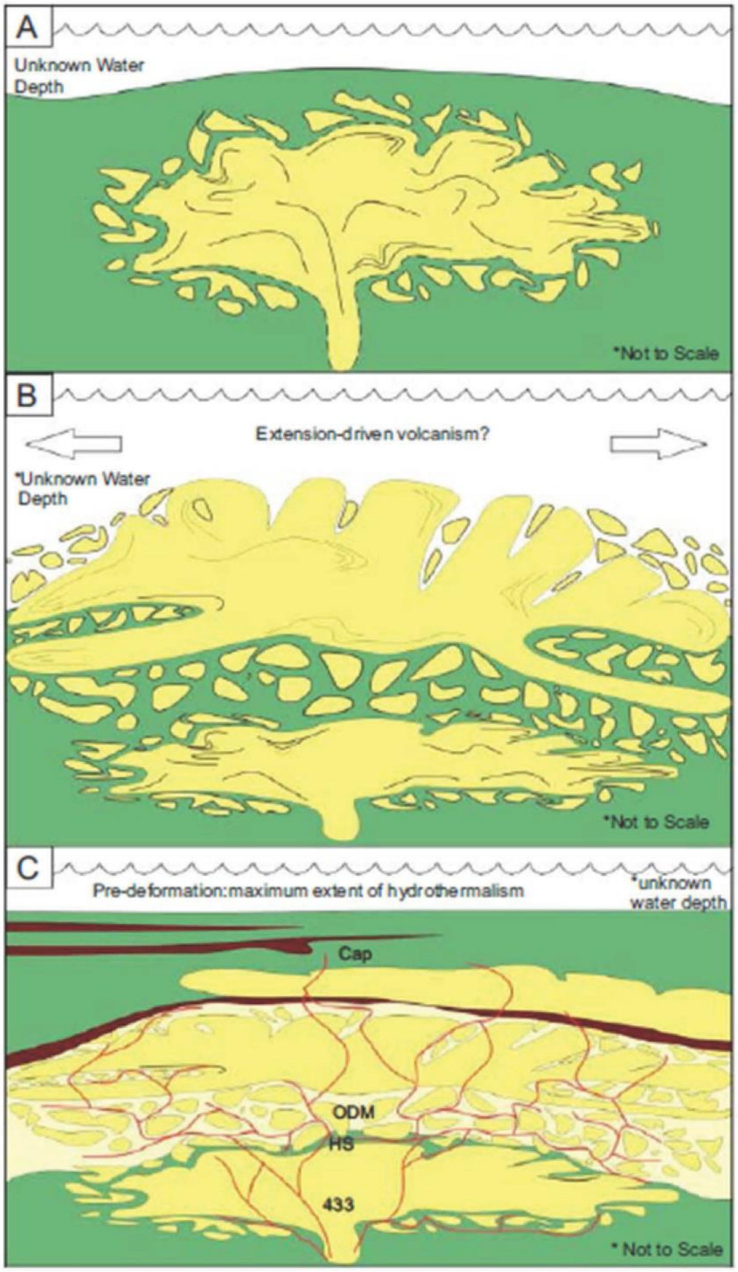
In VMS deposits, the main source of metals are the surrounding volcanic and / or sedimentary rocks, from which circulating hydrothermal fluids collect, enrich and transport the metals and precipitate them in a zone of massive sulphide mineralization at or below seafloor (Franklin et al. 2005).

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. 2007; 2011; 2015).

Figure 8.1 Potential formation of the Rainy River deposit



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.

Source: Pelletier, 2016

POTENTIAL FORMATION OF THE RAINY RIVER DEPOSIT

9 Exploration

New Gold has completed a number of exploration programs at the Property since the announcement of the takeover of RRR in May 2013. New Gold exploration activities are summarized in Table 9.1.

Table 9.1 Summary of New Gold exploration activities at Rainy River

Date	Activity	Performed by
Jul – Oct 2013	2,085 sample MMI geochemical survey	New Gold Geologists
Jul – Nov 2013	56,000 m re-logging program within ODM Zone	New Gold Geologists
Jun-Sep 2013	MSc thesis - style, geometry, timing and structure of mineralization	M. Pelletier, Université du Québec
May – Jul 2014	862 sample MMI geochemical survey	New Gold Geologists
Jan – May 2015	102,380 m re-logging program within Burns Block claim	New Gold Geologists
Apr – Nov 2016	5,000 m Corescan hyperspectral alteration survey	New Gold Geologists
May 2015 – Dec 2016	1,992 sample SWIR spectral alteration survey	New Gold Geologists
2017 - 2018	Drone Airborne UAV-MAG Survey	Abitibi Geophysique
Aug- Dec 2019	174 rock chip samples, 1,136 soil samples	New Gold Geologists

Notes: MMI =mobile metal ion; SWIR=short-wavelength infrared; UAV=unmanned aerial vehicle; MAG=Magnetic.
Results of the MSc thesis have been summarized and referenced in Sections 7 and 8.
Source: New Gold 2019.

9.1 Mobile Metal Ion (MMI) sampling programs

MMI programs initially consisting of 2,085 samples and later 862 samples were completed on various portions of the Property in 2013 and 2014 respectively. The work included 10 sample grids comprising five 100 m spaced reconnaissance lines with a 25 m sample spacing. This work included sampling of the Intrepid Zone.

The test grid over the Intrepid Zone showed a weak to moderate gold anomaly which did not match with the surface projection of Intrepid mineralization. Sporadic gold anomalies were drill tested with no significant results.

9.2 Relogging programs

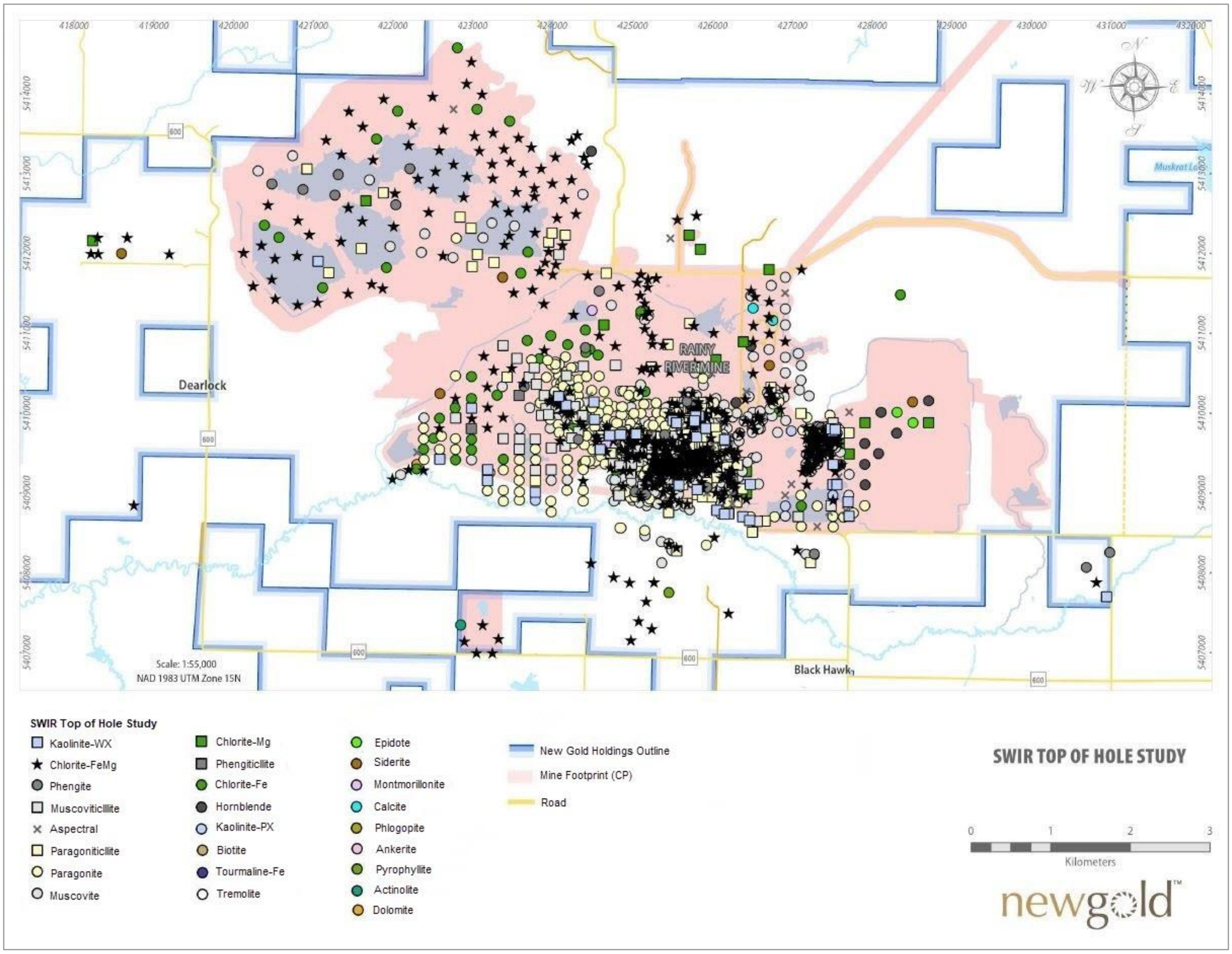
New Gold completed a relogging campaign between July and October 2013. A total of 56,000 m of diamond drill core from key sections of the ODM Zone were relogged to improve the company's understanding of controls on mineralization. All data was incorporated into the digital database.

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

9.3 Short-wavelength infrared (SWIR) alteration study

New Gold completed a 1,992 sample SWIR sampling program between May 2015 and December 2016. The location of this survey is shown on Figure 9.1. Top of hole drillhole samples within the deposit area were analyzed using oreXpress (previously called SpecTERRA) to identify white mica and chlorite compositions. The results of this program were inconclusive and were interpreted to have been affected by thermal overprinting associated with emplacement of the Black Hawk stock.

Figure 9.1 SWIR top of hole survey sample locations



Source: New Gold 2019.

9.4 Hyperspectral alteration study

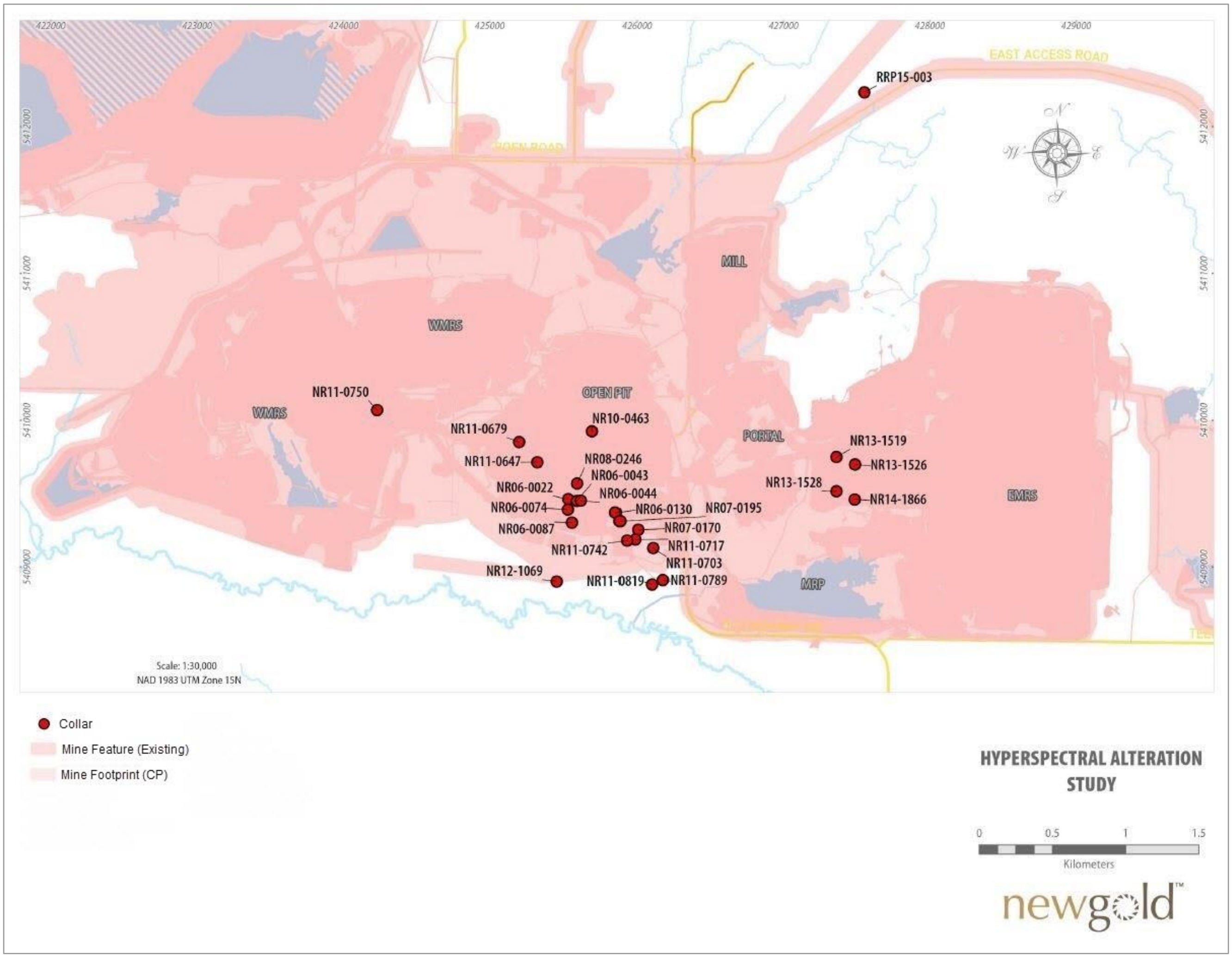
New Gold completed a hyperspectral alteration study to determine potential vectors to gold mineralization in 2016. This program comprised the scanning of approximately 5 km of drill core from the Rainy River deposit and surrounding exploration areas using the Corescan hyperspectral system provided by SGS Analytical Services.

Corescan mineral logs and spectral parameters were compared against sample assays, geochemistry, lithology and magnetic susceptibility and correlations evaluated. In addition, drillholes included in the Corescan study were inspected by site geologists and compared against results. Refinements were made to logging protocols and core was relogged where required.

The Corescan study shows that white micas transition from predominantly phengite peripheral to mineralization zones, to slightly sodic muscovite proximal to mineralization. Similarly, chlorite transitions from Fe-rich to Mg-rich towards the core of the VMS system.

Figure 9.2 shows the location of the hyperspectral alteration study.

Figure 9.2 Corescan hyperspectral alteration study drillhole locations



Source: New Gold 2019.

9.5 MSc research

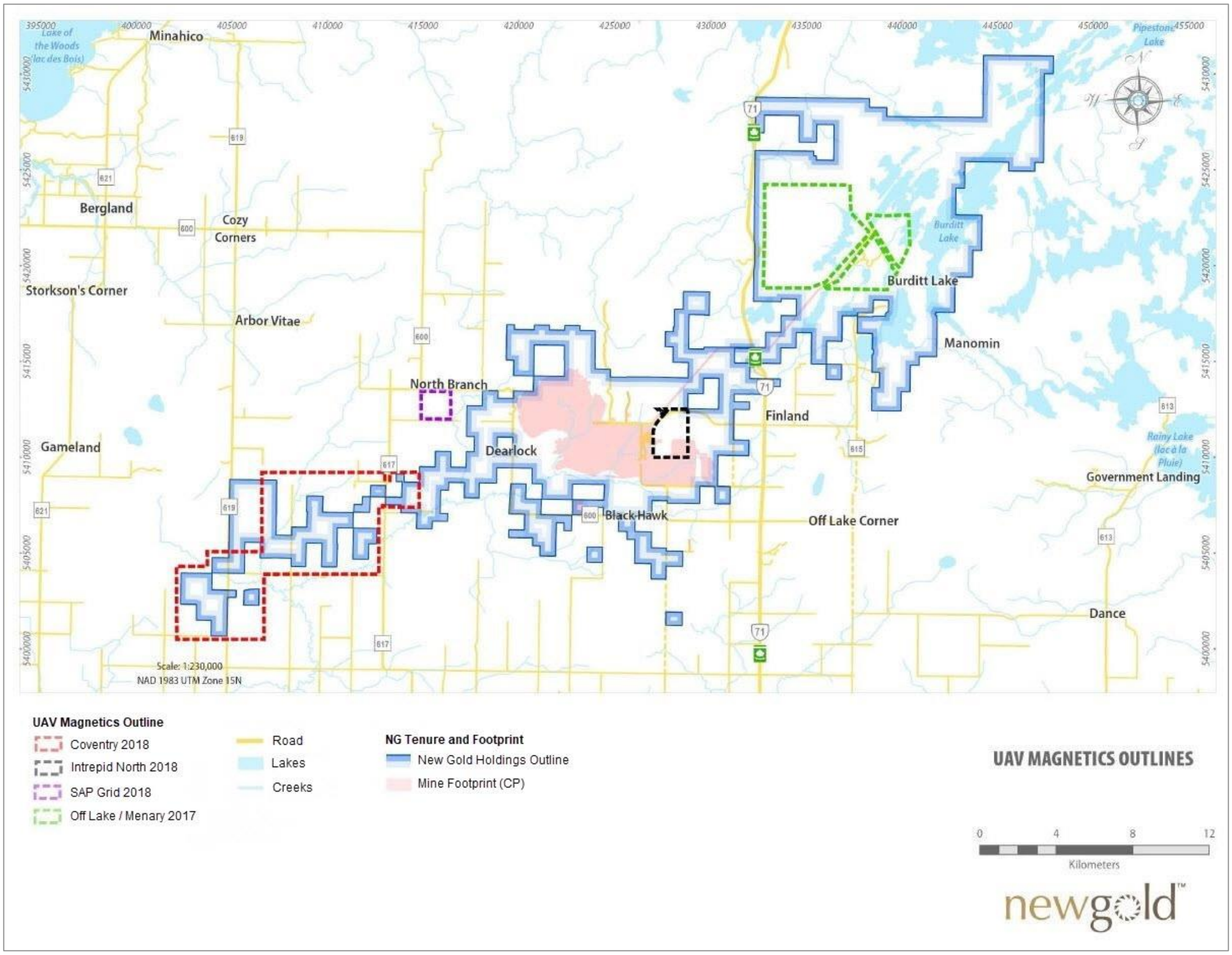
A detailed study of the geology of the Rainy River gold deposit was completed by Ms Mireille Pelletier in 2016 as part of a MSc research with the Université du Québec - Institut National de la Recherche Scientifique (Pelletier 2016). The thesis provided a comprehensive description of deposit geology and controls to mineralization at the Rainy River deposit.

9.6 Unmanned aerial vehicle (UAV) magnetic survey

A high-resolution survey UAV magnetic survey was completed by Abitibi Geophysique for New Gold in 2017 and 2018. A total of 2,041 line-kilometers was flown on 50 m spaced lines over four separate regional targets. The UAV survey improved the understanding of geological framework within target areas including distribution of lithological units, and location of major tectonic features.

The location of the four survey areas is shown in Figure 9.3.

Figure 9.3 2017 – 2018 UAV magnetic survey areas



Source: New Gold 2019.

9.7 Rock chip sampling program

In August 2019 the New Gold exploration team commenced a regional rock chip and soil sampling campaign to generate regional exploration targets. A total of 174 samples and 1,136 soil samples were collected; samples results incorporated within the regional database, combined with geophysical and geological data collected will build the complete data set for follow-up interpretation and drill ready target definition.

10 Drilling

This section describes diamond drilling programs completed by RRR and New Gold from 2005 to present. Drill procedures used by Nuinsco between 1994 and 2004 and Bayfield between 2010 and 2014 are not well documented and are not described in this report.

RRR's and New Gold's drill programs were designed and completed by an experienced exploration team under the supervision of a Project Manager and Vice President, Exploration.

Diamond drill programs completed at the Rainy River deposit and the Intrepid Zone were performed by Bradley Bros. Ltd, Naicatchewenin Development Corporation 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%) comprise the remaining 3% of drillholes.

Rainy River drillholes were drilled predominantly on northerly directed azimuths at inclinations of between 50° and 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 diamond core drilling completed at the Rainy River Mine is included in Table 10.1 and includes all diamond core drillholes drilled on the Property. RC, geotechnical, and abandoned holes are excluded. Drillholes used in the Mineral Resource estimate are a subset of this drilling database. Figure 10.1 shows the location of drillholes in the core portion of the Property.

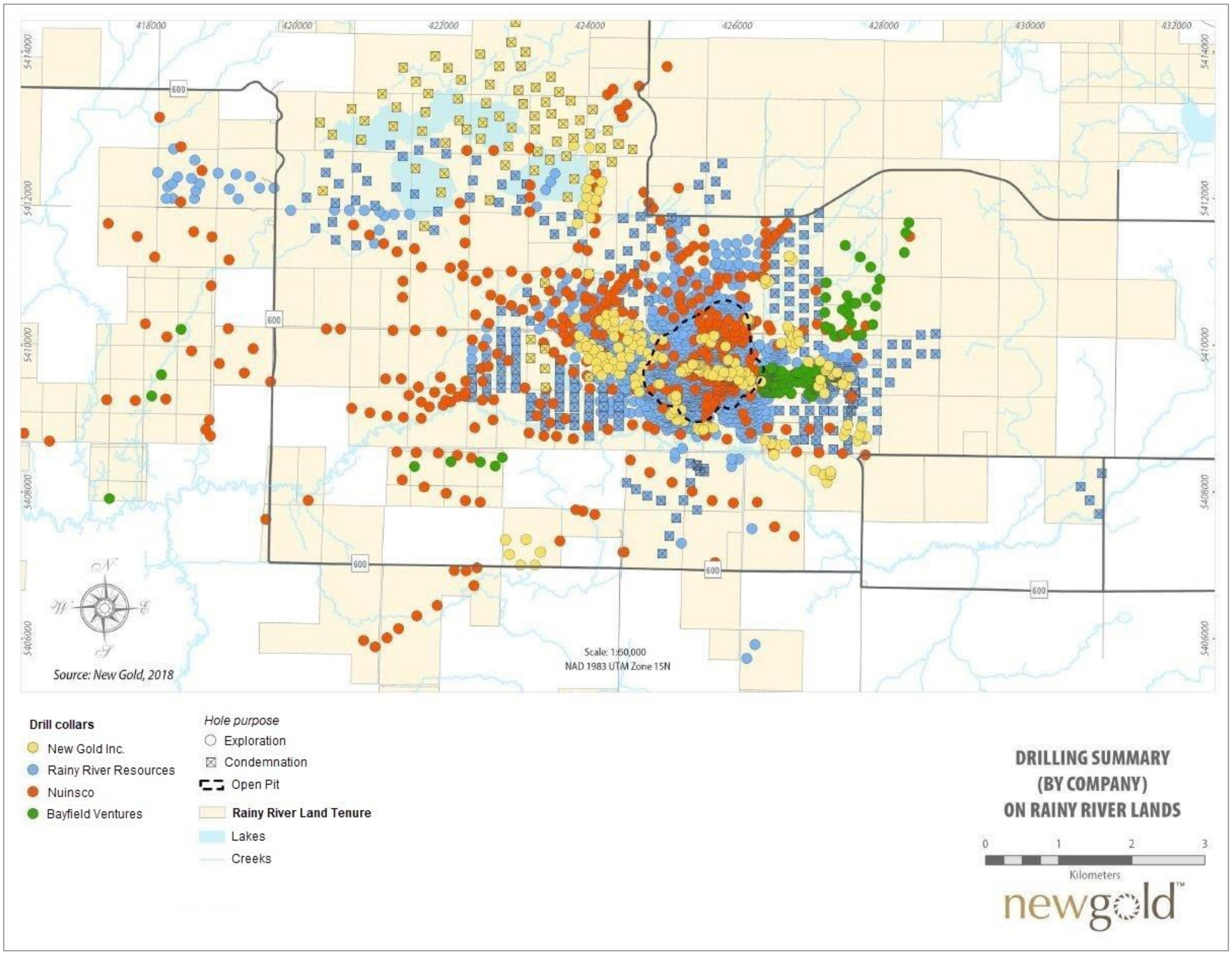
Table 10.1 Summary of diamond drilling at Rainy River

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		
New Gold	2013	27	9,305	37	7,700
	2014	113	44,452	78	15,690
	2015	50	10,592		
	2016	37	5,871		
	2017	31	10,546		
	2019	6	3,358		
	New Gold total	264	84,124	115	23,390
All	**Overall total**	**2,191**	**926,046**	**305**	**66,018**

Notes: This table does not include abandoned, geotechnical, nor RC drillholes.

Drillholes were designed to provide sufficient information to delineate Mineral Resources. Representative cross sections of drilling completed at the Rainy River deposit are presented in Figure 10.2 to Figure 10.4.

Figure 10.1 Drillhole location map



Source: New Gold 2018.

A summary of procedures relating to drilling is provided below.

10.1 Collar surveying

A hand-held global positioning system (GPS) was used to locate and prepare drilling pads in the field. At the completion of each drillhole a Differential GPS (DGPS) was used to survey the casing collar. DGPS accuracy was validated using a known control station location.

10.2 Downhole surveying

Drillhole deviation surveys were completed using a Reflex EZ-SHOT™ instrument. Downhole surveys were collected on 50 m intervals. Downhole surveys show that all drillholes typically shallow with depth. Deeper drillholes also deviate in azimuth.

At the Intrepid Zone, 60 out of the 230 drillholes have been resurveyed with a Reflex Gyro at 5 m intervals. An azimuth pointing system was used to determine the azimuth and inclination at the collar.

To address drillhole deviation in deeper holes RRR utilized Tech Directional Drilling in 2011 to ensure that deeper drillholes intersected planned targets.

10.3 Core processing and logging

All diamond drill core is processed and stored at New Gold's onsite secure core logging facilities which are security monitored 24 hours per day, seven days per week. Core processing and logging procedures have been in effect throughout the RRR and New Gold drill programs.

Core processing includes the collection of core recovery data, magnetic susceptibility, geotechnical data, and geological logging. Core recovery and detailed geotechnical logging including rock quality designation (RQD), joint / fracture analysis, material type, and rock strength were implemented in 2014. Magnetic susceptibility readings are recorded every 3 m. Specific gravity is recorded for both mineralized and non-mineralized material. Geological logging comprises collection of lithology, alteration, mineralization and structure data.

Core is not routinely photographed, although significant intersections and features are photographically recorded.

Core logging data is captured directly onto laptop computers previously using Datamine's DHLogger™ and more recently Maxwell LogChief™. Validation protocols are built into the software to ensure data consistency and minimize data collection errors. LogChief™ logging data is merged into a central Maxwell Datashed™ database where further validation is completed. Geological and assay data is transferred directly from the DataShed™ database into Maptek Vulcan software for three-dimensional (3D) visualization, interpretation, and modelling.

10.4 Sampling

RRR initially selectively sampled parts of the drillholes based on visible observations and interpretation 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. In 2012, RRR adjusted sampling procedures so that the entire drillhole was sampled with predominantly 1.5 m samples. This sample interval as adjusted where required to respect geological boundaries. Under New Gold in 2016 and 2017, sampling was performed at regular 1 m intervals. Shorter samples were collected at the contacts between geological domains.

Sampling is completed following geotechnical and geological logging. A geologist and / or geotechnician marks out sample intervals with a red grease pencil and places two sample tags (with unique pre-printed sample numbers) 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.

Samples are cut using a diamond core saw. After each 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 two sample tags (previously placed at the beginning of each sample interval) is then placed in the sample bag, while the other remains in the core box for reference. Sample bags are stapled closed by the core cutting technician and marked with the unique sample number using a permanent marker.

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 (lab). Each drillhole is separated by placing the rice bags on separate wooden pallets, never combining different holes on one pallet.

Sample shipments are typically dispatched to the lab on 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 by New Gold personnel to the 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.

10.5 Sample recovery

Diamond core sample recovery data has been collected since New Gold acquired the Property in 2013. Core recoveries from New Gold drill programs vary between 2.33% and 100% averaging 99.9%. A total of 219 of the 16746 intervals in the database have recoveries less than 90%.

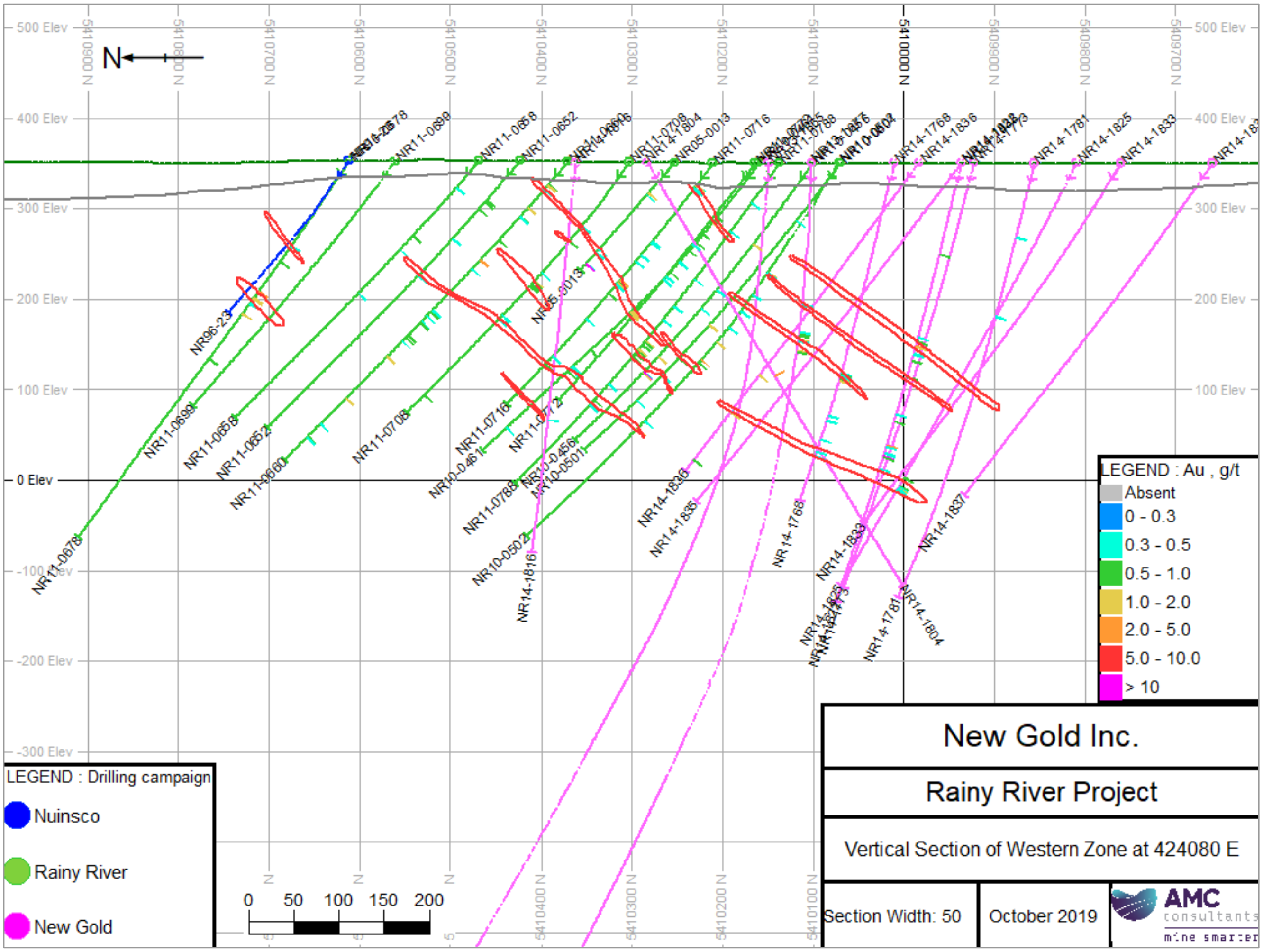
10.6 Representative sections

The following figures show representative sections through the Rainy River Project area from west to east.

- Figure 10.2 is a section through the Western Zone.
- Figure 10.3 is a section through the main zones (including the ODM/17 Zone).
- Figure 10.4 is a cross section through the Intrepid Zone.

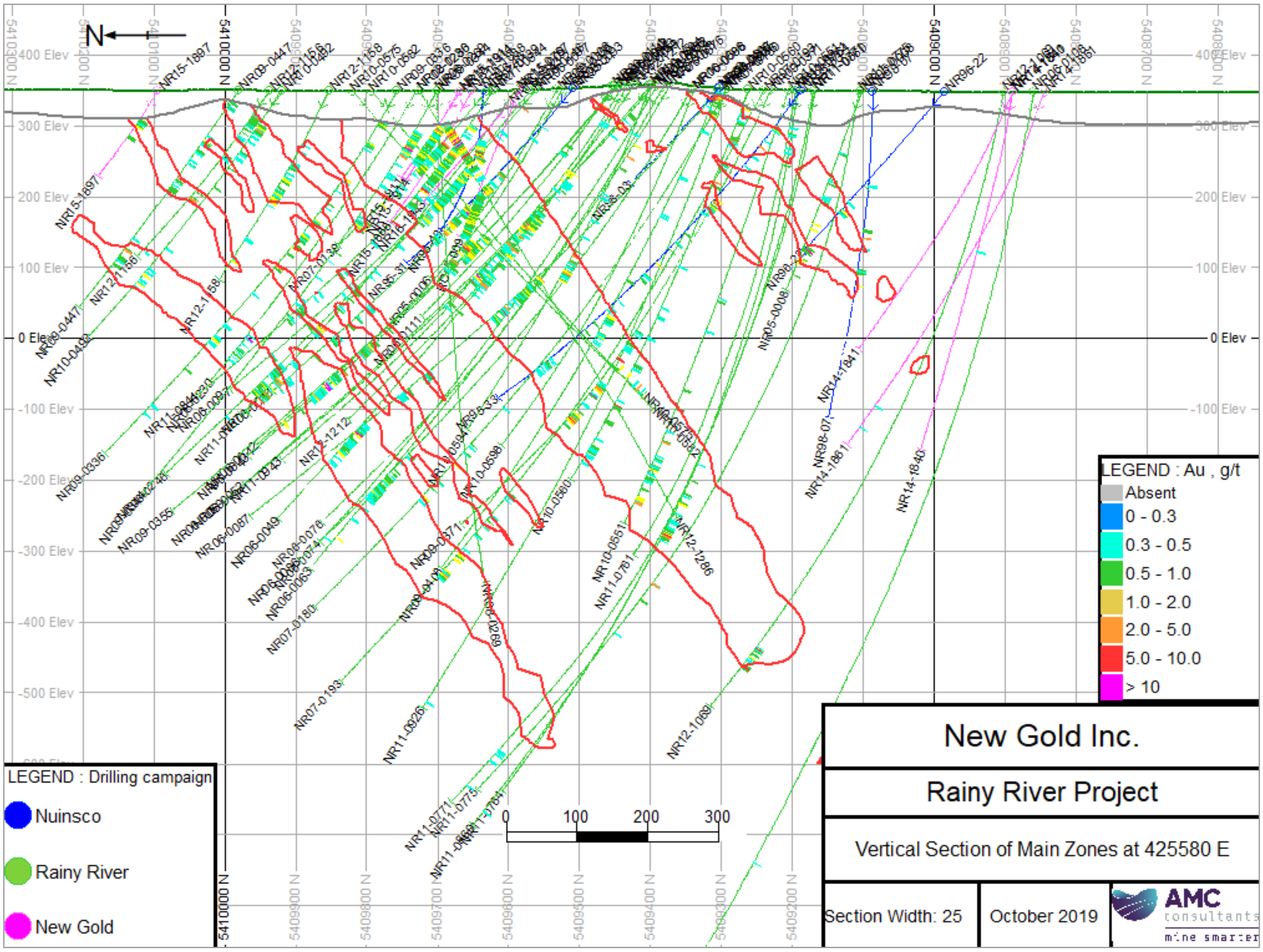
The location of these zones is shown on Figure 7.7 in Section 7.

Figure 10.2 Vertical section through the Western Zone



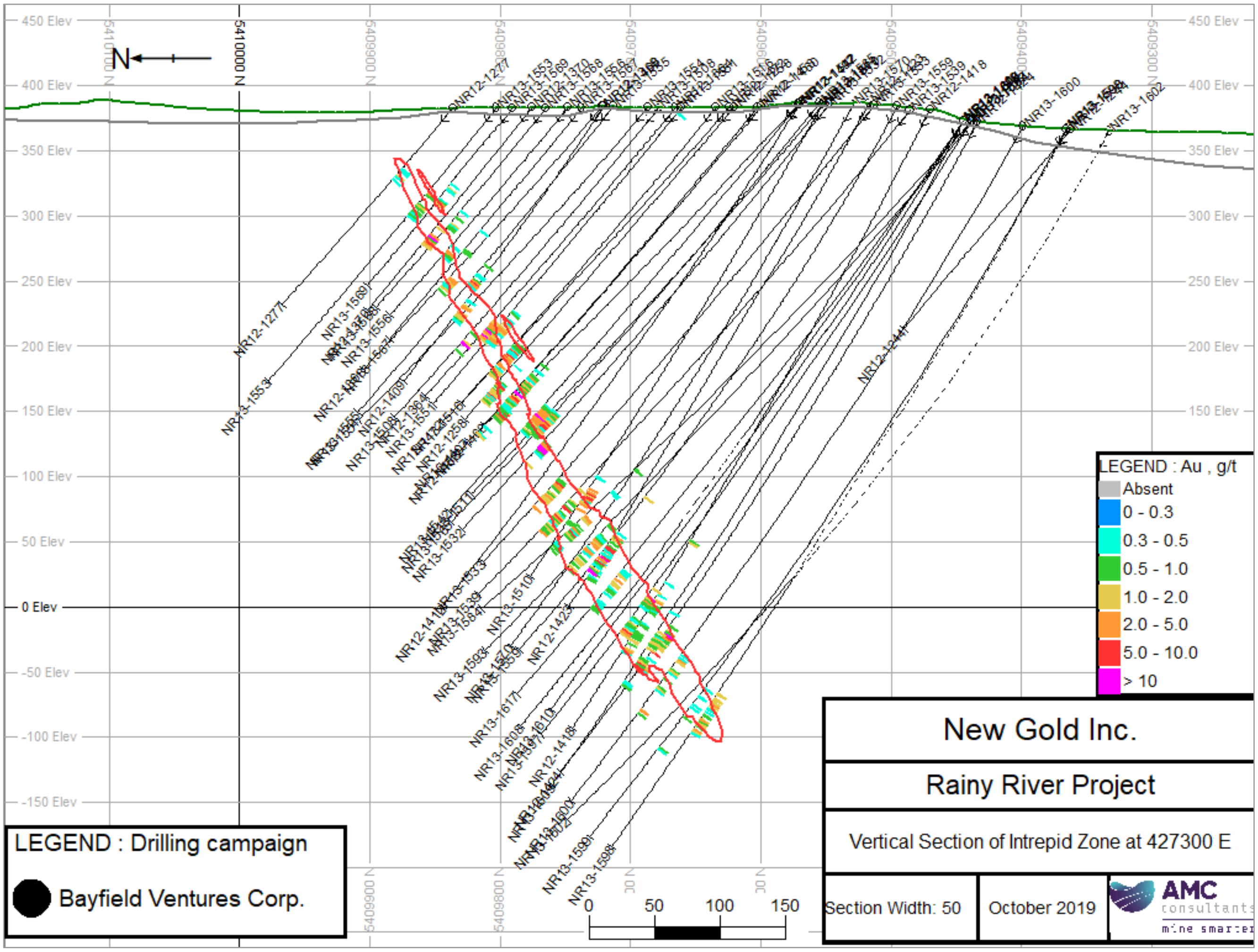
Source: AMC 2019.

Figure 10.3 Vertical section through the main zones (including ODM/17 Zone)



Source: AMC 2019.

Figure 10.4 Vertical section through the Intrepid Zone



Source: AMC 2019.

10.7 Conclusion

In AMC's opinion, there are no drilling, sampling or recovery factors that could materially impact the accuracy and reliability of drill results.

11 Sample preparation, analyses, and security

11.1 Introduction

This section describes the sampling methods, analytical techniques and assay Quality Assurance / Quality Control (QA/QC) protocols followed during the 1994 to 2017 drill programs. Drilling and QA/QC programs are divided into periods, based on the operator at that time. These are: Nuinsco (1994 to 2004), RRR (2005 to 2013), and New Gold (2013 to 2017). Also, New Gold acquired the Bayfield land, which is now part of the current property, from Bayfield in January 2015. The original Bayfield QA/QC data was provided and is treated separately. The location of this ground is shown in Figure 6.1. Sampling methods, preparation, and analyses employed by Bayfield are discussed in Sections 11.2.4 and 11.3.4. All laboratories that have been used are independent of the issuers.

11.2 Sampling methods

11.2.1 Nuinsco Resources Ltd. (1994 – 2004)

Limited information is available for this time period, but Mackie et al. (2003) states that drill core was logged and sampled at the Nuinsco core shack in Richardson Township, with sample splitting achieved through both a hydraulic core splitter and diamond core saw. Samples were bagged and shipped to the ALS Chemex (ALS) preparation lab in Thunder Bay, ON. Accurassay Laboratories Ltd. (Accurassay) also in Thunder Bay, was briefly used. No other sampling methodology information is available for this time period.

11.2.2 Rainy River Resources Ltd. (2005 – 2013)

RRR sampling methodology is summarized from the 2008 Technical Report by CCIC (2008).

RRR initially began sampling entire drillholes at 1.5 m intervals but after approximately eight months, geological understanding improved, and sampling became selective. Sampling focused on specific intervals identified using visual mineralization and alteration criteria. Sampling intervals varied from 1.0 to 1.5 m, with the former used in areas of suspected mineralization.

The logging geologist inserted two sample tags at the beginning of each marked sample interval, with a third tag remaining in the tag book, recording the hole ID and sample interval.

Samples were halved using a core saw, and then rinsed. Half the sample was placed in a bag with one of the tags, the second half remained in the core box with the second tag. Sample bags were stapled shut and packed into labelled rice bags at a frequency of approximately 5 samples per bag.

11.2.3 New Gold Inc. (2013 – 2017)

New Gold sampling methods are similar to those of RRR. Thus, 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 sample tag is placed in the sample bag, and a second remains in the core box for reference. The sample bags are stapled shut 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 lab. Each hole is separated by placing the rice bags on separate wooden pallets, never combining holes on one pallet.

11.2.4 Bayfield Ventures Corp. (2010 – 2014)

Sampling methods are summarized from Duke (2014). Samples with perceived mineralization are cut by core saw, with samples not exceeding 1.5 m in length. Half of the drill core is placed in a labelled plastic sample bag together with a unique sample tag matching the bag label. Samples with no perceived mineralization have no length limit. In these instances, the core is not cut but chipped, with chips collected into a sample bag and labelled in the same way as cut core samples.

11.3 Sample preparation and analysis

Since 1994, the various operators have employed multiple labs with differing sample preparation and analytical methods. Table 11.1 summarizes the analytical labs, Table 11.2 summarizes the preparation methods, Table 11.3 summarized the analytical methods used for Au analyses, and Table 11.4 summarized the analytical methods used for Ag analyses.

The QP notes that all laboratories listed below are independent of New Gold.

Table 11.1 Preparation facilities and analytical laboratories

Company	Years	Laboratory	Location	Accreditation
Nuinsco	1994 - 2004	ALS	Prep - Thunder Bay, ON (?) Analytical – Mississauga, ON	ISO 9002:1994 ISO 9001:2000
RRR	2005 - 2006	ALS	Prep - Thunder Bay, ON Analytical – North Vancouver, BC	ISO 9001:2000 ISO/IEC 17025:2005
	2006 - 2011	Accurassay	Thunder Bay, ON	ISO 9001:2000 ISO/IEC 17025:2005
	2009	Actlabs	Thunder Bay, ON	ISO/IEC 17025
	2010	ALS[1]	Analytical – North Vancouver, BC	ISO 9001:2008 ISO/IEC 17025:2005
	2011 - 2013	ALS	Prep - Thunder Bay, ON Analytical – North Vancouver, BC	ISO 9001:2008 ISO/IEC 17025:2005
New Gold	2014 - 2017	ALS	Prep - Thunder Bay, ON Analytical – North Vancouver, BC	ISO 9001:2008 ISO/IEC 17025:2005
	2014 - 2017	Actlabs[1]	Thunder Bay, ON	ISO/IEC 17025
Bayfield	2010 - 2014	Actlabs	Thunder Bay, ON	ISO/IEC 17025:2005
	2010	TSL	Saskatoon, SK	ISO/IEC 17025:2005 CAN-P-4E CAN-P-1579

Note:[1]Umpire lab.
Source: AMC, using data provided by New Gold.

11.3.1 Nuinsco Resources Ltd. (1994 – 2004)

The following is summarized from Mackie et al. (2003). Samples were prepared at the ALS preparation lab in Thunder Bay, ON. Samples were crushed to ~1 cm sized pieces using a jaw crusher, then put through a roll crusher until >60% passed 10 mesh (2 millimetres (mm)). A 200-250 g riffle split was taken from the crushed sample, and then pulverized in a ring mill until >95% passed 150 mesh. This pulp was then sent to ALS in Mississauga, ON for Au, Cu, Zn, and Ag analysis. Specific analytical method codes are not available. Sample preparation methods are summarized in Table 11.2, and analytical methods are summarized in Table 11.3 and Table 11.4.

ALS Chemex (currently ALS) facilities are accredited (Table 11.1) and were independent of Nuinsco.

11.3.2 Rainy River Resources Ltd. (2005 – 2013)

RRR used multiple labs during their ownership of the Property as shown in Table 11.1.

All labs used by RRR are accredited analytical labs and were independent of RRR. The management system of the ALS Group Laboratories holds quality management accreditation from the

International Organization for Standardization (ISO 9001:2000 (2005 to 2008); ISO 9001:2008 (2008 to 2014)). 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 (2008 to present)) for certain testing procedures, including those used to assay samples submitted from the Rainy River Mine. All ALS preparation facilities also fall under the ISO/IEC 17025:200 accreditation. ALS Laboratories also participated in international proficiency tests such as those managed by CANMET and Geostats Pty Ltd.

The Accurassay facility in Thunder Bay holds accreditations including ISO 9001:2000 and ISO/IEC 17025:2005 for the Mine's relevant analytical tests.

Activation Laboratories Ltd. (Actlabs) holds accreditation ISO/IEC 17025 for certain testing procedures including gold and silver assaying using a fire assay procedure.

11.3.2.1 ALS Chemex (2005 – 2006)

ALS sample preparation involved crushing the sample such that >70% passed through a 2 mm (9 mesh) screen. A 250 g split was then pulverized in a ring mill to achieve > 85% passing through 200 mesh (75 µm) sieve (lab method code PREP-31; Table 11.2).

A 30 g sample was analyzed for gold by fire assay with an atomic absorption spectroscopy (AAS) finish (lab method code Au-AA23). Samples that exceeded the detection limit were re-analyzed by fire assay with a gravimetric finish (lab method code Au-GRA21).

Silver was analyzed by aqua regia (AR) digest with an atomic emission spectroscopy (AES) finish (lab method code ME-ICP41). Samples that exceeded the detection limit were re-analyzed using the same digest and an AES finish, and with a greater upper detection limit (lab method code Ag-OG46).

Analytical methods, including detection limits, are summarized in Table 11.3 and Table 11.4.

11.3.2.2 Accurassay Laboratories (2006 – 2011)

Samples were first entered into a local information management system.

Accurassay preparation method code ALP1 was requested by RRR. The samples were dried in an oven at 50°C prior to crushing with a TM Engineering Rhino Jaw crusher until >90% passed 8 mesh (2 mm). A 500 g split separated using a Jones Riffle Splitter was then pulverized using a TM Engineering ring and puck pulverizer with 500 g bowls until 90% passing 150 mesh (106 µm) was achieved. Pulverized samples were then matted to ensure homogeneity. The homogeneous sample was then sent to the fire assay lab or the wet chemistry lab, depending on the analysis required.

Gold was analyzed by fire assay using lab method code ALFA1. A 30 g sample was mixed with a silver solution and a lead-based flux and fused, resulting in a lead button. 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 produced. This silver bead was digested using AR and bulked up with a distilled de-ionized water and digested lanthanum solution. The solution was then analyzed for gold using AAS. Samples that exceeded the 30,000 parts per billion (ppb) (30 ppm) detection limit for gold were reanalyzed by fire assay but with a gravimetric finish (lab method code ALFA5; Table 11.3).

For silver analysis samples were weighed for geochemical analysis and digested using AR and analyzed for silver using AAS (lab method code ALAR1). Samples that exceeded the

100 parts per million (ppm) detection limit for this method were similarly reanalyzed using an AR digest and AAS finish but with a higher detection limit (lab method ALAR2; Table 11.4).

11.3.2.3 Activation Laboratories (2009)

The sample preparation package requested by RRR was package RX1. This required that the sample be crushed to 90% passing 10 mesh (2 mm), from which a 250 g riffle split was taken. The split was pulverized to 95% passing 105 µm mesh (Table 11.2).

For gold analysis, a 30 g sample was analyzed by fire assay with an AAS finish (lab method code 1A2). If samples exceeded the 5,000 ppb (5 ppm) upper detection limit, a second 30 g sample was taken from the pulp and re-analyzed by fire assay but with a gravimetric finish (lab method code 1A3; Table 11.3).

For silver analysis, a 0.5 g sample was analyzed for through an AR partial extraction. The sample is digested at 95°C, then diluted and analyzed as part of a multi-element suite with an ICP-OES finish (lab code 1E3). Samples that exceeded the 100 ppm upper detection limit for Ag were re-analyzed. A new 30 g sample was taken from the pulp and subjected to fire assay with a gravimetric finish (lab code 1A3-Ag; Table 11.4).

11.3.2.4 ALS (2011 – 2013)

RRR reverted to ALS labs in 2011 and used the same preparation and analytical packages that were originally applied in 2005 and 2006.

Thus, the sample was logged in the ALS tracking system, weighed, dried, and finely crushed to better than 70% passing a 2 mm (9 mesh) screen. A split of up to 250 g was taken using a riffle splitter and pulverized to better than 85% passing a 75 µm (200 mesh) screen (lab method code PREP-31; Table 11.2).

For gold analysis, a 30 g sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents, as required, inquarted with gold-free silver and then cupelled to yield a precious metal bead. The bead was digested using AR, and the cooled solution was diluted with demineralized water, and analyzed by AAS against matrix-matched standards (lab method code Au-AA23; Table 11.3).

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

For silver analysis, a 0.25 g sample underwent decomposition by four-acid digest and was analyzed with an ICP-AES finish (lab method code ME-MS61). Samples that exceeded the upper detection limit of 100 ppm for Ag were re-analyzed. A 0.4 g sample was taken from the pulp, decomposed using a four-acid digest, and analyzed with ICP-AES (lab method code Ag-OG62; Table 11.4).

RRR changed the method of silver analysis in 2012. The decomposition was changed to an AR digestion for both regular and over-limit samples. A prepared sample (0.50 g) was digested with AR for 45 minutes in a graphite heating block. After cooling, the resulting solution was diluted with deionized water, mixed, and analyzed by ICP-AES (lab method codes ME-ICP41). Samples that exceeded the upper detection limit for Ag of 100 ppm were re-analyzed. Overlimit samples were similarly subjected to an AR digest and analyzed by ICP-AES, but with a higher detection limit (lab method code Ag-OG46; Table 11.4).

11.3.3 New Gold (2013 – 2017)

11.3.3.1 ALS (2013 - 2017)

New Gold modified the sample preparation procedure used by RRR at ALS. The sample was logged in the tracking system, weighed, dried, and finely crushed to better than 90% passing a 2 mm (9 mesh) screen. A split of up to 1,000 g was taken and pulverized to better than 90% passing a 105 µm (150 mesh) screen. ALS sample preparation method codes applied were: LOG-21, DRY-21, CRU- 32, SPL-22Y, and PUL-35n (Table 11.2).

Gold analysis methods were also modified by New Gold, with a larger sample size being used.

A 50 g sample was fused with a mixture of lead oxide, sodium carbonate, borax, silica and other reagents, as required, inquarted with gold-free silver and then cupelled to yield a precious metal bead. The bead was digested using AR, and the cooled solution was diluted with demineralized water, and analyzed by AAS against matrix-matched standards (lab method Au-AA24; Table 11.3).

Samples grading over 10 g/t Au were analyzed by gravimetric methods (ALS method code Au-GRA22). A 50 g sample was also selected and subjected to fire assay, but with a gravimetric finish (lab method Au-GRA22; Table 11.3).

New Gold continued to use the same methods for silver analysis that RRR switched to in 2012. Thus, A prepared sample (0.50 g) was digested with AR for 45 minutes in a graphite heating block. After cooling, the resulting solution was diluted with deionized water, mixed and analyzed by ICP-AES (lab method codes ME-ICP41). Samples that exceeded the upper detection limit for Ag of 100 ppm were re-analyzed. Overlimit samples were similarly subjected to an AR digest and analyzed by ICP-AES, but with a higher detection limit (lab method code Ag-OG46; Table 11.4).

11.3.4 Bayfield Ventures Corp. (2010 – 2014)

Bayfield submitted the majority of their samples to Actlabs in Thunder Bay, ON for analysis. During 2010, some samples were submitted to TSL Laboratories Inc. (TSL) in Saskatoon, Saskatchewan (SK). There are no available data summarizing the preparation or analytical methods used at TSL. The analytical methods described below are summarized from Duke (2014).

11.3.4.1 Activation Laboratories (2010 – 2014)

The sampling preparation method utilized by Bayfield is not known.

For gold analysis a 30 g sample was submitted to fire assay with AAS finish. Samples that exceeded the detection limit of >5,000 ppb were re-assayed by gravimetric method. Duke (2014) notes that screened total metallic assays were also performed on samples that exceeded 5,000 ppb, but these data was not available.

Silver analysis was undertaken by AR digest with ICP finish. Fire assay - gravimetric analyses were performed on samples that exceeded the upper detection limit for silver of 100 ppm.

Table 11.2 Summary of sample preparation methods

Company	Lab	Method code	Crush	Split	Pulverize
Nuinsco (1994 - 2004)	ALS	-	>60% passing 10 mesh (1.7 mm)	200 - 250 g	>95% passing 150 mesh (106 µm)
RRR (2005 – 2013)	ALS (2005 – 2006)	PREP-31	>70% passing 9 mesh (2 mm)	250 g	>85% passing 200 mesh (75 µm)
	Accurassay (2006 - 2011)	ALP1	>90% passing 8 mesh (2.36 mm)	500 g	>90% passing 150 mesh (106 µm)
	Actlabs (2009 – 2010)	RX1	>90% passing 10 mesh (2.36 µm)	250 g	>95% passing ~150 mesh (105 µm)
	ALS (2011 - 2013)	PREP-31	>70% passing 9 mesh (2 mm)	250 g	>85% passing 200 mesh (75 µm)
New Gold	ALS (2013 - 2017)	LOG-21 DRY-21 CRU-32 SPL-22Y PUL-35n	>90% passing (2 mm)	1,000 g	>90% passing 150 mesh (106 µm)
Bayfield	Actlabs (2010 - 2014)	RX1	>90% passing 10 mesh (2.36 µm)	250 g	>95% passing ~150 mesh (105 µm)
	TSL (2010)	-	-	-	-

Note: Unavailable data are indicated by "-".
Source: AMC, using data provided by New Gold.

Table 11.3 Summary of analytical methods for gold

Company	Lab	Method code	Sample size	Generic method	Lower detection limit	Upper detection limit
Nuinsco (1994 – 2004)	ALS	-	30 g	FA-ICP	1 ppb	1,000 ppb
		-	30 g	FA-Gravimetric	0.03 g/t	no limit
RRR (2005 – 2013)	ALS (2005 – 2006)	Au-AA23	30 g	FA-AAS	0.005 ppm	10.0 ppm
		Au-GRA21	30 g	FA-Gravimetric	0.05 ppm	1,000 ppm
	Accurassay (2006 – 2011)	ALFA1	30 g	FA-AAS	5 ppb	30,000 ppb
		ALFA5	30 g	FA-Gravimetric	2 g/t	1,000 g/t
	Actlabs (2009 - 2010)	1A2	30 g	FA-AAS	5 ppb	5,000 ppb
		1A3	30 g	FA-Gravimetric	0.03 g/t	10,000 g/t
	ALS (2011 – 2013)	Au-AA23	30 g	FA-AAS	0.005 ppm	10.0 ppm
		Au-GRA21	30 g	FA-Gravimetric	0.05 ppm	1,000 ppm
New Gold (2013 – 2017)	ALS (2014 – 2017)	Au-AA24	50 g	FA-AAS	0.005 ppm	10.0 ppm
		Au-GRA22	50 g	FA-Gravimetric	0.05 ppm	1,000 ppm
	Actlabs (2014 - 2017)	1A2	30 g	FA-AAS	5 ppb	5,000 ppb
Bayfield (2010 – 2014)	Actlabs (2010 – 2014)	1A2	30 g	FA-AAS	5 ppb	5,000 ppb
		1A3-30	30 g	FA-Gravimetric	0.03 g/t	10,000 g/t
		1A4-1000	1,000 g	FA- Metallic Screen	0.03 g/t	10,000 g/t
	TSL (2010)	-	-	-	-	-

Notes: Unavailable data are indicated by "-".
Source: AMC, using data provided by New Gold.

Table 11.4 Summary of analytical methods for silver

Company	Lab	Method code	Sample size	Generic method	Lower detection limit	Upper detection limit
Nuinsco (1994 – 2004)	ALS	-	-	AR digest with AAS finish	0.2 ppm	34 ppm
		-	-	Multi acid digest with AAS finish	17 g/t	500 g/t
		-	30 g	FA - Gravimetric	3 g/t	no limit
RRR (2005 – 2013)	ALS (2005 – 2006)	ME-ICP41	0.5 g	AR digest with ICP-AES finish	0.2 ppm	100 ppm
		Ag-OG46	0.4 g	AR digest with ICP-AES finish	1 ppm	1,500 ppm
	Accurassay (2006 – 2011)	ALAR1	0.25 g	AR digest with AAS finish	1 ppm	100 ppm
		ALAR2	-	AR digest with AAS finish	1 ppm	1,500 ppm
	Actlabs (2009 - 2010)	1E3	0.5 g	AR digest with ICP-OES finish	0.2 ppm	100 ppm
		1A3-Ag	30 g	FA - Gravimetric	3 g/t	1,000 g/t
	ALS (2011 – 2012)	ME-MS61	0.25 g	4A digest with ICP-MS finish	0.01 ppm	100 ppm
		Ag-OG62	0.4 g	4A digest with ICP-AES finish	1 ppm	1,500 ppm
	ALS (2012 – 2013)	ME-ICP41	0.5 g	AR digest with ICP-AES finish	0.2 ppm	100 ppm
		Ag-OG46	0.4 g	AR digest with ICP-AES finish	1 ppm	1,500 ppm
New Gold (2013 – 2017)	ALS (2013 – 2017)	ME-ICP41	0.5 g	AR digest with ICP-AES finish	0.2 ppm	100 ppm
		Ag-OG46	0.4 g	AR digest with ICP-AES finish	1 ppm	1,500 ppm
	Actlabs (2014 - 2017)	1E-Ag	0.5 g	AR digest with ICP-OES finish	0.2 ppm	100 ppm
Bayfield (2010 – 2014)	Actlabs (2010 - 2014)	1E-Ag	0.5 g	AR digest with ICP-OES finish	0.2 ppm	100 ppm
		1A3-Ag	30 g	FA - Gravimetric	3 g/t	1,000 g/t
	TSL (2010)	-	-	-	-	-

Notes: Unavailable data are indicated by "-". AR=aqua regia.
Source: AMC, using data provided by New Gold.

11.4 Metallurgical testing

RRR used the SGS Canada Minerals Services Lakefield Laboratory in Lakefield, ON (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.

11.5 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.

11.6 Chain of custody and security

RRR, New Gold, and Bayfield have followed similar practices with respect to chain of custody and security protocols for core samples. Thus, once bagged samples were bundled into rice bags, they were either immediately driven by company personnel to Fort Frances, ON, or stored in a locked facility prior to transport. Commercial carriers (e.g., Gardewine North, Manitoulin) were utilized to transport samples from Fort Frances to the various laboratories, with samples secured in a locked trailer during transport. All companies placed a copy of the sample submission form inside the first rice bag of each shipment, enabling proper identification and cataloguing by the respective lab on receipt of samples. Descriptions of Nuinsco's chain of custody or security practices are not available.

11.7 QA/QC overview

This section addresses the collection procedures, results, and analysis of QA/QC data collected from 2005 to 2017 from available databases. No QA/QC data is available for the period of 1994 to 2004 when Nuinsco was carrying out their exploration. Drillhole data collected by Bayfield, including QC samples, has been assimilated into the New Gold database, but is addressed separately where appropriate.

Drilling programs completed on the Property between 2005 and 2017 included QA/QC monitoring programs which comprised insertion of certified reference materials (CRMs), blanks, and duplicates into the sample streams on a batch by batch basis. A summary of QA/QC samples included during this period is given in Table 11.5. Table 11.6 summarizes the insertion rates of QA/QC samples between 2005 and 2017.

Table 11.5 Rainy River QA/QC 2005 – 2017

Company	Year	Drill samples	CRMs [1]	Blanks	Field duplicates	Coarse duplicates	Pulp duplicates	Umpire checks
Nuinsco	1994 - 2004	22,371	0	0	0	0	0	0
RRR	2005 - 2013	403,584	9,167	2,956	1,323	0	0	0
New Gold	2014 - 2017	34,359	956	496	406	1,460	1,529	318[2]
Bayfield	2010 - 2014	31,967	1,080	2	0	0	8	226[2]
Total		**492,281**	**11,203**	**3,454**	**1,729**	**1,460**	**1,537**	**544**

Notes:
- Counts of individual samples. Multiple analysis types per sample possible (e.g., fire assay and gravimetric).
- Based on year drilled.
- [1] Gold CRMs only.
- [2] 318 pulps sent from ALS to Actlabs by New Gold for umpire checks as part of regular QC program. 318 pulp duplicates sent by New Gold to ALS as external check on Bayfield data from Actlabs.

Source: AMC, using data provided by New Gold.

Table 11.6 Rainy River QA/QC 2005 – 2017 insertion rates

Company	Year	CRMs	Blanks	Field duplicates	Coarse duplicates	Pulp duplicates	Umpire checks	QA/QC [1]
Nuinsco	1994 – 2004	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%	0.0%
RRR	2005 - 2013	2.3%	0.7%	0.3%	0.0%	0.0%	0.0%	3.3%
New Gold	2014 - 2017	2.8%	1.4%	1.2%	4.2%	4.4%	0.9%	14.9%
Bayfield	2010 - 2014	3.4%	0.0%	0.0%	0.0%	0%	0.7%[2]	4.1%
Overall	**2005 - 2017**	**2.3%**	**0.7%**	**0.4%**	**0.3%**	**0.3%**	**0.1%**	**4.0%**

Notes:
- Counts of individual samples. Multiple analysis types per sample possible (e.g., fire assay and gravimetric).
- Based on year drilled.
- Totals may not compute exactly due to rounding.
- [1] Insertion rate for CRM, blanks, and field duplicates combined.
- [2] Umpire checks are reported as a percentage of Bayfield samples but were submitted by New Gold in 2015.

Source: AMC, using data provided by New Gold.

11.7.1 Certified reference materials

11.7.1.1 Description

A total of 48 different CRMs for gold have been used in the Mineral Resource area between 2005 and 2017. CRMs were supplied by ROCKLABS Ltd. of New Zealand, Canadian Resource Laboratories Ltd. of Canada, Geostats Proprietary Ltd. of Australia, and Ore Research and Exploration Proprietary

Ltd. of Australia. The supplier of several additional CRMs is not known (AUQ1, HGS3, VMS1, and VMS3).

Gold CRMs have been used continuously since 2005 and comprised on average 2.2% of samples submitted to analytical laboratories. Insertion rates have varied, but generally fall between 1 in 20 to 1 in 30 samples. The lower reported insertion rates for this project appear to be from this insertion frequency not being maintained.

The insertion of CRMs for silver was started in 2011 and has continued since that time. Bayfield inserted silver CRMs into their sample stream only between 2010 and 2011.

Between 2005 and 2011, RRR used ROCKLABS CRMs exclusively, with analyses completed by ALS, Actlabs, and Accurassay. In 2011, RRR began using CRMs from Canadian Resource Laboratories in addition to those from ROCKLABS. ROCKLABS CRMs were phased out by the end of 2011. All analyses were completed at ALS from 2011 onwards. In 2014 New Gold began using CRMs from Geostats, in addition to those from Canadian Resource Labs, with the latter phased out by the end of 2014.

Bayfield used CRMs from Ore Research and Exploration (OREAS) exclusively between 2010 and 2014, which were analyzed at Actlabs and TSL. Table 11.7, Table 11.8, Table 11.9, and Table 11.10 summarize gold and silver CRMs by year, lab, and company.

Previous technical reports have presented QA/QC data for the various operators in varying levels of detail. These include Mackie et al. (2003), CCIC (2008), SRK (2008, 2009, 2011a, 2011b, and 2012), and Duke (2018).

QA/QC description and discussion presented herein is from AMC and was derived from the data provided by New Gold in October of 2019.

Table 11.7 Unique gold CRMs used in each year

Year	Company	# CRMs	CRMs used
2005	RRR	2	SH13, SL20
2006		5	SH13, SH24, Si54, SK21, SL20
2007		4	SH24, SH35, SK21, SK33
2008		4	SH24, SH35, SK33, SK43
2009		3	SH35, Si42, SK43
2010		5	Si42, SI54, SK43, SL46, SL51
2011		16	AUQ1, CDN-GS-1H, CDN-GS-1P5D, CDN-GS-5G, CDN-GS-5J, CDN-GS-P4A, HGS3, SE58, SF45, SH24, Si54, SK43, SL46, SL51, VMS1, VMS3
2012		11	CDN-GS-1H, CDN-GS-1J, CDN-GS-1P5D, CDN-GS-1P5E, CDN-GS-5G, CDN-GS-5J, CDN-GS-P3B, CDN-GS-P4A, SE58, SF45, Si54
2013		8	CDN- CM-26, CDN-GS-1J, CDN-GS-1L, CDN-GS-1P5E, CDN-GS-1P5K, CDN-GS-5H, CDN-GS-5J, CDN-GS-P3B
2014	New Gold.	8	CDN-CM-26, CDN-GS-1L, CDN-GS-1P5K, G308-7, G310-6, G311-8, G913-8, GBMS911-1
2015		4	G308-7, G310-6, G311-8, GBMS911-1
2016		4	G308-7, G310-6, G311-8, G913-8
2017		5	CDN-GS-5H, G308-7, G310-6, G311-8, G913-8
2010	Bayfield.	13	OREAS 15d, OREAS 15f, OREAS 15g, OREAS 15h, OREAS 2Pd, OREAS 4Pb, OREAS 52Pb, OREAS 53Pb, OREAS 5Pb, OREAS 60b, OREAS 61d, OREAS 6Pc, OREAS H3
2011		11	OREAS 15d, OREAS 15f, OREAS 15g, OREAS 15h, OREAS 16a, OREAS 52Pb, OREAS 5Pb, OREAS 60b, OREAS 61d, OREAS 6Pc, OREAS H3
2012		3	OREAS 15d, OREAS 15f, OREAS 16a
2013		4	OREAS 15d, OREAS 15f, OREAS 16a, OREAS 2Pd
2014		4	OREAS 15d, OREAS 15f, OREAS 15h, OREAS 16a

Source: AMC, using data provided by New Gold.

Table 11.8 Unique silver CRMs used in each year

Year	Company	#CRMS	CRMs used
2011	RRR	6	CDN-GS-5G, CDN-GS-5J, VMS1, VMS3
2012		2	CDN-GS-5G, CDN-GS-5J
2013		3	CDN-CM-26, CDN-GS-5H, CDN-GS-5J
2014	New Gold	3	CDN-CM-26, GBM310-9, GBMS911-1
2015		2	GBM310-9, GBMS911-1
2016		1	GBM310-9
2017		1	GBM310-9
2010	Bayfield	3	OREAS 60b, OREAS 61d, OREAS H3
2011		3	OREAS 60b, OREAS 61d, OREAS H3

Source: AMC, using data provided by New Gold.

Table 11.9 Timeline of Gold CRM analyses by year, lab, and company (2005 – 2017)

Company			Rainy River Resources									New Gold				
Laboratory			Accurassay													
			ALS				Actlabs		ALS							
CRM	**Expected value (Au ppm)**	**Stdv**	**2005**	**2006**	**2007**	**2008**	**2009**	**2010**	**2011**	**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**Total[1]**
G308-7	0.27	0.02										99	28	58	67	252
CDN-CM-26	0.372	0.024									73	61				134
CDN-GS-P3B	0.409	0.021								535	163					698
VMS1	0.429	0.032							18							18
CDN-GS-P4A	0.438	0.016							389	58						447
SE58	0.607	0.019							269	1						270
G310-6	0.65	0.04										84	27	48	66	225
SF45	0.848	0.028							249	1						250
VMS3	0.922	0.065							14							14
CDN-GS-1J	0.946	0.051								505	131					636
CDN-GS-1H	0.972	0.054							403	82						485
GBMS911-1	1.04	0.11										13	4			17
CDN-GS-1L	1.16	0.05									69	70				139
SH13	1.315	0.034	31	130												161
SH35	1.323	0.044			10	265	2									277
SH24	1.326	0.043		69	137	6			5							217
AUQ1	1.33	0.115							14							14
CDN-GS-1P5K	1.44	0.065									38	87				125
CDN-GS-1P5D	1.47	0.075							396	266						662
CDN-GS-1P5E	1.52	0.055								296	194					490
G311-8	1.57	0.08										59	17	45	75	196
Si42	1.761	0.054					316	350								666
Si54	1.78	0.034		1				392	261	1						655
CDN-GS-5H	3.88	0.14									78				1	79
HGS3	4.009	0.25							17							18
SK33	4.041	0.103			56	167										223
SK21	4.048	0.091		69	71											140
SK43	4.086	0.093				66	287	173	19							545

Company			Rainy River Resources									New Gold				
Laboratory			Accurassay													
			ALS				Actlabs		ALS							
CRM	Expected value (Au ppm)	Stdv	2005	2006	2007	2008	2009	2010	2011	2012	2013	2014	2015	2016	2017	Total[1]
CDN-GS-5G	4.77	0.2							259	2						261
G913-8	4.87	0.16										3		14	30	47
CDN-GS-5J	4.96	0.21							51	610	168					829
SL46	5.867	0.17						473	50							523
SL51	5.909	0.136						3	253							256
SL20	5.911	0.176	33	122												155
								Bayfield Ventures Corp.								
								Actlabs								
								TSL								
OREAS 4Pb	0.049	0.0025						10								10
OREAS 5Pb	0.098	0.003						19	62							81
OREAS 52Pb	0.307	0.019						25	1							26
OREAS 15f	0.334	0.016						17	86	72	23	4				202
OREAS 15g	0.527	0.023						11	69							80
OREAS 2Pd	0.885	0.03						15			1					16
OREAS 53Pb	0.623	0.021						16								16
OREAS 15h	1.019	0.025						7	34			1				42
OREAS 6Pc	1.52	0.07						14	1							15
OREAS 15d	1.559	0.042						13	98	61	20	9				201
OREAS 16a	1.81	0.06							37	68	18	10				133
OREAS H3	2	0.08						31	96							127
OREAS 60b	2.57	0.11						38	39							77
OREAS 61d	4.76	0.14						24	30							54

Notes:
- [1] Counts of individual samples. Multiple analysis types per sample possible (e.g. fire assay and gravimetric).
- Based on year drilled.

Source: AMC, using data provided by New Gold.

Table 11.10 Silver CRM analyses by year, lab, and company (2010 – 2017)

Company			Bayfield		RRR			New Gold				
Laboratory			**Actlabs**		**ALS**							
CRM	**Expected value (Ag ppm)**	**Stdv**	**2010**	**2011**		**2012**	**2013**	**2014**	**2015**	**2016**	**2017**	**Total [1]**
CDN-CM-26	2.5	0					73	61				134
GBM310-9	3.1	0.2						31	7	14	34	86
OREAS H3	4.95	0.3	14	89								103
OREAS 60b	4.96	0.31	15	27								42
OREAS 61d	9.27	0.48	9	19								28
GBMS911-1	11.9	1						13	4			17
VMS1	15.4	1			18							18
VMS3	31	1			14							14
CDN-GS-5H	50.4	1.35					78				1	79
CDN-GS-5J	72.5	2.4			43	592	168					803
CDN-GS-5G	101.8	3.5			217							217

Notes:

- [1] Counts of individual samples. Multiple analysis types per sample possible (e.g. fire assay and gravimetric).
- Individual analyses with Au values but no value for Ag (for CRMs certified for both Au and Ag) were excluded from these counts.
- Based on year drilled.

Source: AMC, using data provided by New Gold.

11.7.1.2 AMC discussion

CRMs are inserted to check the analytical accuracy of the lab. AMC recommends an insertion rate of at least 5% of the total samples assayed. CRMs should be monitored on a batch by batch basis and remedial action taken immediately if required. For each economic mineral, there should be at least three CRMs with values:

- At the approximate cut-off grade (COG) of the deposit.
- At the approximate expected grade of the deposit.
- At a higher grade.

The average grade for the open pit area Mineral Resource is approximately 1.0 g/t Au and ~3.5 g/t Ag at a 0.5 g/t gold equivalent (AuEq) COG. The average grade of the underground area Mineral Resource is approximately 3.0 g/t Au and 8.5 g/t Ag at a 2.0 g/t AuEq COG. The average grade of the low-grade stockpile is approximately 0.35 g/t Au and 2.5 g/t Ag.

CDN-GS-P4A (0.438 ppm Au), G310-6 (0.65 ppm Au), CDN-GS-1H (0.972 ppm Au) and CDN-GS-1L (1.16 ppm Au) cover the approximate grade of the open pit area. CRMs SH13 (1.315 ppm Au), SH35 (1.323 ppm Au), G311-8 (1.57 ppm Au), OREAS 16a (1.81 ppm Au), SK43 (4.086 ppm Au), and G913-8 (4.87 ppm Au) cover the approximate grades of the underground area as well as higher grade samples. CRM G308-7 (0.27 ppm Au) covers the approximate grade of the low-grade stockpile. CRM GBM310-9 (3.1 ppm Ag) covers the approximate silver grade of the open pit area Mineral Resource.

AMC generated and reviewed all CRM charts with specific emphasis on the control charts that demonstrated performance over the entire time span of data collection, including differing CRM manufacturer and assay lab. The following four control charts highlight common patterns in the CRMs including a) a positively biased CRM, b) a negatively biased CRM, c) an acceptably performing

CRM and d) the contrast between the performance of Accurassay and ALS for the same standard. Control charts are for gold, the primary economic element.

Table 11.11 lists the CRMs and discussed the reason they were selected for the control charts.

Table 11.11 CRMs selected for control charts

CRM	Au value (ppm)	No. CRMS	Years	CRM manufacturer	Analytical lab	Notes	Results
CDN-GS-P4A	0.438	447	2011 – 2012	CDN Resource Labs	ALS	Approximate open pit Au COG	Positive bias
G310-6	0.65	225	2014 – 2017	Geostats	ALS	Approximate open pit Au COG	Negative bias
CDN-GS-1H	0.972	485	2011 – 2012	CDN Resource Labs	ALS	Approximate average Au grade of open pit	Well performing CRM
Si54	1.78	655	2010 – 2011	ROCKLABS	Accurassay, ALS	Approximate underground Au COG	Shows the different performance of same CRM between labs.

AMC recommends re-assaying assay batches where two consecutive CRMs occur outside two standard deviations, or one CRM occurs outside three standard deviations of the expected value described on the CRM certificate. Results for gold and silver CRMs used in the QA/QC program are presented in Table 11.12 and Table 11.13.

Control charts are used to monitor the analytical performance of an individual CRM over time. Control lines are also plotted on the chart for the expected value of the CRM, two standard deviations above and below the expected value, and three standard deviations above and below the expected value. CRM assay results are plotted in order of analysis. These charts will show analytical drift and bias should they occur. Control charts for the selected CRMs listed in Table 11.11 are shown in Figure 11.1 to Figure 11.4.

AMC considers a <5% failure rate acceptable for an individual CRM. While several CRMs have not met this criterion, AMC notes that current performance of CRMs used by New Gold is acceptable.

Table 11.12 Rainy River gold CRM results (2005 – 2017)

CRM	Expected Au value (ppm)	Stdv	Years used	Analytical lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
OREAS 4Pb	0.049	0.0025	2010	TSL	10	1	2	20%
OREAS 5Pb	0.098	0.003	2010 – 2011	Actlabs	81	6	1	1%
G308-7	0.27	0.02	2014 – 2017	ALS	252	0	0	0%
OREAS 52Pb	0.307	0.019	2010	Actlabs, TSL	26	1	0	0%
OREAS 15f	0.334	0.016	2010 – 2014	Actlabs	202	7	1	1%
CDN-CM-26	0.372	0.024	2013 – 204	ALS	134	8	1	1%
CDN-GS-P3B	0.409	0.021	2012 – 2013	ALS	698	11	0	0%
VMS1	0.429	0.032	2011	ALS	18	4	1	6%
CDN-GS-P4A	0.438	0.016	2011 – 2012	ALS	447	32	1	0%
OREAS 15g	0.527	0.023	2010 – 2011	Actlabs	80	0	0	0%
SE58	0.607	0.019	2011	ALS	270	8	10	4%
OREAS 53Pb	0.623	0.021	2010	Actlabs, TSL	16	5	1	6%
G310-6	0.65	0.04	2014 – 2017	ALS	225	1	0	0%
SF45	0.848	0.028	2011	ALS	250	2	1	0%
OREAS 2Pd	0.885	0.03	2010	Actlabs, TSL	16	5	5	31%
VMS3	0.922	0.065	2011	ALS	14	1	0	0%
CDN-GS-1J	0.946	0.051	2012 – 2013	ALS	636	52	0	0%
CDN-GS-1H	0.972	0.054	2011 – 2012	ALS	485	20	1	0%
OREAS 15h	1.019	0.025	2010 – 2011	Actlabs	41	13	6	15%
GBMS911-1	1.04	0.11	2014 – 2015	ALS	17	0	2	12%
CDN-GS-1L	1.16	0.05	2013 – 2014	ALS	139	2	0	0%
SH13	1.315	0.034	2005 – 2006	Accurassay, ALS	161	17	2	1%
SH35	1.323	0.044	2007 – 2009	Accurassay	277	39	59	21%
SH24	1.326	0.043	2006 – 2008, 2011	Accurassay, ALS	217	26	40	18%
AUQ1	1.33	0.115	2011	ALS	14	0	0	0%
CDN-GS-1P5K	1.44	0.065	2013 – 2014	ALS	125	5	0	0%
CDN-GS-1P5D	1.47	0.075	2011 – 2012	ALS	662	46	0	0%
CDN-GS-1P5E	1.52	0.055	2012 – 2013	ALS	490	49	1	0%
OREAS 6Pc	1.52	0.07	2010 – 2011	Actlabs, TSL	15	0	1	7%
OREAS 15d	1.559	0.042	2010 – 2014	Actlabs	200	33	24	12%
G311-8	1.57	0.08	2014 – 2017	ALS	196	1	0	0%
Si42	1.761	0.054	2009 – 2010	Accurassay, Actlabs	666	93	94	14%
Si54	1.78	0.034	2010 – 2011	Accurassay, ALS	655	96	241	37%
OREAS 16a	1.81	0.06	2011 – 2014	Actlabs	131	15	7	5%
OREAS H3	2	0.08	2010 – 2011	Actlabs	127	21	13	10%
OREAS 60b	2.57	0.11	2010 – 2011	Actlabs	77	0	4	5%
CDN-GS-5H	3.88	0.14	2013	ALS	79	2	0	0%
HGS3	4.009	0.25	2011	ALS	17	1	0	0%
SK33	4.041	0.103	2007 – 2008	Accurassay	223	39	99	44%
SK21	4.048	0.091	2006 – 2007	Accurassay, ALS	140	15	46	33%
SK43	4.086	0.093	2008 – 2011	Accurassay, Actlabs, ALS	452	62	48	11%
OREAS 61d	4.76	0.14	2010 – 2011	Actlabs	40	0	1	3%

CRM	Expected Au value (ppm)	Stdv	Years used	Analytical lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
CDN-GS-5G	4.77	0.2	2011 – 2012	ALS	261	15	0	0%
G913-8	4.87	0.16	2014, 2016 – 2017	ALS	47	1	0	0%
CDN-GS-5J	4.96	0.21	2011 – 2013	ALS	829	16	0	0%
SL46	5.867	0.17	2010 – 2011	Accurassay, Actlabs, ALS	513	141	147	29%
SL51	5.909	0.136	2010 – 2011	Accurassay, ALS	256	11	5	2%
SL20	5.911	0.176	2005 – 2006	ALS	155	4	3	2%
Total					**11,082**	**927**	**868**	**8%**

Note: Sorted by CRM expected value.
- Fire assay analyses only (gravimetric analyses removed).
- Where a CRM is used by two labs these are at different periods in time, see Figure 11.4.

Source: AMC, using data provided by New Gold.

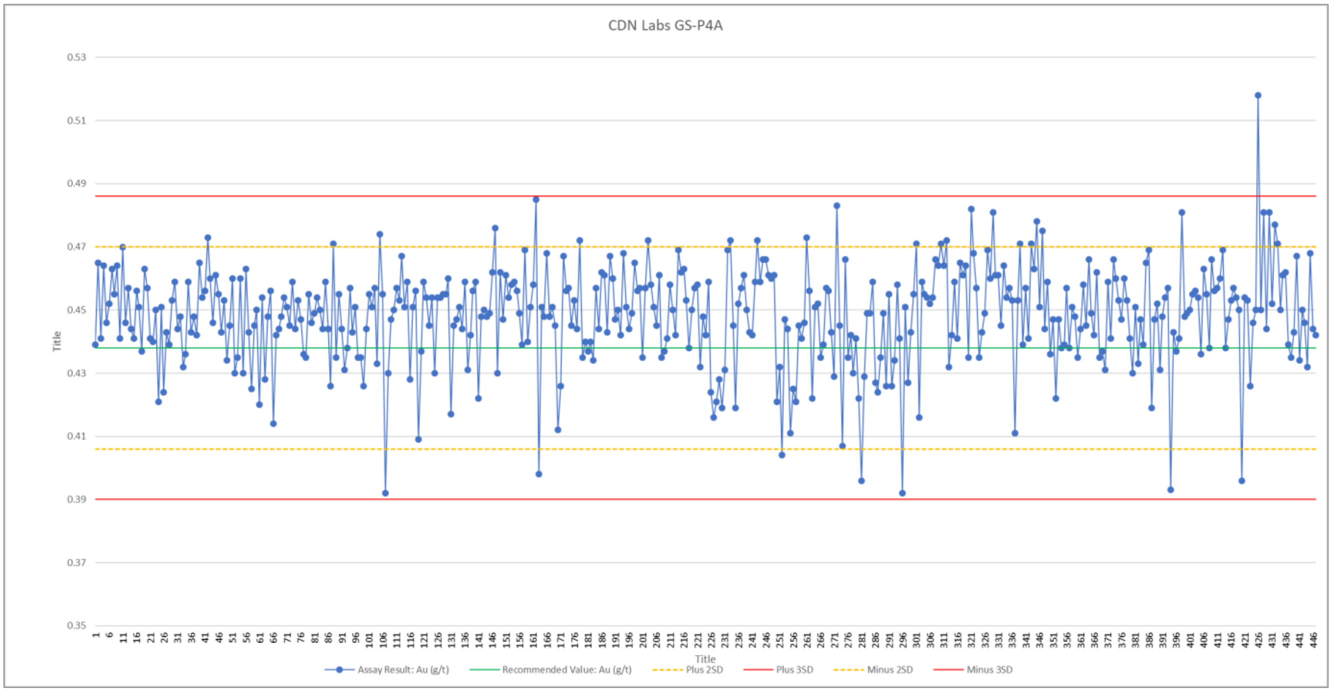
Table 11.13 Rainy River silver CRM results

CRM	Expected Ag value (ppm)	Stdv	Years used	Analytical lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
CDN-CM-26	2.5		2013 – 2014	ALS	134	0	0	0
GBM310-9	3.1	0.2	2014 – 2017	ALS	86	0	0	0%
OREAS H3	4.95	0.3	2010 – 2011	Actlabs	127	17	7	6%
OREAS 60b	4.96	0.31	2010 – 2011	Actlabs	77	7	8	10%
OREAS 61d	9.27	0.48	2010 – 2011	Actlabs	40	2	13	33%
GBMS911-1	11.9	1	2014 – 2015	ALS	17	1		0%
VMS1	15.4	1	2011	ALS	18	2		0%
VMS3	31	1	2011	ALS	14		14	100%
CDN-GS-5H	50.4	1.35	2013	ALS	79	14		0%
CDN-GS-5J	72.5	2.4	2011 – 2013	ALS	829	119	1	0%
CDN-GS-5G	101.8	3.5	2011 – 2012	ALS	262	11	17	6%
Grand total					**1,549**	**173**	**61**	**4%**

Note: Sorted by CRM expected value.
- Fire assay analyses only (gravimetric analyses removed).
- CRM CDN-CM-26 only indicated for Ag analyses. No standard deviation given on certificate. Excluded from total fail calculations.
- CRM VMS3 performed entirely below its expected value as listed in the New Gold database. The certificate was not available for this CRM and the expected value could not be confirmed.
- Individual analyses with Au values but no value for Ag (for CRMs certified for both Au and Ag) were excluded from these counts.

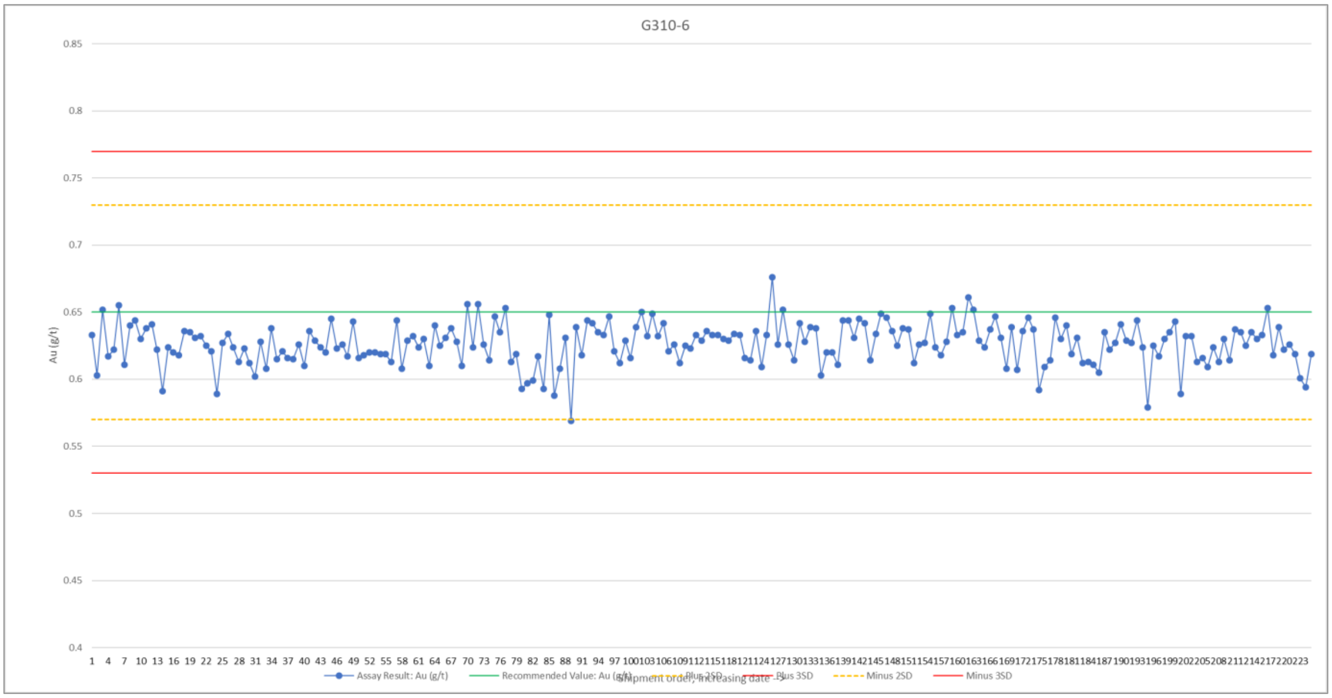
Source: AMC, using data provided by New Gold.

Figure 11.1 Gold CRM CDN-GS-P4A



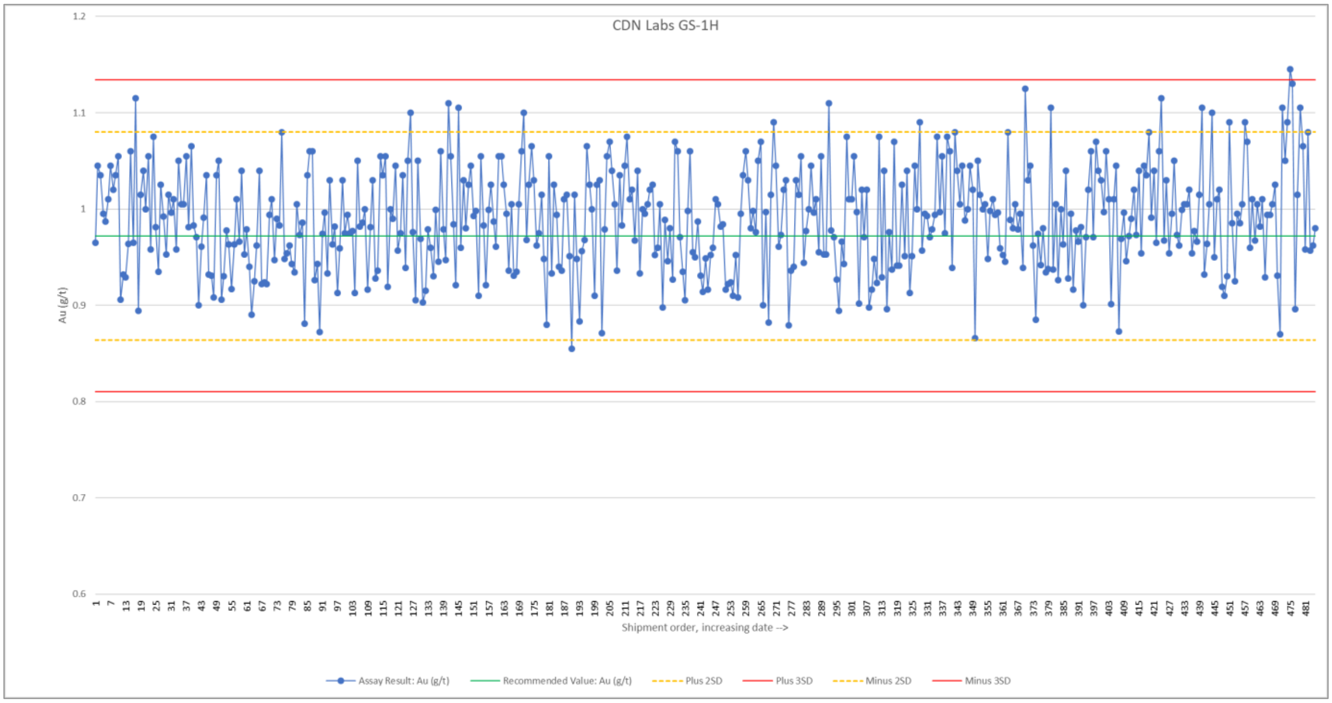
Note: All CRMs analyzed by fire assay with ICP-AAS at ALS Labs. AMC notes a positive bias with this CRM.
Source: AMC, using data provided by New Gold.

Figure 11.2 Gold CRM G310-6



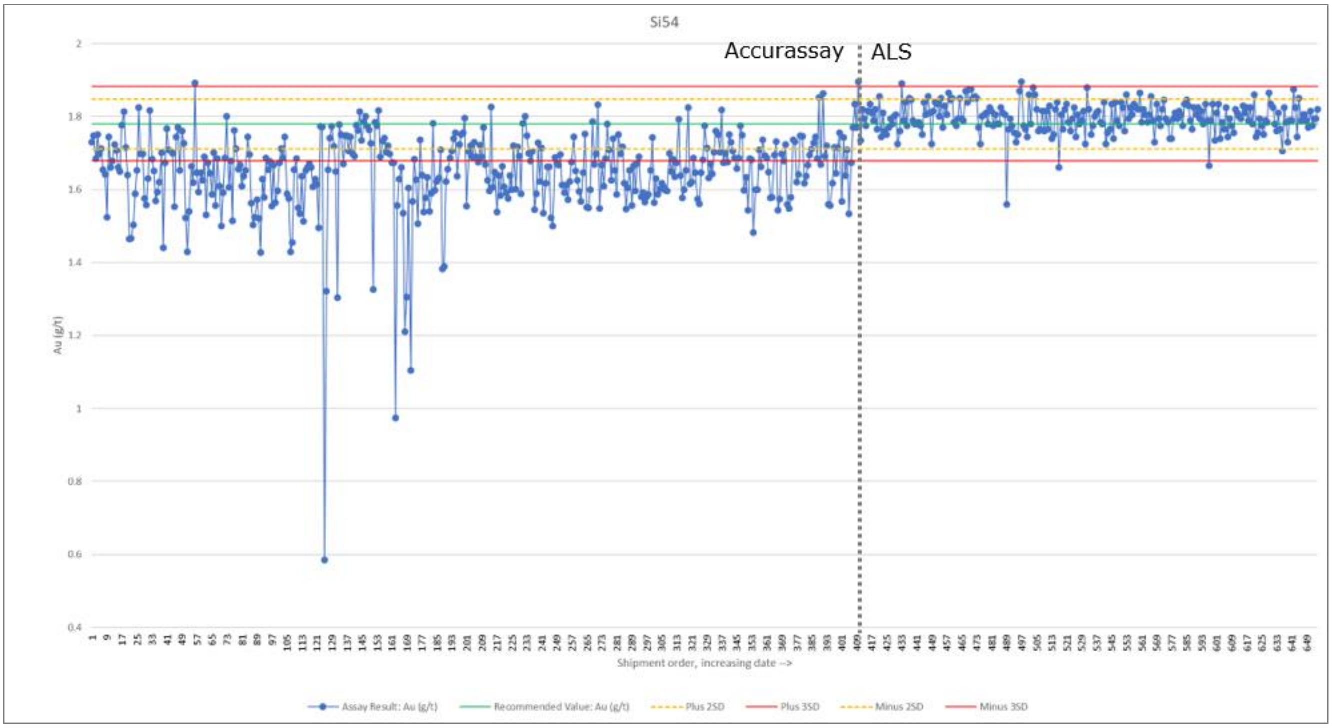
Note: All CRMs analyzed by fire assay with ICP-AAS at ALS Labs. AMC notes a negative bias with this CRM.
Source: AMC, using data provided by New Gold.

Figure 11.3 Gold CRM CDN-GS-1H



Note: All CRMs analyzed by fire assay with ICP-AAS at ALS Labs.
Source: AMC, using data provided by New Gold.

Figure 11.4 Gold CRM Si54



Note: All CRMs analyzed by fire assay with ICP-AAS at ALS and Accurassay, as indicated.
Source: AMC, using data provided by New Gold.

AMC considers the number of different CRMs used historically on the Property to be excessive. It is preferable to limit the number of different CRMs used on a Project to ensure that each CRM has enough results to enable meaningful analysis. In AMCs experience, between three and five different CRMs are usually adequate to monitor lab performance. It is realized that this is exaggerated by the multiple owners and AMC notes that New Gold is using an appropriate number of CRMs.

ROCKLABS CRMs were analyzed between 2005 and 2011 by ALS, Actlabs, and Accurassay, with results demonstrating differing levels of performance by individual laboratories. Specifically, those CRMs analyzed at Accurassay show lower precision and accuracy, with numerous 3SD fails with a dominant, systematic negative bias. This issue was identified and addressed by RRR in 2011. Several suites of samples were re-analyzed at ALS labs, confirming the low bias of 6 – 7% towards Accurassay over the grade range of 0.2 to 2 ppm. Several ROCKLABS standards, however, show a negative bias across labs (e.g., SH24), and across methods (fire assay versus gravimetric, e.g. SK43).

Although negative bias was introduced into the database during this interval of poor lab performance, no adjustment has been made to the original analyses beyond that of re-assaying selected samples. These re-assayed samples were not used in the Mineral Resource. This low bias should lead to a more conservative Mineral Resource estimate (New Gold 2015).

Overall, CRMs supplied by Canadian Resource Labs, all which were analyzed by ALS, performed well. Two of these (CDN-GS-5J (4.96 ppm Au) and CDN-CM-26 (0.372 ppm Au)) show some drift in their earliest analyses, from positively biased results, to those spread more equally around the expected value. Additionally, the two low-grade standards in use between 2011 and 2013 (CDN-GS-P3B, 0.409 ppm Au and CDN-GS-P4A, 0.438 ppm Au)) both yielded data with minor but systematic high biases.

Geostats standards, introduced in 2014 and used exclusively since 2015, have all been analyzed at ALS Laboratories. Both low-grade standards (G208-7, 0.27 ppm Au) and G310-6, 0.65 ppm Au)) both show systematic low biases. New Gold has determined that this negative bias is an issue with the CRM and not a measure of lab performance, based on data collected from other projects and analyzed at different labs.

Geostats CRMs generally have a very low rate of failure when measured against the reported standard deviation on the CRM certificate. The performance of these CRMs suggests that these reported standard deviations are too large, and thus do not accurately track the performance of the analytical lab.

Performance of OREAS standards, in use exclusively by Bayfield, was acceptable. However, due to the large number of unique CRMs in use, many of these CRMs yield small datasets, and their performance over time cannot be evaluated.

Several CRMs were analyzed by different laboratories using methods with differing detection limits, triggering overlimit analyses by gravimetric methods at an individual lab (e.g., SK43: Accurassay upper detection limit: 30 ppm, Actlabs upper detection limit: 5 ppm). Data generated by these differing sample streams cannot be compared, and a CRM's performance over time cannot be properly tracked.

The current highest-grade standard in use (G913-8, 4.87 ppm Au) is not certified for gravimetric analysis and does not have a value sufficiently high to trigger this overlimit analysis at ALS (10 ppm Au). Thus, any sample that exceeds this current analytical upper detection limit does not have a concomitant CRM that monitors this grade range or method. The only certified gravimetric CRMs for gold (CDN-GS-5J and CDN-GS-5H, used between 2011 and 2013) both have values around 5 ppm Au, far below the value required to initiate gravimetric analysis.

It is noted that a high percentage of samples within the mineralized domains have values below 1 ppm Au. For example, within the ODM 100 domain, 91% of the samples are below 1 ppm Au. Since 2005, this grade range has not been satisfactorily monitored by CRMs. Between 2005 and 2010, the lowest grade standard in use was SH13 (1.315 ppm Au). Between 2011 and 2013, low-grade CRMs were introduced (SE58, 0.607 ppm Au; CDN-GS-P4A, 0.438 ppm Au; CDN-GS-P3B, 0.409 ppm Au; and CDN-CM-26, 0.372 ppm Au). Of these, SE58 shows a slight but persistent negative bias while CDN-GS-P4A and CDN-GS-P3B both show systematic positive biases. CDN-CM-26 also shows notable drift over time from positively biased samples towards the expected Au value. Finally, both low-grade standards from Geostats, introduced in 2014 (G308-7, 0.27 ppm Au; G310-6, 0.65 ppm Au), both yield systematic negatively biased values.

AMC notes that only 1% of the gold analyses are greater than 10 g/t gold.

11.7.1.3 AMC recommendations for CRMs

AMC recommends the following for any future programs:

- Ensure that the insertion rate of one CRM every 20 samples (5%) is achieved.
- An additional CRM that covers the COG of the open pit should be acquired.
- If a CRM shows consistent bias at multiple laboratories, this issue needs to be understood and resolved or a new CRM should be obtained. If it isn't practical to discard a large CRM inventory, then internal calculation of the CRM expected value and standard deviation would be appropriate. The rationale should be documented.
- Recalculate standard deviations for Geostats samples based on New Gold data and use these as a measure of performance instead of those indicated on the certificate. These should be used in concert with a recalculated expected value.
- Continue to document warnings, failures, and most importantly any remedial action taken.
- AMC also recommends adding the HoleID to the QA/QC sample database as a cross check to ensure QA/QC samples relate to the dataset and the time period in question. AMC makes this recommendation to minimize future investigative work.

11.7.2 Blank samples

11.7.2.1 Description

Coarse blank samples were inserted into the sample stream of drill programs completed between 2005 and 2017. Available data suggests that Nuinsco (1994 – 2004) and Bayfield (2010 – 2014) did not regularly include blank samples in their drill programs.

Programs run by RRR between 2005 and 2011 used coarse blank material sourced locally from the Black Hawk Stock, an intrusive body outcropping on the Property. Analyses of this material suggests it is at least locally anomalous with low levels of Au, and it was therefore changed to a marble garden stone from Quali-Grow Garden Products Inc. in 2011. The use of coarse marble blank was continued by New Gold to 2017, except for a brief interval in 2016, when coarse blank material was once again sourced from the Black Hawk Stock. New Gold returned to using a coarse marble in early 2017.

Insertion rates for blank materials have varied since 2005, ranging from one blank every 40 samples, to one blank inserted for every 60 samples. New Gold currently inserts a blank every 50 samples.

A total of 3,454 blank samples have been included with drillhole samples from 2005 to 2017. This represents between 0.7% to 1.4% of total samples for RRR and New Gold respectively.

11.7.2.2 AMC discussion

Coarse blanks test for contamination during both sample preparation and assaying. Blanks should be inserted in each batch sent to the lab. In AMC's opinion, 80% of coarse blanks should be less than three times the detection limit. AMC considers New Gold's fail criteria of ten times the lower analytical detection limit to be too high, although it acknowledges that it is probably not a matter of material concern to the Mineral Resource estimates.

Table 11.14 shows the assay results from blank material for drilling completed between 2005 and 2017, and the results of AMC's pass / fail parameters. Results from Accurassay and ALS are presented separately due to the differing performance of these labs during the period of interest.

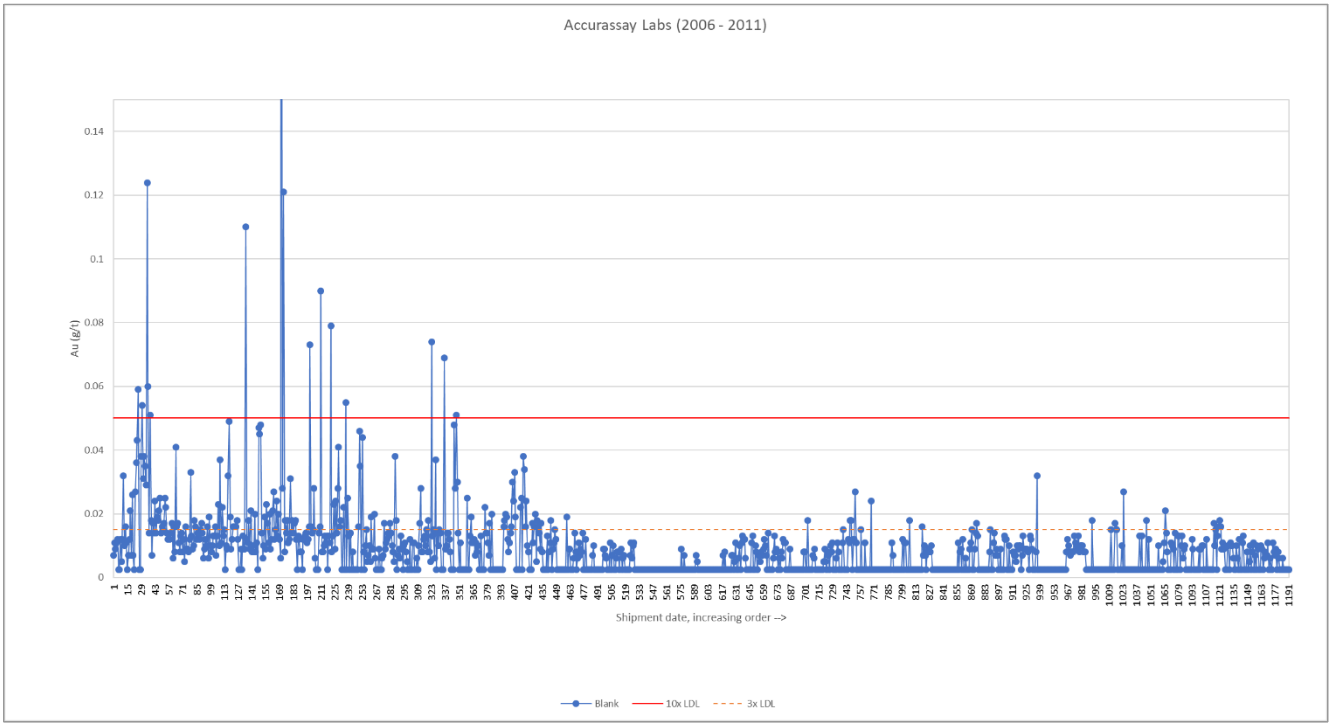
Table 11.14 Rainy River blanks

<table>
<tr><th rowspan="2">Company</th><th rowspan="2">Year</th><th rowspan="2">Lab</th><th colspan="3">Coarse blank (Black Hawk stock)</th><th colspan="3">Coarse marble</th></tr>
<tr><th>Number of samples</th><th># New Gold fail (>10x LDL)</th><th>#AMC fail (>3x LDL)</th><th>Number of samples</th><th># New Gold fail (>10x LDL)</th><th>#AMC fail (>3x LDL)</th></tr>
<tr><td rowspan="14">RRR</td><td rowspan="2">2005</td><td>ACC</td><td>16</td><td>0</td><td>2</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">ALS</td><td>68</td><td>6</td><td>21</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">2006</td><td>187</td><td>4</td><td>40</td><td></td><td></td><td></td></tr>
<tr><td rowspan="4">ACC</td><td>19</td><td>3</td><td>13</td><td></td><td></td><td></td></tr>
<tr><td>2007</td><td>145</td><td>5</td><td>62</td><td></td><td></td><td></td></tr>
<tr><td>2008</td><td>225</td><td>7</td><td>55</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">2009</td><td>252</td><td>0</td><td>18</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">ACT</td><td>10</td><td>0</td><td>0</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">2010</td><td>81</td><td>0</td><td>2</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">ACC</td><td>506</td><td>0</td><td>26</td><td></td><td></td><td></td></tr>
<tr><td rowspan="2">2011</td><td>28</td><td>0</td><td>0</td><td></td><td></td><td></td></tr>
<tr><td rowspan="7">ALS</td><td>131</td><td>2</td><td>14</td><td>560</td><td>1</td><td>6</td></tr>
<tr><td>2012</td><td></td><td></td><td></td><td>527</td><td>0</td><td>1</td></tr>
<tr><td>2013</td><td></td><td></td><td></td><td>200</td><td>0</td><td>1</td></tr>
<tr><td rowspan="4">New Gold</td><td>2014</td><td></td><td></td><td></td><td>175</td><td>0</td><td>0</td></tr>
<tr><td>2015</td><td></td><td></td><td></td><td>30</td><td>0</td><td>0</td></tr>
<tr><td>2016</td><td>40</td><td>2</td><td>5</td><td>61</td><td>0</td><td>0</td></tr>
<tr><td>2017</td><td>4</td><td>0</td><td>0</td><td>186</td><td>2</td><td>4</td></tr>
<tr><td colspan="3">Total</td><td>1712</td><td>29</td><td>258 (15%)</td><td>1739</td><td>3</td><td>12 (0.7%)</td></tr>
</table>

Notes: Year refers to year drilled. Blank samples run by Bayfield not included. Lower detection limit (LDL) is 0.005 ppm Au for fire assay analysis for all listed labs.
Source: AMC, using data provided by New Gold.

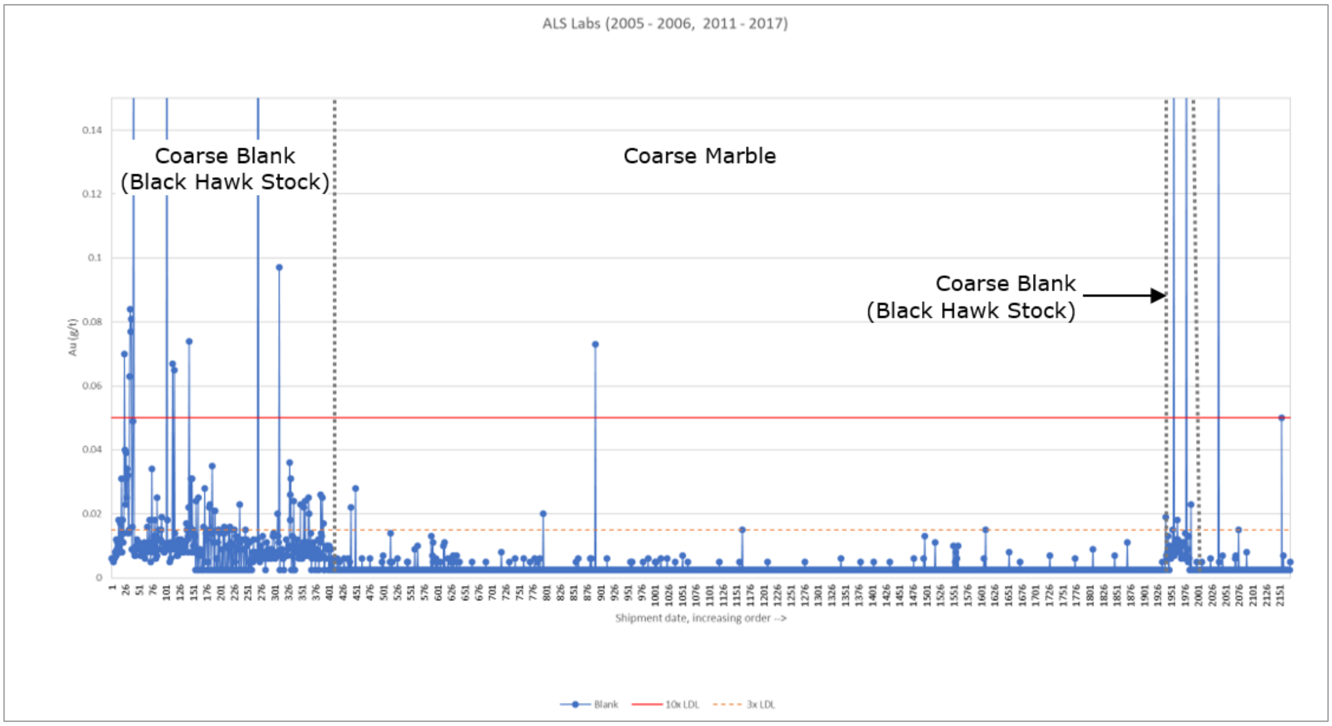
A total of ~15% of coarse blank samples from the Black Hawk Stock reported greater than three times the lower detection limit of 0.005 ppm Au. Analyses from Accurassay and ALS yield similar high percentages of failures, indicating local anomalous gold within the source material. The coarse marble samples performed notably better, with only 0.7% of these samples reporting above three times the detection limit. Figure 11.5 and Figure 11.6 present blank material performance at Accurassay and ALS.

Figure 11.5 Coarse blank performance chart, Accurassay (2006 – 2011)



Note: All data are from coarse blank material sourced from the Black Hawk Stock.
Source: AMC, using data provided by New Gold.

Figure 11.6 Coarse blank and coarse marble performance chart, ALS (2005 – 2006, 2011 – 2017)



Source: AMC, using data provided by New Gold.

11.7.2.3 AMC recommendations

AMC recommends the following for any future programs:

- Continued used of coarse marble.
- Increase blank insertion rate to 5% of the total sample stream.
- Send any potential new blank material to an analytical lab to ensure the material is below analytical detection with respect to any minerals of economic interest.
- Lower the blank failure limit to 3x detection limit.

11.7.3 Duplicate samples

11.7.3.1 Description

The number and type of duplicate samples has varied over time and by operator. Available data indicates that Nuinsco did not submit any samples for duplicate analysis. Similarly, RRR did not regularly submit duplicate samples for analyses before 2010. At that time, they began submitting quarter core (field duplicates) samples. Seventy-five field duplicate samples were analyzed at Accurassay, and an additional 1,248 field duplicates were analyzed at ALS between 2011 and 2013.

RRR did not routinely analyse pulp duplicates as part of their QA/QC program. However, a suite of pulp duplicates was sent to ALS in 2011 as part of RRR's investigation into Accurassay's poor lab performance. This suite of samples was also rerun at Accurassay as part of the investigation and are flagged as pulp duplicates in the New Gold database. Because these data are part of a lab performance investigation, and not part of their regular QA/QC program, they are not presented in this report. No coarse duplicates were analyzed by RRR.

New Gold continued to collect field duplicates, with an additional 406 samples collected between 2014 and 2017. New Gold also routinely analyses both pulp and coarse duplicates as part of their QA/QC program. Between 2014 and 2017, 1,529 pulp duplicates and 1,460 coarse duplicates have been analyzed by New Gold.

New Gold also routinely sends pulp duplicates to an external lab as an umpire check. Between 2014 and 2017 544 pulp duplicates have been sent to Actlabs in Thunder Bay for secondary analyses.

Available data indicates that Bayfield did not routinely analyse duplicate samples as part of their QC program. However, 226 samples from Bayfield were sent to ALS by New Gold in 2015, in order to investigate the Bayfield dataset. Table 11.15 summarizes the duplicate analyses available for the Mineral Resource area.

Table 11.15 Rainy River duplicate analyses

Company	Laboratory	Year	Field duplicates	Coarse duplicates	Pulp duplicates	Umpire samples
Bayfield	TSL	2010	0	0	6	0
	Actlabs		0	0	2	0
RRR	Accurassay		66	0	0	0
		2011	9	0	0	0
	ALS		657	0	0	0
		2012	407	0	0	0
		2013	184	0	0	0
New Gold		2014	184	875	892	0
		2015	25	159	181	226 [1]
		2016	155	245	262	318 [2]
		2017	42	181	194	0
		Total	**1,729**	**1,460**	**1,537**	**544**

Notes:
[1] Bayfield samples originally assayed at Actlabs and sent to ALS by New Gold as an umpire check.
[2] New Gold samples originally assayed at ALS and sent to Actlabs as an umpire check.
Source: AMC, based on data provided by New Gold.

11.7.3.2 AMC discussion

Field duplicates monitor sampling variance, sample preparation and analytical variance, and geological variance. Coarse duplicates monitor sample preparation, analytical variance and geological variance and pulp duplicates monitor analytical precision including homogenization and pulverization quality.

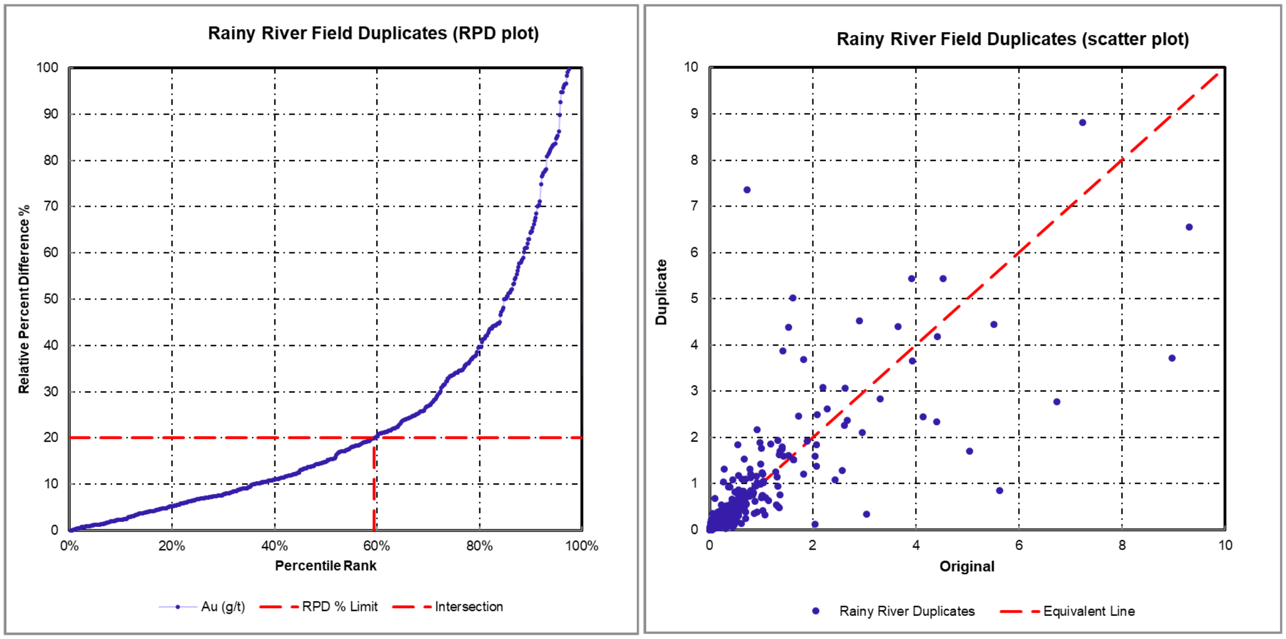
AMC recommends that duplicate samples be selected over the entire range of grades seen at the Project to ensure that the geological heterogeneity is understood. However, the majority of duplicate samples should be selected from zones of mineralization. Unmineralized or very low-grade samples should not form a significant portion of duplicate sample programs as analytical results approaching the stated limit of lower detection are commonly inaccurate, and do not provide a meaningful assessment of variance.

Duplicate data can be assessed using a variety of approaches. AMC typically assesses duplicate data using scatterplots and relative paired difference (RPD) plots. These plots measure the absolute difference between a sample and its duplicate. For field duplicates and coarse duplicates, it is desirable to achieve 80 to 85% of the pairs having less than 20% RPD between the original assay and check assay. For pulp duplicates, it is AMC's opinion that 80% pairs should be within 10% RPD (Stoker 2006). In these analyses, AMC excludes pairs with a mean of less than 15 times the lower limit of analytical detection (0.075 ppm Au; LDL = 0.005 ppm Au for fire assay for all relevant laboratories; Kaufman and Stoker 2009). Removing these low values ensures that there is no undue influence on the RPD plots due to the higher variance of grades expected near the lower detection limit, where precision becomes poorer (Long et al. 1997). AMC notes that a significant portion of the duplicate samples in this dataset (>50% for all duplicate types) are below this limit and are thus excluded from calculations.

RPD and scatter plots for field duplicates are presented in Figure 11.7. These plots show that only 59% of samples are within 20% RPD. Pairs show a weak positive bias towards the duplicate of ~2%. A single pair of high-grade outliers (482 ppm Au, 305 ppm Au) was removed from the calculations as this large absolute difference had a disproportionate effect on the bias calculation.

The proportion of duplicate samples with assay values within 20% RPD is less than desirable. This is most likely due to the combination of the heterogeneous nature of mineralization, as well as sampling variance.

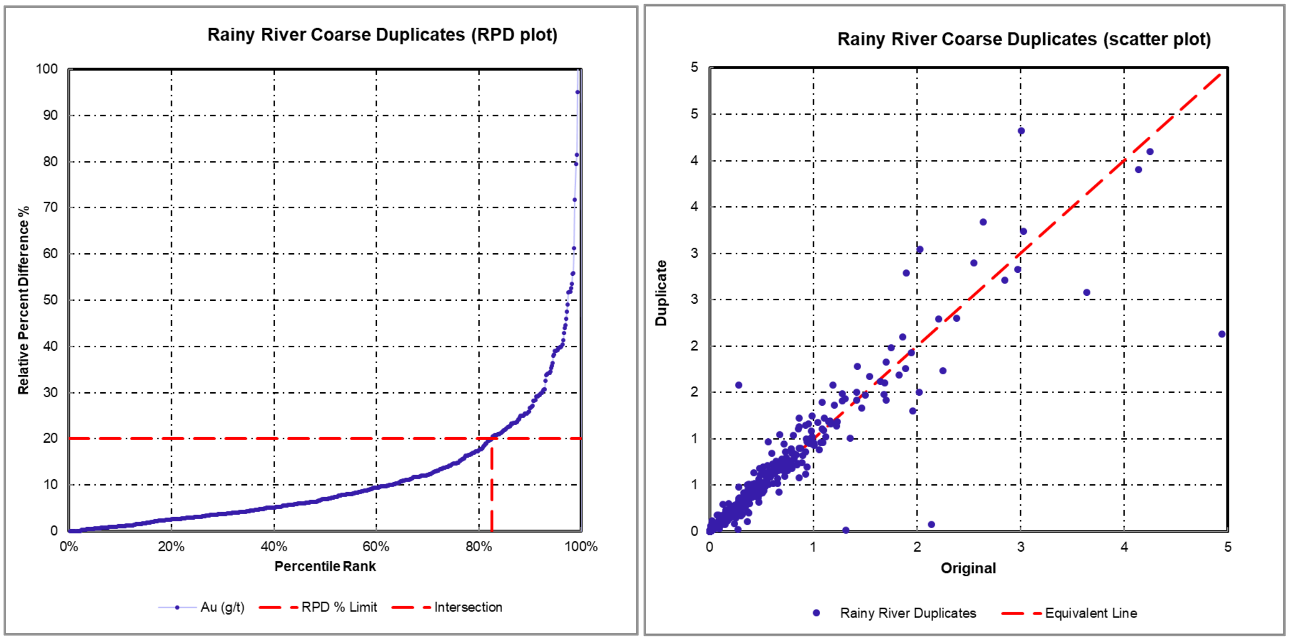
Figure 11.7 Rainy River field duplicate RPD and scatter plot



Note: Data from RRR and New Gold combined; ALS data only (1,653 pairs).
Source: AMC based on data from RRR and New Gold.

RPD and scatter plots for coarse duplicates are presented in Figure 11.8. These plots show that ~82% of samples are within 20% RPD, with a negative bias towards the duplicate of ~12 %. This higher bias is strongly skewed by two duplicate pairs that have an original high-grade analysis (> 50 ppm Au) paired with a much lower grade duplicate. The removal of these two pairs reduces the bias to <1%. The high variance seen in these two samples is likely the result of geological variance.

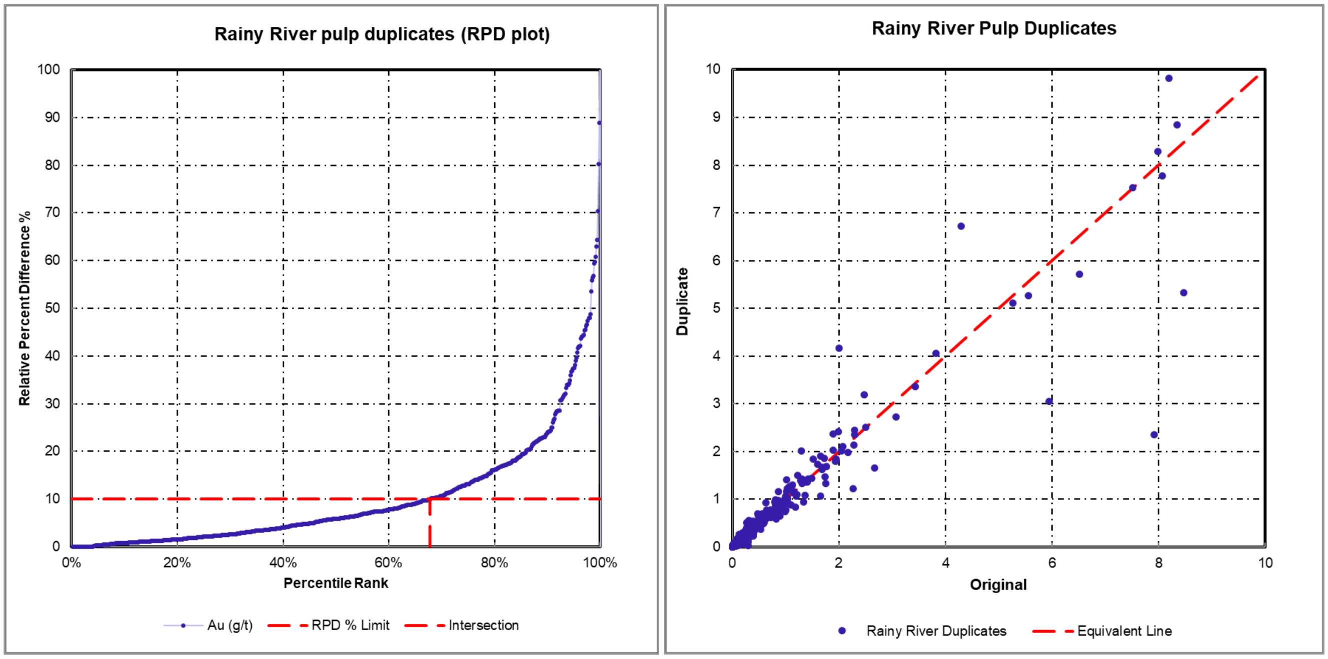
Figure 11.8 Rainy River coarse duplicate RPD and scatter plot



Source: AMC based on data from RRR and New Gold.

RPD and scatter plots for pulp duplicates are presented in Figure 11.9. These plots show that ~68% of samples are within 10% RPD. If the RPD limit is raised to 15%, 78% of the data falls within this range. Again, these results are most likely due to geological variance.

Figure 11.9 Rainy River pulp duplicate RPD and scatter plot



Source: AMC based on data from RRR and New Gold.

11.7.3.3 AMC recommendations

AMC recommends the following:

- Continue the insertion of field, coarse, and pulp duplicates into the sample stream.
- Further investigative work be completed to assess pulp duplicate performance. Such as, applying screen fire assay analyses to a subset of samples in order to better understand the size distribution of gold particles.

11.7.4 Umpire samples

11.7.4.1 Description

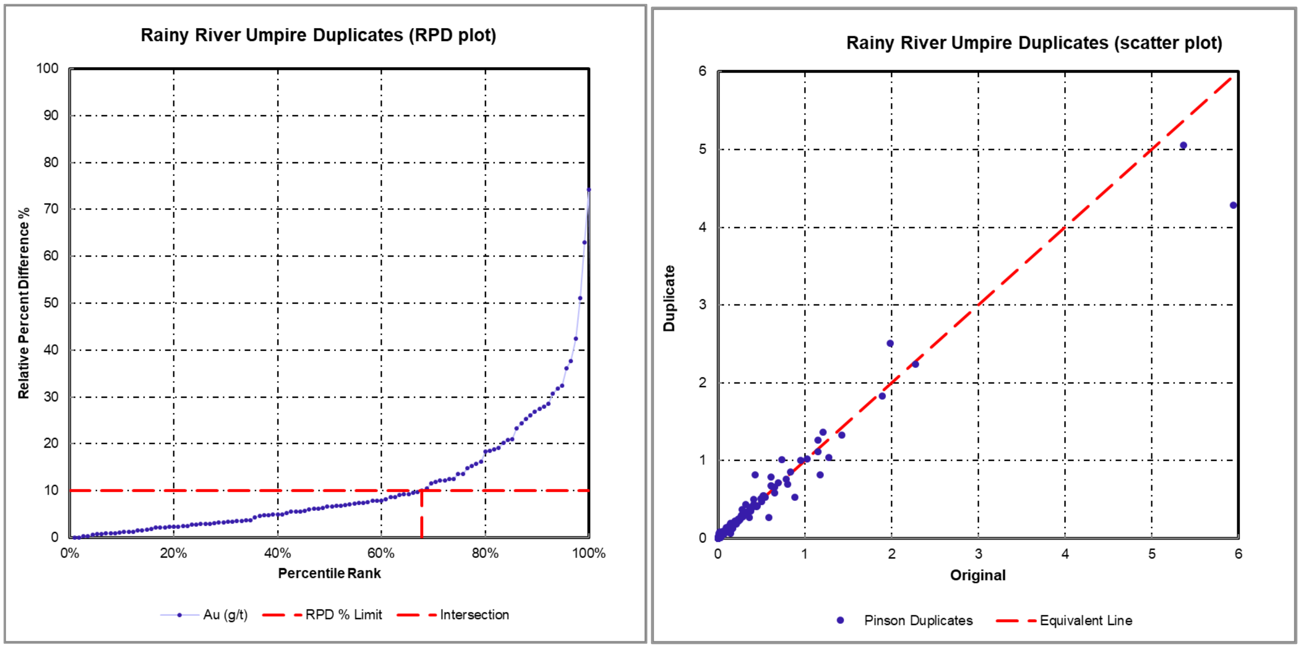
Umpire samples were not regularly submitted as part of the QA/QC programs run by Nuinsco, RRR, or Bayfield. However, New Gold regularly submits such samples, starting in 2014. To date, 318 samples have been sent to Actlabs for umpire testing. Additionally, a subset of samples acquired by Bayfield was also sent by New Gold for umpire testing. Two-hundred and twenty-six (226) samples, originally assayed at Actlabs, were sent to ALS for umpire testing in 2015. Both sample suites appear to have been randomly selected.

11.7.4.2 AMC discussion

Umpire lab duplicates are pulp samples sent to a separate lab to assess the accuracy of the primary lab (assuming the accuracy of the umpire lab). Umpire duplicates measure analytical variance and pulp sub-sampling variance. Umpire duplicates should comprise around 5% of all assays. In AMC's opinion, 80% of umpire duplicates should be within 10% RPD.

RPD and scatter plots for umpire samples submitted as part of New Gold's QC program are shown in Figure 11.10. Sixty-eight percent of samples are within 10% RPD. A slight negative bias of 2% towards the duplicate samples can be reduced to <1% with the removal of a single high-grade outlier with a large absolute difference. Similarly, the suite of umpire samples from the Bayfield dataset (not shown) also yield a comparable 68% pairs within 10% RPD, with no significant bias. Both umpire datasets are comparable to the values seen for pulp duplicates (68% within 10% RPD), further indicating these smaller than expected populations within the accepted RPD limits are primarily the result of geological variance.

Figure 11.10 Rainy River Umpire data RPD and scatter plot – New Gold data



Notes: Original assay lab: ALS. Umpire lab: Actlabs. New Gold data only.
Source: AMC based on data from RRR and New Gold.

11.7.4.3 AMC recommendations

Increase umpire sample submission rate to around 5% of all samples.

11.8 Conclusions

Drilling programs completed on the Property between 2005 and 2017 have included QA/QC monitoring programs which have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams. AMC has compiled and reviewed the available QA/QC data.

In general, the QA/QC sample insertion rates used at Rainy River fall below the general accepted industry standards.

The performance of several CRMs currently in use by New Gold show good precision but poor accuracy. New Gold believes that this is an issue with the CRMs and not a function of lab performance.

The CRMs used by previous operators have not adequately covered the COG grade of the open pit Mineral Resource.

Overall performance of one of the assay labs was inadequate. This was recognized and remedial action taken.

Between 2005 and 2011, blank material was sourced from a local granite Analytical results indicate that this material contained low levels of gold. Blank material was switched to a coarse marble in 2011, and results from this date onwards are considered acceptable and suggest that no systematic contamination occurred throughout the analytical process.

Duplicate sample results show suboptimal performance which is a probable result of the heterogenous nature of the mineralization.

Umpire samples show no bias and indicate that the primary lab currently in use is performing accurately.

Despite the concerns highlighted above, the QP does not consider these issues to be material to the global, long term Mineral Resource estimate. There is however no guarantee that there are no material impacts on the local scale. Overall, the QP considers the assay database to be acceptable for Mineral Resource estimation.

12 Data verification

12.1 Site verification

On 11 April 2018, QP, Ms Dinara Nussipakynova, P.Geo., visited the property to undertake the following verification steps:

1 Review data collection, handling, and manipulation procedures, including:
- Sample collection.
- Sample preparation for grade control.
- Sample storage.
- QA/QC procedures.
- Geological interpretation.

2 Inspect the core shed.

3 Review selected logged and assayed drill core intersections. Table 12.1 lists the inspected drillholes.

Table 12.1 Drillholes inspected on site

Drillhole 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

12.2 Drillhole and assay verification

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 (5.6%). 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.
- Cross-checking a sample set of the database with the original assay results revealed no errors.

12.3 Reconciliation

An important measure of performance at any producing mine is reconciliation of the resource block model to the final mill production figures adjusted for stockpiles as necessary.

There are many reconciliation studies carried out on site on a regular basis. The comparisons selected here attempt to show the performance of the resource model and production by way of the grade control information to the mill figure. Due to large stockpile movements a direct comparison to the mill is not possible, so the notion of Declared Ore Mined (DOM) is used, where:

- DOM = Mill + (closing stocks – opening stocks).

The reconciliation compares the grade control model (GC model) in Table 12.2, the Mineral Resource block model (resource model) and DOM, and all figures are for the full year of 2019. This is only shown here for gold.

Table 12.2 Reconciliation for GC model to DOM

	Tonnes	Au g/t	Au ounces
DOM	8,999,850	1.04	302,355
GC model	9,291,066	0.90	269,722
DOM / GC	97%	116%	112%

In Table 12.2 the GC model tonnes are within 3% of the DOM tonnes and the DOM grade is 16% higher than the GC model, resulting in 12% more gold ounces in DOM.

Table 12.3 Reconciliation for resource model to DOM

	Tonnes	Au g/t	Au ounces
DOM	8,999,850	1.04	302,355
Resource model	7,542,845	1.03	249,839
DOM / resource model	119%	101%	121%

In Table 12.3 the resource model underestimates tonnes by 19% compared to DOM with the grades being within 1%, resulting in 21% more gold ounces in DOM.

The comparison between the GC model and the resource block model is shown in Table 12.4. This demonstrates that the GC model is showing more tonnes (+23%) at a lower grade (-12%), for a gain of 8% more ounces over the resource block model.

Table 12.4 Reconciliation for GC model and resource model

	Tonnes	Au g/t	Au ounces
GC model	9,291,066	0.90	269,722
Resource model	7,542,845	1.03	249,839
GC / res	123%	88%	108%

Note that the resource model used is a regularized block model and thus includes some dilution. All comparisons are at a 0.3 g/t Au cut-off as this is what site uses for grade control (although they are planning on moving to Au equivalent in 2020).

Reconciliation carried out by New Gold is detailed and thorough. It is carried out monthly and year to date figures are presented as tables. Despite some large swings which may be based on carrying large stockpiles and a great deal of rehandling of material, this demonstrates that the block model and thus the Mineral Resources are valid and robust. This in turn validates the data underpinning the model and is by association a good verification of the work done.

AMC recommends that the reconciliation should be done on a rolling 3-month basis and presented graphically, thus reviewing trends and potentially reducing any impacts that come from the large stockpiles' movements in the mine.

In Section 15 there is a series of reconciliations to the diluted or mine planning resource model to which the reader is also referred.

12.4 Conclusion

In the opinion of the QP, 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 practice as defined in the CIM Exploration Best Practice Guidelines and CIM Estimation of Mineral Resource and Mineral Reserve Best Practice Guidelines. As such, the data are suitable for use in the estimation of Mineral Resources.

13 Mineral processing and metallurgical testing

13.1 Metallurgical testwork pre plant start-up

13.1.1 Introduction

Metallurgical testwork programs were conducted on Rainy River drillcore samples to support the development phases of the Rainy River project. These included the Preliminary Economic Assessment (PEA), the Feasibility Study, and the Updated Feasibility Study.

13.1.2 Metallurgical testwork supporting the PEA

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

- Mineralogy.
- Comminution testwork.
- Gravity separation testwork.
- Flotation testwork.
- Cyanide leach testwork of flotation concentrates.
- Whole ore cyanide leach testwork.

In 2012, SGS completed variability sample selection and testwork. Sample selection was guided by SGS geo-metallurgical modelling.

The primary process option that was tested was flotation followed by cyanide leaching of the flotation concentrate. Whole ore cyanide leach tests were also performed to provide a second viable process option.

The overall gold recovery for the leaching of the flotation concentrate option was 89%; with the flotation feed ground to a P_{80} of 150 µm and the flotation concentrate re-ground to a P_{80} of 15 µm.

The gold recovery for the whole ore leach option was approximately 91%, when ground to a P_{80} of 60 µm.

13.1.3 Metallurgical testwork supporting the feasibility study

Metallurgical testwork was performed from October 2011 to November 2012 on samples of the Main Pit. A gravity separation / cyanide leach flowsheet was selected as the preferred flowsheet for the Main Pit ore.

Metallurgical testwork was performed from November 2012 to November 2013 on samples of the Intrepid Zone. The objectives of this testwork program were to determine whether the Intrepid Zone material could be treated successfully using the same gravity separation / cyanide leach flowsheet which was selected for the Main Pit ore; and whether the Intrepid Zone ore would impact plant performance when blended in low tonnages with the Main Pit ore. The testwork program included comminution, gravity separation, cyanidation, carbon adsorption modelling, cyanide destruction, and solid-liquid separation tests.

13.1.4 Sample selection and compositing

13.1.4.1 Master composite sample – 2008 to 2011 testwork

Metallurgical samples were selected from drillcore and drillcore rejects to represent each of the mineralization zones 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 proportions indicated in Table 13.1. The composite consisted of 80% ODM ore, with the balance coming from the remaining zones.

Table 13.1 Master composite sample proportions

Zone composite	Zone composite proportions (%)	Total proportion (%)
CAP	2.0	
Z-433	12.0	20.0
HS	1.0	
NZ	5.0	
ODM-1	35.1	
ODM-2	3.5	80.0
ODM-3	31.4	
ODM-4	9.9	
Master	**100.0**	**100.0**

Totals may not compute exactly due to rounding.

Additionally, composites were made from high-grade areas of the ODM-17 and Z-433 Zones. Two composites were made of each zone including ODM-17 composites of 4 g/t gold and 8 g/t gold, and Z-433 composites of 4 g/t gold and 8 g/t gold.

13.1.4.2 Composite samples for flowsheet confirmation

Three composite samples were selected in March 2012 to represent the major ore types in the deposit and the ore blends to be processed throughout the life-of-mine (LOM). These were:

- ODM Master composite.
- Initial Pit composite.
- Remaining life-of-mine (RLOM) composite.

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 processing ore.

Table 13.2 shows the percentages of each zone type used in each composite as well as the percentages of each zone selected for use in the final design criteria prepared by AMEC.

Table 13.2 Percentages by zone for testwork composites and design criteria

Ore Zone	Composite make-up (%)					
	Initial pit		RLOM		Overall pit	
	Composite	AMEC design	Composite	AMEC design	Composite	AMEC design
ODM	86	82	60	71	68	78
Z433	4	10	14	6	11	8
HS	0	5	6	7	4	6
NZ	4	0	5	0	5	0
CAP	5	0	15	12	12	5
Other	0	3	0	4	0	4

Note: Totals may not compute exactly due to rounding.

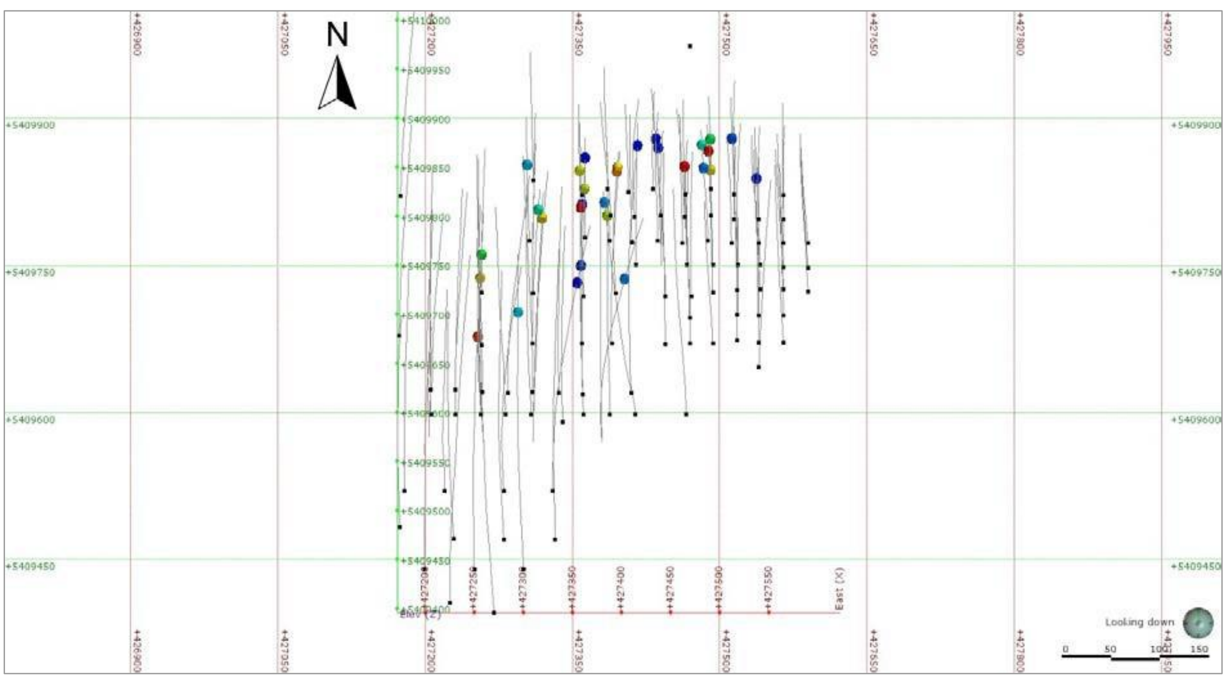
The percentages between the sample composites and the design criteria are similar; however, in the design criteria, the percentage of ODM is higher than the sample composite; and the NZ zone is absent.

13.1.4.3 Variability testwork sample selection

Sample variability testwork was performed, following flowsheet selection and development of the base test criteria. The variability testwork program included 162 comminution samples and 208 cyanide 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, drillhole 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.

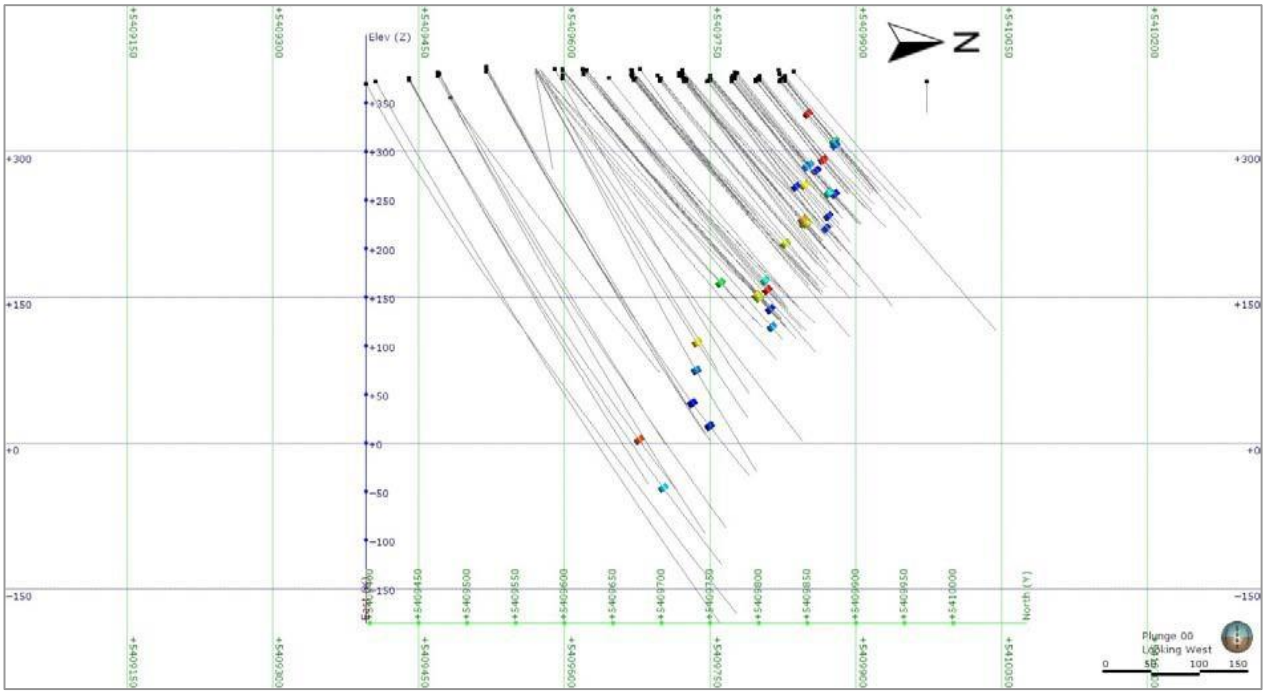
The borehole and sample locations in the Intrepid Zone are presented in Figure 13.1 and Figure 13.2 in plan view and cross-section respectively.

Figure 13.1 Plan view of drillhole and sample locations in the Intrepid Zone



Source: New Gold 2018.

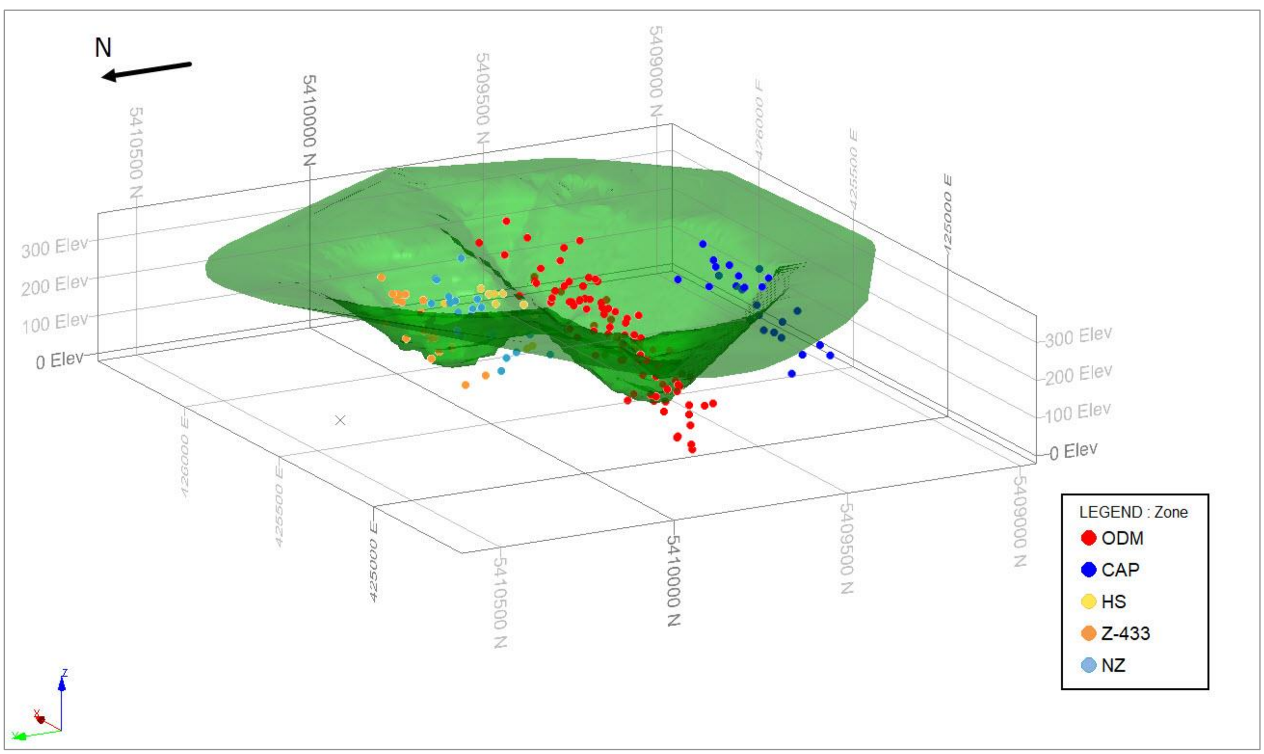
Figure 13.2 Location of Intrepid Zone samples (cross-section looking west)



Source: New Gold 2018.

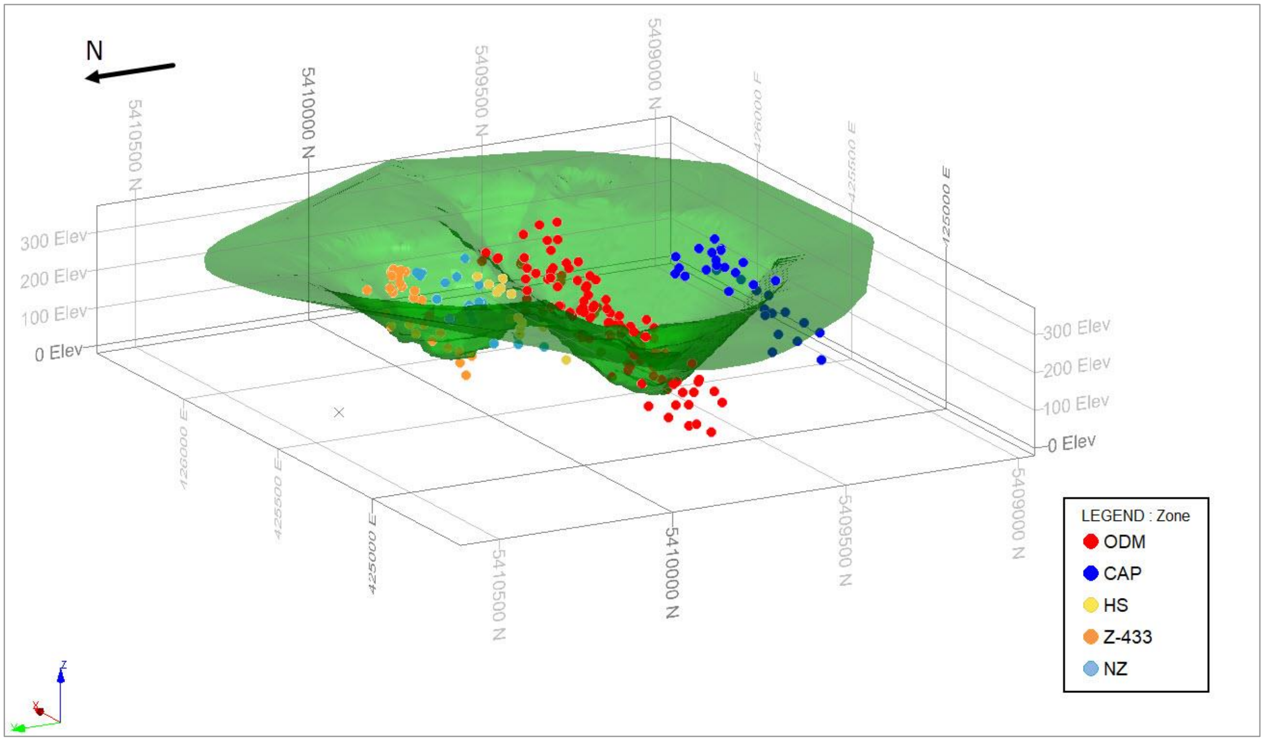
The sample locations for the variability testwork in the Main Pit are presented 3D in Figure 13.3 and Figure 13.4. These figures have been updated with the latest pit shells.

Figure 13.3 Sample locations for comminution variability testwork



Source: New Gold 2020.

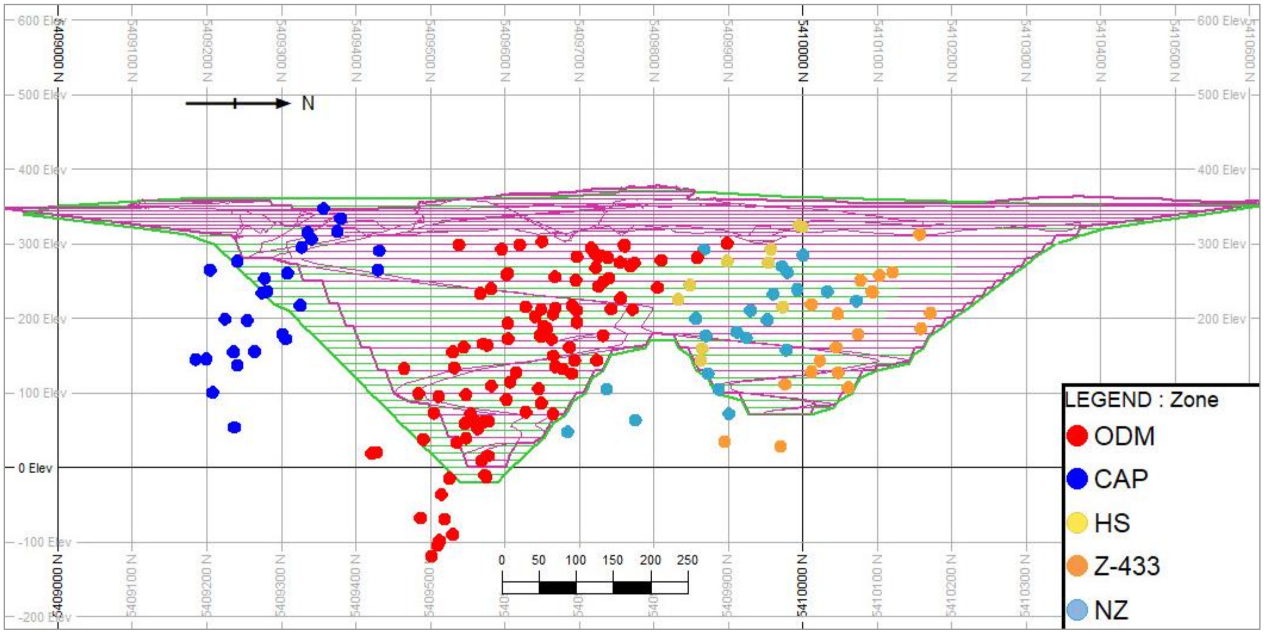
Figure 13.4 Sample locations for cyanide leaching variability testwork



Source: New Gold 2020.

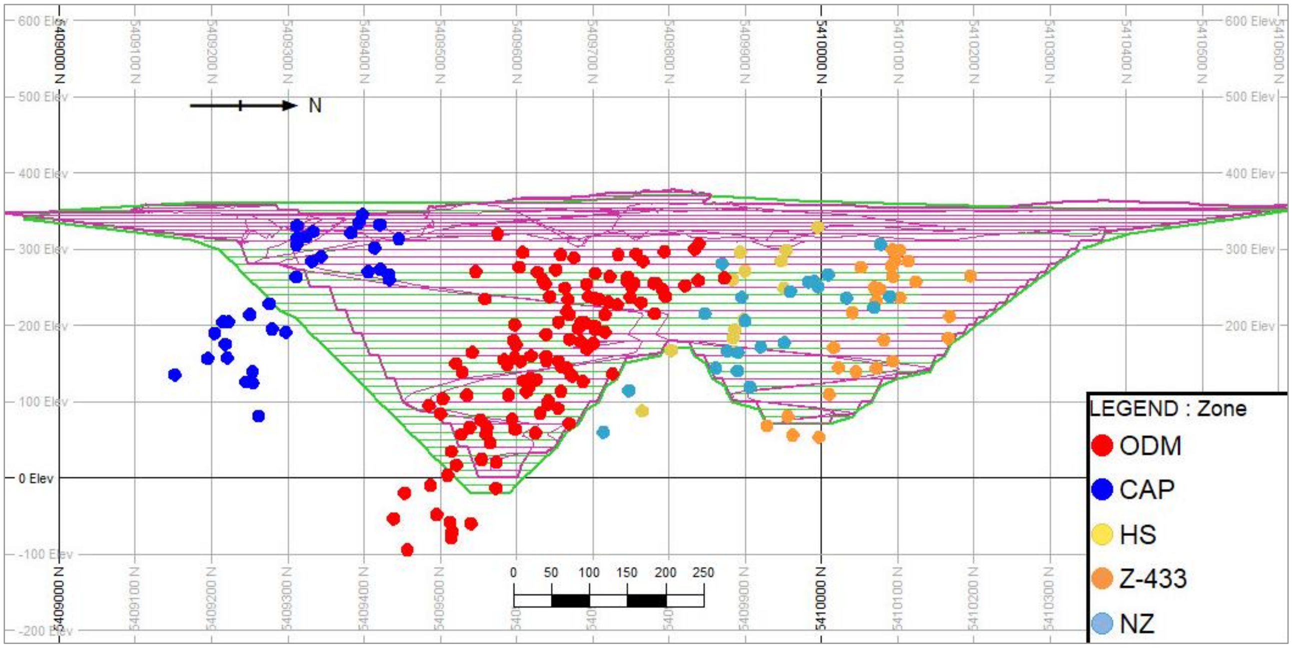
Some of the samples were located outside the proposed pit outline. This is due to a reduction in the size of the engineered pit from the December 2011 PEA to the current NI 43-101 report. Figure 13.5 and Figure 13.6 show the sample locations for variability comminution testing and variability leach testwork respectively with the latest pit shell outlines.

Figure 13.5 Sample locations for variability comminution testwork



Source: New Gold 2020.

Figure 13.6 Sample locations for variability leaching testwork



Note: AMC considers that the samples tested were representative of the Rainy River deposit and adequately cover the variability of the deposit.
Source: New Gold 2020.

13.1.5 Sample characterization

The head grades and major impurity elements for the Master Composites and variability samples are presented in Table 13.3.

Table 13.3 Head analyses for the composite and variability samples

Sample	Number of samples	Screen met. Au (g/t)	Direct Ag (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 LOM	-	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.

13.1.5.1 Mercury assays

Mercury assays were performed on two composite samples. These assays were completed on the feed, residue, loaded carbon and barren solution streams after undergoing leaching and gold adsorption. 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.

AMC does not consider mercury to be a major risk for the economic extraction of Rainy River ore types.

13.1.6 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 quartz-phyric rocks (OMD/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).
- 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 Zone and 433 Zone) with subsidiary zones (HS Zone and NZ Zone), 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 Z-433 Zone samples, one CAP Zone sample, one HS Zone sample, and one NZ Zone 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 Z-433 Zone composites to 9.5% in the CAP Zone 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 the composite samples at a grind size of 150 µm, except for the CAP Zone sample. Liberated and attached gold can be readily extracted with whole ore leaching at the 150 µm grind size.
- The CAP Zone composite contains gold particles present as locked inclusions in pyrite and non-opaque minerals and would require fine grinding to liberate the gold particles prior to leaching.
- The majority of the gold occurs as locked particles in sulphides and silicates minerals. Those composites with locked particles would require very fine grinding to liberate the gold particles prior to whole ore cyanide 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 Zone samples and one of the Z-433 Zone samples contained coarse gold.
- The Z-433 Zone 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.

13.1.7 Comminution testwork

A large comminution testwork program was conducted on the Rainy River composites, in support of the design of the crushing and grinding circuits.

- Crushing characteristics were determined by performing seven crusher work index (CWi) tests at each of three vendor laboratories, for a total of 21 tests. Tests were performed by Metso Minerals Canada Ltd. (Metso), SGS, and FLSmidth Minerals Ltd. (FLSmidth). Only seven of the tests performed at the Metso lab were selected by AMEC for use in the final design.
- A total of 160 bond work index (BWi) tests were performed at SGS, including 140 modified bond work index (ModBWi) tests. Twenty full bond ball mill work index (BBMWi) tests were performed to calibrate the ModBWi tests.
- Unconfined compressive strength (UCS) testwork was performed at Queen's University in Kingston, ON. Most of the UCS samples failed along foliation lines, and as such were not considered to be particularly reliable.
- The abrasion index testwork was performed at SGS.
- 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 the process design, unless otherwise noted.

13.1.7.1 Crusher work index testwork

Crusher work index tests were performed at three separate laboratories, including SGS, Metso, and FLSmidth.

The results are presented in Table 13.4.

Table 13.4 Crusher work index (CWi) test results

Lab	**SGS / Phillips (kWh/t)**					**Metso (kWh/t)**				**FLSmidth (kWh/t)**			
Zone	**ODM**	**Z-433 HS**		**NZCAP**		**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
80th 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 from Metso, which were midway between the results of SGS and FLSmidth. Metso's results were selected for use in the process design. The 80th percentile value of 25 kilowatt-hours per tonne (kWh/t) was selected for design purposes.

13.1.7.2 Unconfined compressive strength testwork

UCS tests were performed at Queen's University to determine the competency of the selected rock samples. Four ODM Zone samples, one Z-433 Zone sample, one HS Zone sample, and one CAP Zone 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 Zone samples. The values from all the tests ranged from 34.5 megapascal (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 most of the samples had low results due to failure along foliation lines, the results were deemed unsuitable for design purposes.

13.1.7.3 Bond ball mill work index testwork

BBMWi testing program consisted of 160 ModBWi tests 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 a standard lab ball mill. The test is run for a specific amount of time after which the feed and product size distributions are determined. The target P_{80} product screen size for the Rainy River samples was 75 µm (200 mesh). The ModBWi results were calibrated by comparing the ModBWi and standard BWi test results. The results of the tests are presented in Table 13.5.

Table 13.5 Results of BWi and ModBWi tests

Description Zone	ODM	BWi, 75 µm kWh/t				Intrepid
		Z-433	HS	NZ	CAP	
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
80th 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
80th 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%	-6.2	11.1	0.64	17.4

The ModBWi results for the ODM Zone, Z-433 Zone, and CAP Zone were within 5% of the BWi. The HS Zone sample had a slightly higher variance of 6.7% and the NZ Zone and Intrepid Zone samples had the highest variances at 11.1% and 17.4% of the BWi respectively. The ModBWi method was considered validated and the remainder of the variability test program was performed using the ModBWi procedure.

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.

In the tests on the Intrepid Zone samples, 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, which is 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 ore in the ODM Zone and NZ Zone 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 basis.

13.1.7.4 Bond abrasion index testwork

SGS performed twenty-four Bond abrasion index tests. Abrasion index data were used to calculate the wear material consumptions for estimating original process plant operating costs. 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 percentile and 90th percentile hardness in the data set. SGS considered the ore to be moderately abrasive when compared to SGS's database. The abrasion index value used in the design basis was 0.25.

The results are presented in Table 13.6.

Table 13.6 Bond abrasion index test results

Description	ODM	Z-433	Zone 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
80th percentile	0.26	0.33	0.38	0.11	0.19
Maximum	0.66	0.51	0.43	0.11	0.21

13.1.7.5 JK Drop Weight and SMC testwork

The SMC Test® (SMC test) program consisted of 13 JK drop weight tests (JK DW) and 175 SMC tests on samples of the Main Pit, and an additional two samples from the Intrepid Zone. The JK DW tests were performed on PQ (85 mm) core drilled specifically for the comminution program; whilst the SMC tests were performed on core samples selected from the exploration drilling program. The SMC test samples were selected by the SGS geometallurgy group, by dividing the deposit into 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 DW testing, for a total of 175 tests.

The JK DW test 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; whilst higher t_a values indicate material that is less resistant to abrasion breakage. The JK DW test results were also used to calibrate the SMC test results. The SMC tests generate A and b factors similar to the JK DW tests, along with Mia, Mic, Mih, and density values. The Mia value is the coarse grinding work index, Mic is the crushing work index, and Mih is the high-pressure grinding rolls (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 DW test to provide a direct comparison between the results of the JK DW tests and the SMC tests. The objective was to confirm that the SMC results are consistent with the JK DW test results and that the method is acceptable for use in the variability testwork program. The results from the JK DW tests and SMC tests performed on fractions of the same sample are presented in Table 13.7.

Table 13.7 Results of JK DW tests and corresponding SMC Test®

	JK DW tests					SMC			
Zone	**A**	**b**	**A x b**	**ta**	**P (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
	50.0	0.79	39.5	0.43	2.75	60.8	0.65	39.5	0.0
CAP	67.0	0.37	24.8	0.35	3.02	58.6	0.45	26.4	6.4
	59.5	0.40	23.8	0.21	2.92	79.1	0.34	26.9	13.0
Z-433	60.6	0.41	24.8	0.44	2.81	69.5	0.35	24.3	-2.0
	60.1	0.42	25.2	0.28	2.82	70.5	0.36	25.4	0.8
NZ	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 DW tests for the same sample, indicating that the SMC results will yield slightly lower resistance to breakage than the JK DW tests. The SMC tests were considered to be acceptable for use in the variability testwork program, rather than using the full JK DW tests.

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 impact on production rates.

The distributions of the Z-433 Zone, HS Zone, and CAP Zone 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 Zone, HS Zone, and CAP Zone, which are consistently harder (Table 13.8).

Table 13.8 SMC A x b values and corresponding M_{IA} values

Description Zone	ODM	Z-433	A x b, and Mia (kWh/t)		CAP	Waste
			HS	NZ		
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
80th 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
Mia (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
80th 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, whilst the other zones and waste rock samples have much higher resistance.

A x b values of 26 and 24 at the 80th percentile were interpolated from JK DW tests 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 JKSimMet simulation program to estimate SAG mill sizing and energy requirements. The A x b value used in the process plant design was 24.

13.1.8 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 testwork 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.
- JKSimMet using the Bond equation method.
- JKSimMet using the phantom cyclone method.
- SAG design method.
- OMC method.

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

- Simulations were performed at a nominal tonnage of 906 tonnes per hour (tph) or 20,000 tonnes per day (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 (T_{80}) was calculated.

- Final grinding circuit P_{80} of 75 µm on the cyclone overflow.
- A x b value of 24.2 and 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 13.9.

Table 13.9 SAG mill and ball mill simulation results

Parameters	Units	Method				
		Morrell's Equations	**JK SimMet + Bonds Equation**	**JK SimMet + Phantom Cyclone**	**SAG Design**	**OMC**
80th percentile						
F_{80}	µm	162,500	162,500	162,500	152,000	<150,000
T_{80}	µm	750	2,400	2,400	1,300	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	13.7
Ball Mill	kWh/t	12.92	13.03	12.2	12.89	12.6
Subtotal	kWh/t	28.18	26.26	25.43	25.45	26.3
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
Annual 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:
- Simulations were performed at 20,000 tpd. Operating powers for 21,000 tpd were calculated using the same operating work index (kWh/t).
- The 79th percentile used for the SAG Design 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 power requirement was obtained using the SAG design method. It was decided to use the JK SimMet + BWi method to determine SAG mill sizing for the Feasibility Study.

Based on these results, 15 megawatt (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 installed power ratio of 90% and a safety factor of 5% was added. The SAG mill was sized for a nominal 13% (v/v) (volume of solute / volume of solution) ball charge and a maximum ball charge of 16% (v/v). The nominal mill load is 25% (v/v) and the maximum load is 30% (v/v).

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 effective grinding length (EGL)) SAG mill and a 7.9 m x 12.3 m (12.2 m 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 T_{80} of the SAG mill circuit would be 2,800 µm, rather than 2,400 µm.

13.1.9 Gravity recoverable gold testwork

Two gravity recoverable gold (GRG) tests were performed by FLSmidth using test-scale Knelson concentrators on ODM Zone and Z-433 Zone composites. The test results are presented in Table 13.10.

Table 13.10 GRG test results

Sample	Grind size P_{80} (µm)	Product	Mass (%)	Assay Au (g/t)	Distribution (%)
ODM Master	650	Stage 1 Conc	0.4	46.7	18.8
	542	Sampled Tails	1.0	0.6	0.6
	275	Stage 2 Conc	0.4	48.7	18.8
	211	Sampled Tails	1.1	0.7	0.8
	141	Stage 3 Conc	0.5	27.1	13.6
	90	Sampled Tails	96.6	0.5	47.7
Total (head)			100.0	1.0	100.0
Final concentrate			**1.3**	**39.7**	**51.2**
Z-433	612	Stage 1 Conc	0.4	56.0	20.6
	546	Sampled Tails	1.0	0.9	0.8
	260	Stage 2 Conc	0.4	52.0	20.8
	247	Sampled Tails	1.0	0.8	0.7
	132	Stage 3 Conc	0.6	35.8	17.9
	92	Sampled Tails	96.6	0.5	39.2
Total (head)			100.0	1.1	100.0
Final concentrate			**1.4**	**46.9**	**59.3**

Note: Totals may not compute exactly due to rounding.

The test results indicated that for samples ground to 90 µm, 51% of the gold in the ODM master composite and 59% of the gold in the Z-433 composite is recoverable by gravity.

The gravity circuit is designed to treat cyclone feed slurry with a P_{80} of 1,000 µm, so the process plant gravity gold recovery will be closer to the values in the coarser range of the tests. At 650 µm, 19% of the gold is recoverable by gravity in the ODM Zone master composite and at 612 µm, 21% of the gold is recoverable by gravity in the Z-433 Zone composite.

In addition to the GRG tests, gravity separation tests were also performed during the variability testing program using 2 kilogram (kg) samples. The gravity recoveries of the variability tests ranged from 1% to 77%, with an average of 27% for the non-CAP Zone 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 gold recoveries, averaging 16%.

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 Zone master composite and the Z-433 Zone samples, the best recovery was from the -90 µm fraction with 48% for the ODM Zone composite and 39% for the Z-433 Zone sample.

13.1.10 Cyanide leaching testwork

13.1.10.1 Gravity concentration and leaching of gravity tailings

Gravity gold recovery tests, followed by cyanide leaching tests on the gravity tailings were 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 gravity tailings.

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

- Grind size P_{80}s ranging from 50 µm to 119 µm.
- Pre-aeration with air for 30 minutes.
- Pulp density of 50% solids (w/w) (weight of solute / weight of solution).
- Pulp pH was maintained between 10.5 and 11.0.
- Cyanide concentration was varied between 0.5 grams per litre (g/L) and 1.0 g/L NaCN.
- Residence time was 48 hours, with kinetic samples taken at 6 hours, 24 hours, and 36 hours.

The results of the leach tests on the gravity tailings for gold and silver are presented in Table 13.11 and Table 13.12, respectively. Results presented for grind sizes with more than one test are averaged values.

Table 13.11 Gold results of leaching tests on gravity tailings

Number of tests	P_{80} (µm)	Reagent consumptions (kg/t)		Au recovery (%)						Au assays (g/t)	
				Cyanide leach [1]				Gravity [2]	Gravity + cyanide leach [2]		
		NaCN	CaO	6h	24h	36h	48h			Residue grade	Head grade
1	119	0.08	0.39	77.7	83.5	85.6	85.8	29.1	89.9	0.10	0.98
1	95	0.12	0.38	76.9	86.3	85.3	86.7	29.1	90.6	0.10	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	0.10	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

Notes:
[1] With respect to test feed.
[2] With respect to ore.

Table 13.12 Silver results of leaching tests on gravity tailings

Number of tests	P_{80} (μm)	Reagent consumptions (kg/t)		Ag recovery (%)						Ag assays (g/t)	
		NaCN	CaO	Cyanide leach [1]				Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
				6h	24h	36h	48h				
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
2	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

Notes:
[1] With respect to test feed.
[2] With respect to ore.

The leach gold recoveries ranged from 86% at 119 μm to 91% at 51 μm, and total gold recoveries from 90% at 119 μm to 93% at 51 μm. Gold recovery at the design grind size of 75 μm was 90% for leaching and 93% for total recovery.

Silver recoveries increased from 67% at 119 μm to 79% at 75 μm; and then dropped for the 62 μm and 51 μm tests.

13.1.10.2 Cyanide leach testwork on gravity tailings

Gravity tailings leach tests were performed on samples from the Initial Pit and the RLOM composites. The tests were performed for fixed times, with kinetic samples taken at 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, 30, and 36 hours.
  - RLOM: 12, 18, and 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 for each leach time for a total of 36 tests.

The results of the leach tests on the gravity tailings are presented in Table 13.13 and Table 13.14 for gold and silver, respectively. The results are presented as averages of the 12 tests performed per grind size per composite.

Table 13.13 Initial Pit and RLOM gravity tailings leach test results for gold

Composite name	Number of tests	P_{80} (µm)	Reagent consumptions (kg/t)		Au Recovery (%)						Au assays (g/t)	
			NaCN	CaO	Cyanide leach [1]				Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
					12h	18h	30h	36h				
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

Notes:
[1] With respect to test feed.
[2] With respect to ore.

Table 13.14 Initial Pit and RLOM gravity tailings leach test results for silver

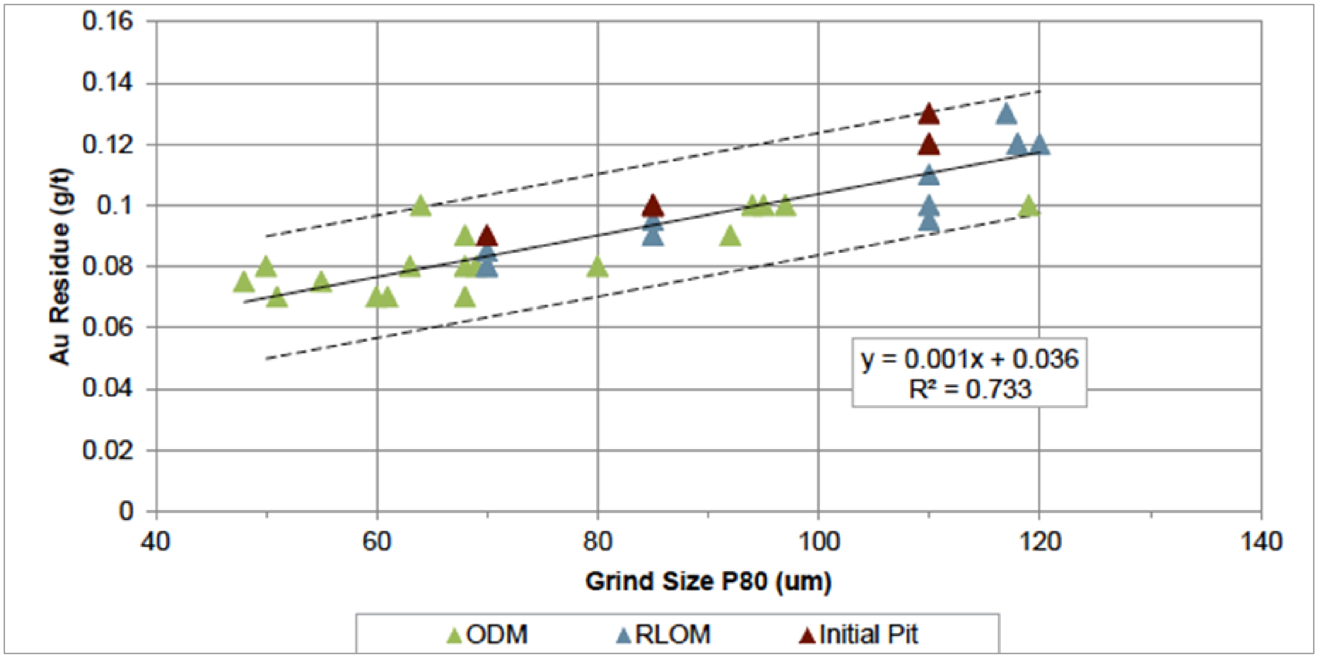
Composite name	Number of tests	P_{80} (µm)	Reagent consumptions (kg/t)		Ag recovery (%)						Ag assays (g/t)	
			NaCN	CaO	Cyanide leach [1]				Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
					12h	18h	30h	36h				
Initial Pit	12	110	0.03	0.32	-	62.3	69.9	61.2	7.4	64.1	1.07	2.80
	12	85	0.04	0.33	-	59.4	70.5	62.8	7.4	65.5	1.07	2.80
	12	70	0.05	0.35	-	58.0	72.1	61.8	7.4	64.6	1.10	2.80
RLOM	12	110	0.02	0.31	61.1	66.4	68.9	-	6.1	70.8	0.80	2.80
	12	85	0.02	0.32	65.1	68.9	71.3	-	6.1	73.1	0.77	2.80
	12	70	0.01	0.32	66.2	72.1	70.8	-	6.1	72.6	0.80	2.80

Notes:
[1] With respect to test feed.
[2] With respect to ore.

The results show that the gold and silver recoveries increase with a reduction in particle size. Gold recovery did not increase beyond 30 hours. Initial Pit silver recoveries increased up to 30 hours but then dropped between 30 and 36 hours.

Figure 13.7 shows the gravity tailings residue assays versus grind size for the ODM, Initial Pit, and RLOM composite leach tests. The dotted lines represent the sensitivity of the assay technique of 0.02 g/t Au.

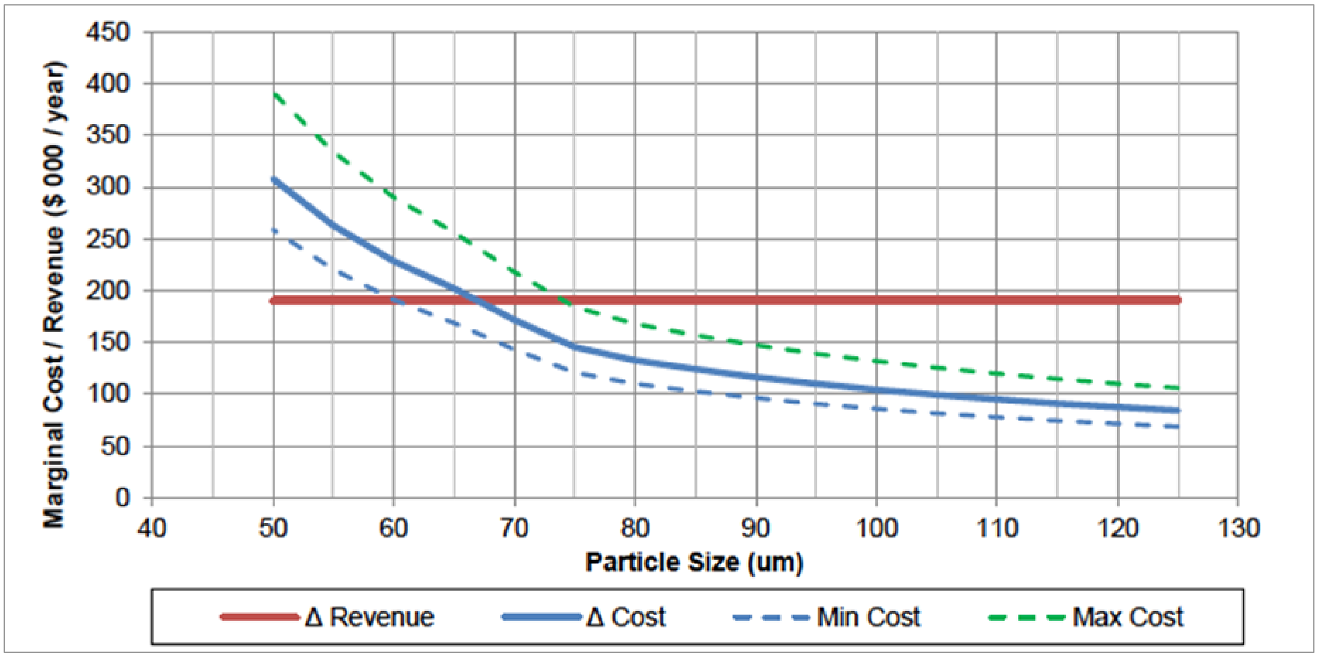
Figure 13.7 Gravity tailings leach residue gold grade versus grind size



A cost versus revenue study was performed during the Feasibility Study to determine a P_{80} for the variability testwork program. The costs included cyanide consumption, grinding energy at a fixed production rate and estimated media wear. Revenue was calculated based on the residue equation shown in Figure 13.7. 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 P_{80} of 75 µm and a retention time of 36 hours were selected for the variability testwork program.

Note, in the current operation, a grind size P_{80} of 75 µm is not targeted, but rather plant grind size P_{80} is a function of SAG and ball mill power draw and plant throughput.

13.1.10.3 Cyanide leach testwork testing the effect of cyanide concentration on gold recovery

The effect of initial cyanide concentration on gold recovery was investigated using RLOM composites samples. The cyanide concentrations were varied between 0.15 g/L and 0.50 g/L NaCN. The tests were conducted for 36 hours and samples were collected at timed intervals.

The results of the tests are presented in Table 13.15.

Table 13.15 Effect of cyanide concentration on gold recovery

Composite name	P_{80} (µm)	NaCn concentration (g/t)	Reagent consumptions (kg/t)		Au recovery (%)							Au assays (g/t)	
			NaCN	CaO	Cyanide leach [1]					Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
					12h	18h	24h	30h	36h				
RLOM	118	0.50	0.11	0.40	77	80	83	81	83	16	86	0.12	0.67
	117	0.30	0.08	0.37	71	77	82	81	82	16	85	0.13	0.69
	120	0.20	0.06	0.40	74	78	82	82	82	16	85	0.12	0.65
	118	0.15	0.06	0.41	70	77	81	80	822	16	85	0.12	0.68

Notes:
[1] With respect to test feed.
[2] With respect to ore.

Figure 13.9 shows cyanide leach gold recovery vs time for these tests.

Figure 13.9 Impact of gold recovery by NaCN concentration

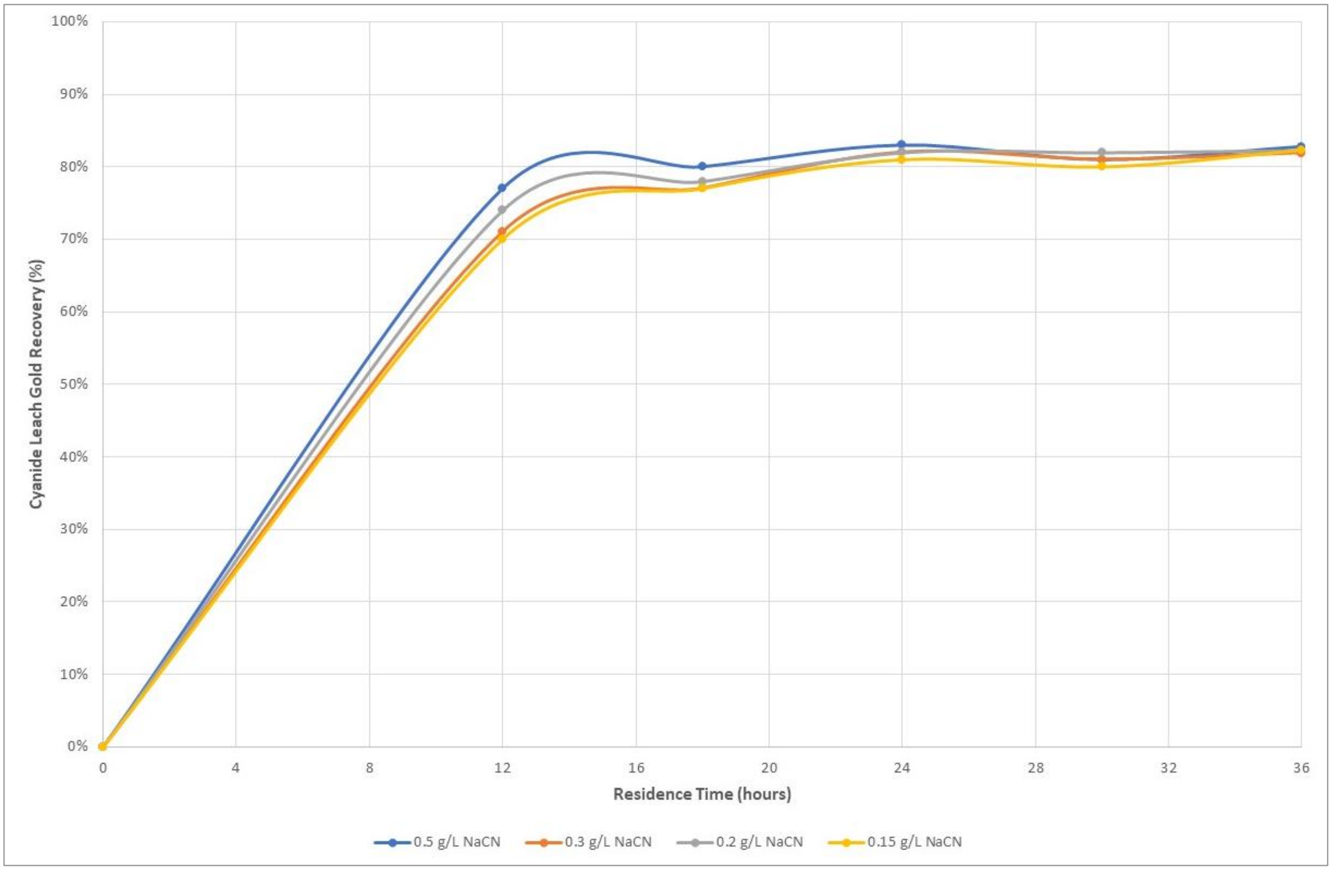


Figure 13.9 showed there was no discernible increase in terminal gold recovery by increasing NaCN concentration.

13.1.10.4 Cyanide leach testwork testing the effect of pre-aeration on gold recovery

Pre-aeration tests using air were performed on the Initial Pit and the RLOM composites to ascertain whether pre-aerating the sample would reduce cyanide and lime consumptions. The tests were performed for 36 hours and kinetic samples were taken throughout the test. The results of the tests are presented in Table 13.16.

Table 13.16 Effect of pre-aeration on leach gold recovery

Composite name	Pre-aeration	P_{80} (µm)	Reagent consumptions (kg/t)		Gold recovery (%)					
			NaCN	CaO	Cyanide leach [1]				Gravity [2]	Gravity + cyanide leach [2]
					6h	12h	24h	36h		
Initial Pit	Y	100	0.07	0.36	79%	83%	83%	84%	31%	89%
	Y	100	0.08	0.36	73%	79%	80%	83%	31%	88%
	N	100	0.22	0.30	74%	82%	86%	84%	31%	89%
	N	100	0.19	0.31	75%	82%	81%	85%	31%	90%
RLOM	Y	118	0.08	0.36	75%	75%	81%	81%	16%	84%
	Y	118	0.07	0.36	76%	82%	83%	82%	16%	85%
	N	118	0.18	0.33	72%	77%	80%	82%	16%	85%
	N	118	0.25	0.29	70%	70%	77%	79%	16%	82%

Notes:
[1] With respect to test feed.
[2] With respect to ore.

For both sets of samples, the pre-aeration tests had significantly lower cyanide and lime consumptions than those without pre-aeration. The gold recoveries were similar in both cases. Based on these tests, the variability tests were run using pre-aeration.

13.1.10.5 Cyanide leach testwork testing oxygen versus air, and impact of lead nitrate

The effect of adding oxygen instead of air in the pre-aeration stage was investigated. Lead nitrate addition was also trialed to ascertain if it could reduce cyanide consumption. The results of the tests are presented in Table 13.17.

Table 13.17 Effect of oxygen, air, and leach nitrate on leach gold test results

Composite name	Aeration	Lead nitrate	P_{80} (µm)	Reagent consumptions (kg/t)		Au recovery (%)						Au assays (g/t)	
				NaCN	CaO	Cyanide leach [1] 12h	18h	30h	36h	Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
Initial Pit	Oxygen	N		0.04	0.37	82	-	-	-	29	87	0.12	0.97
	Oxygen	N	54	0.04	0.36	-	86	-	-	29	90	0.09	
	Oxygen	N	52	0.11	0.41	-	-	89	-	29	92	0.07	
	Oxygen	N	61	0.06	0.38	-	-	88	-	29	92	0.10	
	Oxygen	N	55	0.12	0.38	-	-	-	87	29	91	0.09	
	Oxygen	N	59	0.04	0.39	-	-	-	87	29	91	0.10	
	Oxygen	Y	59	0.16	0.50	-	-	-	88	29	92	0.08	
	Oxygen	Y	45	0.05	0.52	-	-	-	87	29	91	0.08	
	Air	Y	48	0.14	0.56	-	-	-	88	29	91	0.08	
	Air	Y	59	0.06	0.51	-	-	-	87	29	91	0.09	
RLOM	Oxygen	N	66	0.05	0.36	84	-	-	-	39	91	0.08	0.89
	Oxygen	N	59	0.05	0.41	-	87	-	-	39	92	0.07	
	Oxygen	N	79	0.06	0.33	-	-	87	-	39	92	0.09	
	Oxygen	N	68	0.07	0.40	-	-	84	-	39	90	0.08	
	Oxygen	N	57	0.08	0.41	-	-	-	85	39	91	0.08	
	Oxygen	N	66	0.08	0.41	-	-	-	86	39	91	0.08	
	Oxygen	Y	70	0.06	0.53	-	-	-	84	39	90	0.08	
	Oxygen	Y	71	0.03	0.53	-	-	-	85	39	91	0.08	
	Air	Y	72	0.06	0.50	-	-	-	82	39	89	0.11	
	Air	Y	71	0.08	0.49	-	-	-	84	39	90	0.10	

Notes:
[1] With respect to test feed.
[2] With respect to ore.

The data shows that there was no discernible change in cyanide and lime consumption by adding oxygen rather than air in the pre-aeration stage. Based on these results, the variability tests used pre-aeration with air.

The addition of lead nitrate did not reduce cyanide consumption relative to the baseline tests, so it was not used in the variability tests.

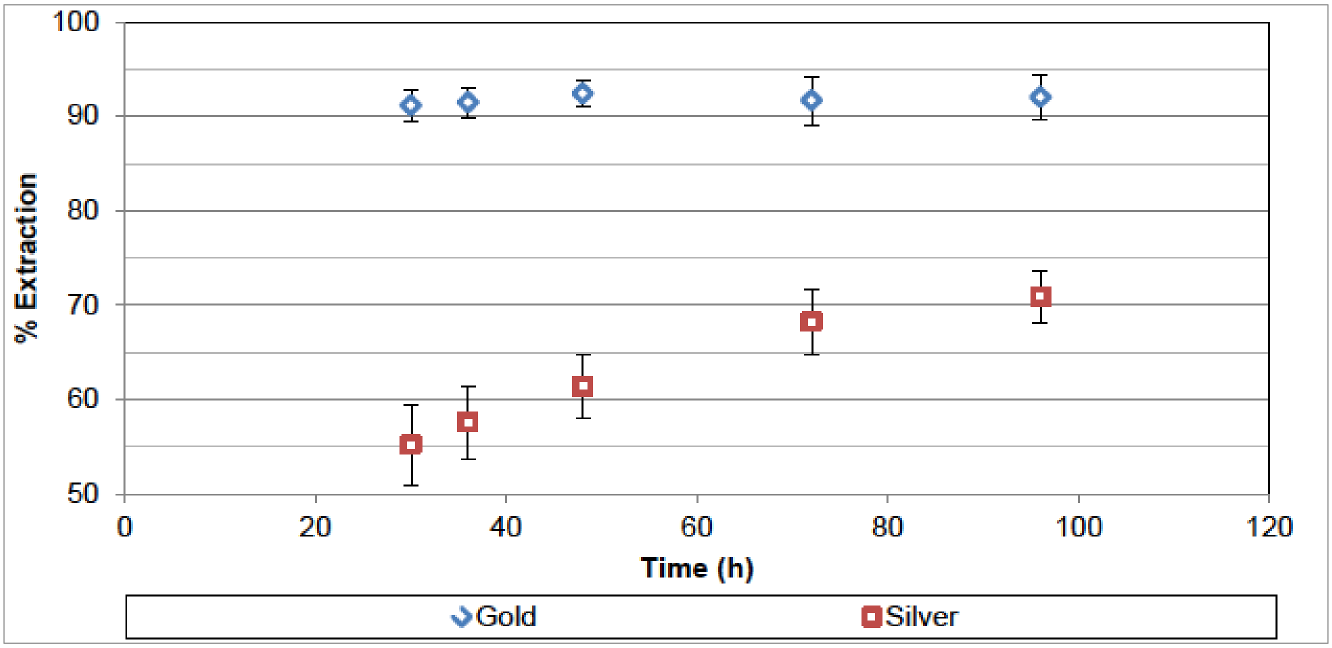
13.1.10.6 Cyanide leach testwork testing Intrepid Zone kinetics

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 kinetic sampling at 30, 36, 48, and 72 hours.
- Target grind size P_{80} of 75 µm.
- Cyanide concentration of 0.5 g/L NaCN.
- 30 minute pre-aeration stage.
- pH of 10.5 – 11.0.

Figure 13.10 shows a boxplot of the Intrepid Zone leach extraction for gold and silver as a function of time. The figure shows the average recoveries, as well as the minimum and maximum recoveries for each time period.

Figure 13.10 Boxplot of Intrepid Zone gold and silver cyanide leaching kinetics



Gold extraction was essentially complete after 30 hours. Silver extraction kinetics were relatively slower and silver extraction was still increasing after 96 hours of leaching.

13.1.10.7 Cyanide leach variability testwork

Variability cyanide leach tests were 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 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.

Table 13.18 summarizes the variability leach test results.

Table 13.18 Averaged variability leach test gold and silver recoveries

Zone	Number of tests	Average reagent consumptions (kg/t)		Average gold recovery (%)				Average silver recovery (%)			
		NaCN	CaO	Cyanide leach [1]		Gravity [2]	Gravity + cyanide leach [2]	Cyanide leach [1]		Gravity [2]	Gravity + cyanide leach [2]
				30h	36h			30h	36h		
ODM	138	0.06	0.37	78%	79%	26%	84%	57%	59%	10%	63%
Z-433	30	0.1	0.41	83%	84%	36%	90%	49%	51%	13%	58%
HS	13	0.06	0.36	84%	86%	24%	89%	48%	48%	9%	53%
NZ	24	0.08	0.4	82%	83%	27%	87%	56%	56%	9%	60%
Intrepid	30	0.1	0.31	86%	87%	16%	88%	60%	60%	5%	61%
Non-CAP	235	0.07	0.37	81%	81%	26%	86%	57%	57%	10%	61%
CAP	40	0.11	0.62	72%	72%	9%	74%	65%	65%	3%	66%

Notes:
[1] With respect to test feed.
[2] With respect to ore.

Table 13.18 shows:

- Most ore zones achieved average total gold recoveries greater than 80%, with the exception of the CAP Zone with 74%.
- The leaching performance was relatively consistent, with the majority of the variability driven by the grind size and the gravity recovery. Gold leaching was generally complete after 30 hours.
- The leaching performance was relatively consistent with the majority of the variability driven by the grind size, gravity separation and leaching retention time. Silver was still leaching at 36 hours in all tests.

13.1.11 Diagnostic leach testwork

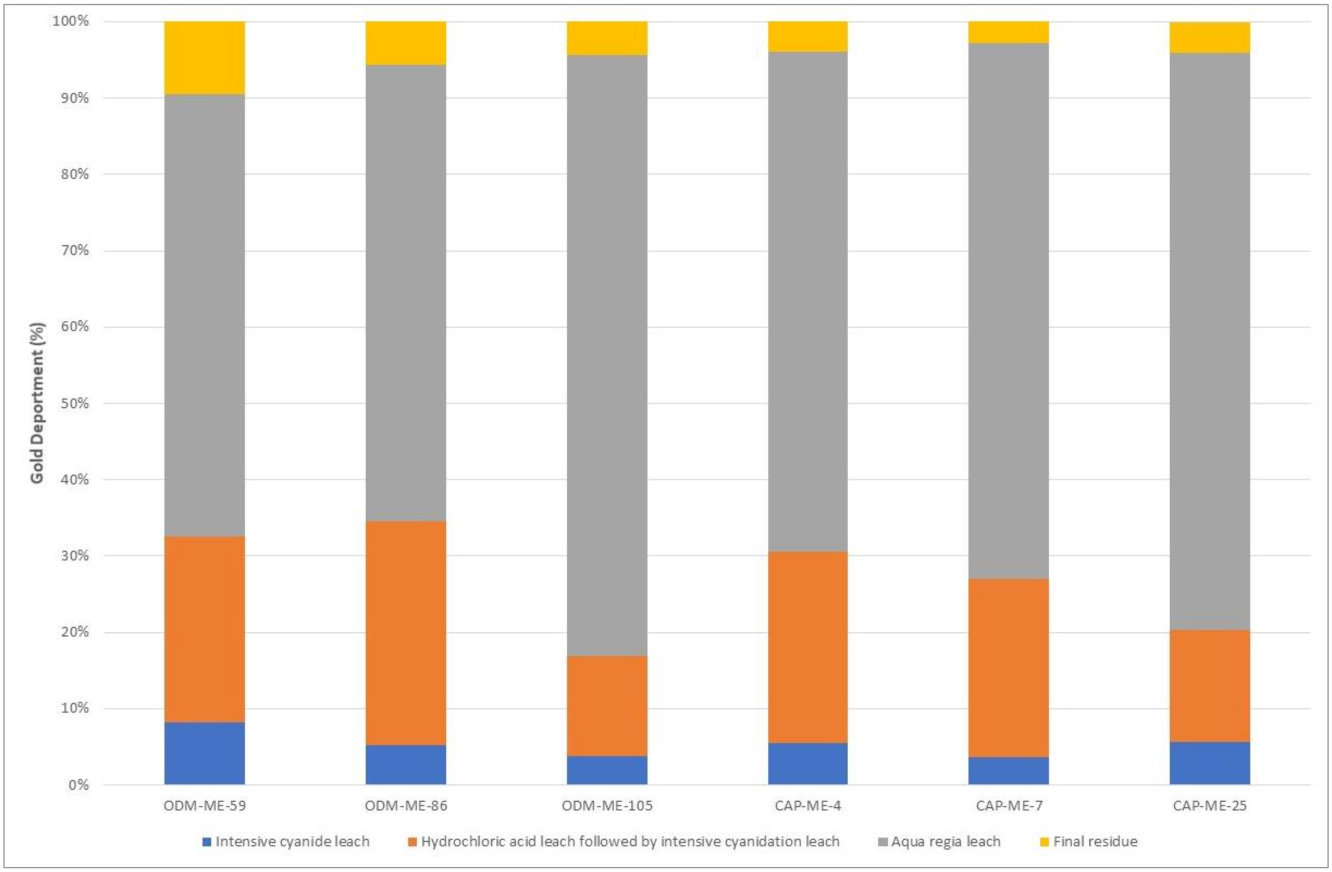
Diagnostic leach tests were performed on the cyanide leach tailings from three ODM Zone samples and three CAP Zone samples to identify the occurrence of the residual gold that did not leach.

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 the 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.
- Agua 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 gold deportments from these tests are shown in Figure 13.11.

Figure 13.11 Diagnostic leach test gold deportments on cyanide leach tails samples



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 Zone and the ODM Zone samples.

- The amount of the residual gold recovered by the agua 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 samples at the detection level.
- Higher percentages of the residual gold were recovered using the hydrochloric acid leach, followed by intensive cyanide leaching with approximately 8% to 24% of the residual gold being leached.
- 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.

In 2017, McLelland completed additional leach diagnostic tests on a composite sample of the ODM Zone ore and a composite sample of the CAP Zone ore. The diagnostic leach test procedure includes the following steps:

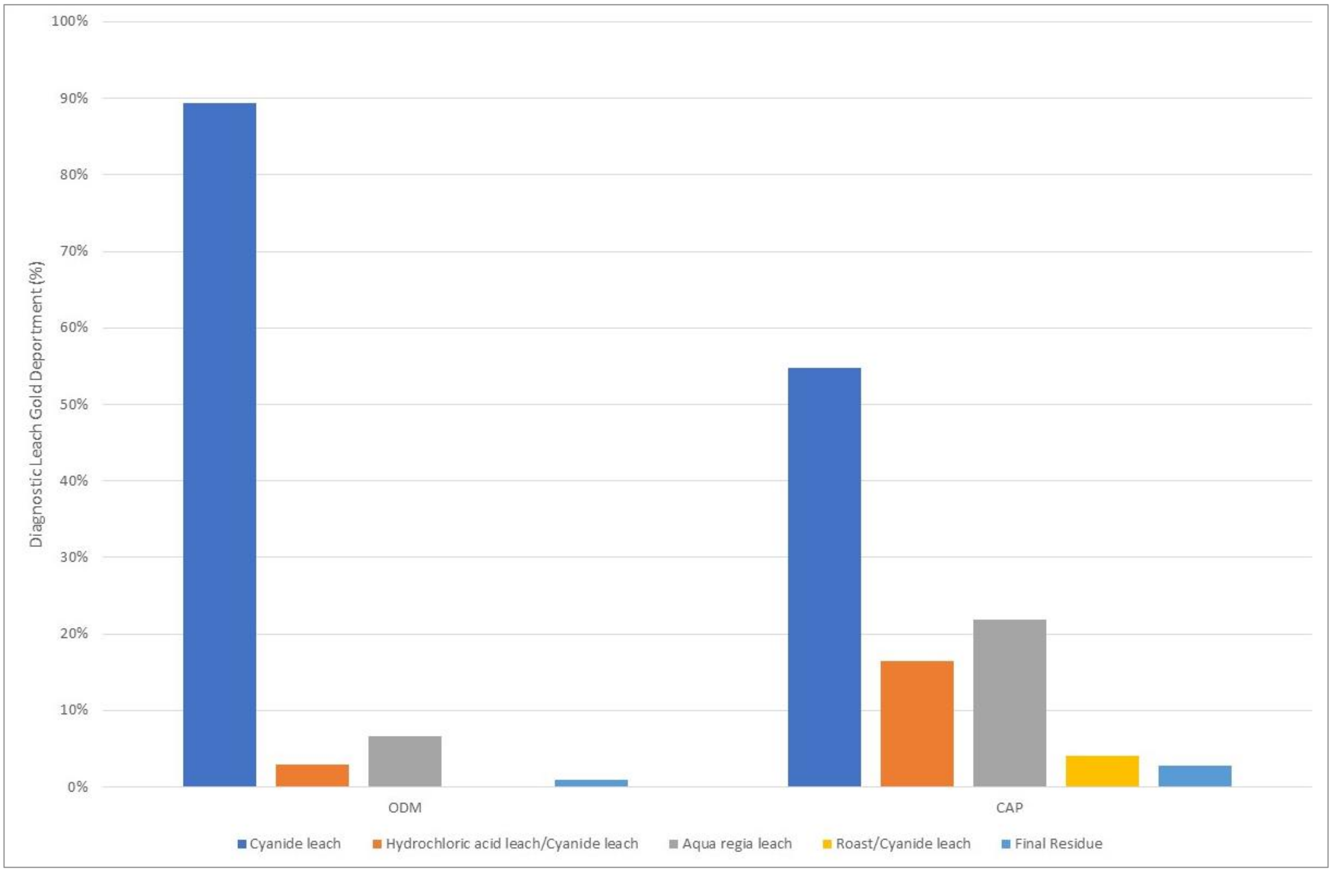
- The samples were ground to a P_{80} of 106 µm.
- Direct cyanide leach: Extraction of free-milling gold.
- Hydrochloric acid leach followed by direct cyanide leach: Extraction of gold that is associated with pyrrhotite, calcite, ferrites, etc. This is done by leaching the tailings using hydrochloric

acid to dissolve the pyrrhotite and other minerals, then performing the intensive cyanide leach to extract the liberated gold.

- Agua regia leach: Extraction of gold associated with or encapsulated in sulphide minerals such as pyrite and arsenopyrite.
- Roast and cyanide leach: Extraction of gold associated with or encapsulated in carbonaceous material.
- The final residue from these tests is considered to be locked in silicates or associated with fine sulphides that are locked in silicates.

The gold deportments from these tests are shown in Figure 13.12.

Figure 13.12 Diagnostic leach test gold deportments on ore samples



The ODM Zone has a large proportion (89%) of free-milling gold (cyanide leach). Approximately 7% of the gold in the ODM Zone ore is locked in sulphides (agua regia leach).

In the CAP Zone ore, there is a moderate proportion of free milling gold. There are relatively large proportions of hydrochloric acid leachable gold (16%) and gold locked in sulphides (22%).

13.1.12 Cyanide destruction testwork

The SO_2 – air cyanide destruction process was investigated on the leach solutions from the 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 13.19.

Table 13.19 Cyanide destruction test results

	Sample	Pulp density (%)	Retention time (min)	Solution phase						Reagent addition (g/g CN_{WAD})		
				pH	CN_1 (mg/L)	CN_{WAD}, standard (mg/L)	CN_{WAD}, picric (mg/L)	Cu (mg/L)	Fe (mg/L)	SO_2	Lime	Cu
Initial Pit	**Feed**	-	-	10.7	152	117	-	9.4	1.8	-	-	-
	Batch											
	CND 3 Continuous	50	90	8.6	-	-	<0.1	-	-	7.52	3.48	0.13
	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
RLOM	**Feed**	-	-	11.1	128	123	-	11.0	-	-	-	-
	Batch											
	CND 4 continuous	50	180	8.5	-	-	0.4	-	-	12.7	14.9	0.24
	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 continuous	50	150	8.6	-	-	0.26	-	-	11.9	7.68	0.13
	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 2-3	50	116	8.5	<0.1	<0.1	<0.1	1.1	0.2	5.69	3.64	0.12

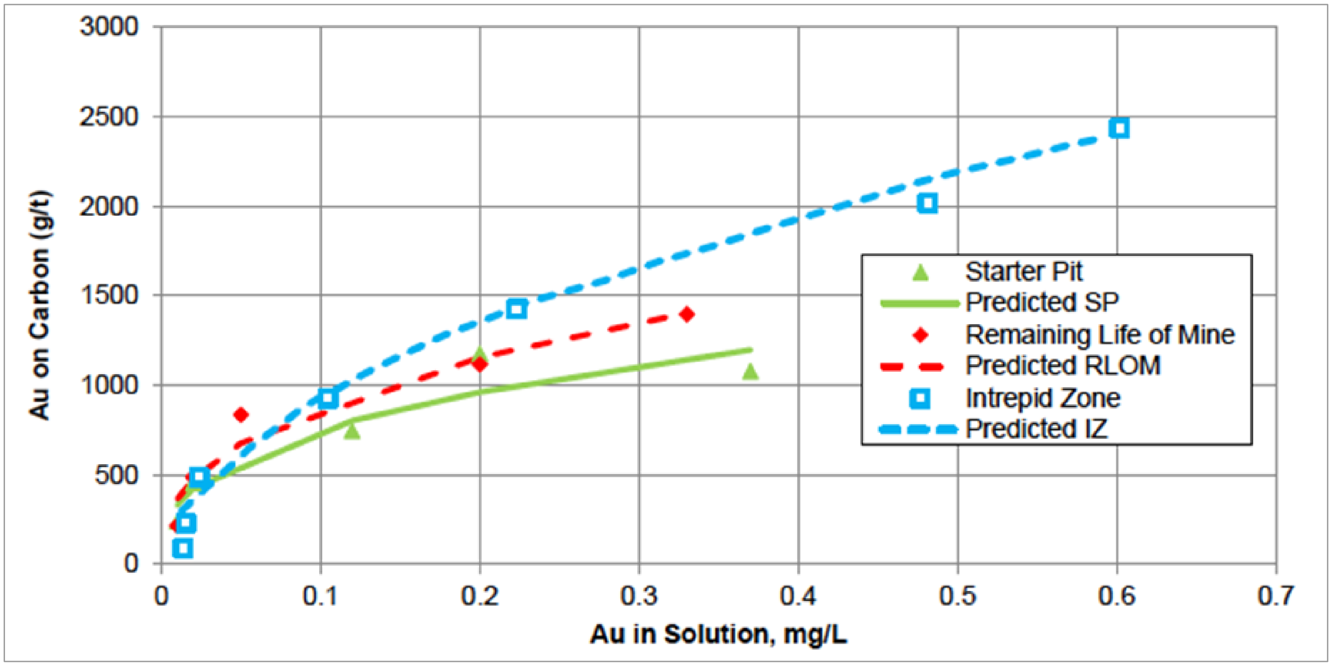
The results show that this process is effective at lowering the weak acid dissociable cyanide (CN_{WAD}) to levels well below 5 ppm CN. The reagent consumptions (Sulphur dioxide (SO_2), lime, and copper) are considered to be in agreement with standard industrial practices.

13.1.13 Carbon-in-pulp modelling

Carbon-in-pulp (CIP) modelling work was performed by SGS to validate the CIP circuit design. 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 13.13.

Figure 13.13 CIP isotherms used for modelling



The isotherms were used to model the kinetics for gold adsorption onto the carbon in a CIP circuit. 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.

SGS modelled 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 milligrams per litre (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 t of carbon) every two days; or one half tank, 10 t of carbon per day.

13.1.14 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 13.20.

Table 13.20 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
RLOM	Settling rate	tph/m^2	-	1.00	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		metres	45	39	46

Based on the test results, the recommended thickener diameter was 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.

SGS performed static and dynamic settling tests on the Intrepid Zone samples. The settling rates were found to be lower than the Initial Pit and RLOM samples at 0.42 tph/m^2 to 0.61 tph/m^2. 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.

13.1.15 Slurry rheology testwork

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 (w/w) and 64% (w/w) for the Initial Pit and RLOM composites, respectively. The design CSDs for the pre-leach and pre-detox thickeners was 61% and 60%, respectively.

13.1.16 Summary and findings from metallurgical testwork program

The results from the SGS testwork program formed the basis for the Mineral Reserve estimate and updated Feasibility Study.

The chosen process flowsheet was gravity separation followed by whole ore leaching. This flowsheet was preferred over the flowsheet with flotation and concentrate leaching. This was due to higher recoveries, lower cyanide consumptions, and the energy costs associated with fine grinding the flotation concentrate.

The grinding testwork indicated significant variation in ore hardness in the ODM Zone.

The testwork demonstrated that 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 circuits if the Intrepid Zone ore is not blended with Main Pit ore.

The CAP Zone material will be placed in the low-grade stockpile and treated toward the end of the mine life, due to the low recoveries the CAP Zone material produced in the testwork program. When the CAP Zone material is processed, it will be blended with other ore types. In later years of the mine life, the CAP Zone ore will report directly to the process plant.

AMEC selected the data for input into engineering design criteria. Vendors selected the data for sizing of major equipment such as the crushers and grinding mills.

During the testwork program, a cost versus revenue study was conducted to identify the optimum grind size P_{80} for the plant process design criteria. This study was based on the testwork data. A grind size P_{80} of 75 µm was chosen, as the cost study demonstrated it was the most economically viable grind size. Despite this, Rainy River's current process philosophy is to target a process throughput rather than a grind size, so the plant typically operates at a grind size P_{80} of 90 µm to 110 µm (dependent on throughput). Rainy River determined that it is more economically beneficial to operate at higher throughputs and lower gold recoveries (through coarser grinds) over lower throughputs and higher gold recoveries (through finer grinds).

It is AMC's opinion that the metallurgical test programs for the Rainy River deposit were comprehensive and have taken into consideration the major ore types and the mine plan 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 plant.

13.2 Metallurgical testwork post plant start-up

13.2.1 Introduction

Metallurgical testwork programs have been conducted since the start-up of the Rainy River process plant in 2017.

OMC completed an audit of the Rainy River process plant in April 2019. OMC used the comminution data that was collected from the audit for creating a JKSimMet model. The purposes of the JKSimMet model were to forecast the process plant throughput based on comminution testwork data, and to simulate different comminution circuit flowsheet configurations. OMC also developed multivariate regression formulas for forecasting process plant gold recovery. OMC developed these regression formulas based on actual process plant data including process plant feed gold grades, cyclone overflow P_{80}s, and total gold recoveries.

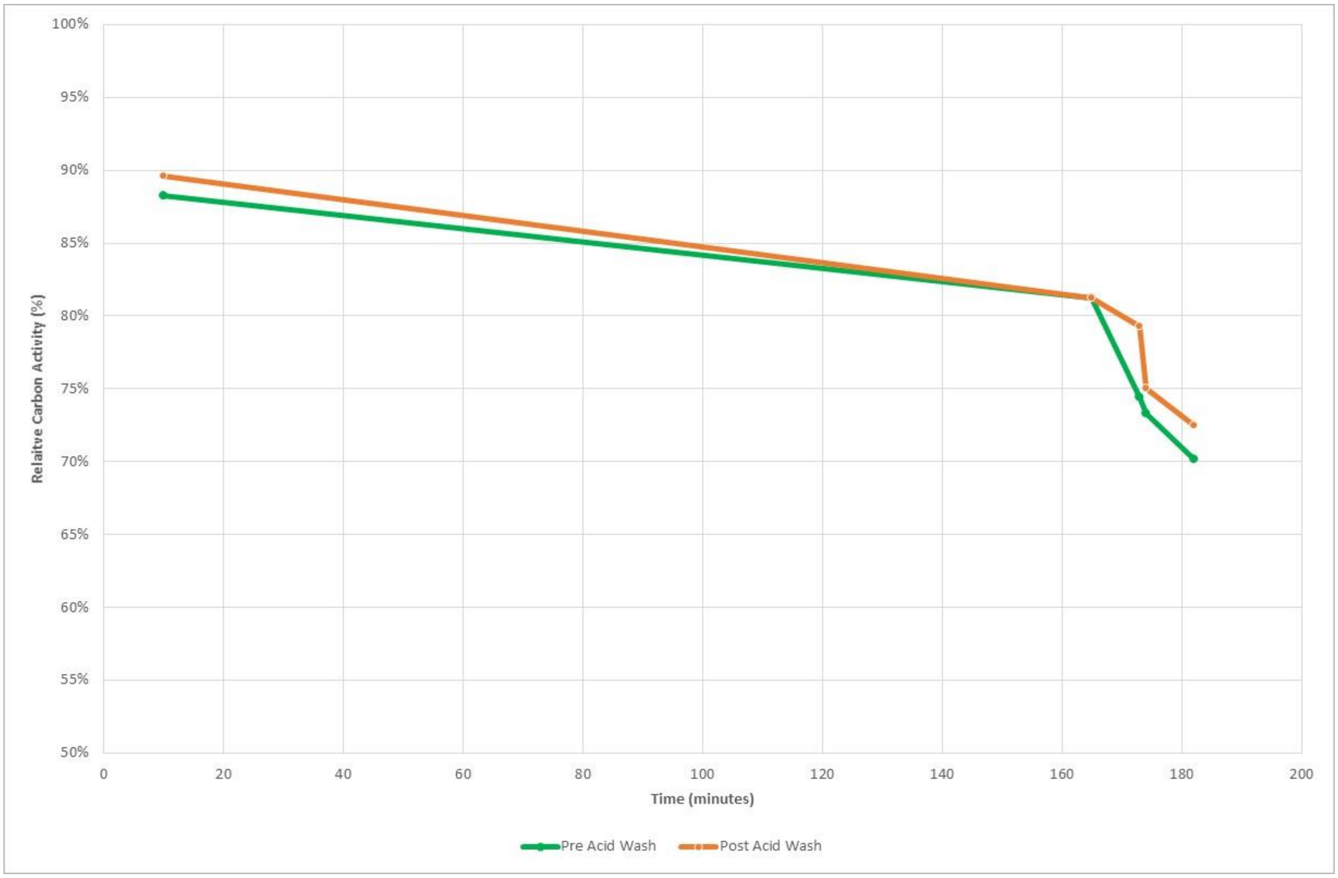
13.2.2 Acid wash testwork

Calcium carbonate (lime) is one of the major causes of carbon fouling. As a general guide, the activity of the carbon may be severely reduced where calcium content is greater than 3%. To control the calcium content on the carbon, the acid wash process is commonly used as it removes the calcium from the fouled carbon.

To ascertain the usefulness of the Rainy River acid wash circuit, in 2019 carbon activity tests were completed on samples of carbon that had been acid washed and carbon samples that had not been acid washed. The relative activity of the carbon is then used to assess the effectiveness of the acid wash process.

Figure 13.14 shows carbon activity vs time for these tests.

Figure 13.14 Carbon activity vs time for acid wash tests



There was no significant difference in terms of carbon activity observed between the pre-acid wash samples and the post-acid wash samples. Rainy River concluded that the activity of the carbon is not being severely reduced from the absorption of calcium carbonate.

Based on these tests, Rainy River has stopped using the acid wash circuit in the process plant. Rainy River notes that this has removed all acid costs and reduced the carbon attrition due to the reduction in carbon movement.

13.2.3 Flocculant screening testwork

Settling rates in the pre-leach thickener have been identified as a plant bottleneck. When the plant experiences excessive grinding circuit throughput, the thickener tends to discharge solids to the thickener overflow launder. From 2017 to 2019, a number of flocculant screening testwork programs have been completed in an attempt to understand and rectify these issues.

Quadra Chemicals Ltd. (Quadra) completed thickening testwork and an audit of the pre-leach thickener in September 2017. Quadra made the following recommendations:

- The Flocculant A-100 was the best performing flocculant relative to all other flocculants tested.
- The use of coagulants in conjunction with a flocculant did not reduce settling time but could be used to improve thickener overflow quality.
- Rheology flocculants could be used to augment settling and improve pump-ability of the settled slurry.

Quadra completed another testwork program in June 2019 to identify a flocculant for the pre-leach thickener. Quadra completed testwork evaluating different flocculants and recommended Magnafloc 5250 due to faster dissolution rates, lower consumption rates and faster settling rates than the other flocculants trialed.

SNF Canada Ltd. (SNF) completed flocculant screening tests for the pre-leach thickener in September 2019. The testwork demonstrated that the FO 905VHM flocculant had a slightly faster settling rate of 14.97 metres per hour (m/h) compared to the P A250L-K with 14.22 m/h. The P A250L-K is the flocculant that is currently being added to the pre-leach thickener. Both these flocculants were trialed at 20 g/t.

These programs identified that during winter periods, the cold water reduces flocculant dissolution rates. Rainy River will trial a polymer slicing unit and an alternative flocculant supply system with a goal of improving the dissolution of flocculant and subsequently increase flocculant flowrates.

13.3 Grade-recovery predictive formulas for gold recovery and silver recovery

Grade-recovery predictive formulas were developed for plant gold recovery and silver recovery. The purpose of these predictive formulas was to forecast gold and silver recovery in Rainy River LOM and financial models.

The deposit was divided into three zones to develop the grade-recovery formulas: non-CAP Zone ore, Intrepid Zone ore, and CAP Zone ore.

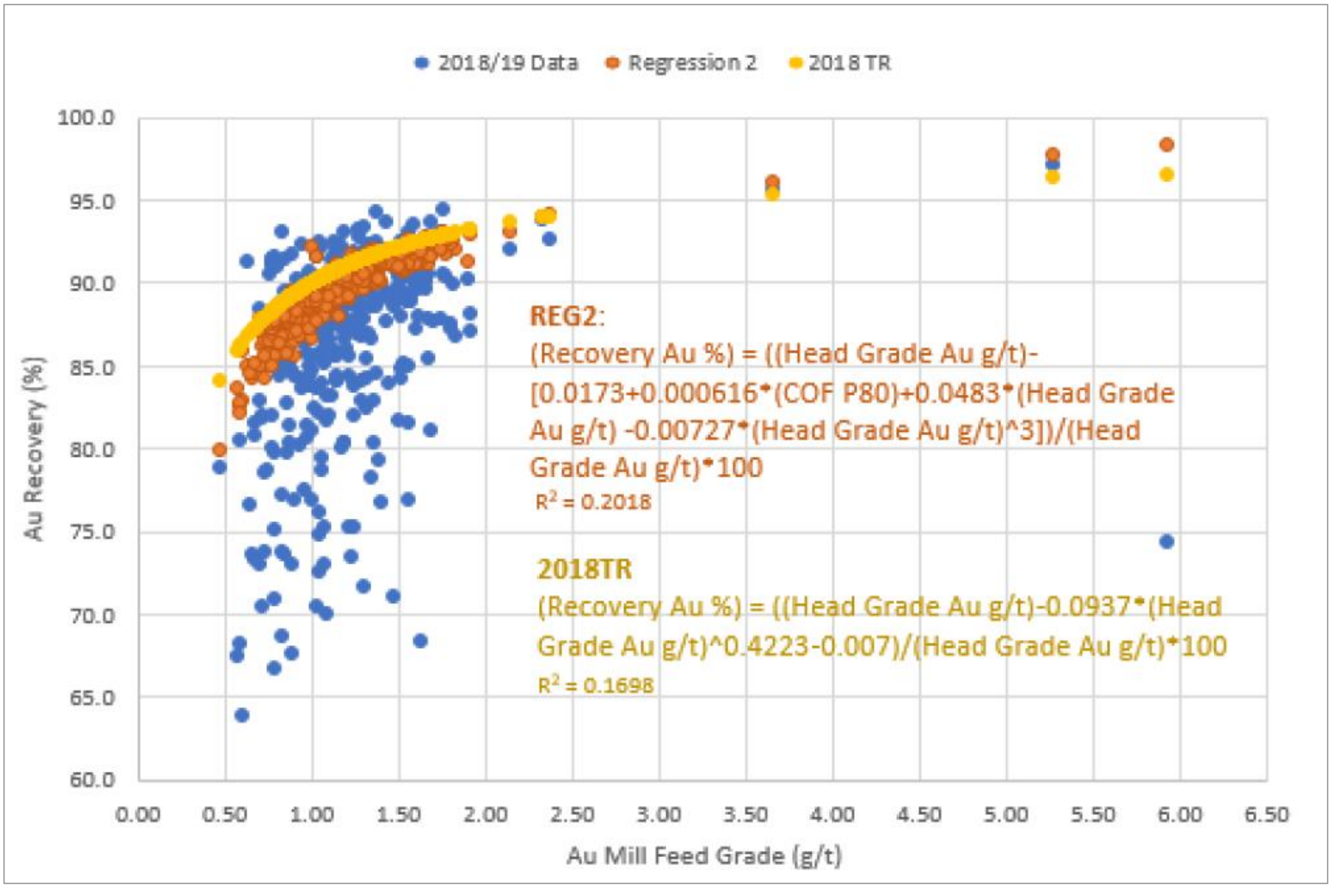
As part of their process plant audit, OMC developed multivariate regression formulas for forecasting process plant gold recovery. OMC developed these regression formulas based on actual process plant data including process plant feed gold grades, cyclone overflow P_{80}s, and total gold recoveries.

OMC reviewed the 2018/19 plant production data from the grinding circuit and cyanide leach circuit to establish the grind-recovery relationship between process plant feed gold grade, cyclone overflow P_{80}, tails solid gold grade and gold recovery. The analysis yielded two distinct regression formulas to predict gold recovery:

- Gold recovery at grind size.
- Gold recovery calculated from tails grade multivariable analysis. This formula uses grind size and process plant feed grade as inputs.

Figure 13.15 shows gold recovery vs process plant feed gold grade and compares the Regression 2 model to the overall 2018/19 process plant data as well as the previous grade-recovery relationship from the 2018 Technical Report (2018 TR).

Figure 13.15 Process plant gold recovery vs. process plant gold feed grade



Source: OMC 2019.

OMC observed that plant gold recovery appears to be more sensitive to changes in feed grade compared to changes in grind size. The analysis shows a general relationship, but the correlation is not strong.

OMC recommended using the 2nd regression formula due to its multivariable functionality for economic analysis of throughput, grind size and recovery. The Regression 2 and 2018 TR models have relatively similar responses. At a constant mill feed grade, the regression 2 model output is +/-0.3% change in recovery for every +/-5 µm increment in cyclone P_{80}, which is similar to the Regression 1 behaviour. Regression 2 is valid for a process plant feed grade range of 0.47 g/t to 2.0 g/t. OMC considered that the model is most accurate within this feed grade range, however OMC considered that it could be used for extrapolations outside of this range with a reasonable level of confidence.

The predictive gold recovery formulas are as follows:

The gold recovery formula for the CAP Zone was based on the model from the 2018 NI 43-101 report. To date, CAP Zone ore has not been processed.

CAP Zone:

$$\text{Au Rec} = ([\text{AuHG} - (0.2497 * \text{AuHG}^{1.015}) - 0.007)] / \text{AuHG}] * 100$$

Non-CAP Zone:

$$Au\ Rec = ([AuHG - (0.0173 + (0.000616 * P_{80}) + (0.0483 * AuHG) - (0.00727 * AuHG^3)] / AuHG) * 100$$

The Non-CAP Zone formula has been capped at a maximum gold recovery of 95%.

Intrepid Zone:

$$Au\ Rec = ([AuHG - (0.0937 * AuHG^{0.4223}] - 0.007) / AuHG) * 100$$

Where:

- Au Rec is the process plant gold recovery in %.
- AuHG is the process plant gold head grade in g/t.
- P_{80} is the hydrocyclone overflow P_{80} in µm. As process plant throughputs increase, the P_{80} will be coarser.

New Gold has developed similar predictive formulas for silver recovery from metallurgical testwork programs (Kenny 2016). These predictive formulas are as follows:

CAP Zone:

$$Ag\ Rec = [([AgHG - (0.3868 * AgHG^{0.9174})] / AgHG) * 100] * 0.966$$

Non-CAP Zone:

$$Ag\ Rec = [([AgHG - (0.4409 * AgHG^{0.9285})] / AgHG) * 100] * 0.966$$

Intrepid Zone:

$$Ag\ Rec = [([AgHG - (0.4409 * AgHG^{0.9285})] / AgHG) * 100] * 0.966$$

Where:

- Ag Rec is the process plant silver recovery in %.
- AgHG is the process plant silver head grade in g/t.

The forecast gold and silver grade and recoveries based on the mine plan for 2020 to 2028 are shown in Table 13.21.

Table 13.21 Forecast annual head grades and recoveries for gold and silver

Year	Gold		Silver	
	Head grade (g/t)	**Recovery (%)**	**Head grade (g/t)**	**Recovery (%)**
2020	0.95	87.7%	2.5	57.1%
2021	1.02	88.4%	2.6	56.9%
2022	1.17	89.6%	2.3	56.9%
2023	1.19	90.4%	2.6	57.1%
2024	1.26	90.6%	3.4	59.1%
2025	1.25	90.0%	2.4	57.7%
2026	0.85	86.4%	2.2	56.4%
2027	0.84	86.6%	2.2	56.4%
2028	0.61	83.0%	2.2	56.5%
Total	**1.06**	**88.9%**	**2.5**	**57.3%**

14 Mineral Resource estimates

14.1 Introduction

The Mineral Resource estimates for the Rainy River Mine are based on two block models. These are for the Main and Intrepid Zones. The Main Zone was modelled and estimated by Mr Mauro Bassotti (formerly of New Gold), and the estimate for the Intrepid 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 Resources and Mineral Reserves dated 10 May 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 2018 and 2015 block model estimates include the results of all drilling and updated geologic interpretation carried out on the Property to 31 December 2017, given that no drilling was carried out on the Intrepid Zone after 2015. The Main Zone model has been depleted to reflect remaining Mineral Resources as of 31 December 2019: the Intrepid Zone has not been mined to date.

Table 14.1 Mineral Resource estimates at Rainy River

Area	Year of estimate	Author	Responsibility	Statement date
Intrepid	2015	El-Rassi	Nussipakynova	31 December 2019
Main Zone	2018*	Bassotti	Nussipakynova	31 December 2019

Note: *Based on the 2017 block model

The Mineral Resource estimate of the Main Zone is based on a block model completed in 2017 using Maptek's Vulcan software, and the estimate of the Intrepid Zone is based on a block model completed in 2015 using GEMS software.

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 Mineral Resource categories used in this report are consistent with those defined by CIM Definition Standards (2014).

Open pit Mineral Resources are reported at COGs of 0.3 g/t and 0.44 g/t AuEq for low-grade material and for direct processing material, respectively with the exception of the CAP Zone, which as seen in Section 13 has lower metallurgical recoveries. CAP Zone has a COG of 0.45 g/t AuEq for direct processing material. Underground Mineral Resources for all zones are reported at a COG of 2.0 g/t AuEq. Measured and Indicated Mineral Resources are estimated to total 23.1 million tonnes (Mt) at grades of 2.57 g/t Au and 6.9 g/t Ag, containing 1,914 koz of gold and 5,120 koz of silver. Inferred Mineral Resources are estimated to total 3.5 Mt at grades of 1.77 g/t Au and 2.4 g/t Ag, containing 198 koz of gold and 268 koz of silver.

Table 14.2 Mineral Resources as of 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000's)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Total Mineral Resources					
Measured	989	1.13	2.6	36	82
Indicated	22,139	2.64	7.1	1,878	5,037
Total M + I Mineral Resources	**23,127**	**2.57**	**6.9**	**1,914**	**5,120**
Total Inferred Mineral Resources	3,479	1.77	2.4	198	268

Notes:
- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Resources.
- The Mineral Resources are stated exclusive of Mineral Reserves.
- Mineral Resources are estimated using a long-term gold price of US$1,375 per troy oz and a long-term silver price of US$19 per troy oz. The exchange rate used was 1:1.30 US$/C$.
- Direct processing open pit Mineral Resources are estimated at an AuEq COG of 0.45 g/t for the CAP Zone and 0.44 g/t for the Non-CAP Zone. Low grade open pit Mineral Resources were estimated at an AuEq cut-off of 0.30 g/t. Gold equivalency was estimated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 90)].
- Open pit assumptions include:
  - Metal recoveries are variable dependent on metal head grade. At COG, the gold recoveries are as follows:
    - Direct Processing Ore
      CAP zone gold = 73.8%
      Non-CAP zone gold = 77.0%
    - Low Grade Ore
      CAP zone gold = 73.1%
      Non-CAP zone gold = 68.9%
  - Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
  - Open pit Mineral Resources are constrained by a conceptual pit shell.
  - Inferred open pit Mineral Resources include inferred material from within the Mineral Reserve open pit.
- Underground Mineral Resources are estimated at an AuEq COG of 2.00 g/t. Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 95)].
- Underground assumptions include:
  - Average gold and silver recoveries of 95% and 60%, respectively.
  - Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
- Effective date of Mineral Resources is 31 December 2019.
- The QP for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC.
- Totals may not compute exactly 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.

14.2 Mineral 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 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. 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.

For the Main Zone, 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 the Mineral Resource estimation. Vulcan software was used to prepare assay data for geostatistical analysis, construct the block model, prepare composite samples, estimate metal and bulk density values, and validate and tabulate the Mineral Resources. The geostatistical software Snowden Supervisor was used for variography, geostatistical analysis, and validation.

The Mineral Resource estimate of the Intrepid Zone is based on a block model completed in 2015 using GEMS software.

Interpolation of gold and silver grades for all models was completed using ordinary kriging (OK). Bulk density values were interpolated in the Main Zone using inverse distance squared (ID^2) and were assigned to the Intrepid Zone based on rock type.

In addition, a hardness block model was produced by AMC in 2019. Measurements of SAG hardness (A x b) and Bwi for 202 drill core samples were provided by New Gold. The samples were collected from 175 drillholes, mostly within mineralized domains. AMC estimated the hardness values using the ID^2 method. The estimation was carried out using Datamine software. The hardness samples were mainly collected from the four main mineralization zones at Rainy River: ODM, 433, HS, and CAP. No estimation of hardness in the Intrepid Zone was carried out.

The mean values of A x b and Bwi were applied for the host rocks and Intrepid Zone. The estimated and default values were added into the current open pit and underground block models, but not used in the estimation of Mineral Resources.

14.2.1 Mineral Resource database

The Rainy River Mineral Resource database has been exported as a series of Microsoft Excel files and includes drillhole 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 Mineral Resource models is provided in Table 14.3.

Table 14.3 Summary of Mineral Resource database

Item	Record count / details
Drillholes	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

Source: AMC from New Gold data.

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 inconstancies 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.

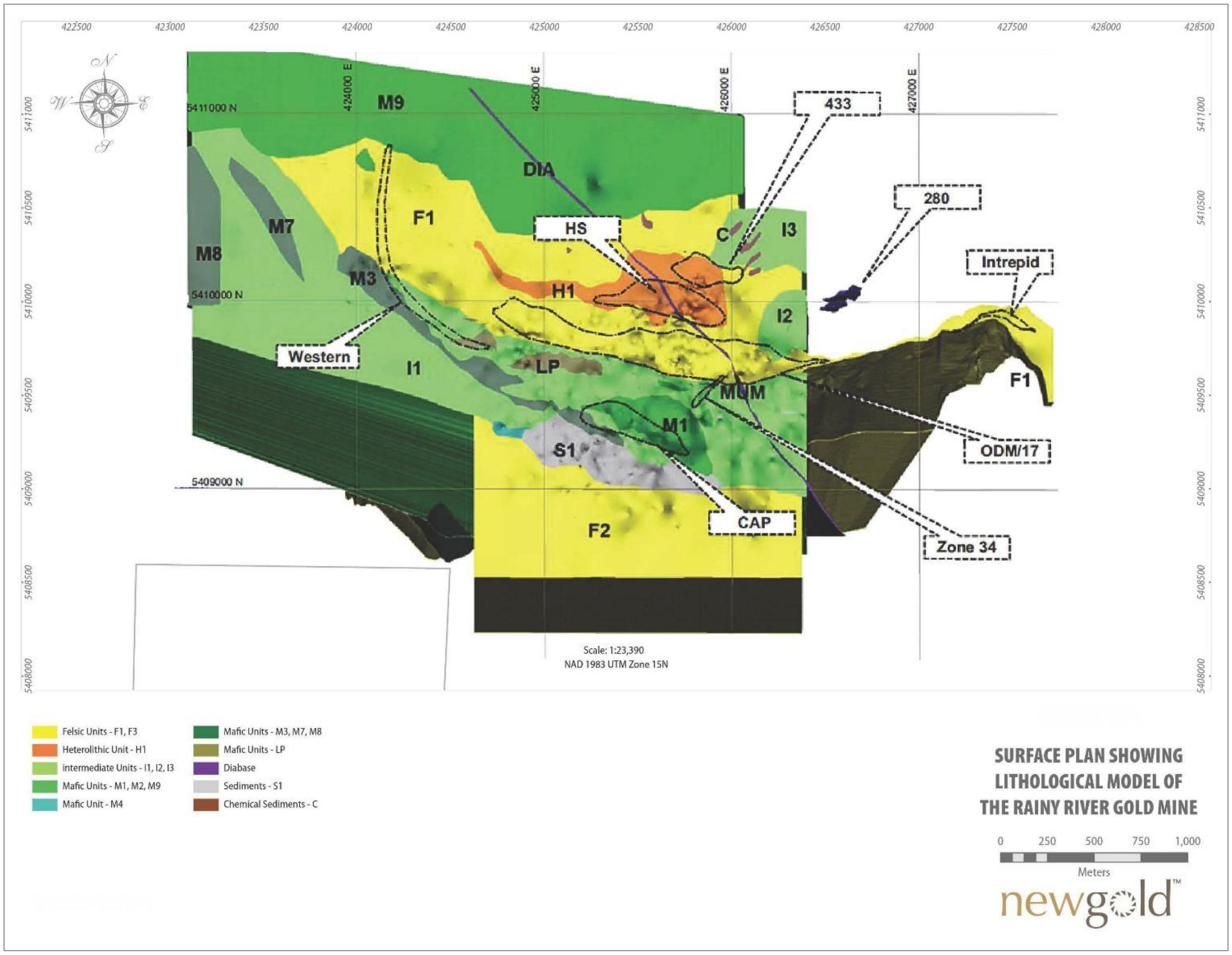
- Checked that collar locations plot in the correct location against the topography and there are no collars that are above or below the surface. Below topography collars were confirmed to match with open pit pre-stripping activities.
- Checked that all downhole survey dips are negative (no upward holes present).
- Checked that downhole survey azimuth readings are all in range of expected drilling deviation and not impacted by any erroneous effects.
- 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).

14.2.2 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. Intrepid Zone represents a satellite deposit located 1 km 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.

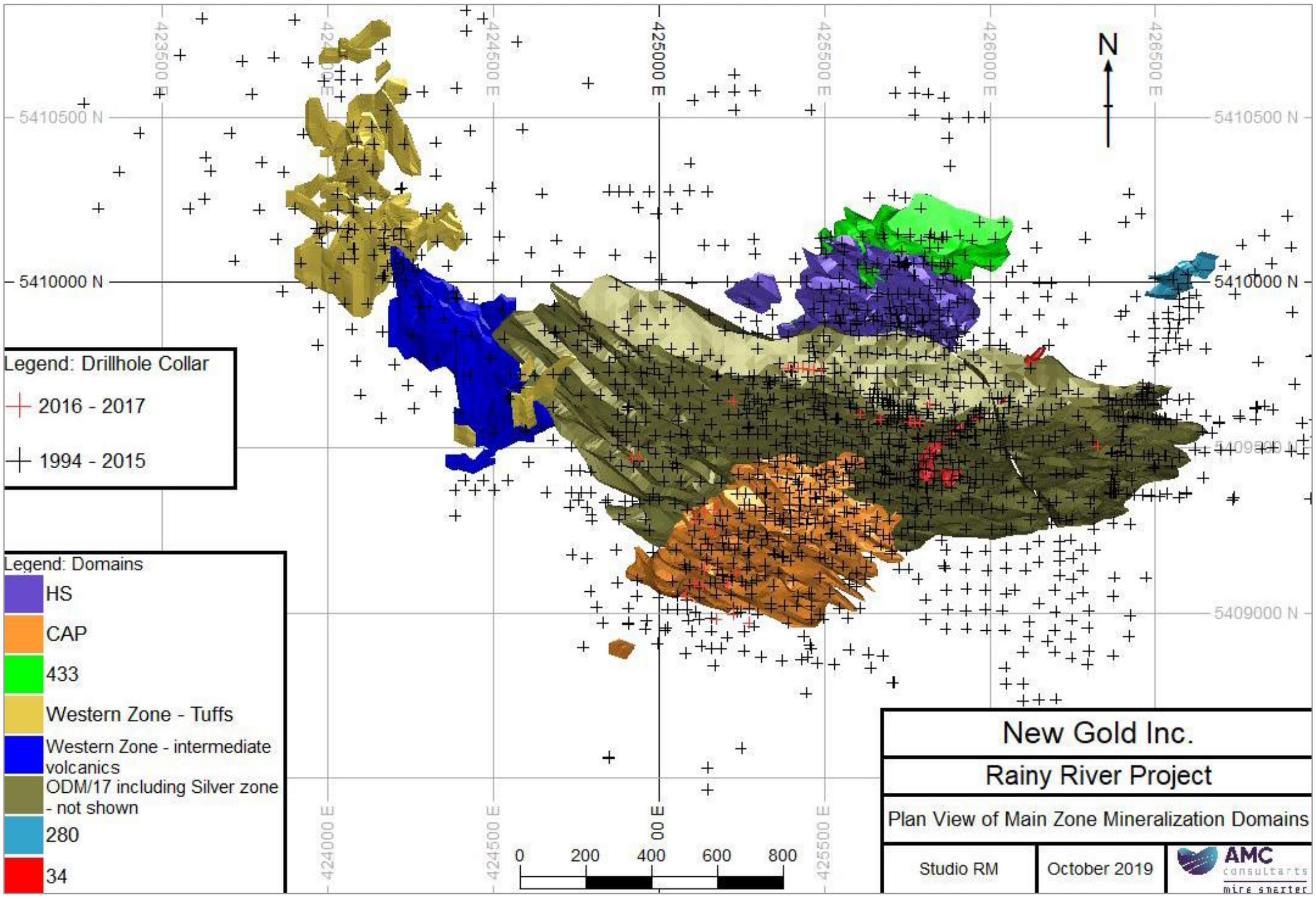
In 2017, New Gold updated the geological model for the deposit. The model comprises 3D wireframes delineating the major lithological units and zones of significant gold and silver mineralization. Lithologic domains were modelled in Leapfrog, and mineralization domains were modelled in GEMS, guided by drillhole 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 (ODM/17, 433, HS, CAP, Western, 280, and 34 zones) are shown in plan and isometric views in Figure 14.2 and Figure 14.3, respectively. The wireframes delineating the Intrepid and 34 Zones remain unchanged since the geologic model prepared by SRK in 2015. Mineralization domains defining the Intrepid Zone are shown in Figure 14.4.

Figure 14.1 Surface plan showing lithological model of the Rainy River Gold Project



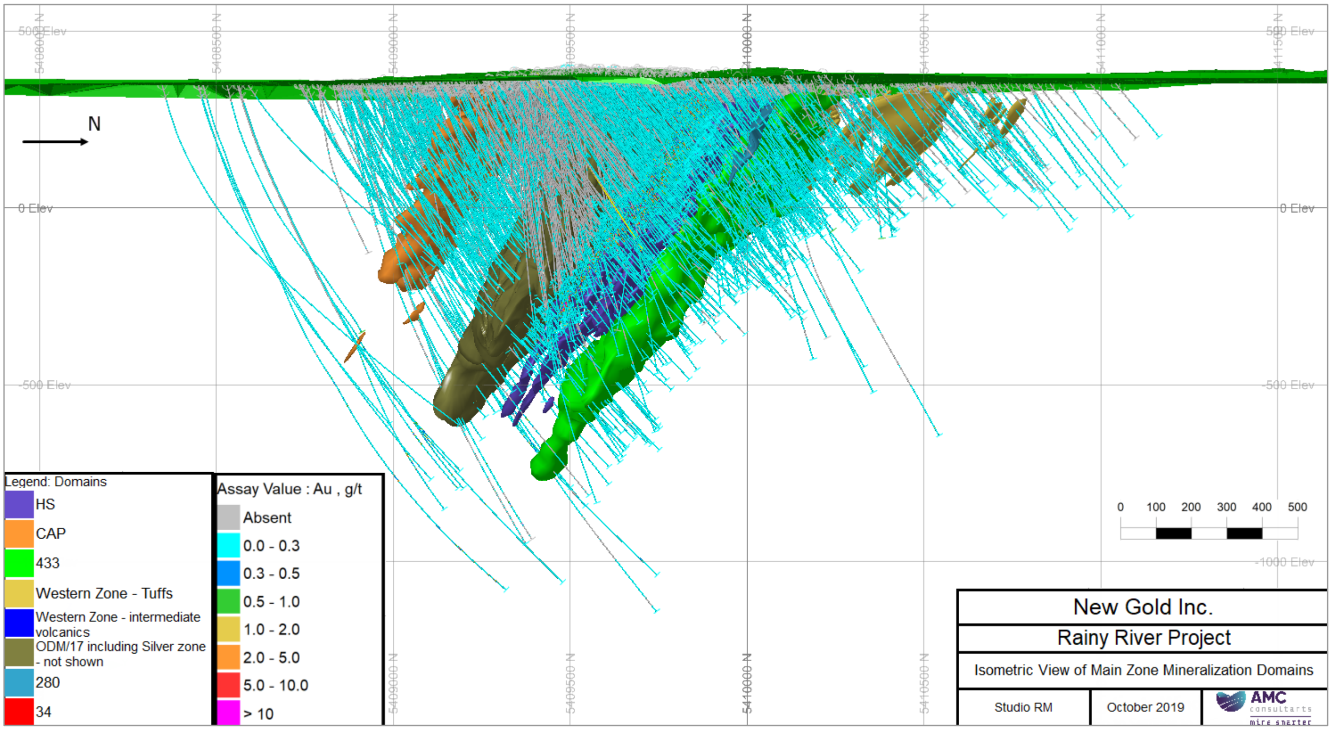
Source: New Gold 2019.

Figure 14.2 Plan view of Main Zone mineralization domains



Source: AMC 2019.

Figure 14.3 Isometric view of Main Zone mineralization domains



Source: AMC 2019.

14.2.2.1 ODM/17 Zone

The ODM/17 Zone is interpreted as a generally east-west trending, south-west 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 HW of the zone coincides with the top of a felsic fragmental volcaniclastic 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 HW 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 south-westerly 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 FW 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.

14.2.2.2 433 and HS zones

The 433 and HS zones form two zones of gold mineralization in the Main Zone. They are located within the FW 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 south-westerly 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.

14.2.2.3 Silver Zone

The Silver Zone (not shown in Figure 14.3) occurs in the FW 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 south-west 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 COG (3.0 m composites; less than 4.0 m waste), on inclined cross-sections oriented perpendicular to the south-westerly plunge of the silver mineralization.

14.2.2.4 Western Zone

The Western Zone represents a north-westerly 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 volcaniclastic rocks and conglomerate.
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.

14.2.2.5 CAP Zone

The CAP Zone occurs in the HW 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. North-east to north-west 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.

14.2.2.6 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.

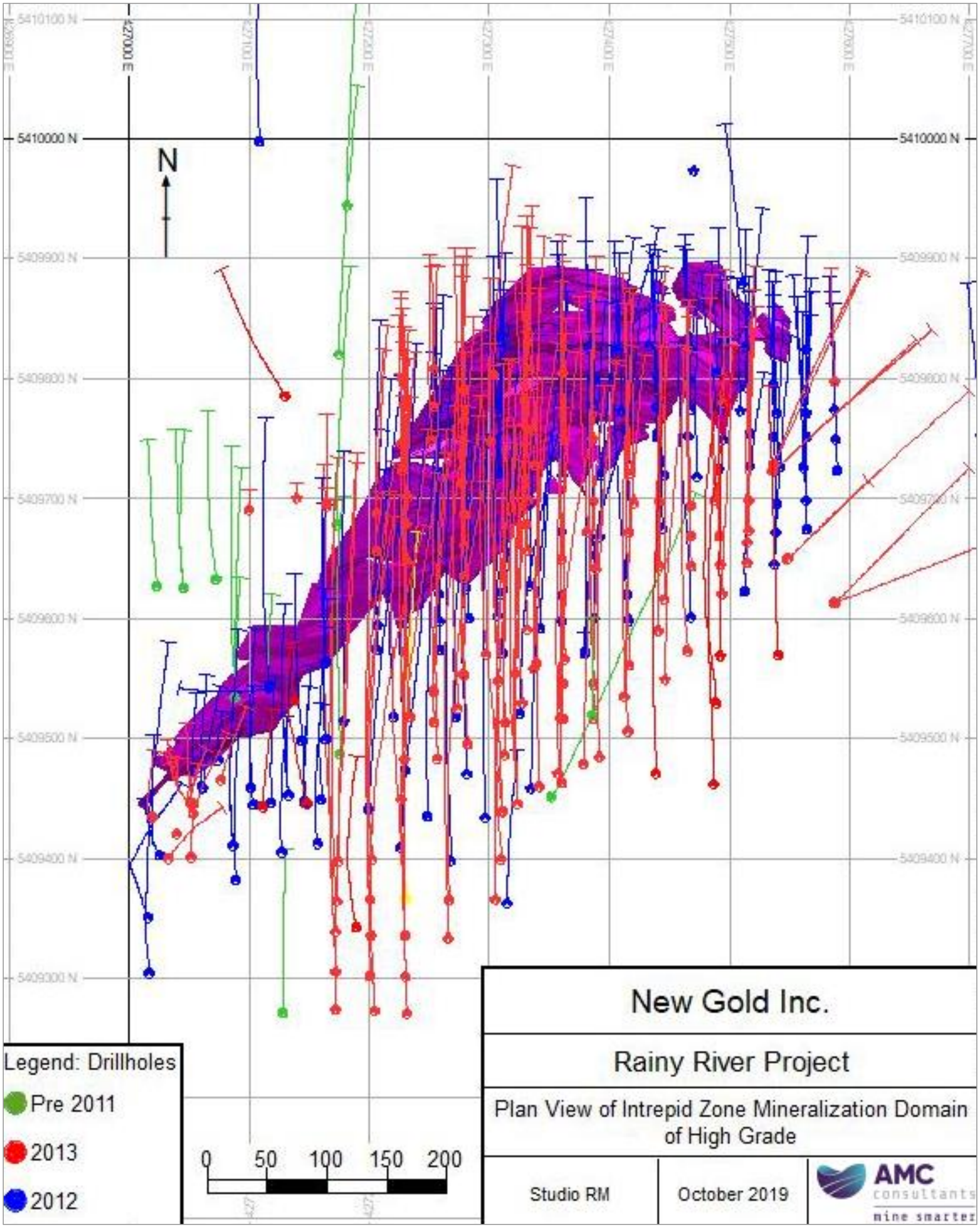
The Intrepid Zone has remained unchanged in the 2019 Mineral Resource estimation, with no new drilling data available. The general shape of the Intrepid Zone is shown in Figure 14.4.

Table 14.4 lists the associated domain codes for the different mineralization zones and grade domains at Rainy River.

14.2.2.7 34 Zone

The 34 Zone was modelled by site geologists initially and modified by SRK in 2015 using Leapfrog software and incorporating logged drillhole data. 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.

Figure 14.4 Plan view of Intrepid Zone high-grade domain



Source: AMC 2019.

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 to 803
	Silver	901 to 904
Lithology	Felsic Units	1001 / 1002
	Heterolithic Unit	2001
	Intermediate Units	3001 / 3002
	Mafic Units	4001 / 4011
	Mafic Units – LP	5001
	Mafic Intrusion	6001
	Diabase Dike	7001
	Sediments	8001
	Chemical Sediments	9001

Source: AMC from New Gold data.

14.3 Exploratory data analysis

14.3.1 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 Table 14.5 and Table 14.6 for gold and silver respectively.

Table 14.5 Statistical summary of gold assay data

Grade domain	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
ODM/17 Zone							
Low	101	76,384	0.00	448.56	0.24	2.20	9.22
Medium	110	4,319	0.00	104.51	0.67	2.46	3.66
	111	5,887	0.00	168.50	0.72	3.38	4.69
	112	4,739	0.00	166.00	0.72	3.01	4.18
	113	2,943	0.00	125.73	0.73	2.76	3.78
	114	1,103	0.03	1,080.00	1.97	33.69	17.12
	115	1,028	0.00	84.40	1.25	4.51	3.62
	116	804	0.00	7.49	0.65	0.82	1.25
High	120	2,693	0.01	746.33	1.96	9.38	4.79
	121	4,457	0.00	1,221.19	2.35	21.97	9.37
	122	4,457	0.01	482.00	2.68	12.99	4.85
	123	798	0.01	2,559.00	2.54	45.20	17.79
	124	2	0.03	1.54	0.79	0.92	1.17
	125	104	0.10	31.03	2.05	2.99	1.46
	126	324	0.01	281.00	7.57	21.36	2.82
34 Zone							
	200	246	0.00	10.00	0.22	0.68	3.17
280 Zone							
	280	269	0.01	51.68	0.62	2.97	4.79
433 Zone							
Low	300	11,456	0.00	1,000.00	0.29	5.65	19.80
Medium	310	3,069	0.00	121.20	0.88	4.01	4.57
High	320	1,113	0.01	4,158.63	5.68	108.86	19.17
HS Zone							
	400	10,114	0.00	707.80	0.58	7.50	12.95
CAP Zone							
	500	12,102	0.00	192.72	0.43	1.52	3.57
Intrepid Zone							
Low	700	2,691	0.00	37.60	0.40	0.97	2.45
Medium	710	1,385	0.01	25.80	1.11	1.90	1.72
High	720	1,053	0.02	528.00	4.28	18.01	4.21
Western Zone							
	801	125	0.01	13.40	0.35	0.78	2.24
	802	713	0.01	14.90	0.47	0.84	1.81
	803	1,256	0.00	1335.00	1.83	38.44	21.04
Silver Zone							
	901	227	0.00	9.56	0.28	0.91	3.18
	902	86	0.11	1,088.45	18.13	122.04	6.73
	903	215	0.00	28.87	0.98	2.42	2.46
	904	537	0.00	18.44	0.45	0.92	2.03

Grade domain	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
Lithological domains							
	1001	69,371	0.00	255.00	0.09	1.21	13.57
	1002	10,461	0.00	74.10	0.04	0.88	24.91
	2001	38,743	0.00	188.00	0.13	1.35	10.60
	3001	66,413	0.00	48.79	0.04	0.27	6.04
	3002	8,356	0.00	15.91	0.04	0.18	4.76
	4001	13,362	0.00	79.60	0.08	0.74	9.14
	4002	3,101	0.00	7.83	0.06	0.19	3.11
	4003	6,460	0.00	7.39	0.10	0.19	1.93
	4004	755	0.00	1.02	0.02	0.06	3.33
	4007	269	0.00	0.12	0.00	0.01	1.68
	4009	11,884	0.00	32.80	0.07	0.37	5.39
	4011	2,168	0.00	2.42	0.05	0.10	2.06
	5001	8,052	0.00	8.53	0.07	0.20	3.09
	6001	352	0.00	0.51	0.03	0.06	1.88
	7001	1,562	0.00	8.07	0.09	0.28	3.32
	8001	13,987	0.00	3.78	0.02	0.09	3.54
	9001	4,391	0.00	8.56	0.10	0.28	2.85

Notes: Stdv=standard deviation, CV= coefficient of variation. Gold is in g/t for minimum, maximum, and mean.

Table 14.6 Statistical summary of silver assay data

Domain code		Count	Minimum	Maximum	Mean	Stdv	CV
Low	101	75,786	0.01	2,020.00	2.03	9.03	4.45
Medium	110	4,318	0.03	430.00	2.94	8.33	2.84
	111	5,887	0.09	181.00	1.48	3.68	2.48
	112	4,739	0.09	65.00	1.44	2.53	1.76
	113	2,905	0.08	135.00	2.75	6.28	2.29
	114	1,103	0.22	256.00	4.84	10.64	2.20
	115	1,007	0.13	1,760.00	13.85	67.40	4.87
	116	776	0.50	773.00	14.38	32.92	2.29
High	120	2,687	0.10	332.00	4.80	12.45	2.60
	121	4,457	0.10	230.00	2.26	4.91	2.17
	122	4,457	0.09	190.00	2.39	5.11	2.14
	123	788	0.10	655.00	3.21	12.68	3.95
	124	2	0.80	20.80	10.80	12.25	1.13
	125	104	0.50	100.00	9.77	16.10	1.65
	126	316	0.60	2,580.00	77.20	202.79	2.63
34 Zone							
200		230	0.10	59.00	2.45	5.60	2.29
280 Zone							
280		269	0.10	11.40	0.95	1.34	1.41
433 Zone							
Low	300	11,419	0.01	294.00	0.78	2.23	2.87
Medium	310	3,069	0.01	100.00	0.99	3.20	3.22
High	320	1,113	0.10	439.00	1.62	12.03	7.45

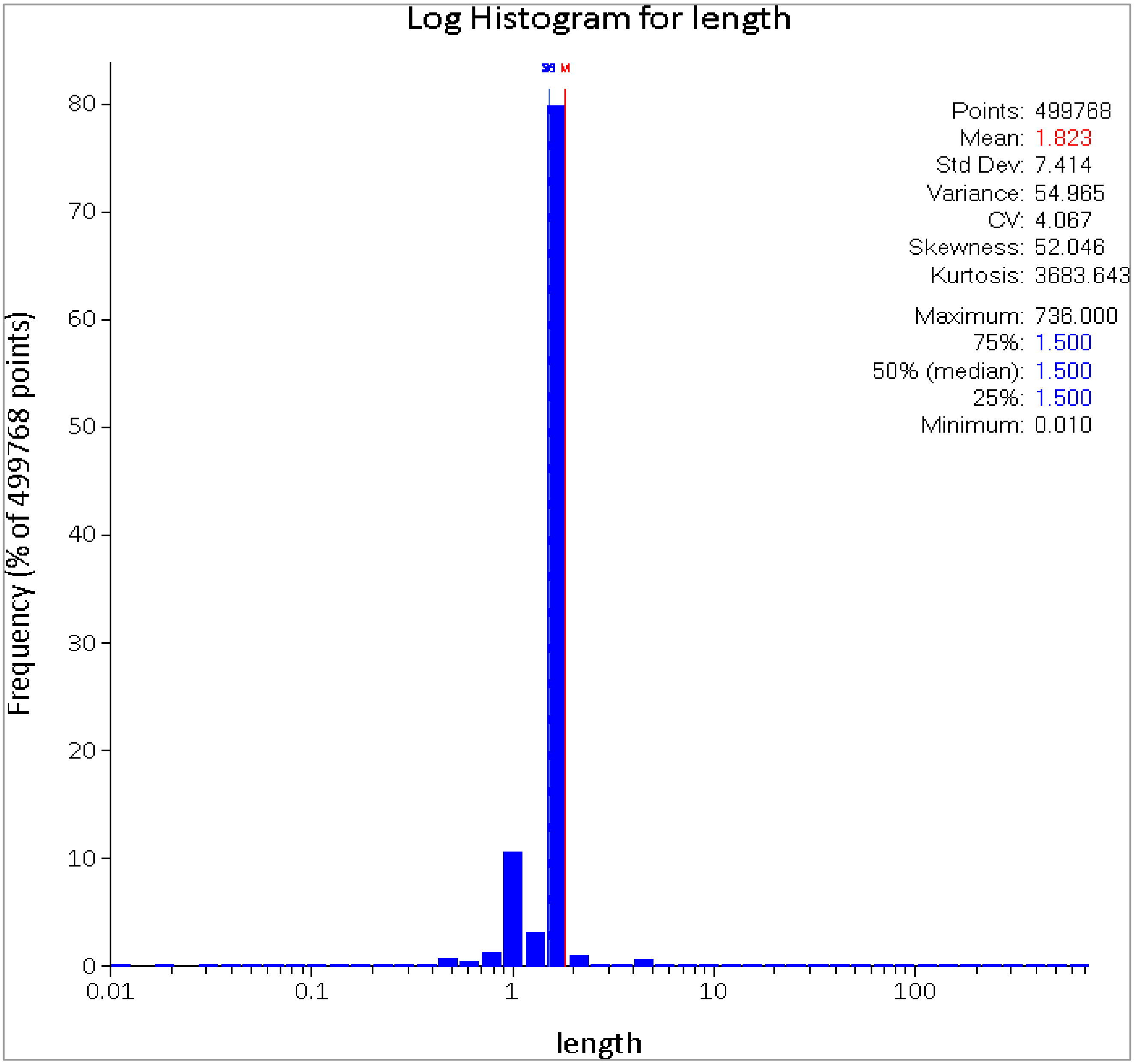
Domain code		Count	Minimum	Maximum	Mean	Stdv	CV
HS Zone							
400		10.114	0.03	1,000.00	1.27	8.83	6.97
Cap Zone							
500		12,101	0.04	1,288.98	2.29	8.91	3.89
Intrepid Zone							
Low	700	2,691	0.10	139.00	5.40	8.54	1.58
Medium	710	1,385	0.10	207.00	12.36	17.35	1.40
High	720	1,053	0.30	464.00	26.61	42.93	1.61
Western Zone							
	801	125	0.10	14.20	0.70	0.93	1.33
	802	713	0.06	48.40	1.03	2.80	2.71
	803	1,255	0.03	166.00	2.39	9.80	4.10
Silver Zone							
	901	227	0.45	1,050.00	58.31	95.38	1.64
	902	86	0.50	312.00	24.57	43.73	1.78
	903	214	0.40	384.00	18.44	31.60	1.71
	904	534	0.50	437.00	18.75	31.31	1.67
Lithological domains							
	1001	68,960	0.01	920.00	0.88	4.15	4.73
	1002	10,462	0.01	33.00	0.34	0.94	2.76
	2001	38,699	0.01	182.00	0.70	1.71	2.43
	3001	64,343	0.01	875.00	0.51	4.08	8.08
	3002	7,397	0.01	18.00	0.50	0.77	1.54
	4001	13,361	0.01	1,398.00	0.81	12.51	15.51
	4002	3,100	0.01	150.00	0.83	3.33	4.01
	4003	6,460	0.01	48.50	0.64	1.09	1.69
	4004	755	0.03	30.00	0.40	1.16	2.92
	4007	269	0.01	12.30	0.16	0.54	3.40
	4009	11,736	0.02	45.70	0.69	1.47	2.13
	4011	2,159	0.03	25.10	0.80	0.99	1.23
	5001	8,042	0.01	21.00	0.58	0.98	1.68
	6001	329	0.10	6.20	1.04	1.14	1.10
	7001	1,543	0.03	274.00	1.45	8.93	6.18
	8001	13,932	0.01	86.10	0.59	1.42	2.41
	9001	4,391	0.01	39.50	0.86	1.48	1.71

Notes: Stdv=standard deviation, CV= coefficient of variation. Silver is in g/t for minimum, maximum, and mean.

14.4 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



Source: New Gold, 2018.

14.5 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. No capping was applied to calcium or sulphur.

Table 14.7 Summary of gold and silver capping thresholds

Zone	Domain	Gold cap (g/t)	Gold percentile	No. capped	Silver 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
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

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 Table 14.9.

Table 14.8 Statistical summary of gold composites

Zone	Domain	Code	Count	Minimum	Maximum	Mean	Cut mean	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
		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
	High	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
		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

Notes: CV= coefficient of variation. Gold is in g/t for minimum, maximum, mean, and cut mean.

Table 14.9 Statistical summary of silver composites

Zone	Domain	Code	Count	Minimum	Maximum	Mean	Cut mean	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
		122	3,967	0.09	100.00	2.39	2.36	1.93	1.74
	High	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
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		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

Notes: CV= coefficient of variation. Silver is in g/t for minimum, maximum, mean, and cut mean.

14.6 Bulk density

The bulk density 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	Minimum	Maximum	Mean	Stdv	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	-	-	-	-	-	-
		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
	High	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		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	-	-	-	-	-	-
		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	-	-	-	-	-	-

Notes: Stdv=standard deviation, CV= coefficient of variation.

14.7 Block model parameters

Two block models, representing the open pit and underground volumes within the Main Zone, were created using Vulcan software. 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, which is unchanged since the previous Mineral 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

Source: AMC from New Gold data.

The sub-blocked model for the open pit Mineral 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. 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 Zone 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 Mineral Reserve pit shells, underground stopes, and infrastructure were assigned to the integrated block model.

Table 14.12 lists the block model parameters of 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

Source: AMC from New Gold data.

14.7.1 Variography

Variogram model parameters are unchanged from an earlier Mineral Resource estimate (SRK 2015) and are listed in Table 14.13 for gold, the primary economic metal. The gold and silver variogram parameters were published in the 2018 New Gold Technical Report and are not reproduced here. Variogram models were also completed by lithology domain (irrespective of mineralization domain) for calcium and sulphur.

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	10	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					
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
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

14.7.2 Interpolation parameters

Gold and silver grade interpolation was carried out using OK, and capped composite data. Grade interpolation was completed in two or three successive passes using search ellipse orientations and dimensions as described in Table 14.14 and composite sample selection and limits as described in Table 14.15. Interpolation parameters have remained largely unchanged from the earlier resource estimation by SRK in 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 domains using a three-pass approach and search ellipse and orientations based upon variogram models.

Table 14.14 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
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. The lithologic domains are used for the background model constraints.

Table 14.15 shows the block model interpolation parameters.

Table 14.15 Block model interpolation parameters

Model	Interpolation parameters	1st Pass	2nd Pass	3rd Pass
Open pit	Search type	Octant	Ellipsoidal	Ellipsoidal
	Minimum number of octants	2	-	-
	Maximum number of composites per octant	5	-	-
	Minimum number of composites	7	5	2
	Maximum number of composites	12	12	15
	Maximum number of composites per drillhole	5	3	-
Underground	Search type	Octant	Ellipsoidal	-
	Minimum number of octants	2	-	-
	Maximum number of composites per octant	5	-	-
	Minimum number of composites	3	2	-
	Maximum number of composites	8	15	-
	Maximum number of composites per drillhole	2	-	-
Intrepid	Search type	Octant	Ellipsoidal	Ellipsoidal
	Minimum number of octants	2	-	-
	Maximum number of composites per octant	5	-	-
	Minimum number of composites	5	3	2
	Maximum number of composites	10	15	15
	Maximum number of composites per drillhole	3	2	-

Bulk density was interpolated into the Main Zone mineralization domains using a single pass, ID^2 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 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 14.16.

Table 14.16 Main Zone default bulk density values

Domain	Bulk density (t/m^3)	Domain	Bulk density (t/m^3)
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

Source: AMC from New Gold data.

Gold and silver search orientations and ranges for the Intrepid Zone are listed in Table 14.17. These parameters remain unchanged since the estimate prepared by SRK in 2015, since which there has been no new data.

Table 14.17 Intrepid 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
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

Source: AMC from New Gold data.

14.8 New Gold block model validation

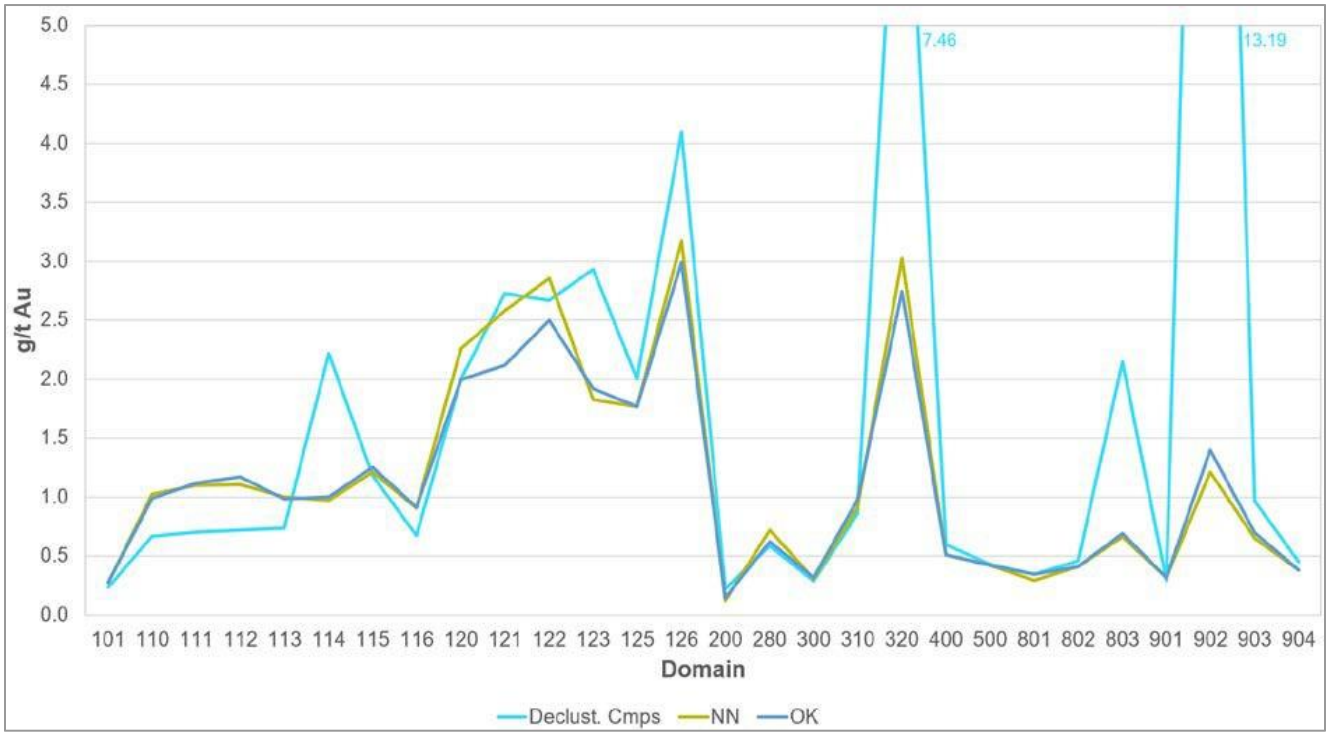
New Gold validated various modelling aspects of the Main Zone estimation. A list of the block model validations is provided below:

- Validation of wireframes.
- Volume comparison by domain between wireframes and block models.
- Validation of OK estimate by comparison to inverse distance cubed (ID^3) and nearest neighbour (NN) results.
- Swath plots.
- Visual inspection.
- Graphical comparison (histograms plots) of gold grades in block model and composites.
- Comparison of block model and composite statistics.

All validation methods showed satisfactory results.

Selected comparative statistics are shown for gold in Figure 14.6.

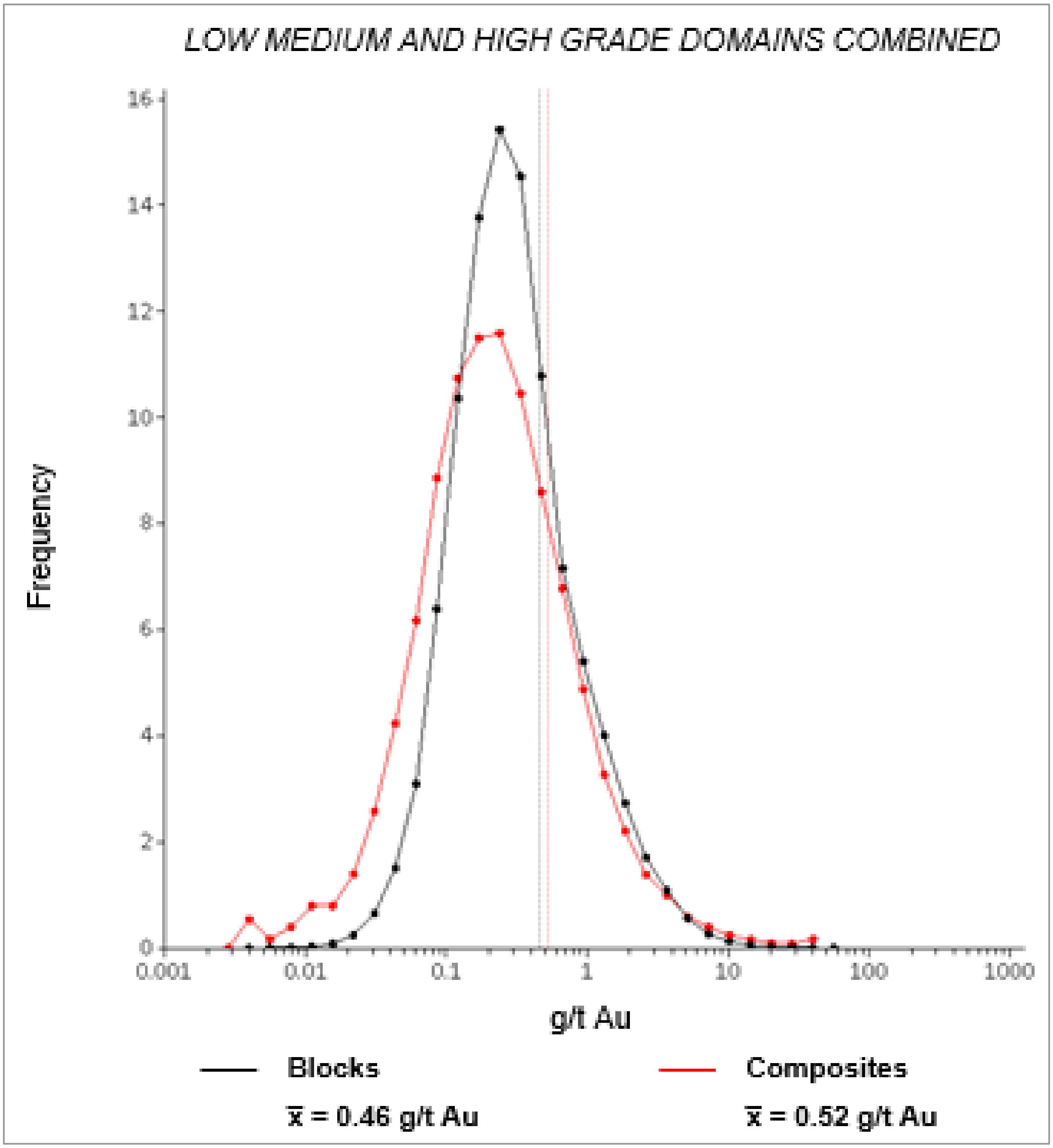
Figure 14.6 Graphical comparison of gold statistics for the Main Zone domains



Source: New Gold.

Figure 14.7 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 14.7 Gold histogram of blocks and composites within the ODM/17 Zone



Source: New Gold.

14.9 AMC block model validation

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

- Validation of drillhole database.
- Validation of wireframes and digital terrain mapping topographic surfaces.
- Review and checking of the statistics of selected raw samples and composites.
- 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 Mineral Resource and Mineral Reserve reporting.

14.9.1 Drillholes

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

Validation of drillhole 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.
- Downhole surveys.

AMC is of the opinion that the drillhole database is valid and suitable to estimate Mineral Resources.

14.9.2 Mineralized domains

Validation of the mineralized domains included the following checks:

- Verifying the mineralization domains for intercept, crossovers, and duplicates.
- Verifying the domaining code name.
- 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.

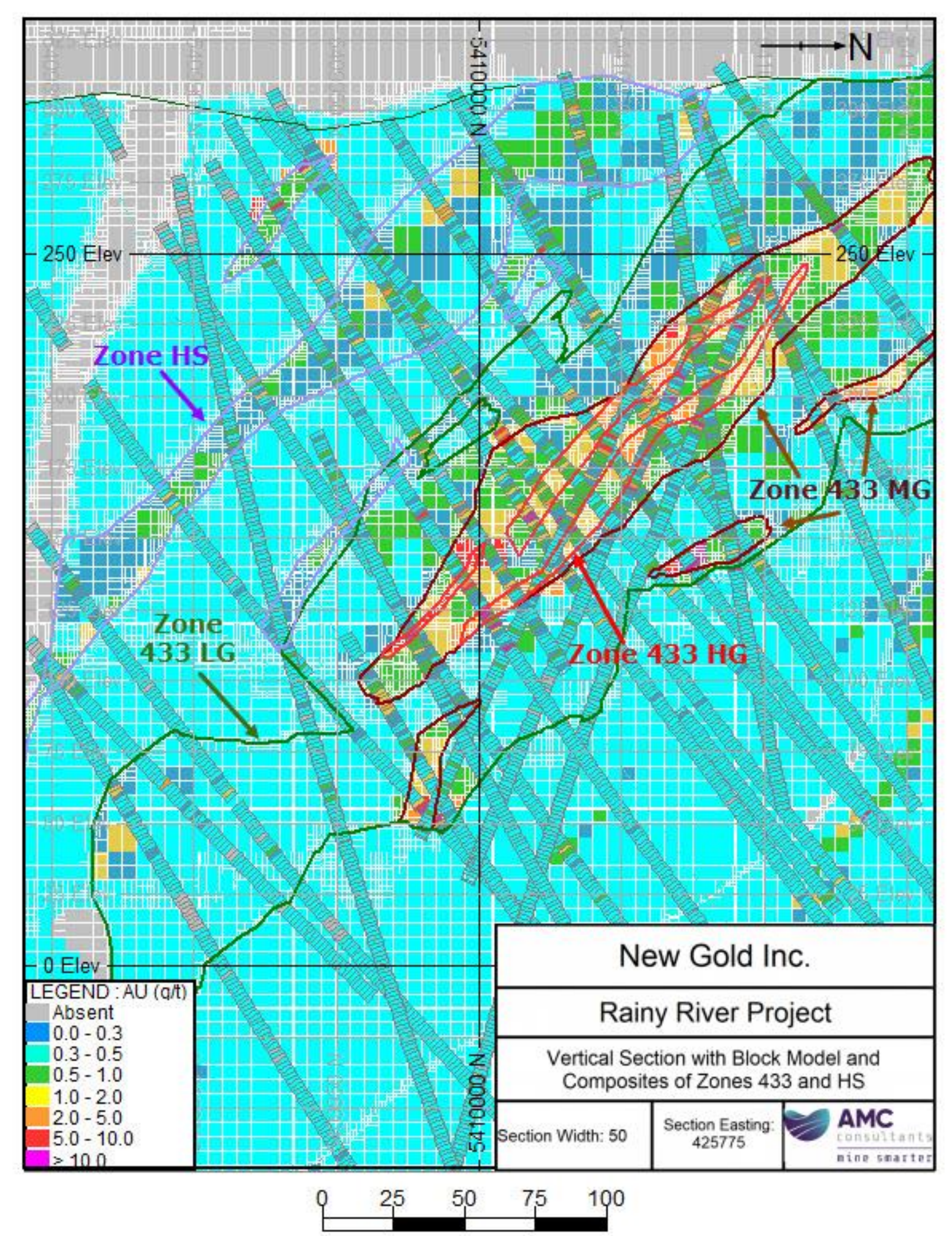
14.9.3 Lithology domains

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.

14.9.4 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 14.8 shows an example of the drillhole composite gold grades compared to the estimated block grades for the HS and 433 Zone domains.

Figure 14.8 Vertical section with block model and composites of zones 433 and HS



Source: AMC 2019.

AMC compared the average composite and block gold and silver grades by domain and found them to show good agreement as shown in Table 14.18.

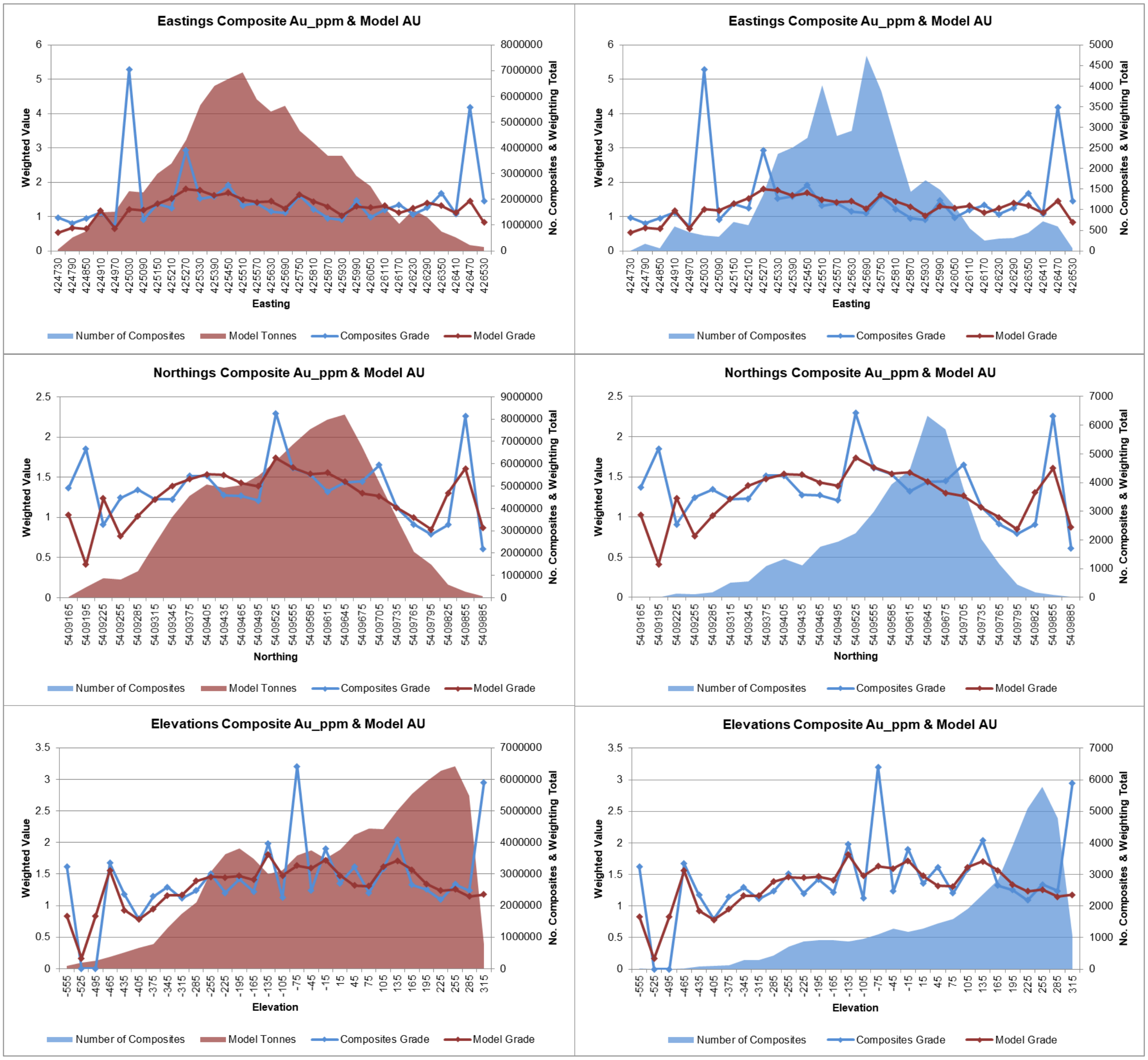
Table 14.18 Comparison of average composite and block gold and silver grades by domain

Domain	Mean (Au g/t) composite	Model	Mean (Ag g/t) 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 capped composites and blocks using swath plots on a domain basis. The swath plots show good agreement between drillhole and model grades. Figure 14.9 shows swath plots created by AMC for the gold grade distribution in the high- and medium-grade domains of the ODM/17 Zone.

AMC is of the opinion that the methods used to produce the Mineral Resource estimate at the Main Zone are in line with accepted industry practices.

Figure 14.9 Swath plots of gold grades for ODM/17 Zone



Source: AMC.

14.9.5 Intrepid model validation

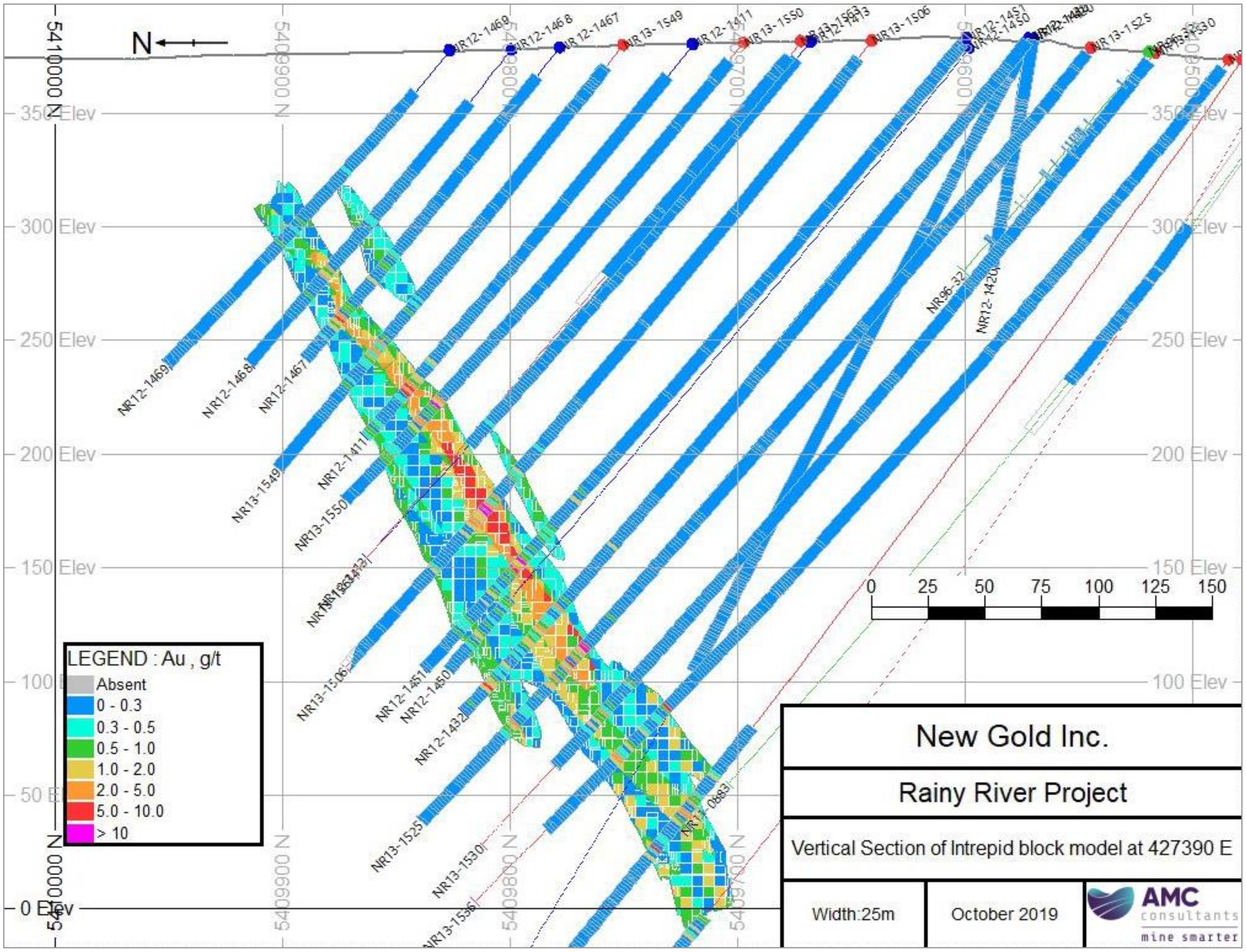
AMC's review of the Intrepid Zone block model included the following:

- Drillhole 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 drillhole data and mineralization style.
- Reasonability of the interpolation method for the volume of data and style of mineralization.
- Classified blocks with absent grade values.
- Agreement of block grades with supporting drillhole data.

In general, AMC found there to be good visual agreement between the block model and drillhole grades as shown in Figure 14.10.

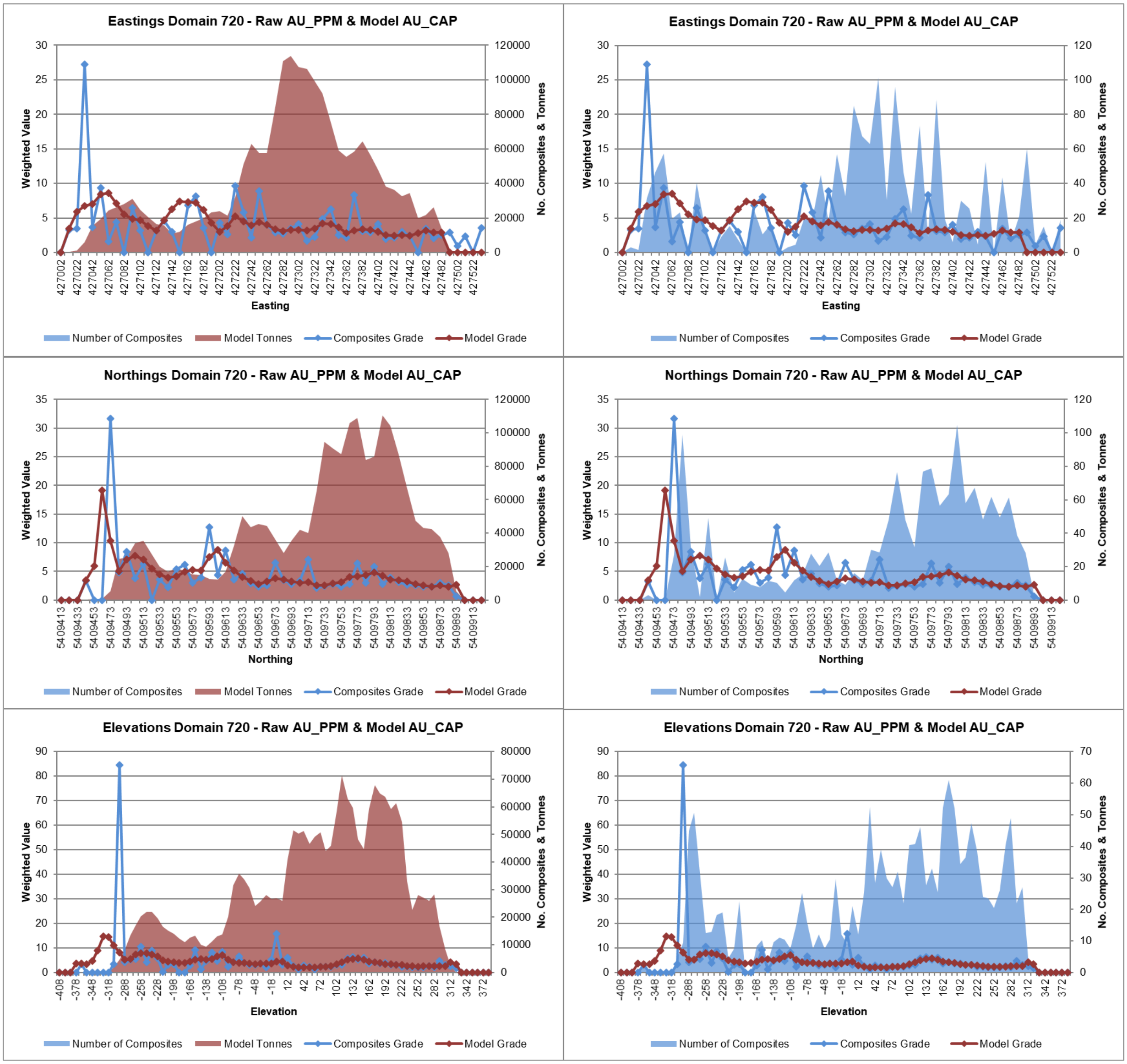
Figure 14.10 Vertical section showing gold in block model and drillholes at the Intrepid Zone



Source: AMC 2019.

Swath plots of the raw data for the capped composited data 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 14.11.

Figure 14.11 Swath plots of gold grades for Intrepid Zone



Source: AMC.

AMC did not find any significant errors that would have an adverse material impact on Mineral Resources. AMC is of the opinion that the methods used to produce the Mineral Resource estimate for the Intrepid Zone are in line with accepted industry practices.

14.10 Mineral Resource classification

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

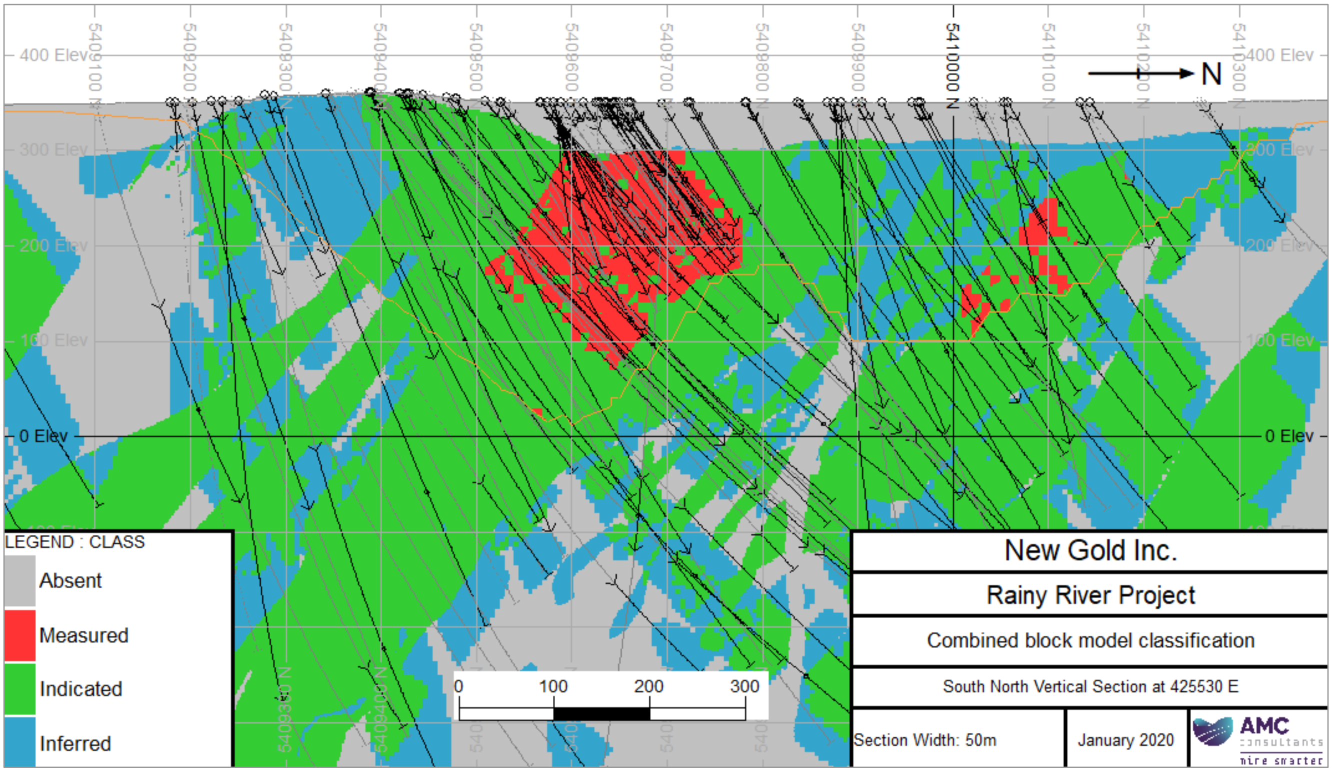
New Gold has assigned Measured classification where both drillhole density and bulk density measurements provide a high level of confidence in the geologic interpretation, grade continuity, and local grade and bulk density estimates. Currently, the ODM/17 and 433 Zones are the only areas with sufficient exploration drilling to support the classification of Measured Mineral Resources. The parameters used for Measured classification are summarized in Table 14.19.

Table 14.19 Classification criteria for Measured Mineral Resources

Interpolation parameters	**Criteria**
Zone	ODM17 / 433 Zone
Interpolation Method	Ordinary Kriging
Search Type	Octant (25 x 25 x 25)
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 drillhole	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. Confidence in the geological interpretation was also considered during the classification process. A vertical section displaying block class is shown in Figure 14.12. AMC is of the opinion that the classification criteria used to categorize blocks at Rainy River is reasonable.

Figure 14.12 Vertical section showing block model classification



Source: AMC 2020.

14.10.1 Cut-off grade

The Mineral Resource COG is expressed as an AuEq grade. The gold equivalency formula used to calculate COGs is provided below for both the OP and UG areas:

Open Pit AuEq = Au g/t + ((Ag g/t*19*60)/1375*90))

Underground AuEq = Au g/t + ((Ag g/t*19*60)/1375*95))

Where:

- Gold price = $1,375 per ounce
- Gold recovery = 90% open pit and 95% underground
- Silver price = $19 per ounce
- Silver recovery = 60%

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

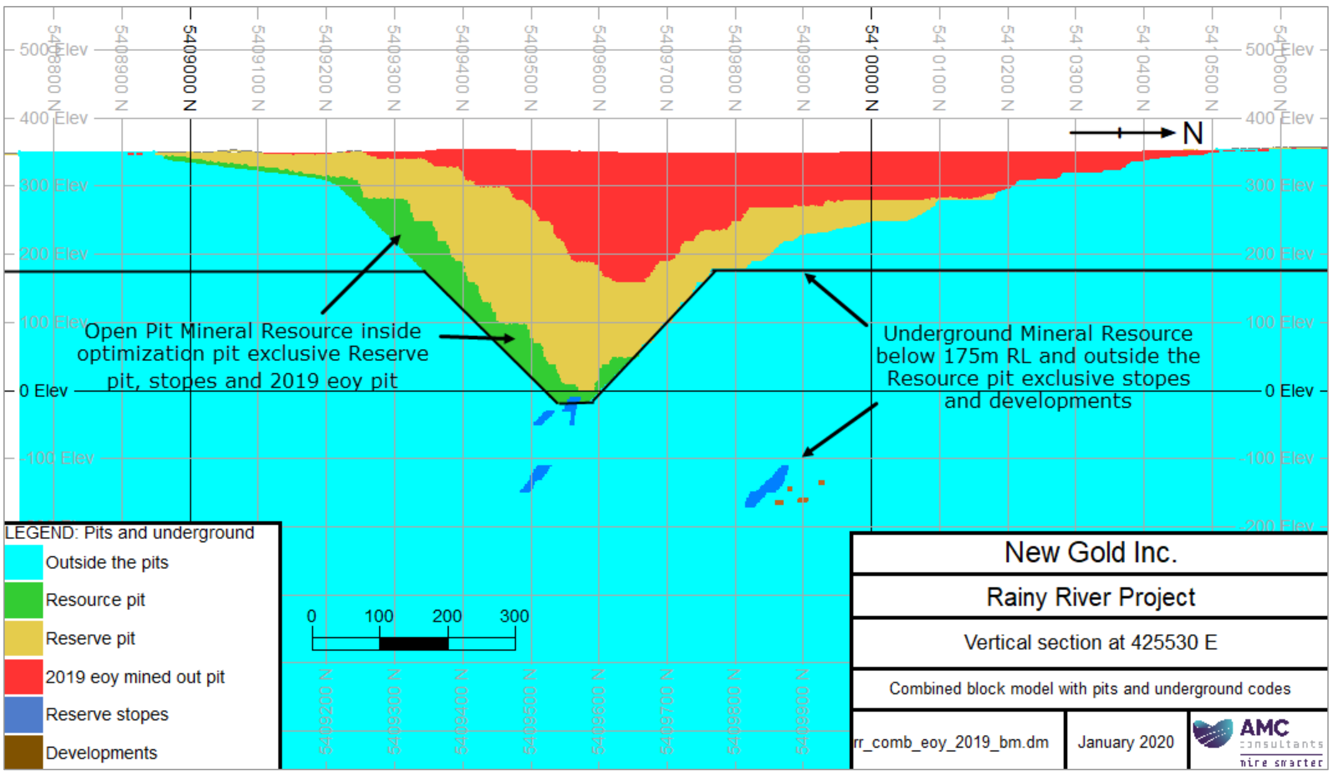
14.11 Mineral Resource reporting

Mineral Resources for the Rainy River Mine have been updated to 31 December 2019. They are reported based on AuEq COGs consistent with the mining methods envisioned for possible extraction in the future. The Mineral Resources at Rainy River are presented in Table 14.20. The Mineral Resources reported herein supersede the Mineral Resources reported previously in the 2018 New Golds Technical Report. 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 90% for gold and 60% 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 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 14.13 provides a schematic vertical section of the constraining limits of the open pit and underground Mineral Resources reported for the Rainy River Mine.

Figure 14.13 Mineral Resource reporting criteria



Source: AMC 2020.

Table 14.20 Mineral Resources as of 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000's)	Gold (g/t)	Silver (g/t)	Gold (k oz)	Silver (k oz)
Direct processing Mineral Resources					
Open pit					
Measured	695	1.46	2.9	33	64
Indicated	4,813	1.18	3.4	182	531
Sub-total open pit M + I	**5,508**	**1.21**	**3.4**	**214**	**596**
Inferred	2,015	0.61	1.8	39	114
Underground					
Measured	-	-	-	-	-
Indicated	14,866	3.49	9.1	1,669	4,331
Sub-total underground M + I	**14,866**	**3.49**	**9.1**	**1,669**	**4,331**
Inferred	1,297	3.76	3.5	157	146
Low grade Mineral Resources					
Open pit					
Measured	293	0.34	1.9	3	18
Indicated	2,460	0.34	2.2	27	175
Sub-total open pit M + I	**2,753**	**0.34**	**2.2**	**30**	**193**
Inferred	167	0.35	1.4	2	8
Total Mineral Resources					
Measured	989	1.13	2.6	36	82
Indicated	22,139	2.64	7.1	1,878	5,037
Total M + I Mineral Resources	**23,127**	**2.57**	**6.9**	**1,914**	**5,120**
Total Inferred Mineral Resources	3,479	1.77	2.4	198	268

Notes:

- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Resources.
- The Mineral Resources are stated exclusive of Mineral Reserves.
- Mineral Resources are estimated using a long-term gold price of US$1,375 per troy oz and a long-term silver price of US$19 per troy oz. The exchange rate used was 1:1.30 US$/C$.
- Direct processing open pit Mineral Resources are estimated at an AuEq COG of 0.45 g/t for the CAP Zone and 0.44 g/t for the Non-CAP Zone. Low grade open pit Mineral Resources were estimated at an AuEq cut-off of 0.30 g/t. Gold equivalency was estimated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 90)].
- Open pit assumptions include:
  - Metal recoveries are variable dependent on metal head grade. At COG, the gold recoveries are as follows:
    - Direct Processing Ore
      CAP zone gold = 73.8%
      Non-CAP zone gold = 77.0%
    - Low Grade Ore
      CAP zone gold = 73.1%
      Non-CAP zone gold = 68.9%
  - Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
  - Open pit Mineral Resources are constrained by a conceptual pit shell.
  - Inferred open pit Mineral Resources include inferred material from within the Mineral Reserve open pit.
- Underground Mineral Resources are estimated at an AuEq COG of 2.00 g/t. Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,375 * 95)].
- Underground assumptions include:
  - Average gold and silver recoveries of 95% and 60%, respectively.
  - Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
- Effective date of Mineral Resources is 31 December 2019.
- The QP for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC.
- Totals may not compute exactly due to rounding.

14.12 Comparison to previous Mineral Resource estimate

A comparison between the current Mineral Resource estimate, which is effective 31 December 2019, and the Mineral Resource statement dated 31 December 2018 is presented in Table 14.21. Principal changes since the 31 December 2018 estimate are:

- Ongoing depletion of Mineral Resources due to mining.
- Updated to account for changes in Mineral Reserves.
- Updated costs reflecting the current cost of operation at the mine (mine, process, G&A, and relevant sustaining capital requirements). Overall, costs have increased.
- Updated metallurgical models for gold and silver, resulting in lower average metal recoveries.
- Updated geotechnical model resulting in slightly lower overall pit slope angles.
- Updated methodology in the estimation of COGs.

Note that both estimates are based on the same 2017 and 2015 block models discussed above.

Table 14.21 Comparison of 2019 and 2018 Mineral Resources

Resource estimate date	Category	Tonnes & grade			Contained metal	
		Tonnes (000's)	Gold (g/t)	Silver (g/t)	Gold (k oz)	Silver (k oz)
Combined direct processing and stockpile Mineral Resources						
31 December 2019	Measured	989	1.13	2.6	36	82
	Indicated	22,139	2.64	7.1	1,878	5,037
	Measured & Indicated	23,127	2.57	6.9	1,914	5,120
	Inferred	3,479	1.77	2.4	198	268
31 December 2018	Measured	5,455	0.78	4.5	137	782
	Indicated	57,412	1.08	3.5	2,002	6,539
	Measured & Indicated	62,867	1.06	3.6	2,139	7,321
	Inferred	13,202	1.05	2.4	444	1007
Difference %	Measured	-82%	45%	-42%	-74%	-90%
	Indicated	-61%	144%	103%	-6%	-23%
	Measured & Indicated	-63%	142%	92%	-11%	-30%
	Inferred	-74%	69%	0%	-55%	-73%

Notes for the 31 December 2019 estimate are shown in the footnotes under Table 14.20.
Notes for the 30 June 2018 estimate:

- CIM (2014) Definition Standards were followed for Mineral Resources.
- 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.
- Mineral Resources are reported at COGs 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)/1,375*95)).
- Bulk density ranges from 2.70 t/m^3 to 3.08 t/m^3.
- Mineral Resources are exclusive of Mineral Reserves
- Mineral Resources that are not Mineral Reserves do not have demonstrated economic viability
- Open pit Mineral Resources are constrained by a conceptual pit shell.
- Totals may not compute exactly due to rounding.

Comparison of the current and previous Mineral Resource estimates indicates the following for total combined Mineral Resources:

- Total Measured and Indicated Mineral Resource tonnes have decreased by 63%, while gold grade has increased by 142% and the silver grade have increased by 92% resulting in contained gold metal decreasing by 11% and silver metal decreasing by 30%.
- Of note is that while there is a reduction of 63% in tonnes the gold metal has only decreased by 11%.
- Total Inferred Mineral Resource tonnes have decreased by 74%, gold grade increased by 69% and silver grade have not changed. The metal content of gold and silver decreased by 55% and 73% respectively.

The principal reasons for the reduction in Mineral Resources is the net result of the use of the updated input parameters, resulting in the updated resource conceptual pit shell being significantly smaller than the end-2018 conceptual pit shell, with a subsequent decrease in open pit Mineral Resources.

The updated underground Mineral Resources increased and has offset some of the open pit Mineral Resource losses. See Section 15 for details.

15 Mineral Reserve estimates

The Mineral Reserve estimates conform to CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) and only include Measured and Indicated Resources.

The open pit Mineral Reserves have been prepared by New Gold under the guidance of Mr Francis J. McCann, P.Eng., a mining engineer employed by AMC. Mr McCann is independent of New Gold and takes QP responsibility as defined in NI 43-101 for the open pit Mineral Reserve estimate.

The underground Mineral Reserves have been prepared by AMC under the guidance of Mr Herbert A. Smith, P.Eng., a mining engineer employed by AMC. Mr Smith is independent of New Gold and takes QP responsibility as defined in NI 43-101 for the underground Mineral Reserve estimate.

A summary of the Mineral Reserve estimates at Rainy River is presented in Table 15.1.

Table 15.1 Summary of Mineral Reserves – effective 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	**Gold (g/t)**	**Silver (g/t)**	**Gold (koz)**	**Silver (koz)**
Total Mineral Reserves					
Open pit (including stockpile)					
Proven	27,331	0.88	2.0	779	1,740
Probable	46,145	0.88	2.4	1,308	3,492
Sub-total open pit	**73,476**	**0.88**	**2.2**	**2,087**	**5,231**
Underground					
Proven	-	-	-	-	-
Probable	4,096	4.17	7.8	549	1,034
Sub-total underground	**4,096**	**4.17**	**7.8**	**549**	**1,034**
Total					
Proven	27,331	0.88	2.0	779	1,740
Probable	50,241	1.15	2.8	1,857	4,526
Total Mineral Reserves	**77,572**	**1.06**	**2.5**	**2,636**	**6,266**

Notes:
- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Reserves.
- Refer to the footnotes to Table 15.7 for prices, cut-off, recoveries, etc.
- Totals may not compute exactly due to rounding.

The Mineral Reserves reported herein supersede the Mineral Reserves reported previously at year-end 2018 by New Gold for the Rainy River Mine.

AMC is not aware of any known mining, metallurgical, infrastructure, permitting, and / or other relevant factors that could materially affect the stated Mineral Reserve estimates.

15.1 Open pit Mineral Reserve estimates

Open pit Mineral Reserves were estimated by New Gold through the application of a mine design, phasing sequence and subsequent mine plan to convert the Measured and Indicated Mineral Resources to Proven and Probable Mineral Reserves. The estimate is based upon the application of a typical truck-shovel open pit mining operation to extract the Mineral Reserve.

15.1.1 Material type classification

There are two principal ore type classifications used to identify Mineral Reserves at Rainy River: direct processing ore (DPO) and low-grade ore (LGO). DPO is subsequently sub-classified as high-grade ore (HGO) and medium-grade ore (MGO). Additionally, material is identified as being associated with the CAP Zone or labelled as Non-CAP Zone. The CAP Zone is described in Section 7 and has been identified as having a different metallurgical recovery response than other mineralized zones within the deposit in Section 13. Non-Cap Zone is a reference to all other mineralized zones not identified as CAP Zone within the open pit deposit. Table 15.2 presents a breakdown of the material classifications and their respective COG.

Table 15.2 Material classifications

Ore type	AuEq (g/t)	
	CAP Zone	Non-CAP Zone
Direct processing ore		
High-grade ore	≥ 0.80	≥ 0.80
Medium-grade ore	≥ 0.49 & < 0.80	≥ 0.46 & < 0.80
Low-grade ore		
Low-grade ore	≥ 0.30 & < 0.49	≥ 0.30 & < 0.46

DPO is material that meets the requirements of the breakeven COG definition as stated in the CIM Estimation of Mineral Resources and Reserves Best Practice Guidelines (2019), being "The lowest grade or value of material that can be mined and processed at an operating profit, considering all applicable costs".

LGO is material that meets the requirements of the marginal COG definition as stated in the CIM Estimation of Mineral Resources and Reserves Best Practice Guidelines (2019), being "…cut-off grades or values can consider current metal prices, sunk costs, appropriate variable costs, material destinations, and equipment capacities. These COGs or values apply only to material which must be excavated due to the normal course workflow of the mining operation". In the case of Rainy River, the LGO covers all costs including re-handle, apart from G&A and fixed process costs. This material is used only when excess process plant capacity exists, and predominantly to supplement process plant feed at the end of the open pit operating life when the underground is still operational.

15.1.2 Open pit resource mine planning block model

The resource model used for open pit mining is a regularized block model (regularized model) that was developed by New Gold in Vulcan from the resource model discussed in Section 14 of this report. The regularized model has block dimensions of 10 m in the X (east) direction by 10 m in the Y (north) direction by 10 m in the Z (vertical) direction. These block dimensions were selected by New Gold to adequately represent the dimension of a selective mining unit appropriate for the size of the chosen loading units.

In 2019 New Gold contracted G Mining Services Inc. (G Mining) to undertake two block model dilution studies (February 2019 G Mining Services; August 2019 G Mining Services) on the regularized model to improve the prediction of tonnes and grade of ore to be extracted from the mine. Based on the results, New Gold selected the following modifying factors to be applied to the regularized block model:

- A 4.0 m dilution skin was applied to blocks above 290 m. This relatively high dilution skin represents the high variability of the overburden-rock contact.
- A 1.3 m dilution skin was applied to blocks below 290 m to -250 m. This dilution skin was selected as a best representation of the dilution applicable to blocks ≥ 0.8 g/t gold in the

model, while maintaining an overall good representation of model dilution for blocks ≥ 0.3 g/t gold.

- Below -250 m, the regularized model was not modified. This elevation is deemed sufficiently low to ensure the diluted model covers the area of interest for any open pit.
- Dilution flags were assigned to blocks with < 0.3 g/t gold that were surrounded by eight ≥ 0.3 g/t gold blocks on the bench, implying that these blocks could not be mined independently.
- Ore loss flags were assigned to blocks with ≥ 0.3 g/t gold that were surrounded by eight < 0.3 g/t gold blocks on the bench, implying that these blocks could not be mined independently.

Note that the dilution skins are applied on a hierarchical designation as follows:

- Blocks ≥ 0.8 g/t gold can be diluted with dilution skins of blocks with lower grades only.
- Blocks 0.5 g/t ≤ gold < 0.8 g/t gold can be diluted with dilution skins of blocks with lower grades only.
- Blocks 0.3 g/t ≤ gold < 0.5 g/t gold can be diluted with dilution skins of blocks with lower grades only.

Overall net impact of the modifying factors applied against the regularized model at a fixed gold COG of 0.3 g/t is an increase of 3% in ore tonnes and a 4% decrease in grade. This diluted model is designated as the mine planning resource model.

2019 Reconciliation of Measured and Indicated Resource within the mine planning resource model to ore mined ex-pit, as measured by the fleet management system installed at Rainy River, is as follows:

Table 15.3 Reconciliation January – December 2019

Model	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Model results & ex-pit values					
Regularized resource model (OR-1)	7,543	1.03	1.71	250	414
Mine planning resource model (OR-2)	6,393	1.06	1.74	218	357
GC model	7,120	1.00	1.51	229	345
Ex-pit mined (EP)	6,845	0.97	1.83	214	403
Reconciliation regularized resource model					
GC vs OR-1	94.4%	97.1%	88.3%	91.6%	83.3%
EP vs OR-1	90.7%	94.5%	107.4%	85.7%	97.4%
EP vs GC	96.1%	97.3%	121.6%	93.6%	116.9%
Reconciliation mine planning resource model					
GC vs OR-2	111.4%	94.2%	86.7%	104.9%	96.5%
EP vs OR-2	107.1%	91.7%	105.4%	98.2%	112.9%
EP vs GC	96.1%	97.3%	121.6%	93.6%	116.9%

The mine planning resource model reconciliation provides a better overall prediction of tonnes, grade, and contained metal when compared to the regularized resource model. Modifying factors should be reviewed as new mining areas are exposed and additional reconciliation information is gathered to continue validating the model performance.

15.1.3 Open pit metallurgical recoveries

Predictive metal recovery curves have been developed for the open pit ores being extracted from the CAP Zone and Non-CAP Zone. Details of the development of these curves are provided in Section 13.

The predictive gold recovery formulae are as follows:

CAP Zone:

$$\text{Au Rec} = ([\text{AuHG} – (0.2497 * \text{AuHG}^{1.015}) - 0.007)] / \text{AuHG}] * 100$$

Non-CAP Zone:

$$\text{Au Rec} = \min(95,([\text{AuHG} - (0.0173 + (0.000616 * P_{80}) + (0.0483 * \text{AuHG}) – (0.00727 * \text{AuHG}^3)] / \text{AuHG}) * 100)$$

Note that the proceeding Non-CAP Zone formula has been capped at a maximum gold recovery of 95%.

Where:

- Au Rec is the gold recovery in %.
- AuHG is the gold head grade in g/t.
- P_{80} is the hydrocyclone overflow P_{80} in µm. For recovery estimation within the block model, this is set to the expected 100 um average.

New Gold has developed similar predictive formulas for silver recovery from metallurgical testwork programs (Kenny 2016). These predictive formulas are as follows:

CAP Zone:

$$\text{Ag Rec} = [([\text{AgHG} – (0.3868 * \text{AgHG}^{0.9174})] / \text{AgHG}) * 100] * 0.966$$

Non-CAP Zone:

$$\text{Ag Rec} = [([\text{AgHG} – (0.4409 * \text{AgHG}^{0.9285})] / \text{AgHG}) * 100] * 0.966$$

Where:

- Ag Rec is the silver recovery in %.
- AgHG is the silver head grade in g/t.

15.1.4 Open pit COG

The open pit COG was calculated by AMC using metal prices, operating costs, applicable sustaining capital costs and exchange rates provided by New Gold or developed from New Gold's preliminary 2020 Budget and LOM financial and cost models. Table 15.4 summarizes the open pit COG assumptions. An AuEq COG of 0.49 g/t for CAP Zone and 0.46 g/t for Non-CAP zone material was used for the estimation of Direct Processing Mineral Reserves. Low grade Mineral Reserves are set at an AuEq COG of 0.30 g/t for all rock types.

Table 15.4 Open pit COG calculation parameters

Parameter field	Unit	Open pit parameter value	
Metal prices			
Gold	$/oz	1,275.00	
Mining cost			
Ore	$/t mined	2.27	
Waste	$/t mined	2.43	
Incremental per bench (below 340 m elevation)	$/t mined	0.03	
Re-handle	$/t ore	1.47	
Sustaining capital	$/t mined	0.66	
Process cost			
Process base cost	$/t milled	7.65	
Process variable cost	$/t milled	3.52	
Sustaining capital	$/t milled	0.05	
Tailings management	$/t milled	2.29	
Treatment & refining			
Gold	$/oz recoverable	1.89	
Silver	$/oz recoverable	0.95	
Royalties			
Gold	$/oz recoverable	8.19	
Silver	$/oz recoverable	0.10	
G&A			
G&A	$/t processed	3.39	
Gold recovery at cut-off			
CAP zone			
Direct processing ore	%	73.9	
Low grade-ore	%	72.6	
Non-CAP zone			
Direct processing ore	%	78.2	
Low grade-ore	%	66.0	
COG	**Unit**	**Calculated**	**Utilized**
CAP zone			
Direct processing ore	g/t AuEq	0.49	0.49
Low grade-ore	g/t AuEq	0.24	0.30
Non-CAP zone			
Direct processing ore	g/t AuEq	0.46	0.46
Low grade-ore	g/t AuEq	0.27	0.30

The COG is expressed as an AuEq grade which is estimated as follows:

AuEq = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 90)]

Where, the factors in the equivalence calculation are:

- Gold price $1,275/oz
- Silver price $17/oz
- Gold recovery 90% (estimated preliminary overall average)
- Silver recovery 60% (estimated preliminary overall average)

Low-grade ore COGs utilized are slightly higher than the calculated COGs due to uncertainty related to the applicability of the metal recovery curves at lower grades where testwork is more limited.

The QP finds the open pit COG calculation is considered to be appropriate for the deposit based upon the assumptions used.

Low grade Mineral Resources do not cover fixed process costs or site G&A costs. As such, they are not included within the open pit optimization process but are rather estimated based on the material contained within the resulting pit design developed. These Mineral Reserves are included in the mine plan when excess process plant capacity exists, principally at the end of life of the open pit to supplement the process plant feed coming from the underground.

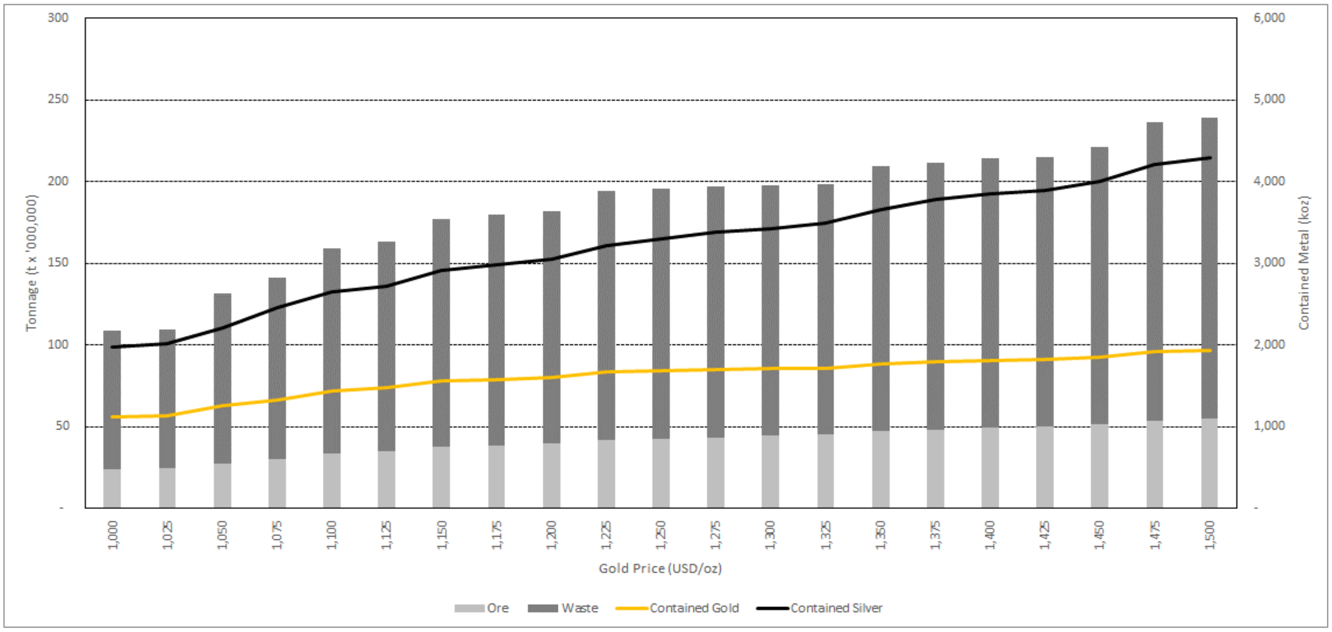
15.1.5 Open pit optimization

The pit optimization was conducted by AMC on the mine planning resource model described in Section 15.1.2 using metal prices of $1,275/oz gold and $17/oz silver. The parameters used for open pit optimization are provided in Table 15.4. Only Measured and Indicated Mineral Resources were included in the pit optimization process. Low grade Mineral Resources, as described in the previous section, are excluded from the pit optimization process. GEOVIA Whittle™ was the software used for the open pit optimization.

Preliminary slope estimates were included in the open pit optimization process utilizing overburden slope recommendations from Golder, and hard rock slope recommendations from SRK as presented in Section 16. The overburden slope angle was maintained at a constant 8:1 slope (horizontal:vertical) in all directions for the optimization; however, the hard rock design criteria of SRK were modified to represent overall slope angles, based on the impact resulting from the conceptual superposition of mine haul road and geotechnical safety berm positions required per the design criteria. Hard rock overall slope angles vary by zone from 39° to 53°.

Open pit optimization results at incremental gold metal price are provided in Figure 15.1.

Figure 15.1 Open pit optimization results at incremental gold metal price



Note: Ore tonnages and contained metal reflect DPO material only. Impact of LGO and existing stockpiles is not represented in the open pit optimization charts.

As the optimization results illustrate, the design metal price of $1,275/oz gold is in the centre of a relatively flat area of the optimization results, indicating a relative stability of the open pit size over a range of gold metal prices.

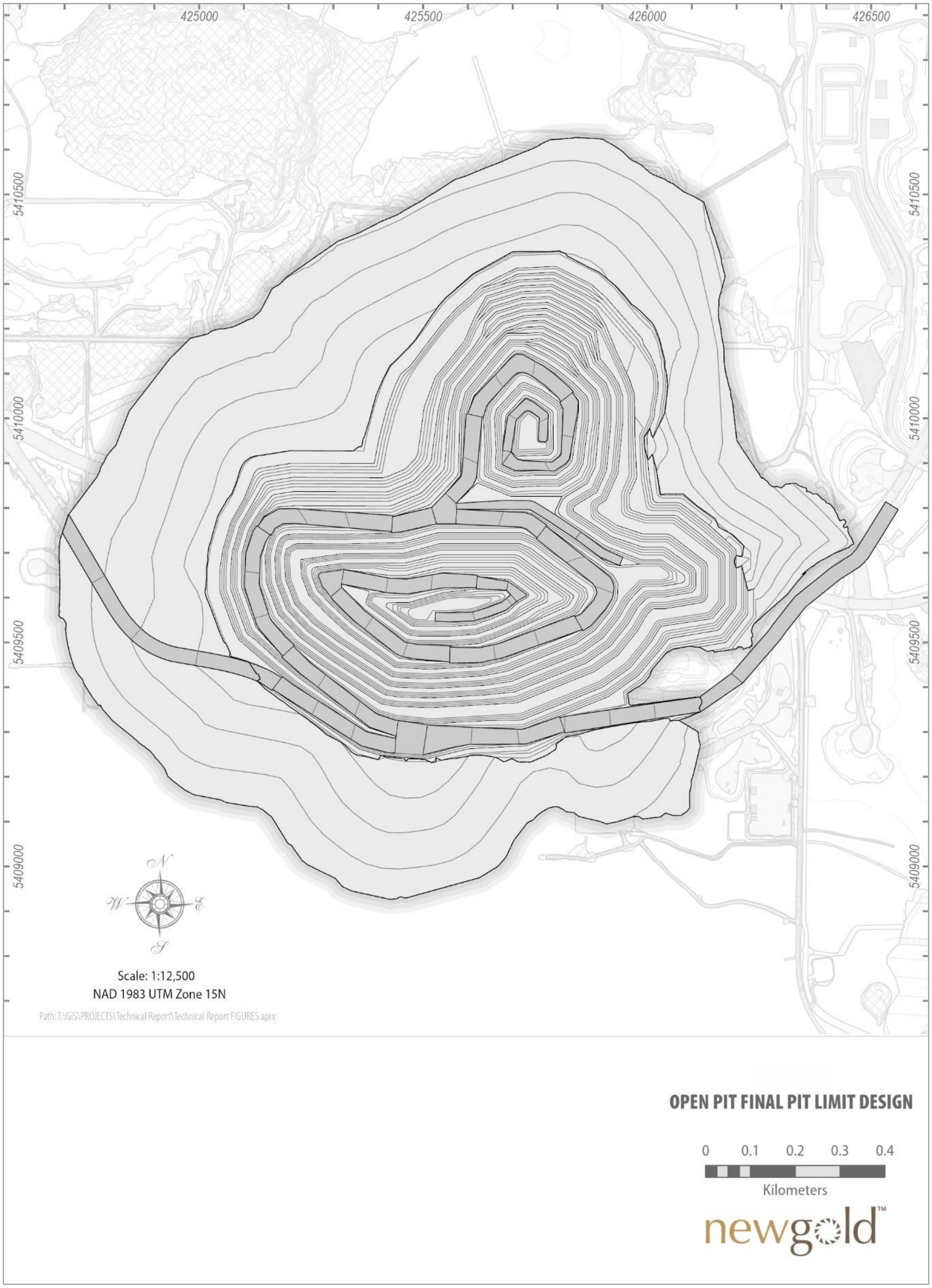
15.1.6 Reserve pit design

The optimized pit solution resulting from the criteria presented in the preceding sections was rationalized into a feasible mining geometry, and haulage ramps were superimposed. Haulage ramps were designed nominally at a 33 m width and with a maximum ±10% grade, except for the bottom few benches where widths were permitted to be reduced to one-way traffic of 20 m and ± 12% grade. Figure 15.2 illustrates the resulting open pit final limit design.

The reserve pit limit spans approximately 1,650 m in the east to west direction and 1,450 m in the north to south direction. Maximum depth is approximately 350 m.

Total material within the final pit limit design as of end-2019, including waste, is 238 Mt.

Figure 15.2 Open pit final limit design



Source: New Gold 2020.

15.2 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 longhole stoping, with a combination of open stopes and backfilled stopes.

15.2.1 Extraction ratio

Based upon a recent geotechnical review and update of longhole open stope designs, and assessment of allowable spans and related rib-pillar requirements, designed mining extraction in each zone, as a percentage of stopes-plus-pillars (see Figure 16.2), is projected as shown in Table 15.5.

Table 15.5 Underground design extraction

Zone	Design extraction
ODM Main	99%
ODM East	93%
17 East Upper	95%
433 Zone	96%
Intrepid	87%

15.2.2 Dilution and recovery

Unplanned stope dilution has been assessed using the empirical estimation of wall slough after Clark and Pakalnis (1997). Relative to considerations of rock mass quality, stope dimensions, structure, dip and depth, average values of 0.3 m footwall (FW) and 0.6 m hangingwall (HW) of equivalent linear overbreak / slough (ELOS) were estimated for a sublevel spacing of 20 m and a strike length up to 40 m. Average backfill dilution from stope end walls and floor mucking on fill for downhole stopes has been estimated at 0.2 m. Average overall unplanned stope dilution, inclusive of FW dilution, HW dilution and backfill dilution, has been estimated at approximately 12%.

A mining recovery of 95% has been applied to the estimates. In addition, the mining recovery of sill pillars, where such are needed, is estimated to be 60%.

15.2.3 Cut-off grade

A COG of 2.2 g/t AuEq was used for the estimation of Mineral Reserves, based on the cost estimates, metal prices and exchange rate summarized below in Table 15.6.

A COG of 0.8 g/t AuEq was used for development ore, recognizing this as incremental material that must be mined in the stope development process.

Table 15.6 Underground COG calculation parameters

Parameter	Unit	Underground parameter value
Gold price	$/oz	1,275
Gold recovery	%	95
Mining cost	$/t mined ore	52.42
Processing cost	$/t mined ore	8.13
G&A cost	$/t mined ore	0.96
Sustaining capital	$/t mined ore	20.00
Royalties	$/t mined ore	1.61
Total operating cost	$/t mined ore	83.12
Break even COG	g/t AuEq	2.2
Exchange rate	US$:C$	1:1.30

The COG is expressed as an AuEq grade, which is estimated as follows:

AuEq = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 95)]

Where the factors in the equivalence calculation are:

- Gold price $1,275/oz
- Silver price $17/oz
- Gold recovery 95%
- Silver recovery 60%

The COG calculation is considered to be appropriate for the deposit based upon the assumptions used and the current company strategy relative to metal prices and combined open pit and underground operations.

15.3 Mineral Reserves

Open pit and underground Mineral Reserves at Rainy River are summarized in Table 15.7.

Table 15.7 Mineral Reserves – effective 31 December 2019

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Direct processing Mineral Reserves					
Open pit					
Proven	15,700	1.21	2.4	612	1,187
Probable	30,675	1.15	2.5	1,136	2,416
Sub-total open pit	**46,375**	**1.17**	**2.4**	**1,748**	**3,602**
Stockpile					
Proven	1,922	0.71	1.0	44	59
Probable	-	-	-	-	-
Sub-total stockpile	**1,922**	**0.71**	**1.0**	**44**	**59**
Underground					
Proven	-	-	-	-	-
Probable	4,096	4.17	7.8	549	1,034
Sub-total underground	**4,096**	**4.17**	**7.8**	**549**	**1,034**
Total direct processing Mineral Reserves	**52,393**	**1.39**	**2.79**	**2,341**	**4,696**
Low grade Mineral Reserves					

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Open pit					
Proven	5,702	0.35	1.9	65	341
Probable	15,470	0.35	2.2	172	1,076
Sub-total open pit	**21,172**	**0.35**	**2.1**	**237**	**1,417**
Stockpile					
Proven	4,007	0.45	1.2	58	153
Probable	-	-	-	-	-
Sub-total stockpile	**4,007**	**0.45**	**1.2**	**58**	**153**
Total low grade Mineral Reserves	**25,179**	**0.36**	**1.9**	**295**	**1,570**
Total Mineral Reserves					
Open pit (including stockpile)					
Proven	27,331	0.88	2.0	779	1,740
Probable	46,145	0.88	2.4	1,308	3,492
Sub-total open pit	**73,476**	**0.88**	**2.2**	**2,087**	**5,231**
Underground					
Proven	-	-	-	-	-
Probable	4,096	4.17	7.8	549	1,034
Sub-total underground	**4,096**	**4.17**	**7.8**	**549**	**1,034**
Total					
Proven	27,331	0.88	2.0	779	1,740
Probable	50,241	1.15	2.8	1,857	4,526
Total Mineral Reserves	**77,572**	**1.06**	**2.5**	**2,636**	**6,266**

Notes:

- CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Reserves.
- Mineral Reserves are estimated using a long-term gold price of US$1,275 per troy oz and a long-term silver price of US$17 per troy oz. The exchange rate used was 1:1.30 US$:C$.
- Direct processing open pit Mineral Reserves are estimated at an AuEq COG of 0.49 g/t for the CAP Zone and 0.46 g/t for Non-CAP Zones. Low grade open pit Mineral Reserves were estimated at an AuEq cut-off of 0.30 g/t. Gold equivalency was estimated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 90)].
- Open pit assumptions include:
  - COGs applied to a regularized 10 m x 10 m x 10 mine planning block model, which was generated from re-blocking the original resource model. Modifying factors representing a planned dilution of 1.3 m below 290 m and 4.0 m above 290 m were applied, the latter factor being higher as it reflects the uncertainties in the geometry of the rock / overburden contact. Ore blocks surrounded by waste blocks were accounted as ore loss, while waste blocks surrounded by ore blocks were included as additional dilution.
  - Metal recoveries are variable dependent on metal head grade. At COG, the gold recoveries are as follows:
    - DPO
      CAP zone gold = 73.9%
      Non-CAP zone gold = 78.2%
    - LGO
      CAP zone gold = 73.1%
      Non-CAP zone gold = 68.9%
  - Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
- Underground Mineral Reserves are estimated at an AuEq COG of 2.20 g/t for stoping and 0.80 g/t for development. Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 95)].
- Underground assumptions include:
  - Planned HW and FW dilution of 0.6 m and 0.3 m, respectively, with total unplanned dilution of approximately 12%.
  - Average mining recovery estimated as 95%.
  - Average gold and silver recoveries of 95% and 60%, respectively.
  - Cut-off value of $83.12/t, inclusive of costs for mining, processing, G&A, refining & transport, royalties, and sustaining capital allowance.
- Effective date of Mineral Reserves is 31 December 2019.
- The QP for the open pit estimate is Mr F. McCann, P.Eng., and for the underground estimate is Mr H.A. Smith, P.Eng., both of AMC.
- Totals may not compute exactly due to rounding.

The Mineral Reserves reported herein supersede the Mineral Reserves reported previously at year-end 2018 by New Gold for the Rainy River Mine.

15.4 Comparison with previous Mineral Reserve estimate

The most recent Mineral Reserve estimate published by New Gold was in a press release titled 'New Gold Reports Fourth Quarter and Year-End Financial Results', sub-title 'Provides Updated Reserves and Resources', dated 14 February 2019. This estimate provided Mineral Reserves effective 31 December 2018.

The current Mineral Reserve estimate described in this NI 43-101 Technical Report provides Mineral Reserves effective 31 December 2019.

Table 15.8 and Table 15.9 provide a comparison of the end-2018 and end-2019 Mineral Reserve estimates for the open pit and underground, respectively.

Table 15.8 Comparison with previous Mineral Reserve estimate – open pit

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Open pit + stockpile					
Effective 31 December 2018					
Proven	34,000	0.89	2.4 [1]	989	2,291
Probable	80,385	0.84	2.8	2176	7,097
Total open pit	**114,385**	**0.86**	**2.6**	**3,165**	**9,388**
Effective 31 December 2019					
Proven	27,330	0.89	2.0	779	1,739
Probable	46,145	0.88	2.4	1,308	3,492
Total open pit	**73,476**	**0.88**	**2.2**	**2,087**	**5,231**
Difference over 2018					
Proven	-20%	0%	-18%	-21%	-24%
Probable	-43%	5%	-14%	-40%	-51%
Total open pit	-36%	3%	-13%	-34%	-44%

Notes:
- [1] An error exists in silver grade as published on New Gold's website regarding Total Proven Mineral Reserves, effective 31 December 2018. The reported Proven silver ounces of 2,291 koz are correct.
- Totals may not compute exactly due to rounding.

Changes to the open pit Mineral Reserve estimate from end-2018 to end-2019 are due predominantly to:

- 2019 Mineral Reserve depletion from mining activities of 8.0 Mt @ 1.08 g/t gold and 1.81 g/t silver, totalling 277 thousand ounces (koz) of contained gold and 467 koz of contained silver.
- Updated costs reflecting the current cost of operation at the mine (mine, process, G&A, and relevant sustaining capital requirements). Overall, costs have increased.
- Updated metallurgical models for gold and silver, resulting in lower average metal recoveries in general.
- Updated geotechnical model resulting in slightly lower overall pit slope angles.
- Updated mine planning block model with modifying factors impacting dilution and ore loss.
- Updated methodology in the estimation of COGs.

The net result of these items has been a significant shrinking of the open pit and subsequent decrease in the Mineral Reserve.

Table 15.9 Comparison with previous Mineral Reserve estimate - Underground

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	**Gold (g/t)**	**Silver (g/t)**	**Gold (koz)**	**Silver (koz)**
Underground					
Effective 31 December 2018					
Proven	-	-	-	-	-
Probable	8,954	3.55	9.5	1,021	2,728
Total underground	8,954	3.55	9.5	1,021	2,728
Effective 31 December 2019					
Proven	-	-	-	-	-
Probable	4,096	4.17	7.8	549	1,034
Total underground	4,096	4.17	7.8	549	1,034
Difference over 2018					
Proven	-	-	-	-	-
Probable	-54%	17%	-18%	-46%	-62%
Total underground	-54%	17%	-18%	-46%	-62%

Note: Totals may not compute exactly due to rounding.

Changes to the underground Mineral Reserve estimate from that reported at the end of 2018 to that reported currently at the end of 2019 are primarily a reflection in the reduction of the mine life of the open pit. The underground operation depends significantly on the open pit's ability to deliver ore to supplement the mill feed during the underground operation in order to absorb some of the related costs. With the decrease in open pit mine life and quantity of stockpiled LGO to supplement the mill feed, the underground Mineral Reserves have decreased, as the operation was re-sized to match the shorter open pit mine life.

AMC recognizes that there is a significant quantity of what currently is considered marginal material in underground Mineral Resources and recommends that regular reassessment of that material be undertaken relative to the metal price environment, company strategy, and possibilities for adjusted ratios of open pit and underground mining over time.

15.5 Conversion of Mineral Resources to Mineral Reserves

Table 15.10 shows the proportions of total Measured and Indicated Mineral Resources that have been converted to Mineral Reserves in terms of contained gold ounces.

Table 15.10 Mineral Resource to Mineral Reserve conversion ratios for contained gold

Mineral Resources inclusive of Mineral Reserves		**Mineral Reserves**		**Conversion rate**
Category	**Contained gold (koz)**	**Category**	**Contained gold (koz)**	
Direct processing				
Open pit				
Measured	689	Proven	656	95%
Indicated	1,318	Probable	1,136	86%
OP M&I	2,006	OP P&P	1,792	89%
Underground				
Measured	-	Proven	-	-
Indicated	2,218	Probable	549	25%
UG M&I	2,218	UG P&P	549	25%
Total direct processing M&I	**4,224**	**Total direct processing P&P**	**2,341**	**55%**
Low grade				
Open pit				
Measured	126	Proven	123	98%
Indicated	199	Probable	172	86%
OP low grade M&I	**325**	**OP low grade P&P**	**295**	**91%**
Total M&I	4,550	Total P&P	2,636	58%

Note: Totals may not compute exactly due to rounding.

16 Mining methods

Mining at Rainy River is currently conducted using open pit mining methods and will transition into a combined open pit and underground operation over the next two years, with underground production commencing in 2022. An average processing rate of approximately 25,800 tpd is scheduled over the LOM.

Over the LOM, the open pit (including stockpile rehandle) and underground operations are scheduled to provide 95% and 5% of the ore tonnage processed with 79% and 21% of the contained gold ounces processed, respectively.

The open pit mine is a conventional truck and shovel mining operation, with a fleet of 220 t payload haul trucks combined with diesel powered hydraulic excavators and large front-end loaders (FELs) as primary loading units. The open pit operates at a peak mining rate of 151,000 tpd of ore and waste and has an overall strip ratio of 2.53:1 (waste:ore).

The underground mine is planned to be accessed from five portals, targeting separate ore zones. One portal, with 166 m of existing decline development currently exists for the Intrepid Zone and is scheduled to re-commence development in 2020 as part of an orebody investigation project.

The underground operations will be accessed via declines and follow a mechanized longitudinal longhole open stoping (LLHOS) technique to exploit the underground Mineral Reserves. Underground ore production rates will be variable but are planned to reach a maximum of approximately 3,200 tpd in year 2026.

The combined open pit and underground operation have a remaining mine life through to Q1-2028.

16.1 Open pit mining

16.1.1 Production to end-2019

The open pit operation at Rainy River commenced stripping activities in 2016, ore processing in September 2017 and commercial production in mid-October 2017. Open pit and mill production to end-2019 are provided in Table 16.1 and Table 16.2, respectively.

Table 16.1 Open pit mine production to end-2019

Year	Ore tonnes (000s)	Grade		Contained metal		Waste tonnes (000s)	Total tonnes (000s)	Strip ratio (w:o)
		Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)			
2017	1,808	1.05	1.9	61	110	5,013	6,821	2.77
2018	12,296	1.30	2.3	514	897	27,267	39,563	2.22
2019	6,830	1.00	1.5	220	332	36,387	43,217	5.33
Total	**20,934**	**1.20**	**2.1**	**795**	**1,339**	**68,668**	**89,601**	**3.28**

Note: Totals may not compute exactly due to rounding.

Table 16.2 Mill production to end-2019

Year	Ore tonnes processed (000s)	Grade processed		Produced metal	
		Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
2017	977	0.94	2.2	29	44
2018	6,546	1.25	2.0	227	248
2019	8,023	1.08	1.8	254	282
Total	**15,546**	**1.15**	**1.9**	**510**	**575**

Note: Totals may not compute exactly due to rounding.

16.1.2 Hydrologic considerations

In 2019, SRK was contracted by New Gold to provide a LOM pit water management plan for the open pit operations at Rainy River. The open pit dewatering plan considers three sources of inflow: rainfall, snowmelt and seepage with all sources contributing to both the surface water and ground water inflows. The New Gold dewatering system has been designed to handle surface water that could originate from a 2-year freshet event.

The current dewatering system includes pumps, sumps, pipes and staging tanks that remove water from the open pit and the surrounding area. Water diversion ditches are developed around the open pit limit to minimize surface inflow into the pit. Based on a preliminary analysis of the current pumping system and regional hydrological trends, it is envisioned that the current dewatering system will continue to be expanded as the mine develops with the focus being on achieving the following objectives:

- Maintain a dry working area for pit operations and mining activities.
- Minimize the cost of extra pumping systems.
- Optimize sump and pipeline locations to collect all reporting inflow.

16.1.3 Open pit geotechnical considerations – overburden

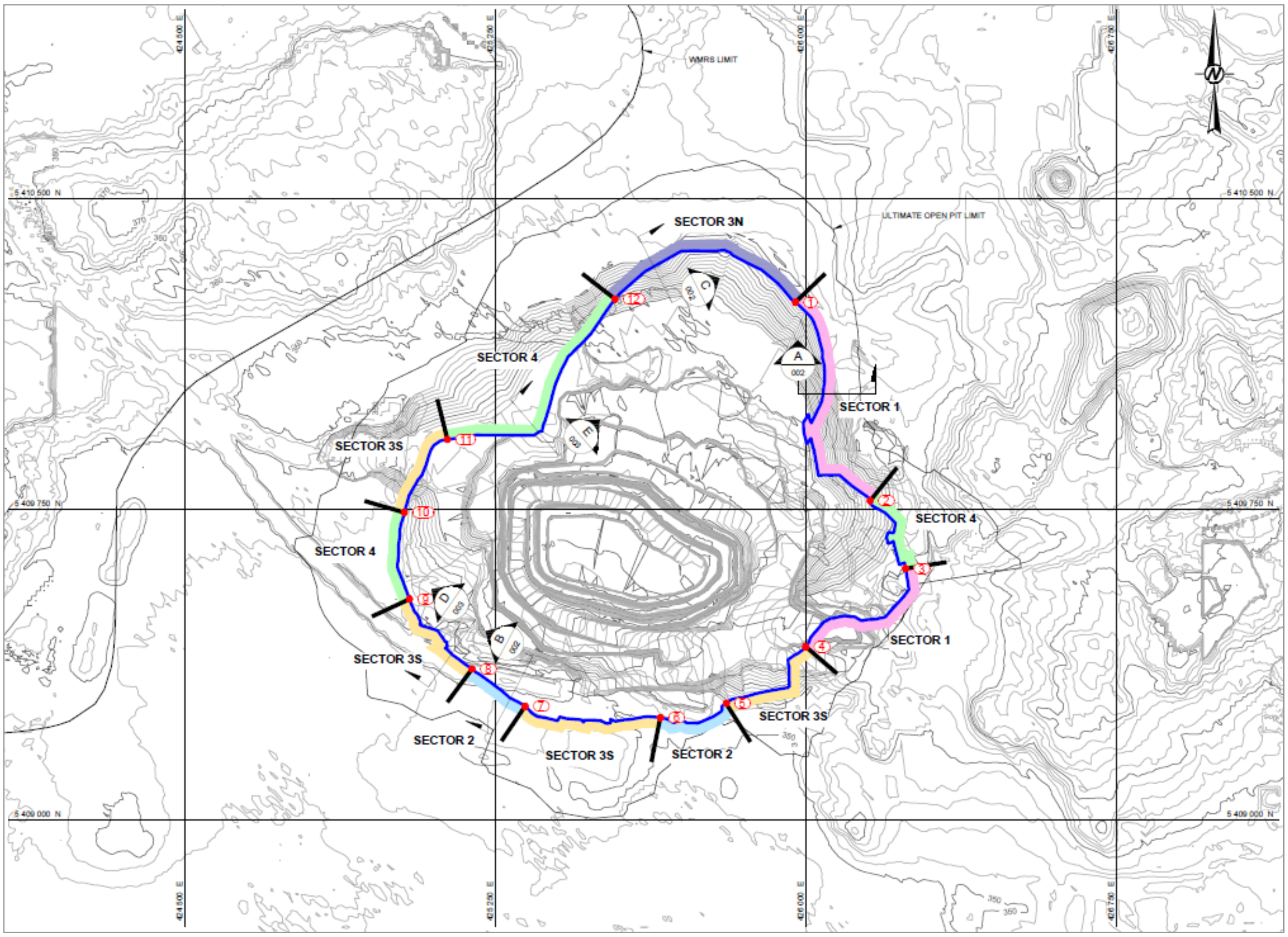
The depth of overburden varies around the ultimate open pit perimeter up to approximately 42 m. Except for the sandy basal Whiteshell Till (WST) formation which directly overlays bedrock, the overburden is largely comprised of clay deposits. The total clay thickness varies from approximately 4 m to 38 m. The final overburden slopes have been designed to meet or exceed slope stability criteria. The perimeter open pit overburden was divided into five design sectors based on the bedrock geometry and interpreted clay thickness. The design sectors are as follows:

- Sector 1: 0 m – 10 m clay thickness
- Sector 2: 10 m – 20 m clay thickness
- Sector 3: 20 m – 30 m clay thickness
- Sector 4: 30 m – 40 m clay thickness

A rockfill toe berm and slope buttress have been designed for all the sectors. The advantage of the rock toe berm and slope buttress is to achieve long term stability, thus allowing steeper cut slopes and reduced excavation volume and potentially shorter waste rock haulage distance (in comparison to transporting material to the waste rock stockpiles). The initial excavation through the overburden takes advantage of the short-term strength of the clays; however, placement of the rockfill toe / slope buttress is required for longer term stability. Excavation of the final overburden slopes should be top down in order to prevent excessive strain. Placement of the rockfill toe / slope buttresses is to be carried out in a progressive segment by segment manner once each segment is excavated to final grade.

The design sectors are presented in plan-view on Figure 16.1 and a summary of the design geometry for each of the sectors is provided in Table 16.3. The clay deposits have design slopes that vary between 3.5H:1V to 8H:1V and the bottom sand till deposit (WST) has a constant design slope of 3H: 1V slope for all the sectors. The overburden slope design summarized represents an optimization relative to the overburden slopes that were assumed in the project economic model for sectors 2 and 3S where a reduced volume of overburden will be mined.

Figure 16.1 Design sector layout plan



Source: Golder 2020.

Table 16.3 Summary of design geometries

<table>
<tr><th colspan="2" rowspan="2">Design geometries</th><th colspan="5">Open pit overburden design sector</th></tr>
<tr><th>Sector 1</th><th>Sector 2</th><th>Sector 3N</th><th>Sector 3S</th><th>Sector 4</th></tr>
<tr><td rowspan="2">Overburden cut slope grade</td><td>Through WST unit</td><td colspan="5">3H:1V</td></tr>
<tr><td>Through Clay units (BRE, WML, and WYL)</td><td>3.5H:1V</td><td>4H:1V</td><td>7H:1V</td><td>7H:1V</td><td>8H:1V</td></tr>
<tr><td rowspan="5">Rockfill toe berm and slope buttress</td><td>Base width (at the overburden / bedrock contact)</td><td>15 m</td><td>15 m</td><td>15 m</td><td>18 m</td><td>23 m</td></tr>
<tr><td>Lower slope grade (from bedrock surface up to buttress bench)</td><td colspan="5">1.3H:1V</td></tr>
<tr><td>Buttress bench distance below the crest</td><td>2.5 m</td><td>2.0 m</td><td>19.5 m</td><td>12 m</td><td>13 m</td></tr>
<tr><td>Upper slope grade (from buttress bench up to existing ground surface)</td><td>10.8H:1V</td><td>5.5H:1V</td><td>9.2H:1V</td><td>9.2H:1V</td><td>11.0H:1V</td></tr>
<tr><td>Crest width (at existing ground surface)</td><td colspan="5">10 m</td></tr>
</table>

Notes: WST = Whiteshell Till; BRE = Brenna Formation; WML = Whitemouth Lake Formation; and WYL = Wylie Formation

16.1.4 Open pit geotechnical considerations – hard rock

SRK carried out an open pit slope stability assessment and design study in 2019. The study was carried out to develop revised rock slope design criteria for the Phase 3 and Phase 4 Pit. In summary, the Rainy River pit slope stability and resulting design is defined by:

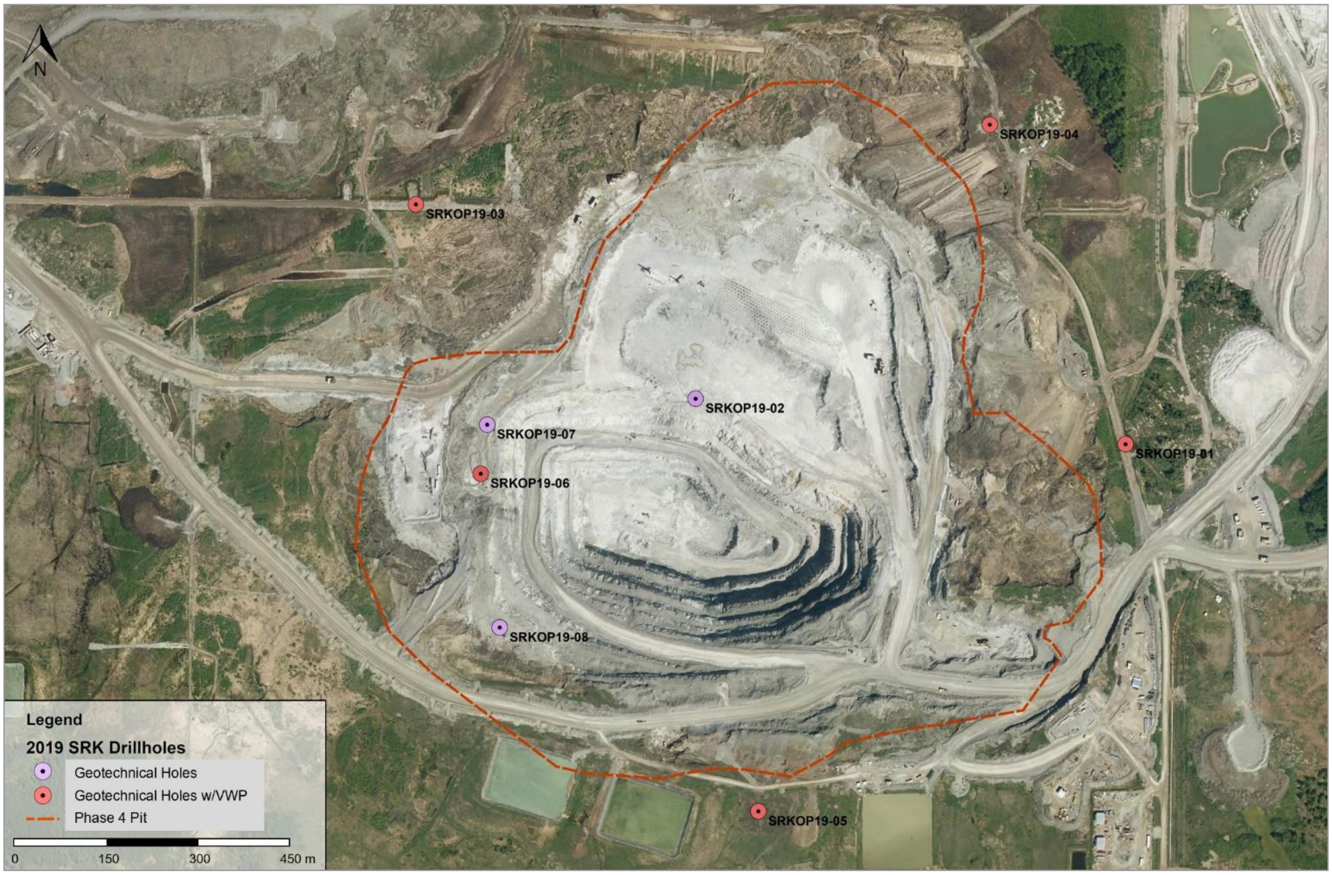
- The orientation of the regional south-southwest dipping foliation structures (North Wall).
- The kinematic stability related to the major joint sets (all pit walls).

With consideration to the above, the revised design recommendations were predominantly driven by the bench-berm stability assessments with shallower bench face angles (BFAs) adopted to reduce the undercutting potential from planar sliding and wedge intersections.

16.1.4.1 Field and lab investigation

SRK conducted additional pit slope geotechnical and hydrogeological investigations in 2019 to address data gaps and improve the reliability of the previously collected data. These investigations comprised eight DDHs, geotechnical logging and field testing, hydrogeological testing, and five vibrating wire piezometer (VWP) installations. The field investigations were carried out from February to March 2019. The investigation locations are relative to the approximate phase perimeters and are shown in Figure 16.2. Details are provided in Table 16.4. The locations of the previous geotechnical drillholes and televiewer surveys are shown in Figure 16.3.

Figure 16.2 Collar locations of 2019 SRK geotechnical drillholes



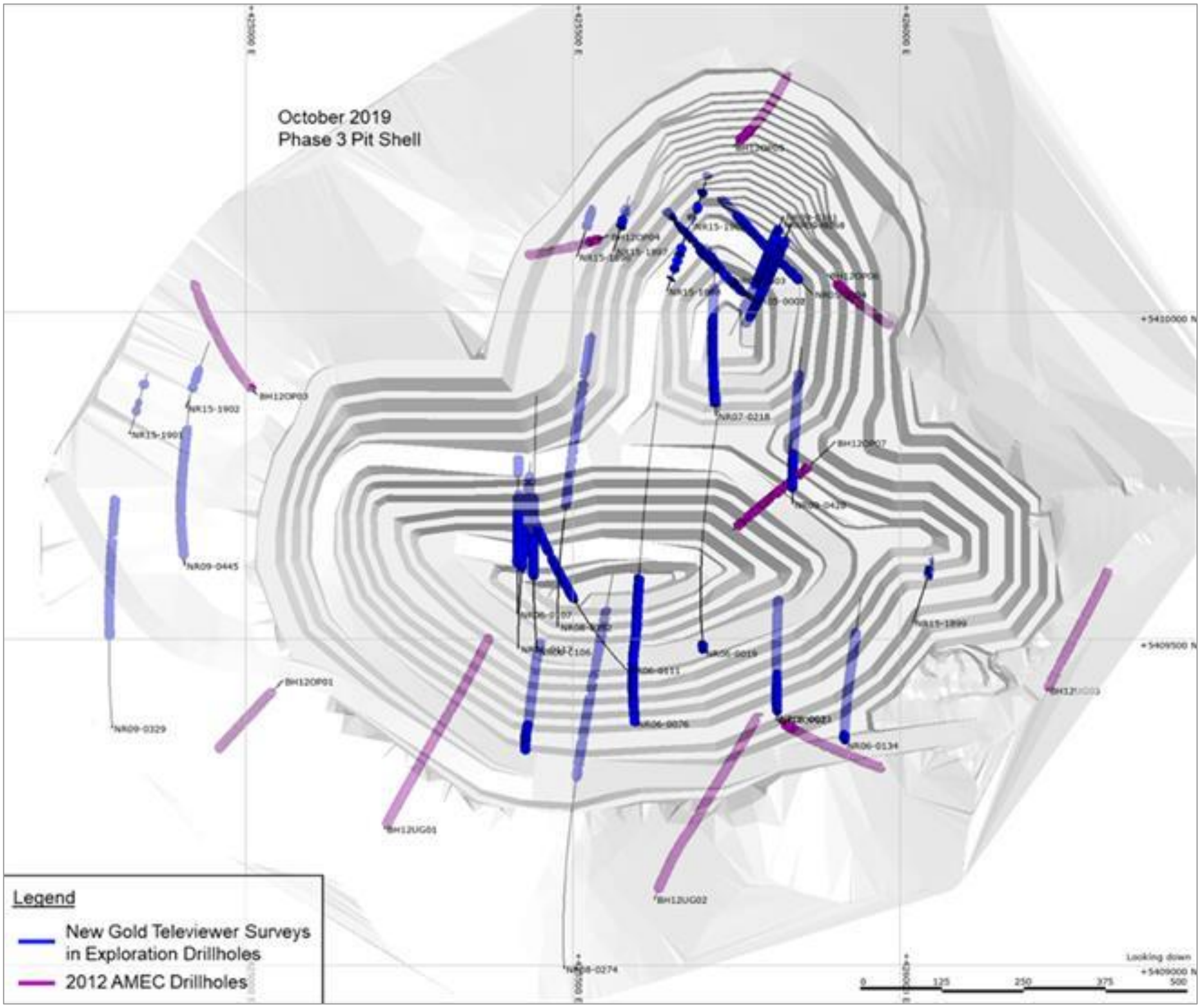
Source: SRK 2020.

Table 16.4 As-built SRK geotechnical drillholes

Hole ID	Easting (m)	Northing (m)	Elev. (masl)	Azimuth [1] (degree)	Inclination [1] (degree)	Depth (m)	Hydrogeology testing / install	Televiewer
SRKOP19-01	426279	5409779	359	245	60	320	Packer, VWP	Yes
SRKOP19-02	425573	5409854	290	300	80	251	-	Yes
SRKOP19-03	425114	5410173	351	150	55	344	Packer, VWP	Yes
SRKOP19-04	426056	5410304	356	230	55	365	Packer, VWP	Yes
SRKOP19-05	425676	5409176	350	355	60	374	VWP	Yes
SRKOP19-06	425220	5409731	313	270	75	299	Packer, VWP	Yes
SRKOP19-07	425230	5409811	314	015	80	224	-	Yes
SRKOP19-08	425251	5409478	330	160	70	341	-	Yes

Note: [1] As-built azimuth and inclination reported were measured at top of hole.

Figure 16.3 Geotechnical drillholes and televiewer surveys completed between 2006 and 2015



Note: Orientation data annotated in blue or purple, with the drillhole traces in black.
Source: SRK 2020.

16.1.4.2 Stability assessment

A rock slope stability assessment was carried out using multiple approaches with a combination of software packages, as summarized in Table 16.5.

Table 16.5 Overview of stability assessment approach and software

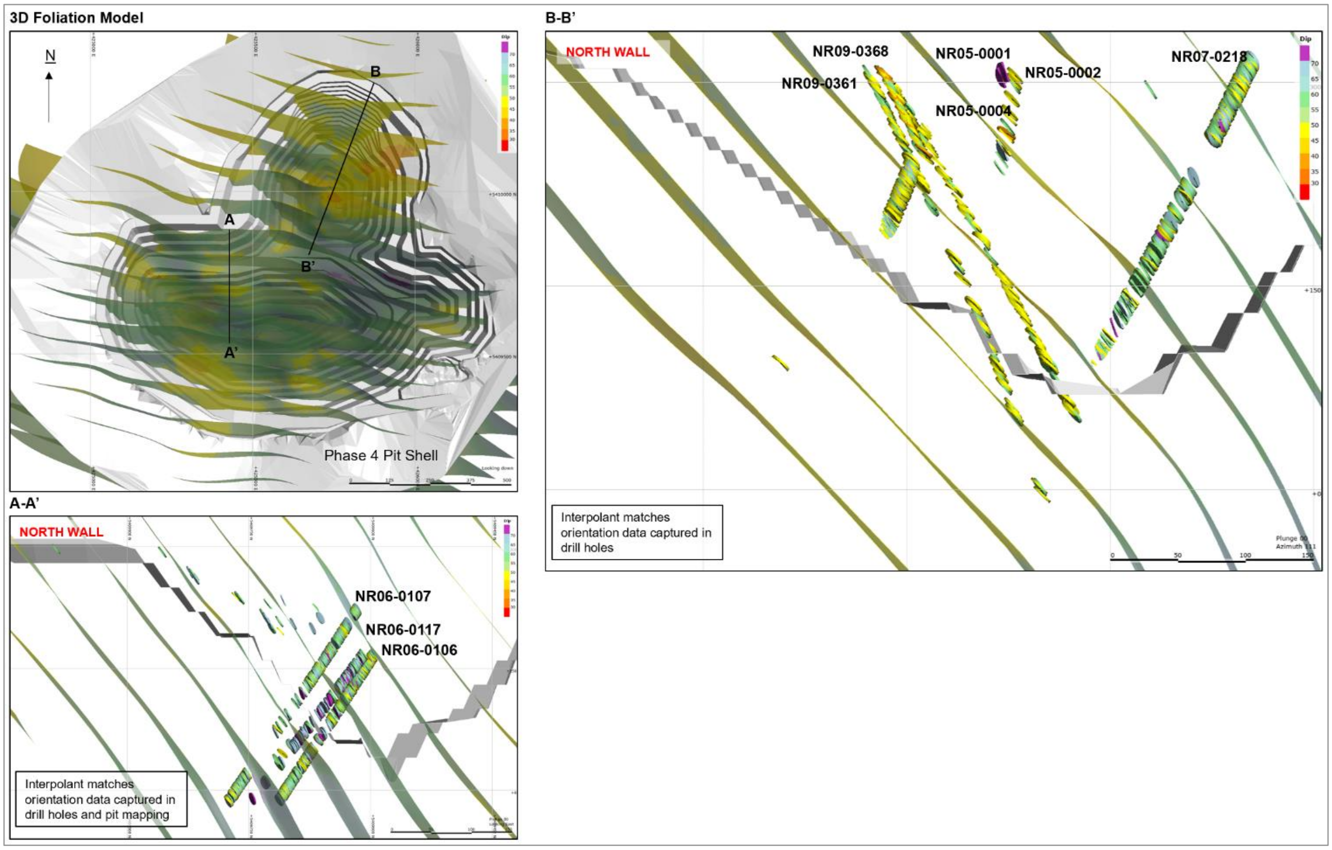
Pit slope scale	Approach	Software utilized
Bench	• Observed bench and blast slope performance • Kinematic stability analyses (deterministic and probabilistic) • 3D interpolant (foliation) models • Modified Richie criteria	• DIPS™ • Leapfrog™ • SBlock™
Inter-Ramp	• Observed bench and blast slope performance • 3D fault / joint geometric intersections • 3D interpolant (foliation) models • Kinematic stability analyses (deterministic and probabilistic) • Limit equilibrium (LE) stability analyses • Finite element (FE) stability analyses	• DIPS™ • Leapfrog™ • SWedge™ • Slide2D™ • RS2™
Overall	• LE stability analyses • FE stability analyses	• Leapfrog™ • Slide2D™ • RS2™

16.1.4.3 Foliation model

Orientation, shear strength and fracture spacing components are critical stability controls for the foliation-parallel, south-facing pit slopes along the North Walls at Rainy River. The design of these slopes will be defined by the character of the persistent foliation structures, and the requirement to reduce the probability of undercutting that can result in planar sliding mechanisms.

All valid orientation data was compiled in Leapfrog™. The orientation data was then filtered by type to define a valid foliation dataset for use in generation of a 3D model. The 3D foliation model was created using a form interpolant in Leapfrog™ and comprises a series of meshes at approximately 100 m spacing over the entirety of the Phase 4 Pit as shown in Figure 16.4.

Figure 16.4 Pit scale 3D foliation model



Source: SRK 2020.

16.1.4.4 Bench to inter-ramp kinematic stability

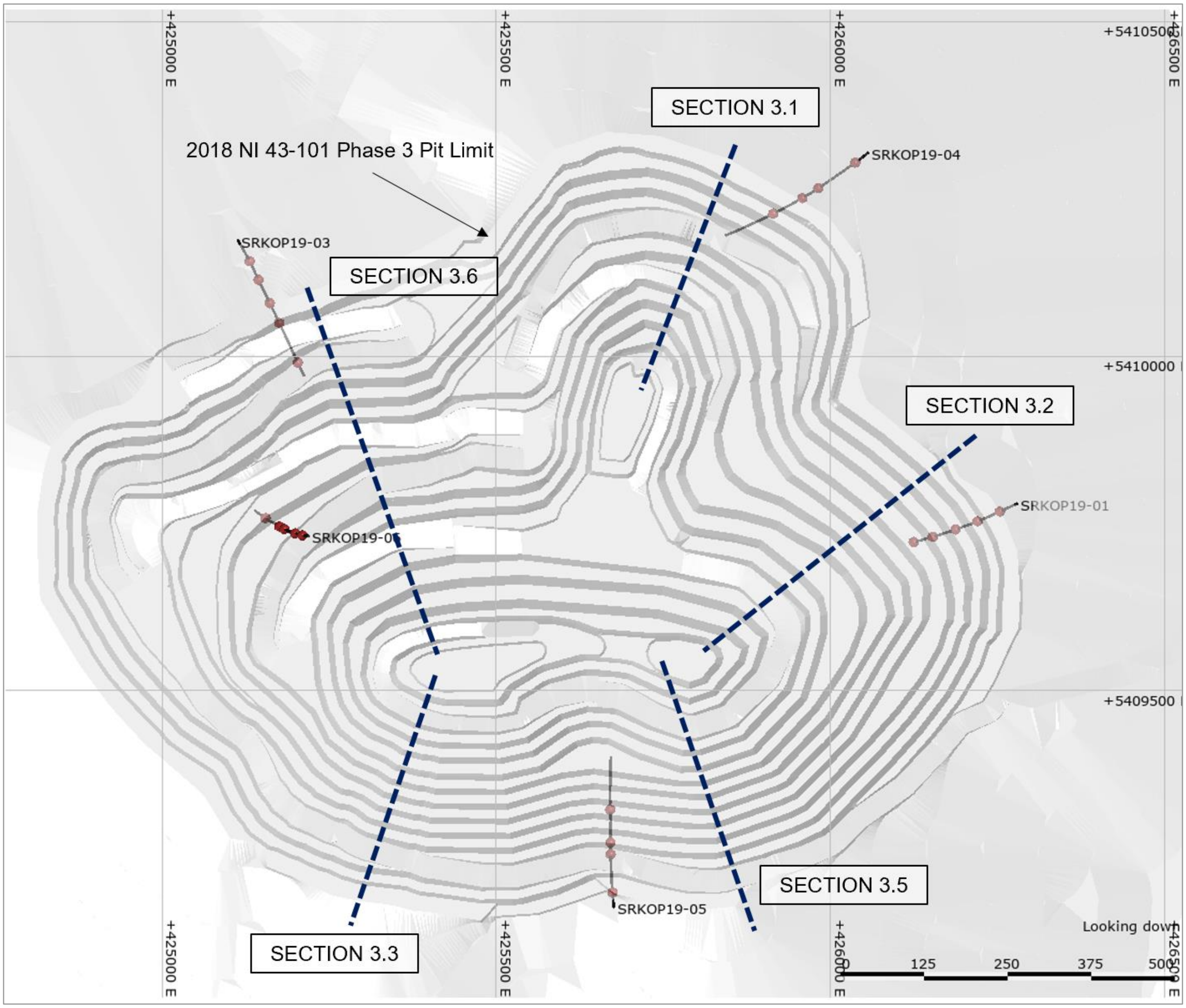
Kinematic analyses were carried out using Dips™ for defined litho-structural domains and all applicable slope face directions. The analyses were carried out for 30° segments to identify the potential kinematic failure modes that could limit the design. Both wedge intersection and planar sliding mechanisms were at the bench slopes for most of the analyzed slope aspects.

Bench scale probabilistic stability analyses were then carried out using the software program SBlock™ by Esterhuizen 2004. The software analyses key-block formation of the discontinuity sets probabilistically using the Monte-Carlo method. SBlock™ analyzes the expected bench break-back from planar and wedge sliding failure mechanisms. The results of the SBlock™ analyzes were used to build the design bench and inter-ramp configurations.

16.1.4.5 Stability analyses

Two-dimensional slope stability analyses were used to evaluate the expected design rock slope stability conditions. The analyses were conducted using Slide2D™ and RS2™. The stability analyses considered the potential for overall non-circular failure through the anisotropic rock mass (i.e. step-path failure mechanism). Stability analyses were carried out for a total of nine design sections, including four sections through the Phase 2 Pit and five sections for the Phase 3 Pit (Figure 16.5). These stability sections were based on the 2018 NI 43-101 Phase 2/3 Pit with the results being used to inform the updated 2020 stability and design work.

Figure 16.5 2018 NI 43-101 Phase 3 pit stability section locations



Source: SRK 2020.

The design sections were intended to target the higher pit slopes located near the VWP's to confirm the pore pressure modelling inputs with future groundwater monitoring. Three groundwater (GW) cases were modelled, including:

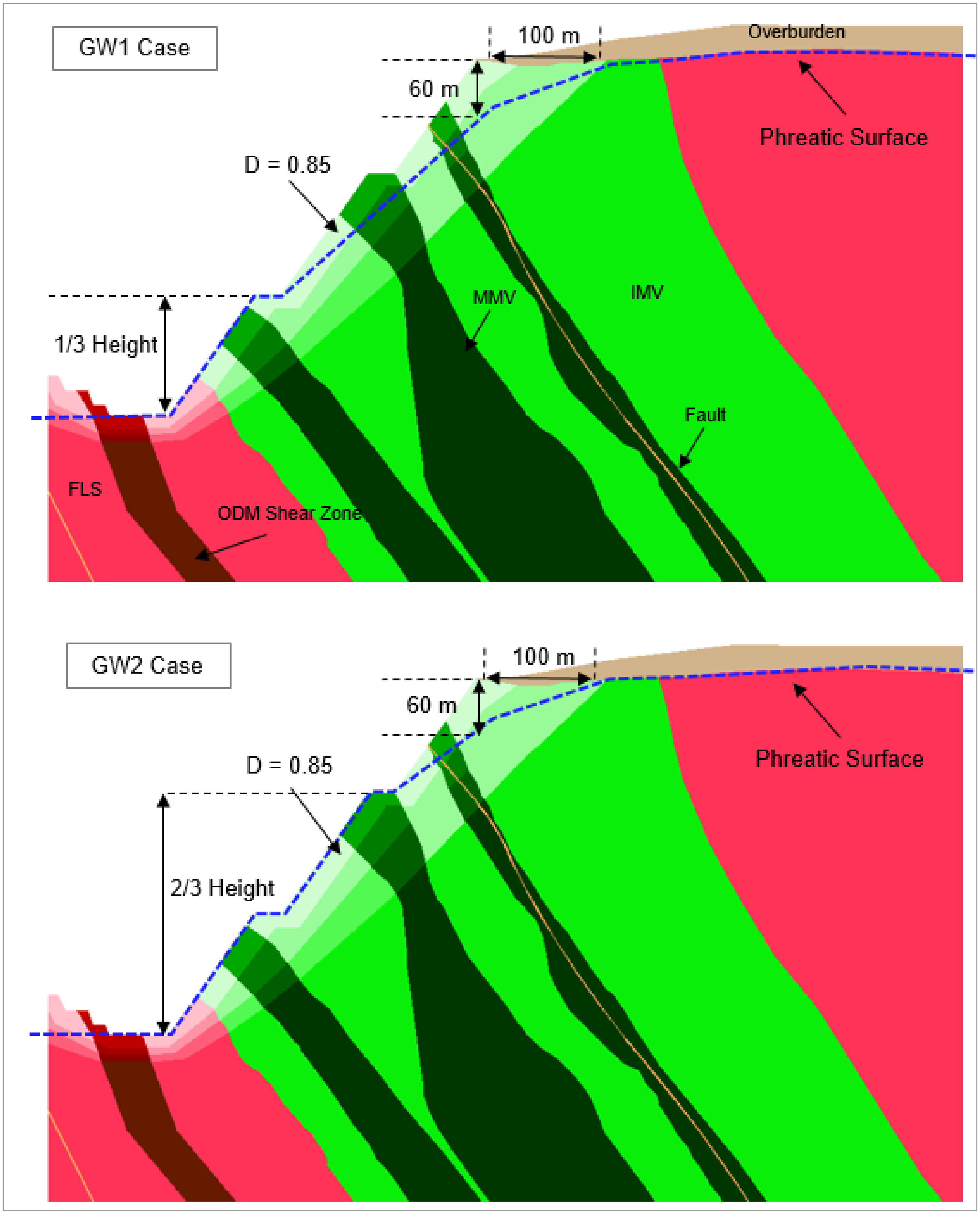
1 **GW1 Case**: phreatic surface approach that represents full saturation of the lower one-third of the slope and the partial saturation of the upper two-thirds.

2 **GW2 Case**: phreatic surface approach that represents full saturation of the lower two-thirds of the slope and the partial saturation of the upper one-third.

3 **Pore pressure model (ppm)**: steady-state PPM that utilizes water elevations from nearby VWPs, at their peak water elevation and hydraulic conductivities from the hydrogeological field testing. Infiltration was set at 7.2 x 10^{-4} metres per day (m/d) which represents 25% of the total precipitation pro-rated through the wettest months (May through September).

Schematic figures showing the two phreatic surface cases are presented in Figure 16.6.

Figure 16.6 Two phreatic surface ground water modelling cases



Notes: IMV = Intermediate Metavolcanics; MMV = Mafic Metavolcanics; FLS = Felsic Metasediments.
Source: SRK 2020.

The results of the LE stability analyses are summarized in Table 16.6 and the FE results in Table 16.7. The results were compared against a minimum acceptance criterion of a factor of safety (FOS) equal to or greater than 1.3 that represents an inter-ramp or overall slope with a medium consequence of failure. The results indicated that a FOS of equal or greater than 1.3 for all sections except for Section 3.5 analyzed under the GW2 case. This groundwater case is considered the most conservative of the three cases and is supported by the current VWP monitoring data that shows evidence of step draw-down response to excavation. The longer-term monitoring record needs to be reviewed against these analyses to validate this GW2 condition. As the excavation becomes closer to the current VWP's, the understanding of the near slope groundwater conditions will increase. A phreatic surface approach is considered more conservative than a steady-state PPM approach.

Table 16.6 Summary of LE results for Phase 3 and Phase 4 pit slopes

Pit phase	Pit wall	Stability section	Pit height (m)	Modelled IRA (°)	Minimum design FOS	Factor of safety		
						GW1	GW2	PPM
Phase 4	North	3.1	250	44	1.3	2.4	2.4	2.6
	East	3.2	360	49	1.3	1.8	1.8	1.8
	South	3.3	360	54	1.3	1.3	1.3	1.5
	South	3.5	350	54	1.3	1.3	1.2	1.5
	North	3.6	360	46	1.3	1.6	1.4	1.5

Table 16.7 Summary of FE method results for Phase 4 pit slopes

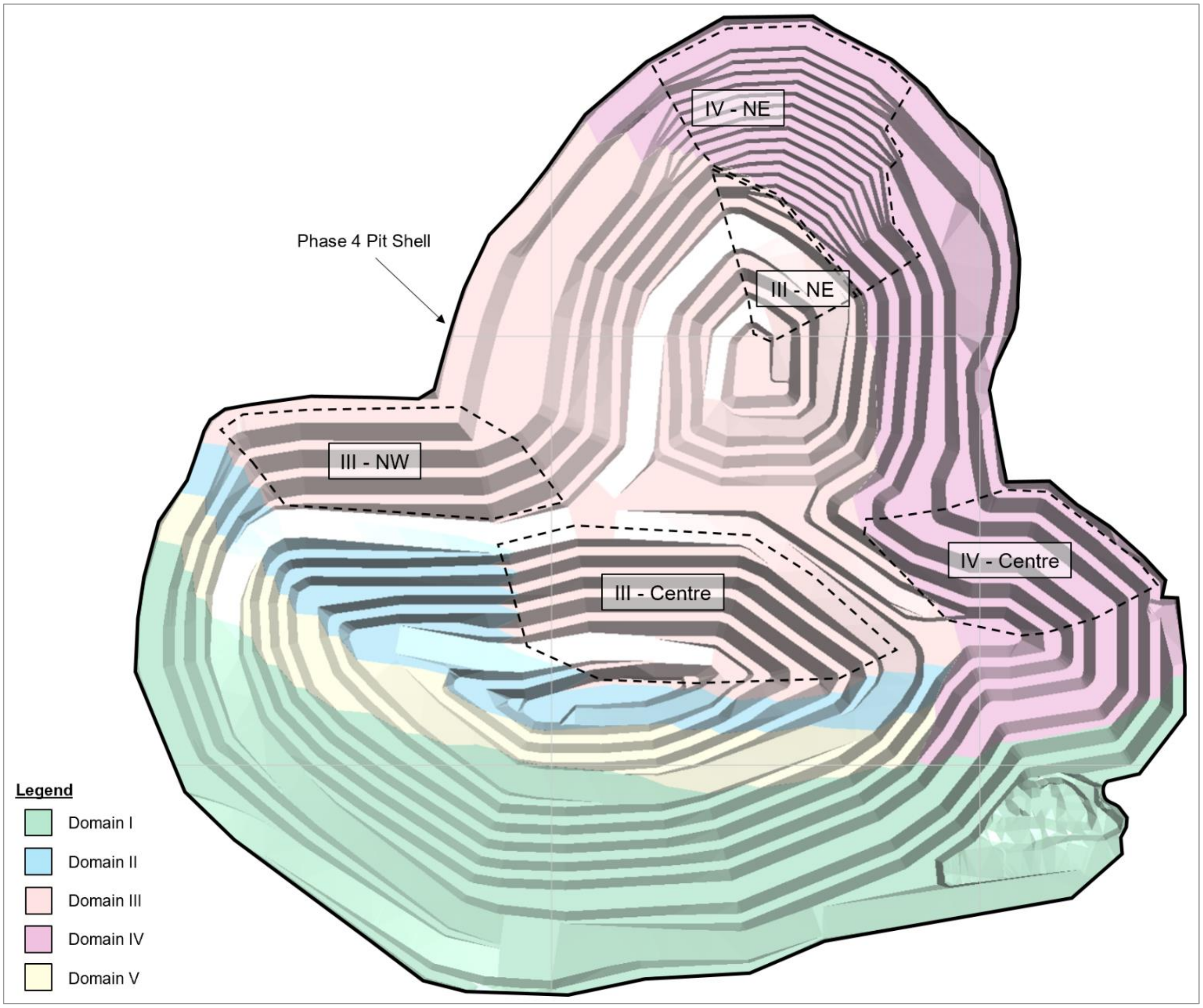
Pit phase	Pit wall	Stability section	Pit height (m)	Modelled IRA (°)	Minimum design SRF	Strength reduction factor	
						GW1	GW2
Phase 4	North	3.1	250	44	1.3	2.8	2.7
	East	3.2	360	49	1.3	1.3	2.1
	South	3.3	360	54	1.3	1.8	1.7
	South	3.5	350	54	1.3	1.4	1.3
	North	3.6	360	46	1.3	2.3	2.1

16.1.4.6 Rock slope design criteria

Pit slope design recommendations are presented in Table 16.8 and are based on the litho-structural domains shown in Figure 16.7.

Note that the 60° BFA recommendation presented in Table 16.8 was actually modified by New Gold to 62° in their open pit designs to reflect drill fleet capabilities. Inter-ramp angles were maintained.

Figure 16.7 Litho-structural design domains



Source: SRK 2020.

Table 16.8 Summary of rock slope recommendations

Design sector			Design recommendation						Kinematic stability limitations
SRK domain	Slope dip direction (°) From	Slope dip direction (°) To	BFA (°)	Bench height[1] (m)	Planned berm width (m)	Inter-ramp angle IRA (°)	Maximum stack height (m)	Geotechnical berm width (m)	
I	270	300	60	30	12.5	45	120	25	Wedges failure models expected at the bench scale on multiple joint set intersections.
	300	350	65	30	10.5	51			Wedges failure models expected at the bench scale on multiple joint set intersections.
	350	070	70	30	10.5	54			Wedges failure models expected at the bench scale on multiple JS1 / JS4 and JS5. Planar sliding on JS3 resulting in crest loss.
	070	090	65	30	10.5	51			Wedges failure models expected at the bench scale on multiple joint set intersections.
II/V	080	130	70	30	10.5	54	120	25	Foliation expected to dominant rock fabric through West Wall. Joint sets dis-continuous.
	130	150	60	30	10.5	47			Bench scale wedge intersection on JS3 / JS5.
	160	230	60	30	16.0	43			Within ODM Shear (Domain II). Orientation of south-dipping Foliation (FOL) structures. Design configuration to limit planar sliding mechanisms. Benched along foliation structures.
	230	260	60	30	12.0	46			Bench scale wedge intersection on FOL and JS3 / JS5.
	260	330	65	30	10.5	51			Bench scale wedge intersection on FOL and JS1.
	330	030	70	30	10.5	54			Wedges failure models expected at the bench scale on multiple JS1 / JS4 and JS5. Planar sliding on JS3 resulting in crest loss.
III	090	130	70	30	12.5	52	120	25	Foliation expected to dominant rock fabric through West Wall. Joint sets dis-continuous. High-angle planar sliding on joint set (JS) 6 sets.
	130	160	65	30	12.0	49			Wedges failure models expected at the bench scale on multiple joint set intersections and foliation.
	160	230	55 (NW) 60 (Centre) 50 (NE)	30 30 10	10.5 (NW) 10.5 (Centre) 5.0 (NE)	44 (NW) 47 (Centre) 37 (NE)			Orientation of south-dipping Foliation (FOL) structures. Design configuration to limit planar sliding mechanisms. Benched along foliation structures.
	230	250	60	30	14.0	44			Significant bench scale wedge intersection on FOL and JS3 / JS5. Interaction with Southeast dipping JS6 set.
	250	330	65	30	10.5	51			Wedges failure models expected at the bench scale on multiple joint set intersections.
	330	030	70	30	10.5	53			Wedges failure models expected at the bench scale on multiple JS1 / JS4 and JS5. Planar sliding on JS3 resulting in crest loss.
IV	160	230	50 (NE) 60 (Centre)	10 (NE) 30 (Centre)	75.0 (NE) 10.5 (Centre)	37 (NE) 47 (Centre)	120	25	Orientation of south-dipping FOL structures. Design configuration to limit planar sliding mechanisms. Benched along foliation structures.
	230	270	65	30	12.0	49			Wedges failure models expected at the bench scale on FOL and joint set intersections.
	270	320	65	30	10.5	51			Wedges failure models expected at the bench scale on FOL and joint set intersections.
	320	360	65	30	10.5	51			Wedges failure models expected at the bench scale on multiple joint set intersections. Planar sliding along JS3 set result in crest loss.

Note: [1] 10 m bench heights recommended for shallower FW designs.

In addition, the following design guidelines were provided:

- Incorporation of a minimum 15 m berm width at the overburden-rock contact with surface water interception ditches to capture and divert water flow from the gravelly till units.
- The irregular bedrock-overburden profile will need to be considered in the pit design work.
- The initial bench slope should be limited to a single bench height due to the increased fracturing and irregular joint orientations observed in shallow bench slopes.
- The North Wall BFA's are based on achievable blasting approaches that will need to be trialed, including the stab-hole approach. This includes Domain II, III, and IV.
- Bullnoses (convex slopes) of one or more stack heights should be stepped-out and assigned a lower IRA, depending on their size, location, and radius of curvature.
- Implementation of a two-ramp approach through Phase 3/4 to reduce consequences of an instability location above or below critical access. Acceptable design criteria are linked to the consequence of an instability event. Where a two-ramp access strategy is incorporated, a significant reduction in the consequence component can be identified for stability evaluation and design.

16.1.5 Open pit mine design

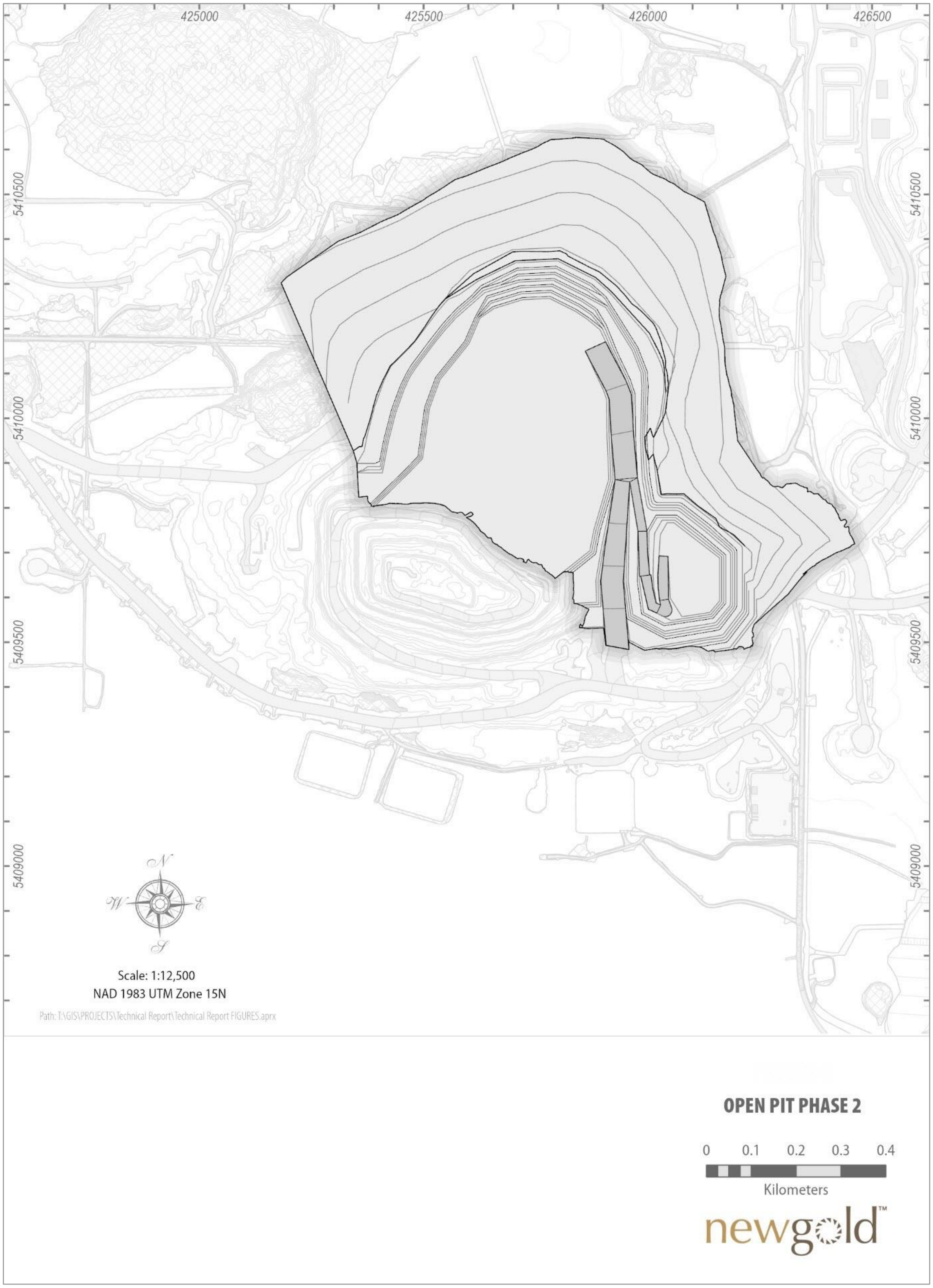
The mine design was developed on the optimized pit shell detailed in Section 15 by rationalizing the shape into a feasible mining geometry, and incorporating haulage ramps and detailed slope design criteria as presented in Section 16.1.3 and Section 16.1.4. Haulage ramps were designed nominally at 33 m width and a maximum ± 10% grade, except for the bottom few benches where widths were permitted to be reduced to one-way traffic of 20 m and ± 12% grade.

Following the design of the ultimate pit, the pit was subdivided into a series of mining phases. Phases are mining shapes, which except for the final pit limit, never exist exactly as depicted during the LOM. The phases determine the conceptual development of the open pit, with the objective of outlining the feasible mine development which will dictate, along with a mine plan, the order of presentation of ore and waste materials required to maximize net present value (NPV).

The selection of the mining phases was based upon an incremental analysis of optimized pit solutions generated at increasing gold prices, as well as geometric considerations for safe and efficient mining and access to the primary crusher, ore stockpiles and waste storage facilities.

Three additional mining phases were developed beyond the current Phase 1 which has already been excavated. These are designated Phase 2, Phase 3, and Phase 4, with Phase 4 also representing the final pit limit design. Figure 16.8 through Figure 16.10 illustrate the phase designs developed for Rainy River.

Figure 16.8 Open pit Phase 2



Source: New Gold 2020.

Figure 16.9 Open pit Phase 3

Source: New Gold 2020.

Figure 16.10 Open pit Phase 4

Source: New Gold 2020.

16.1.6 Mining method

The open pit mine is a conventional truck and shovel mining operation, with a fleet of 220 t payload haul trucks combined with diesel powered hydraulic excavators and large FELs as the primary loading units. The open pit operates at a peak mining rate of 151,000 tpd of ore and waste and has an overall strip ratio of 2.53:1 (waste:ore).

16.1.6.1 Drilling

Production drilling is carried out by a fleet of Sandvik diesel powered blasthole drill units. The fleet consists of four Sandvik D75KX down-the-hole drills which drill 216 mm diameter holes, two Sandvik DR580s and one DR560 down-the-hole drills which drill 171 mm diameter drillholes. Blasthole drills are configured to drill the 10 m height of the bench plus one additional metre of subdrill. Drill patterns vary from 5.2 m x 6.0 m for the 216 mm drillholes to 4.5 m x 5.2 m for the 171 mm drillholes.

Presplit drilling of pit walls is accomplished primarily using the Sandvik DR580s and DR560 drills with a 140 mm diameter drillhole spaced every 1.8 m linearly.

A single, Sandvik Pantera DR1500i top hammer, diesel powered drill rig which drills 140 mm diameter holes is primarily used for pioneering on the overburden / bedrock interface where more maneuverability is required. If additional support is required, the Sandvik DR580s and DR560 drills may be utilized.

Drill productivities are estimated at a rate of 21 m/operating hour.

16.1.6.2 Blasting

A complete down-the-hole explosives loading and initiation service is performed by a contractor. Services include the provision of explosive products, accessories and storage magazines as well as a mixing plant for the creation of emulsion. Emulsion explosives are used exclusively due to the general expectation of wet holes and design energy requirements. Explosive delivery trucks and in-hole explosive priming (non-electric detonators and boosters), emulsion pumping, and electronic initiation services are provided by the contractor’s blasting crew.

Powder factors range from 0.33 to 0.37 kilogram per tonne (kg/t) dependent on hole diameter and blasting pattern.

16.1.6.3 Loading

Primary loading activities are performed using a fleet consisting of large diesel powered hydraulic excavators in a front-shovel configuration accompanied by large FELs. The excavator fleet consists of one Komatsu PC8000 (42 m^3 bucket – 3,500 tonnes per operating hour (tpoh)) and two Komatsu PC5500 (30 m^3 bucket - 2,500 tpoh) units. The FEL fleet consists of one Komatsu WA1200 (18 m^3 bucket - 1,500 tpoh) with an additional CAT 994HL (18 m^3 bucket – 1,500 tpoh) planned to be incorporated during 2020 to assist primarily with rehandle. Preferentially, the PC8000 is attempted to be scheduled in waste, with the PC5500s scheduled in ore and waste. The FELs, due to their mobility, are assigned to ore or waste as required and are utilized for stockpile rehandle.

An additional Komatsu P3000 (15 m^3 bucket – 1,250 tpoh) diesel powered hydraulic excavator is also part of the fleet, and supports loading operations, stockpile rehandle, face cleaning, etc.

16.1.6.4 Hauling

Hauling is performed by a fleet of Komatsu 830E / 830E-AC electric drive rear-dump haul trucks in the 220 t payload class. The fleet is primarily used for mine production and stockpile rehandle, however, it is also involved in tasks such as clean-up, snow-handling and other support functions. Under certain circumstances, and providing hauling capacity exists, the fleet may be used to support transport of waste to the tailings management area (TMA) for construction purposes.

16.1.7 Mine planning

The mine plan is executed to take advantage of the installed mine fleet productive capacity, allowing an elevated COG policy to be adapted, whereby higher-grade ores are preferentially sent to the mill for processing while lower grade ores are sent to stockpile for deferred processing. This results in an open pit mine life extending to Q1-2025 with stockpile rehandling occurring in parallel to the underground operations through to Q1-2028 to fulfill available process plant capacity.

A significant amount of operational stockpile rehandle is included within the mine plan as a net result of the use of an elevated COG policy, the sequence of ore / waste presentation during the mine plan and processing rate limitations. Operational rehandle is included at approximately 40% of ore mined on average during peak mining years. AMC recognizes that this amount of operational rehandle appears excessive and recommends a review of strategic and tactical planning options to limit this quantity while maintaining or improving the mine plan NPV.

In-pit rehandling of waste is also incurred during mining whereby mined waste rock is dumped in-pit for the preparation of road and platform foundations for equipment excavating overburden. The requirement of waste rock to be rehandled in-pit for this use is estimated as 15% of the overburden tonnes excavated.

Waste from the open pit is identified as either overburden (including glacial tills and clays), non-acid generating waste (NAG) or potentially acid generating waste (PAG). Waste is stored at two locations, the East mine rock stockpile (EMRS) and the West mine rock stockpile (WMRS).

The EMRS is designed to accommodate a combination of overburden and PAG waste mine rock, while the WMRS is designed to accommodate a combination of overburden waste and NAG waste mine rock, as detailed in Section 18. In addition, the EMRS is designed to accommodate the mid- to long-term ore stockpiles. The TMA and east outcrop (EOC) also have capacity to accommodate materials from the mine. Facility designs and capacities updated by Rainy River to year-end 2019 and storage requirements based on the mine plan are indicated in Table 16.9.

Table 16.9 Mine rock stockpile requirements and capacities

Stockpile	Required capacity (Mm^3)					
	OVB	**NAG**	**PAG**	**Stockpile**	**Total**	**Design [1]**
EMRS	7.8		41.7	9.0	58.5	59.0
WMRS	3.0	23.1			26.0	26.0
TMA		7.7	2.3		10.0	10.0
EOC				2.0	2.0	2.0
Total	**10.8**	**30.8**	**44.0**	**11.0**	**96.5**	**97.0**

Note: [1] Design capacities may differ from those in Section 18 as these have been updated based on year-end 2019 status.

Commencing in 2020, NAG requirements for TMA construction are entirely fulfilled from in-pit mine production. NAG has been found to more consistent and recoverable on the south side of the orebody (HW) than on the north side of the orebody (FW) where most of the mining has focused in the recent past. At a recovery rate averaging 65% of modeled NAG, there is sufficient NAG to fulfill the LOM TMA construction schedule requirements. No further mining of the EOC for NAG construction rock is included in the mining schedule.

Table 16.10 presents the open pit mine production schedule.

Table 16.10 Open pit mine production schedule

Year	Ore tonnes (000s)	Grade		Contained metal		Waste tonnes (000s)	Total tonnes (000s)	Strip ratio (w:o)	Rehandle tonnes [1] (000s)	Total tonnes moved (000s)
		Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)					
2020	13,232	0.74	2.8	313	1,185	42,011	55,243	3.2	6,647	61,890
2021	15,452	0.82	2.3	408	1,164	39,772	55,224	2.6	5,309	60,533
2022	10,333	1.00	2.2	332	723	40,806	51,139	3.9	6,177	57,317
2023	11,924	0.97	2.4	373	900	34,520	46,444	2.9	4,617	51,061
2024	15,000	1.04	2.0	501	950	13,339	28,340	0.9	4,247	32,586
2025	1,607	1.13	1.9	58	98	323	1,930	0.2	7,337	9,267
2026	-	-	-	-	-	-	-	-	8,285	8,285
2027	-	-	-	-	-	-	-	-	8,331	8,331
2028	-	-	-	-	-	-	-	-	1,537	1,537
Total	**67,547**	**0.91**	**2,3**	**1,985**	**5,019**	**170,772**	**238,319**	**2.5**	**52,488**	**290,807**

Notes: Totals may not compute exactly due to rounding.
[1] Rehandle tonnes include inpit and ex-pit rehandle. Excludes underground ore rehandle from portal stockpiles.

16.1.8 Equipment requirements

Mine equipment requirements were developed by Rainy River from the annual mine production schedule, equipment availability, utilization and equipment productivities.

Equipment productivities were determined for drills, shovels, and loaders based on historical operating parameters and reasonable productivity improvements currently being implemented. Haul truck productivity is also dependent on annual cycle times. Required production hours were calculated for all primary equipment as well as support equipment. A summary of peak principal open pit mining equipment requirements is presented in Table 16.11.

Table 16.11 Peak principal open pit mining equipment requirements

Description	Manufacturer	Model	Units
Production drill	Sandvik	DR580	2
Production drill	Sandvik	DR560	1
Production drill	Sandvik	D75KS	4
Presplit drill	Sandvik	DR1500i	1
Hydraulic excavator	Komatsu	PC8000	1
Hydraulic excavator	Komatsu	PC5500	2
Hydraulic excavator	Komatsu	PC3000	1
Wheel loader	Komatsu	WA1200	1
Wheel loader	CAT	994HL	1
Wheel dozer	Komatsu	WD900	1
Haul truck	Komatsu	830E / 830E-AC	25
Dozer	Komatsu	D475	2
Dozer	CAT	D10T	3
Dozer	CAT	D9T	4
Dozer	CAT	D8T	1
Dozer	CAT	24M	1
Grader	CAT	16M	4
Excavator	CAT	390F	1
Tire handler	Komatsu	WA600	1
Fuel & lube truck	Komatsu	HM400	3

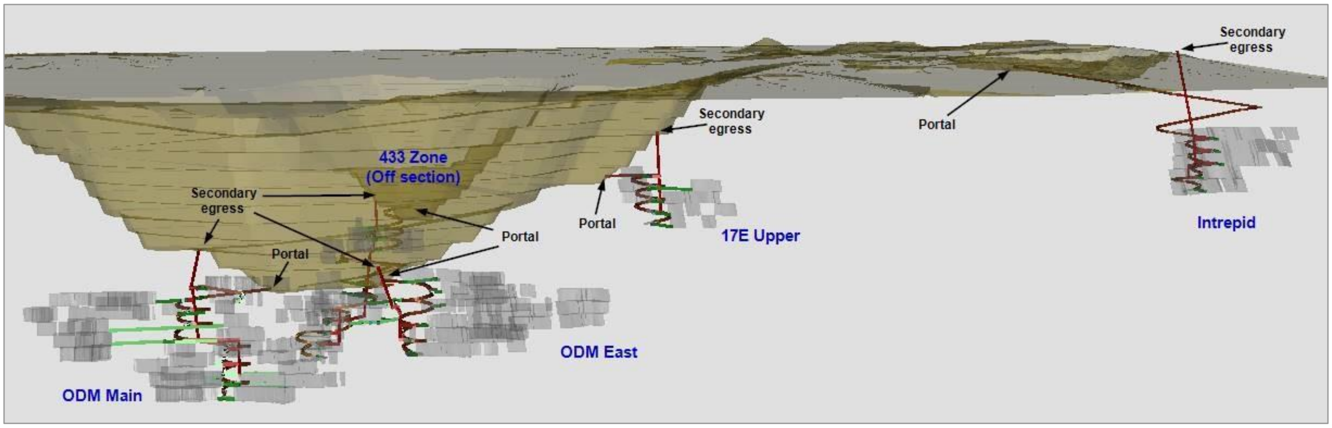
Except for the addition of a leased CAT 994HL to the fleet in 2020 to assist principally with stockpile rehandle, peak open pit mining equipment requirements correspond to the current fleet size.

Note that in addition to the principal fleet, a support fleet of smaller equipment is available for miscellaneous activities and jobs at the mine site. This miscellaneous fleet consists of small FEL's, graders, compactors, etc.

16.2 Underground mining

The Rainy River underground mining operation is planned to extract ore from five portals and / or zones: ODM Main Zone, ODE East Zone, 17 East Upper Zone, 433 Zone, and Intrepid Zone. These are shown in Figure 16.11.

Figure 16.11 Isometric view of the Rainy River underground (looking north)



Source: AMC 2019.

Each of the underground zones is planned to be mined using one or both of the following mining methods:

- Blind uphole LLHOS without backfill (uphole).
- Downhole LLHOS with backfill (downhole).

The smallest of the mining areas is the 17 East Upper Zone, which is the first underground area planned for ore mining, with the decline portal starting in July 2022. The zone has a dip between 55° and 70°, a vertical depth of 80 m (170 RL to 90 Reduced Level (RL)), and a lateral extent of approximately 170 m. The mining method used in this area will be uphole stoping. The mining methods (i.e. uphole and downhole stoping) are further discussed in Section 16.2.2.

The Intrepid Zone is the only zone with an existing portal and has approximately 166 m of decline development currently executed. The portal is at 365 RL and planned stopes are located between 225 RL and 100 RL. The zone extends approximately 250 m horizontally and dips between 50° and 70°. This area will be mined using an uphole stoping method. The decline development is planned to start up in Q2-2020 as an orebody investigation project for the Intrepid Zone. The total length of decline for the orebody investigation project will be 600 m. The Intrepid Zone decline advance is then planned to recommence in June 2023, with stoping to begin in December 2023.

The 433 Zone is located beneath the open pit and will commence mining (portal collaring) in July 2024 once the northern lobe of the open pit is completed. The zone has a vertical depth of 240 m (70 RL to -170 RL) and a lateral extent of 70 m to 200 m. The zone has a dip between 55° and 65° and stopes will be mined by uphole stoping.

Approximately one third (~36%) of the underground Mineral Reserve tonnage will be extracted from the three zones discussed above (17 East Upper Zone, Intrepid Zone, and 433 Zone). The remaining ore (~64%) will be mined from the two remaining zones: ODM Main Zone and ODM East Zone, which are further described below.

The ODM Main Zone will commence portal collaring for the decline in April 2025, immediately after the completion of the open pit. The ODM Main Zone extends approximately 400 m horizontally and has a vertical depth of 180 m (30 RL to -150 RL). The zone dips between 50° and 70° and is planned to be mined using both uphole and downhole mining methods.

The proportion of stope tonnages in the ODM Main Zone that are uphole and downhole are 10% and 90%, respectively. The only zone that utilizes downhole with backfilling is the ODM Main Zone.

The ODM East Zone also starts a portal and decline in April 2025 after the open pit has finished mining. The zone dips between 45° and 65° and has a horizontal extent of approximately 550 m. The stopes are located between 30 RL and -90 RL and are planned to be mined using upholes.

16.2.1 Geotechnical considerations for underground

Site investigations and initial Feasibility Study level work for underground geotechnical assessment and design (stopes, access development, ground support, and backfill) were performed and reported by AMEC as part of the updated Feasibility Study for the Mine completed in 2014 on behalf of New Gold (AMEC 2014). During the geotechnical investigation study, a Map3D^{TM} linear elastic modelling exercise was undertaken to review the proposed mining design and sequence, and stress elevation around the planned development and stopes, and to estimate the levels of ground support required.

In the 2012 drilling campaign, three main zones of what was termed 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 selected cores of exploration holes, were subsequently geotechnically logged by AMEC, and representative core samples in the HW, ore zone (OZ), and FW of each zone (depth > 400 m) were tested. The core logging data and lab test results were the basis of the initial underground geotechnical assessment and design parameters.

In 2016 and 2017, BGC completed Feasibility Study level studies for the open pit (BGC 2017); much of the data and information from the BGC study is relevant to underground mining and has been incorporated into the current study.

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 conducted additional data collections including geotechnical logging of selected representative core intervals and mapping of open pit, quarry walls and the Intrepid Zone portal site. NRMS reviewed and updated the underground geotechnical assessment and design (stopes, equivalent linear overbreak / slough (ELOS), dilution, ground support, etc.). NRMS also reviewed the AMEC Map3D stress model and performed additional two-dimensional (2D) and 3D stress modelling of the updated Feasibility Study mining shapes.

In 2019, AMC conducted a geotechnical review and update for underground mine design criteria, with focus on the stability assessment of open stopes. The open stope stability assessment was conducted based on existing geotechnical data.

16.2.1.1 Underground geotechnical considerations

The overall rock mass quality in terms of RQD at Rainy River underground is classified as "Fair" to predominantly "Good", with RQD typically ranging from 90% to 100% throughout all stoping domains. With respect to the Modified NGI Q-system, Q' (after Barton et. al. 1974) average values of 23, 17, and 19, were obtained characterizing the HW, OZ, and FW 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 HW, OZ, and FW domains, respectively.

Although general rock mass conditions in all domains can be characterized as "Fair" to predominantly "Good" (Barton et. al. 1974), 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 as currently planned does not intersect these zones.

Table 16.12 summarizes Rock Mass Classification data (Q', RMR_{76}, and Geological Strength Index (GSI)) by mining zone (AMEC 2013).

Table 16.12 Summary of underground rock mass classification

Zone	Zone length		Run	Q'				RMR				GSI			
	From (m)	To (m)	(#)	Avg	Stdv	Min	Max	Avg	Stdv	Min	Max	Avg	Stdv	Min	Max
HW	0	50	49	19.2	7.5	5.8	50.3	70	4	60	79	70	6	58	82
OZ	0	34*	36	16.1	6.8	5.2	36.9	68	4	59	76	70	5	54	81
FW	0	50	50	17.7	7.5	6.6	40.0	69	4	61	77	72	4	61	80
OZ + FW	0	84*	86	17.0	7.2	5.2	40.0	69	4	59	77	71	5	54	81

Notes:
- *Average Length.
- Avg: average.
- Stdv: standard deviation.
- Min: minimum.
- Max: maximum.
- Q: Tunneling Quality Index.
- Q': modified Q with SRF = 1, where SRF denotes stress reduction factor.
- RMR: Rock Mass rating.
- GSI: Geological Strength Index.

Lab testing consisting of 30 uniaxial compressive strength (UCS) tests, 27 triaxial tests and 24 Brazilian tensile tests was used to determine the elastic and strength parameters for each rock unit and develop rock mass failure criteria. The test results of the data reduction are presented in Table 16.13 (AMEC 2013).

Table 16.13 Summary of underground intact rock properties and derived strength parameters

Orebody	Zone	Density (kg/m^3)	E (GPa)	μ	UCS (σ_c) (MPa)		Brazilian Tensile Strength σ_t (MPa)		Hoek-Brown	Mohr-Coulomb	
					Lab	RocLab [1]	Lab	RocLab [1]	m_i	c (MPa)	φ (deg)
ODM (UG-01)	HW	2,898	116	0.34	67.8	72.3	-14.9	-17.2	4.20	18.7	32.2
	OZ	2,810	N/A	N/A	105.8	101.5	-15.8	-16.6	6.10	22.9	38.5
	FW	2,826	51	0.14	108.3	123.9	-18.5	-19.3	6.41	27.2	40.0
17 (UG-02)	HW	2,834	76	0.31	110.8	106.3	-13.8	-16.5	6.44	23.5	39.3
	OZ	2,785	46	0.39	110.3	105.0	-17.8	-23.9	4.40	26.3	34.4
	FW	2,773	47	0.31	73.8	70.8	-11.2	-12.6	5.62	16.8	35.7
17 E Ext (UG-03)	HW	2,726	75	0.27	75.5	87.7	-10.8	-11.0	7.99	18.5	41.0
	OZ	2,764	74	0.33	165.7	143.3	-20.0	-21.0	6.83	30.6	41.4
	FW	2,806	70	0.33	126.3	119.0	-12.5	-15.8	7.53	24.8	41.8

Notes:
- [1] Rocscience software program for determining rock mass strength parameters.
- E: Young's modulus.
- μ: Poisson's ratio.
- σ_c: compressive strength.
- σ_t: tensile strength.
- m_i: material constant for the intact rock.
- c: cohesion.
- φ: friction angle.

The overall average UCS results for the HW, OZ, FW, and OZ + FW, are presented in Table 16.14, indicating strong to very strong rocks. Field assessments by NRMS confirmed these ranges as accurate and representative.

Table 16.14 Summary of UCS test results for the stoping domains

Zone	Rock type	Test #	Avg	Stdv	CV	Min	Max
HW	MMV / IMV	10	87.3	42.6	49%	36.1	171.4
OZ	FLS	10	125.1	39.4	32%	87.3	221.2
FW	FLS	10	103.4	28.7	28%	68.3	168.7
OZ + FW	FLS	20	114.2	35.4	31%	68.3	221.2

Note: CV = coefficient of variation.

In 2019, AMC reviewed and updated the stope stability design based on the updated mine design and existing rock mass classification data using the same empirical modified stability graph method (after Potvin 1988, Nickson 1992, and Hadjigeorgiou et. al. 1995). Several scenarios have been analyzed to evaluate the stability of the HW and the back of an open stope with respect to stope inclination, stope width and rock mass classification for maximum 'unsupported' and 'supported' stable stope dimensions (in terms of strike length of an individual stope for a given sublevel interval of 20 m and typical stope width of 8 m). The maximum stable, unsupported and supported strike lengths for both HWs and backs have been projected. Table 16.15 summarizes the hydraulic radius (HR) design limits and associated maximum strike length for both unsupported and supported cases for anticipated rock mass conditions.

Table 16.15 Design limits for a stable open stope

Q'	Dip of stope face	A	B	C	N'	Unsupported HR (m)	Maximum unsupported strike length (m)	Supported HR (m)	Maximum supported strike length (m)
40 Upper	HW (55)	1	0.3	4.6	54.7	10.6	161	14.3	Infinite
	HW (60)	1	0.25	5.0	50.0	10.2	124	14.1	Infinite
	HW (65)	1	0.2	5.5	43.7	9.8	99	13.5	Infinite
	Back (0)-260 mbgs	0.43	0.7	2.0	24.1	7.6	Infinite [1]	11.6	Infinite [1]
	Back (0)-400 mbgs	0.26	0.7	2.0	14.6	6.4	Infinite [1]	9.8	Infinite [1]
	Back (0)-800 mbgs	0.1	0.7	2.0	5.6	4.5	Infinite [1]	7.3	Infinite [1]
10 Typical	HW (55)	1	0.3	4.6	13.7	6.2	25	9.8	97
	HW (60)	1	0.25	5.0	12.5	6.1	24	9.6	90
	HW (65)	1	0.2	5.5	10.9	5.8	22	9.2	75
	Back (0)-260 mbgs	0.43	0.7	2.0	6.0	4.6	Infinite [1]	7.7	Infinite [1]
	Back (0)-400 mbgs	0.26	0.7	2.0	3.6	3.9	312 [1]	6.6	Infinite[1]
	Back (0)-800 mbgs	0.1	0.7	2.0	1.4	2.7	17 [1]	5.0	Infinite[1]
3 Lower	HW (55)	1	0.3	4.6	4.1	4.0	12	6.9	32
	HW (60)	1	0.25	5.0	3.8	3.9	12	6.7	32
	HW (65)	1	0.2	5.5	3.3	3.8	11	6.4	27
	Back (0)-260 mbgs	0.43	0.7	2.0	1.8	3.0	22 [1]	5.4	Infinite [1]
	Back (0)-400 mbgs	0.26	0.7	2.0	1.1	2.6	15 [1]	4.8	Infinite [1]
	Back (0)-800 mbgs	0.1	0.7	2.0	0.4	1.7	7 [1]	3.8	152 [1]

Notes:
- [1] For 8 m wide stope.
- A: rock stress factor; determined by the ratio of max. induced stress of stope face to intact rock UCS (114 MPa); where max. stress in the back was estimated by doubling the pre-mining horizontal stress perpendicular to the ore strike proposed by Yves and Hadjigeorgiou (2001); A is assumed to be 1 for stope walls.
- B: joint orientation factor; determined based on the orientation of dominant joints relative to the stope surface (AMEC 2013).
- C: gravity adjustment factor; determined based on dip of stope face.
- N: modified stability number; given by $N' = Q' \times A \times B \times C$.
- HR: hydraulic radius; given by $HR = \frac{area}{perimeter} = \frac{w \times l}{2(w+l)}$; where *w* and *l* are surface width and surface length, respectively.

Rib pillars have been incorporated within longhole open stoping (LHOS) zones to break exposed excavation spans into "permissible" and "stable" dimensions. NRMS conducted 2D and 3D elasto-plastic stress modelling to assess the stope and pillar stability as per mining methods. Based on the modeling results, NRMS concluded that a nominal pillar width of 8 m is suitable for the majority of stopes, while the pillar dimensions can and should be reviewed and modified as local knowledge of geotechnical and hydrogeological conditions, in situ stress, structure features, etc. become better understood. It should be noted that the numerical modelling conducted by NRMS has not been calibrated to actual stope and pillar behavior and can only be used as a guide to inform the design process. A cavity monitoring survey (CMS) and geotechnical instrumentation and monitoring program should be implemented to monitor the responses of stopes and pillars to mining. The numerical models should be further calibrated based on the geotechnical monitoring data and CMS data. Additional numerical modelling (forward analysis) should be undertaken to investigate the stope and pillar stability as per mine design and sequence using the calibrated model.

16.2.1.2 Ground support designs for underground

NRMS assessed the ground support requirement as per development profile and type of development (permanent with service life more than 1 year and temporary with service life less than 1 year) and anticipated rock mass conditions using the Q system chart for tunnel support guideline (after Grimstad and Barton 1993) and Unwedge analysis (Rocscience 2018).

The minimum ground support for typical rock mass conditions anticipated ('Good' with Q ranging from 10 to 40, and GSI >75, and 'Fair' with Q ranging from 4 to 10 and GSI ranging from 55 to 75) are presented in Table 16.16.

Table 16.16 Proposed ground support requirements for permanent lateral development

Support class (SC)	**Ground support requirements**	
	Less than 6 m span	**6 m span or above**
Q 10 – 40 GSI 75 - 100	**Primary support:** • Minimum 9 Ga wire mesh or chain-link equivalent to springline. • 1.8 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.5 x 1.5 m spacing in back. • 1.5 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required.	**Primary support:** • Minimum 9 Ga wire mesh or chain-link equivalent to springline. • 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.5 x 1.5 m spacing in back. • 1.5 -2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls ad required. **Secondary support for span 8 m to 15 m:** • Minimum 6 m long 0.6''-0.7" twin-strand bulbed cablebolt on 1.5 m to 2.5 m square pattern or an approved equivalent.
Q 4 – 10 GSI 55 - 75	**Primary support:** • Minimum 9 Ga wire mesh and six 1.2 m long split sets at the face. • Minimum 6 Ga wire mesh or chain-link equivalent to 1.8 m above sill. • 1.8 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.4 x 1.6 m spacing in back and wall, with 0.8 m 'Dice-5' offset row. • 1.5 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required.	**Primary support:** • Minimum 9 Ga wire mesh and six 1.2 m long split sets at the face. • Minimum 6 Ga wire mesh or chain-link equivalent to 1.8 m above sill. • 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt on 1.4 x 1.6 m spacing in back and wall, with 0.8 m 'Dice-5' offset row. • 2.4 m long #6 fully encapsulate rebar bolt or equivalent bolt in walls as required. **Secondary support for span 8 m to 15 m:** • Minimum 6 m long 0.6''-0.7" twin-strand bulbed cablebolt on 1.5 m to 2.5 m square pattern or an approved equivalent.

Note: AMC advises that final definitive grounds support requirements should be aligned with actual conditions experienced in underground operations.

For any rock mass conditions encountered of 'Fair to Poor', or 'Poor', or 'Extremely Poor', additional ground support may be required for long term stability, which may include the addition of the following support elements, progressively:

- Steel / mesh straps.
- 5 to 10 cm of plain or fibre reinforced shotcrete applied to the back, walls, or other specified target areas.
- 3 to 10 m long bulbed twin-strand cablebolt on 1.5 to 3.0 m square pattern. These may be required to support intersections and larger spans, potential wedges and blocks formed by persistent structure, fault zones, etc.

16.2.1.3 Underground dilution

Discussed in Section 15.

16.2.1.4 Open pit – underground interaction

The underground mine designs for the 17 East Upper Zone, OME Main Zone, ODM East Zone, and 433 Zone are all beneath the open pit or along the open pit walls. Underground stopes will not daylight into the open pit to avoid stability issues. A separation distance of 20 m has been utilized for design purposes. This can be reassessed as underground mining progresses and operational experience is gained relative to pillar stability between open pit and underground. For extraction estimates, the recovery factor for stopes in close proximity to the 20 m open pit crown pillar has been reduced to 60%.

Most stopes in areas close to the open pit are relatively narrow and, as such, are not anticipated to affect the open pit wall stability.

16.2.2 Mining method

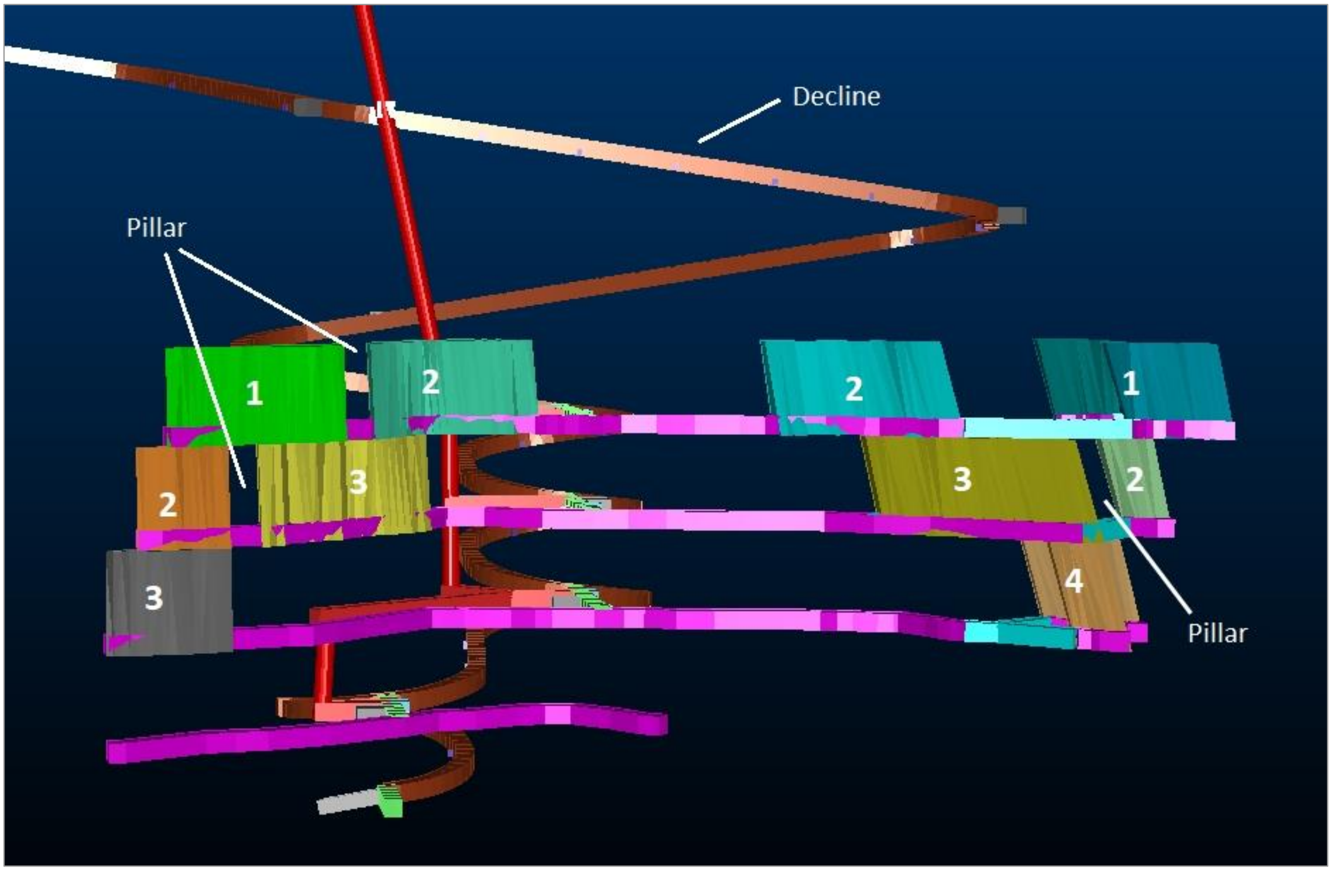
The selected mining method for the Rainy River underground mine is LLHOS. There are two types of LLHOS employed:

- Blind uphole LLHOS without backfill (uphole).
- Downhole LLHOS with backfill (downhole).

16.2.2.1 Blind-uppers LLHOS (upholes)

Uphole is a top-down mining method that does not require the use of backfill. Figure 16.12 shows an isometric view of a typically ongoing (i.e. snapshot in time) uppers top-down mining sequence using numbers to illustrate the order in which the stopes are mined. Uphole mining is planned to be utilized in all five mining zones at Rainy River and represents 68% of all stope tonnages.

Figure 16.12 Isometric view of uppers

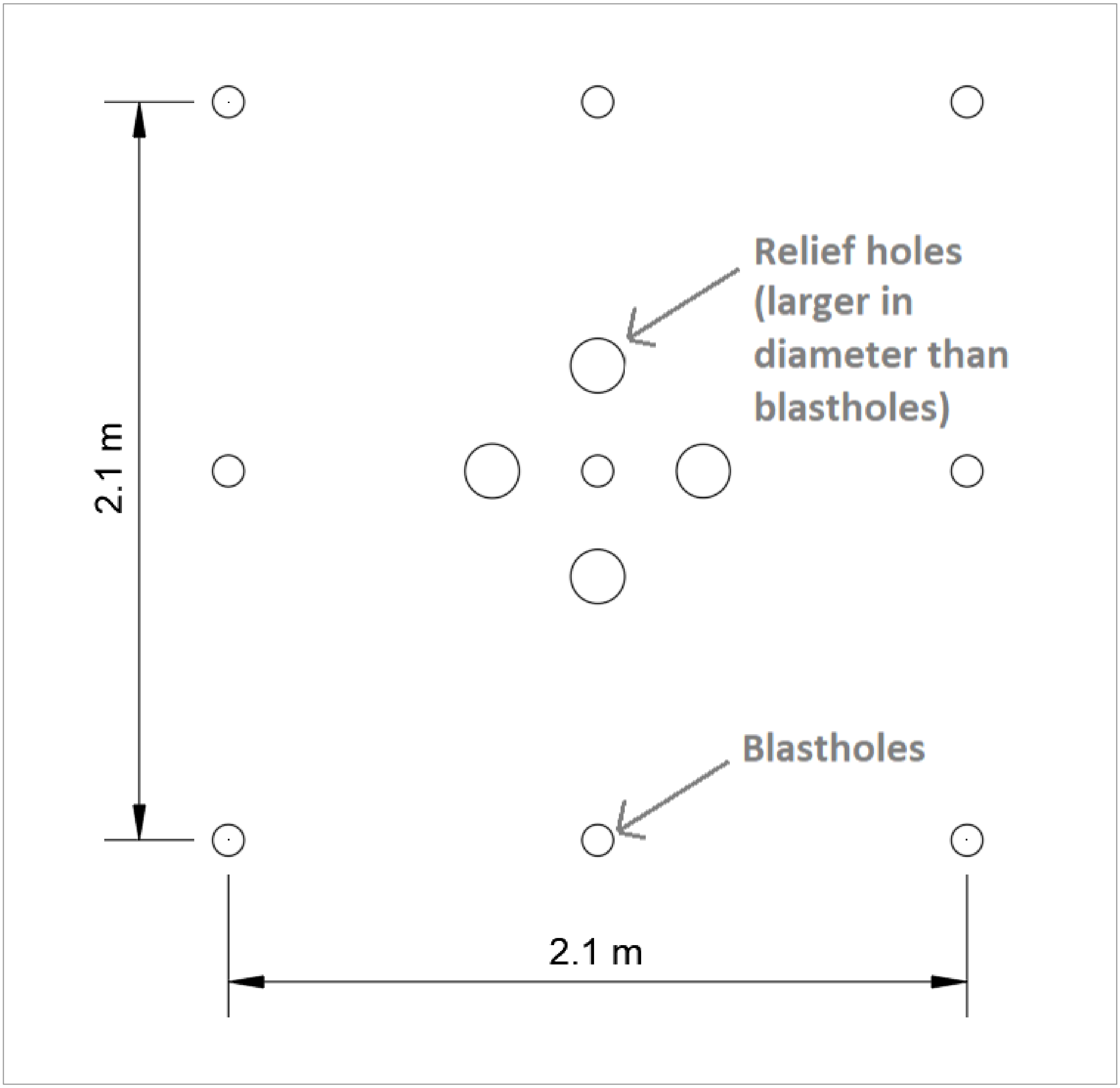


Source: AMC 2019.

The uphole stopes have a minimum mining width of 3 m, a maximum width of 15 m, and an overall average width of 7.5 m. The sublevels will be spaced at 20 m intervals (floor to floor) in all zones except for the Intrepid Zone, where the stope heights have been increased to 25 m. Upholes are mined in a longitudinal retreat sequence, with open stopes up to 48 m long separated by a minimum 8 m wide pillar.

The undercut for the uphole stope is driven at 5.0 m x 5.0 m in ore at the base of the stope. A blind-uppers raise (see drill pattern in Figure 16.13) is driven at the end of the stope followed by longhole drilling of the slot holes and production rings.

Figure 16.13 Uppers raise drilling pattern



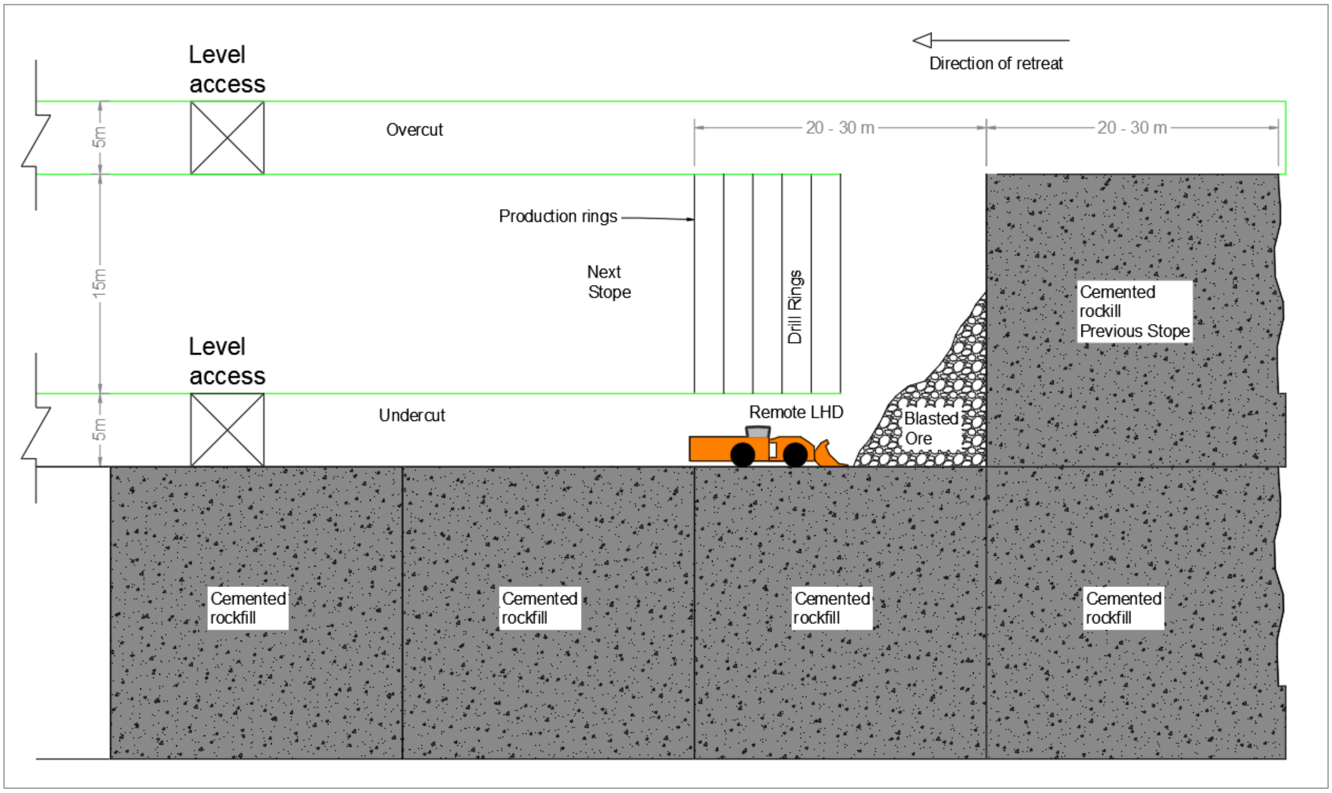
Source: AMC 2019.

Slot blasting will occur once drilling of all the raise and slot holes, as well as the production rings, is completed. All ore will be mucked from the stope undercut for both slot and production blasts using a mixture of manual mucking when the brow is choked with ore, and remote scoop operation when the brow is open. Confirmation of complete extraction of the stope by operations supervision will initiate the engineering department to complete the stope CMS and finish the stope mining cycle. The next uphole stope in sequence will then commence mining, repeating the steps described above, which include drill, blast, muck, and CMS. This mine sequence will continue until the entire mining zone is completed.

16.2.2.2 Downhole LLHOS with backfill (downhole)

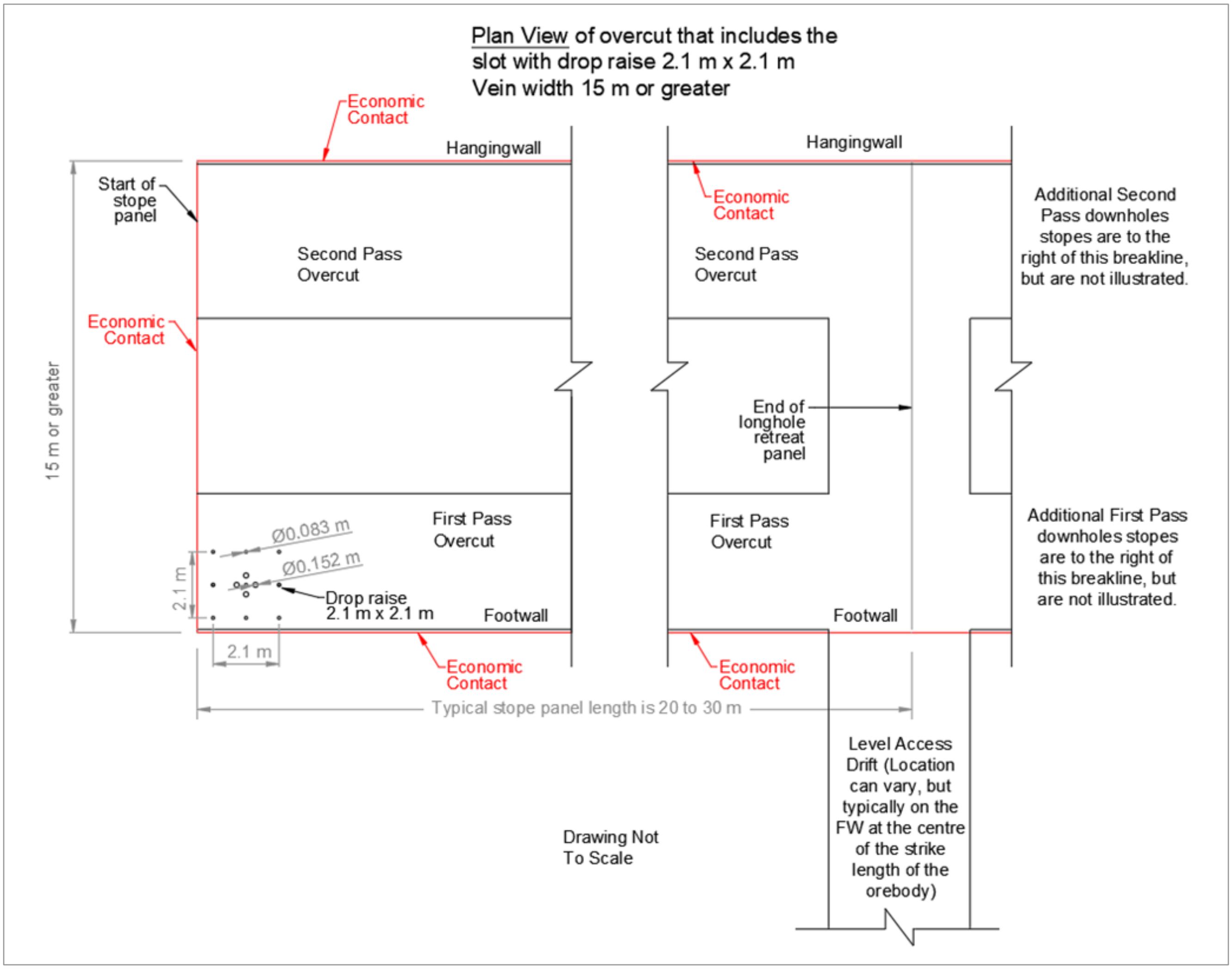
The downhole mining method is a bottom-up mining method that uses backfill (Figure 16.14 and Figure 16.15). Downhole stopes are only used in the ODM Main Zone, where 90% of the stopes are planned as downhole and 10% are planned as uphole. The downhole stopes represent 32% of all projected stope tonnages for the underground LOM.

Figure 16.14 Downhole long section



Source: AMC 2019.

Figure 16.15 Downhole plan view



Source: AMC 2019.

Downholes are mined as a longitudinal retreat (with a first- and second-pass sequence) and are utilized when the stope width exceeds 15 m from HW to FW contacts. The sublevel intervals are 20 m (floor to floor) and stope lengths are from 20 m to 30 m. Overcut and undercut drifts are driven, at 5.0 m x 5.0 m, on the FW side on the stope initially (see Figure 16.14 and Figure 16.15).

Once all ore drives (overcut and undercut) are completed for the first pass, blastholes are drilled for the raise and slot at the end of the stope that is furthest from the level access drift. Next, production rings are drilled for the remainder of the stope. Blasted muck for the raise / slot and production blasts will be mucked from the undercut using a mixture of manual (when brow is choked off) and remote (when the brow is open) techniques. When operations supervision confirms the complete extraction of the stope, a CMS is initiated by engineering. Once the CMS is completed, then backfilling begins using cemented rockfill (CRF). After the fill has cured, the next downhole stope in the first-pass sequence commences and repeats the drill, blast, muck, CMS, and fill process. This sequence of mining keeps going until all the first-pass stopes are completed.

The second-pass extraction begins by driving the overcuts and undercuts at 5.0 m x 5.0 m, on the HW side on the stope. The second-pass downhole stopes follow the same sequence of mining activities, namely: drill, blast, muck and CMS. However, the backfill utilized is uncemented rockfill (URF). The illustration Figure 16.14 would be the same except the "Cemented rockfill" would be substituted with the word "Uncemented rockfill". After the second-pass stope is filled with URF, the next downhole stope in the second-pass sequence begins and repeats the drill, blast, muck, CMS and rockfill process. This sequence of mining keeps going until all the second-pass stopes are completed.

It is noted that additional amounts of dilution are anticipated for downhole second-pass stopes, which has been accounted for in the mine production schedule.

16.2.3 Stope design

AMC used the software Mineable Shape Optimizer^{TM} (MSO) on the Mineral Resource block model to produce conceptual stope shapes. The estimation is based on the application of the LLHOS mining method (uphole and downhole). Stope wireframes were generated using MSO on a 2 m increment. Once the preliminary stopes were generated, a check was made to remove any outlying stopes that would not be economic when the cost of access development was included.

After the projected economic value stopes were selected, the wireframes were combined into stopes 48 m in length for uppers, with adjacent 8 m wide pillars. Where possible, pillars were strategically placed in areas without MSO stope shapes or in locations that had MSO shapes with lower grades / tonnes.

Similarily, the downhole stopes were created after determining economically viable MSO shapes. As noted earlier, downhole stopes were created in zones that are greater than 15 m in width. The MSO shapes were split (to be less than 15 m in width from HW to FW) and were then combined into 20 m to 30 m lengths and mined in a first-pass / second-pass sequence.

For both upholes and downholes, any stopes overlapping with the open pit or within the 20 m open pit crown pillar were removed. AMC has assumed that selected stopes adjacent to the crown pillar would be partially extracted (60%) to ensure that stopes do not daylight into the open pit.

As described in Section 15, a mining recovery factor of 95% has been applied to all stope extractions. In addition, the mining recovery of sill pillars, where such are needed, is estimated to be 60%. The overall average unplanned stope dilution, inclusive of FW dilution, HW dilution, and backfill dilution, has been estimated at approximately 12%. A final check was made to remove any outlying stopes that would not be economic when the cost of access development was included.

The MSO parameters used for the generation of stope wireframes are summarized in Table 16.17. Key design parameters by mining method are shown in Table 16.18.

Table 16.17 MSO parameters for LLHOS shapes

Parameters	Field	Default	Units
Density (waste)	Density	2.76	t/m^3
Optimization field	AuEq*	0	g/t
COG (stopes)	AuEq	2.2	g/t
Slice interval		2.0	m
Default dip		90	degrees
Strike azimuth		15	degrees
Sub-blocking		No	
Optimization length		2.5	m
Minimum mining width		3	m
HW dilution		0.6	m
FW dilution		0.3	m
Minimum HW angle		55	degrees
Minimum FW angle		43	degrees
Maximum strike change		30	degrees
Stope maximum side-length ratio		2.25	ratio

Notes:
*Au price US$1,275 per troy ounce, Ag price US$17 per troy ounce.
*The exchange rate used was 1:1.30 US$/C$.
*AuEq is equal to Au (g/t) + [(Ag (g/t) * 17 * 60)/ (1,275 * 95)]

Table 16.18 Key design parameters by mining method

	Units	Uphole	Downhole
AuEq	g/t	2.2	2.2
Minimum mining width	m	3	3
Mining height	m	20	20
Mining length	m	Up to 48	Up to 30
Minimum HW angle	degrees	55	55
Minimum FW angle	degrees	55	55
HW unplanned dilution	m	0.6	0.6
FW unplanned dilution	m	0.3	0.3
Floor unplanned dilution	m	0.0	0.2
LHOS endwall unplanned dilution	m	0.0	0.2
Unplanned dilution*	%	12	13
Mining recovery	%	95	95

Note: *The overall unplanned dilution is 12%.

16.2.4 Development access

The underground mine has five portals for the five separate underground mining zones (ODM Main Zone, ODE East Zone, 17 East Upper Zone, 433 Zone, and Intrepid Zone), as previously discussed.

Each of the respective portals immediately transitions into a main decline driven at a -15% gradient. Generally, declines are 5 m x 5 m, but there was a requirement to enlarge the ODM Main Zone decline to 6 m x 6 m starting from the portal down to -70 RL to increase fresh air intake. Similarly, the 433 Zone has a decline of 5.5 m x 5.5 m from the portal down to -50 RL.

Each respective mining zone includes Alimak exhaust raise development (equipped with manway ladders) to establish and extend the primary air circuit, as well as providing secondary egress from the underground. As the mining levels are developed from the decline, raising from level to level will be undertaken.

16.2.5 Drill and blast

16.2.5.1 Introduction

The following sections discuss the assumptions used by AMC for blasting practices at Rainy River with respect to explosives selection, drill and blast designs, and logistics and explosives management.

16.2.5.2 Explosives selection

This section examines explosives utilization for underground stoping and development.

Bulk products and packaged explosives

Emulsion is a commonly utilized bulk explosive product that can be used for lateral development and can also be pumped both up and down into stoping blastholes. Emulsion is water resistant, produces low levels of ammonium nitrate in leachate, can be handled in bulk deliveries to reduce costs, has reliable and consistent performance and, depending on the product, can be left in the ground for up to four weeks before firing. Bulk emulsion is proposed as the primary explosive product to be used for both lateral development and stoping (uphole and downhole) at Rainy River.

ANFO (ammonium nitrate fuel oil) is the most basic bulk explosive product. It is widely available and can be poured or blown into development headings or used for uphole and downhole stoping methods. The equipment required to use ANFO is basic and easily maintained. However, the total ammonium nitrate in the leachate from blasting is higher for ANFO than bulk emulsion. If the total ammonium nitrate found in the leachate from explosives detonation is to be minimized, then ANFO should be replaced with emulsion. Further, ANFO is not water resistant, which makes it less versatile than emulsion when wet conditions are encountered. ANFO is also less desirable for blind upholes as it is limited to a blasthole size of 76 mm because of the explosive tending to fall out under gravity. Conversely, bulk emulsion can effectively be used in dry upholes to a maximum diameter of 102 mm. Emulsion is therefore the preferred bulk explosive.

Packaged explosive products are typically three to five times more expensive on a per unit basis compared to bulk explosive products such as emulsion or ANFO. Although packaged explosives can be used for small narrow blind uphole open stopes (such as in a typical narrow vein mine) they are not commonly used for large uphole stopes (such as Rainy River), primarily due to inefficiencies in loading large quantities of explosives. Therefore, packaged explosive products have also been eliminated in preference to using bulk emulsion.

Detonators

Non-electric detonators are the most basic detonation system and are proposed for lateral development and stoping. They are widely available, relatively simple to use, and less expensive than electronic detonators.

Another viable blast initiation system is electric detonators (or electronic detonators), which have significant advantages over non-electric detonators; these include:

- Electric detonators are a single product (where the delay is programmable). This reduces the required site stock and increases stock turnover.

- The detonator delay is programmable. This allows increased blast size, precision blasting, and better fragmentation.
- The delay scatter (prevalent in non-electric detonators) is eliminated.
- The complexity of the charge process for operators is reduced.
- Sensitivity of detonators to handling during the charge process is significantly reduced.
- Multiple security features make them virtually immune to interference, unauthorized use, or unplanned detonation.

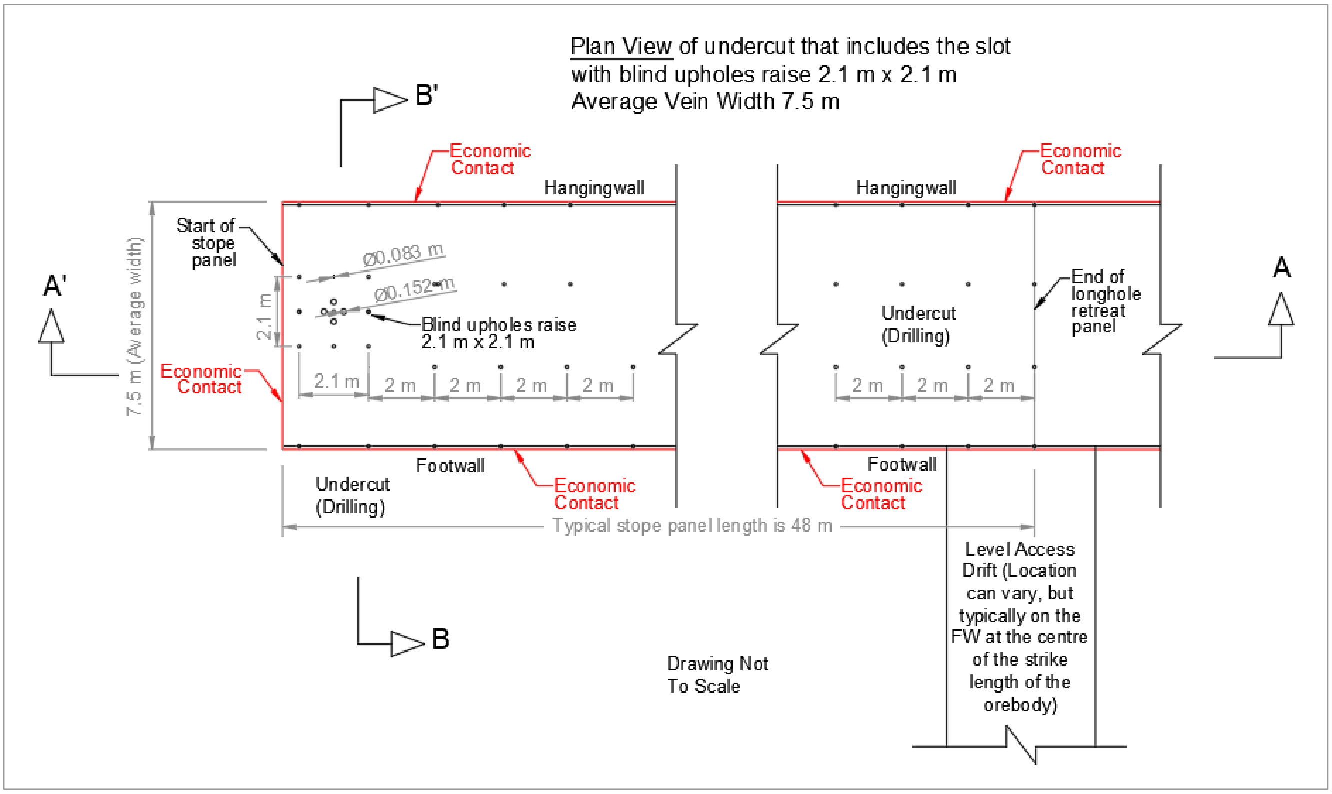
Two commonly available electric detonator systems are Dyno Nobel's SMARTSHOT system and Orica's IKON system. All electric detonator systems are more complex than non-electric systems and require both operator and engineer training to be used safely and effectively. Training is provided by the system suppliers.

Electric detonators (or electronic detonators) are the highest quality detonating system available but are four to five times more expensive than non-electric systems; they are, therefore, not recommended as the proposed main detonating system for Rainy River.

16.2.5.3 Upholes drill and blast design

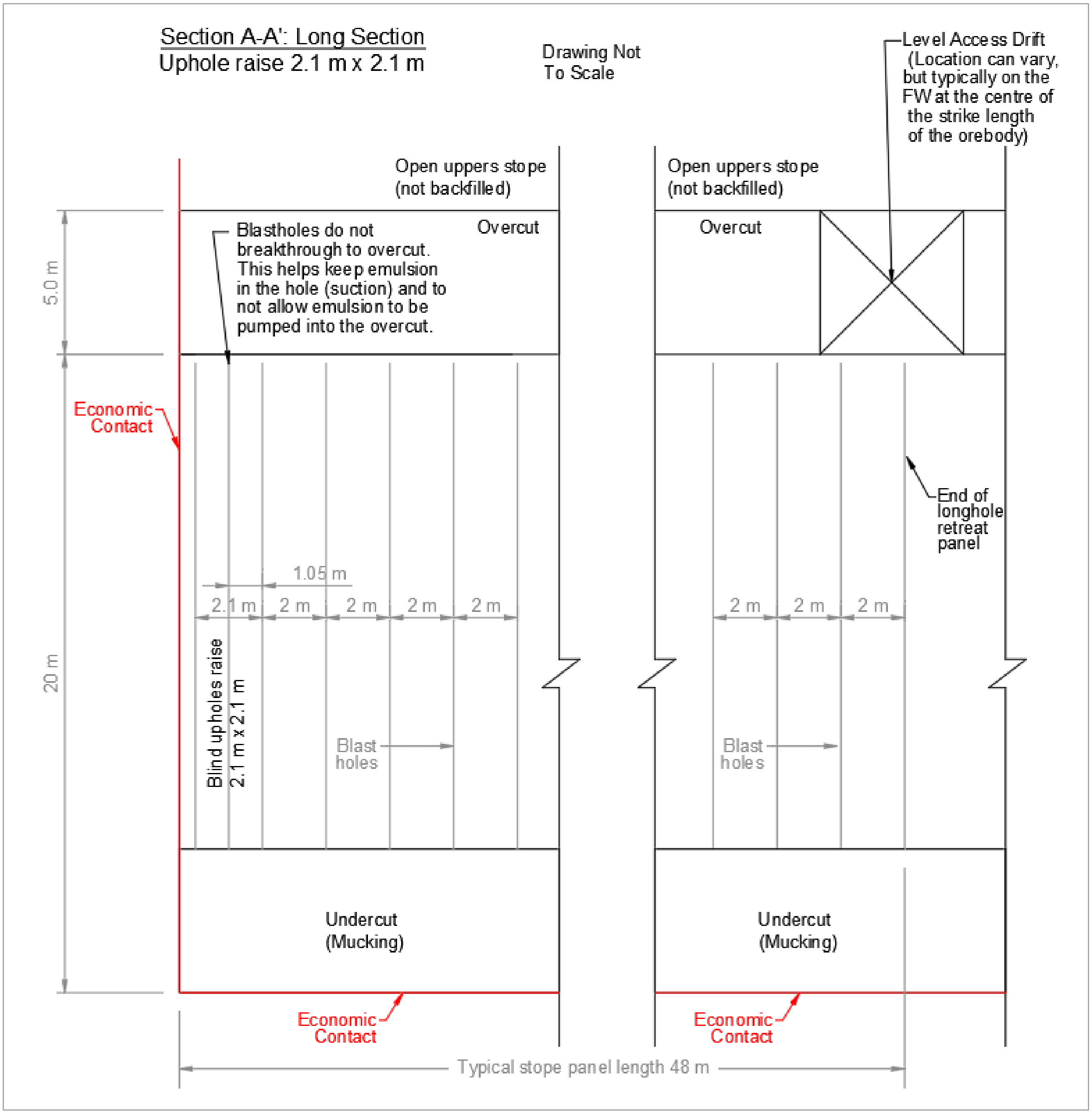
Figure 16.16, Figure 16.17, and Figure 16.18 respectively, illustrate a typical plan view, longitudinal section, and cross section of an uphole stope of 20 m height (floor to floor), fixed width of 7.5 m and length of 48 m. Uphole stopes account for approximately 68% of the planned stope tonnage for the underground mine. The remaining 32% of planned stope tonnage will be from downhole stopes.

Figure 16.16 Upholes plan view



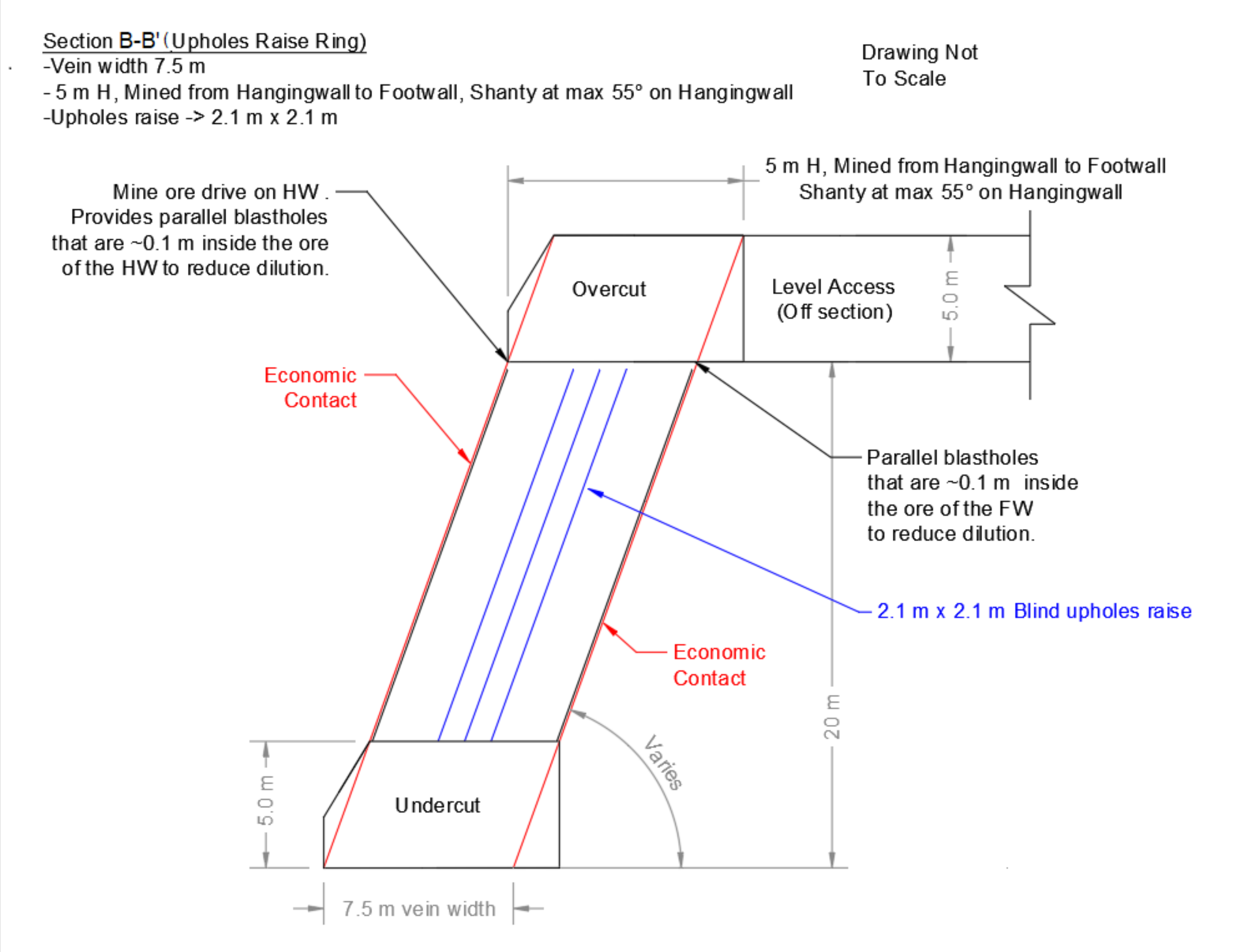
Source: AMC 2019.

Figure 16.17 Upholes long section



Source: AMC 2019.

Figure 16.18 Upholes cross section



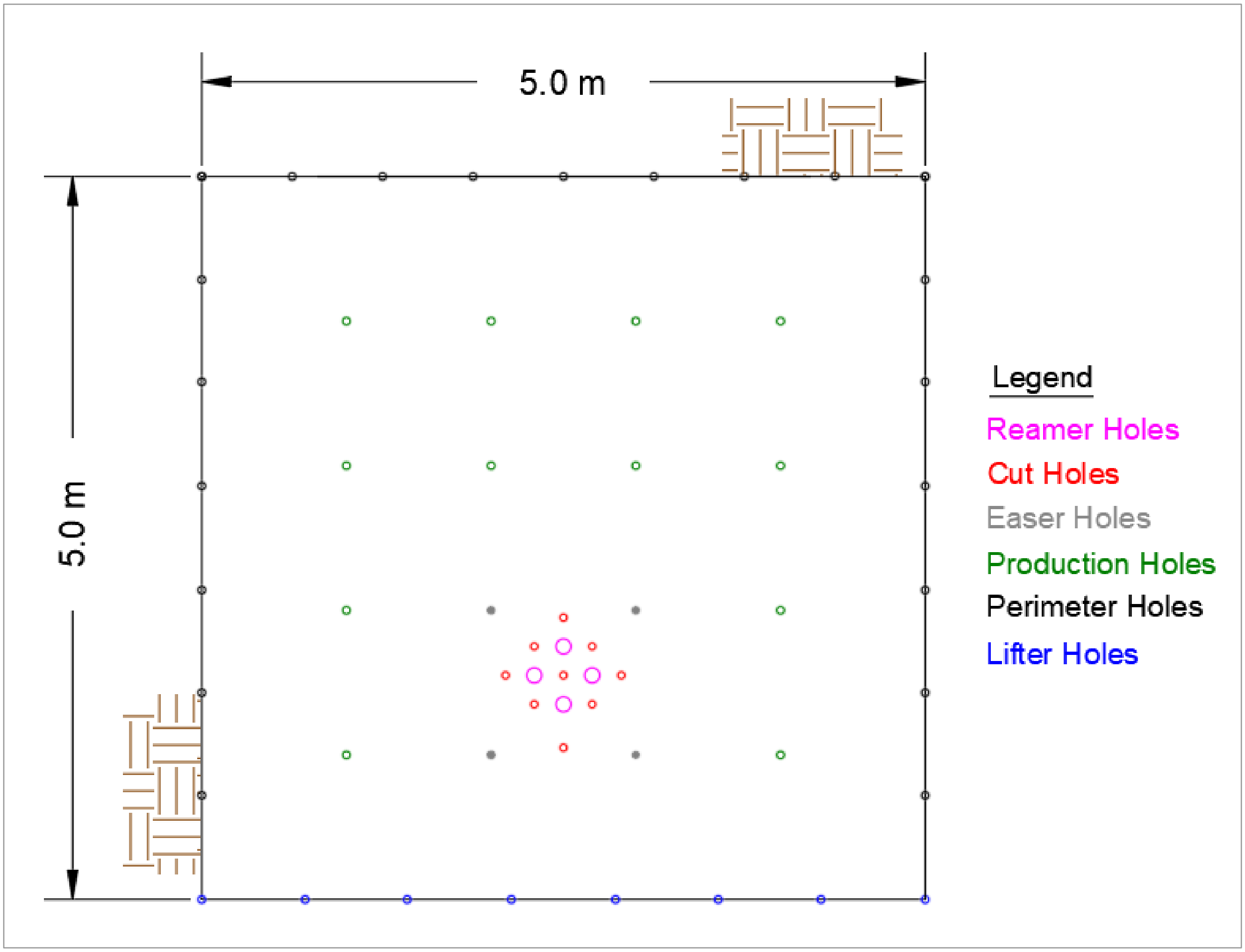
Source: AMC 2019.

The overall powder factor for the uphole drilling stope design shown in Figure 16.16 is estimated at 0.40 kg/t. Depending on the stope shape and drill pattern, the design powder factor ranges from 0.35 to 0.50 kg/t for all types of longhole open stopes (uphole and downhole).

16.2.5.4 Lateral development blast design

Figure 16.19 shows the drill layout for a typical lateral development round measuring 5.0 m x 5.0 m x 3.9 m deep.

Figure 16.19 Typical lateral development drill pattern



Source: AMC 2019.

The overall powder factor for the lateral development design shown is 1.10 kg/t. The powder factor range for all lateral development is from 0.95 to 1.20 kg/t.

16.2.5.5 Explosives consumption

At steady-state ore production of around 3,100 tpd (2025 – 2028) and peak development advance of 30.0 m/d in year 2025, the monthly mine explosives consumption will be approximately 95 t of bulk emulsion, 13,700 detonators, and 13,700 boosters. Table 16.19 shows typical powder factors for LLHOS (uphole and downhole) and lateral development.

Table 16.19 Powder factors for longhole stoping and lateral development

Description	Ore tonnes (t)	Explosives (kg)	Powder factor (kg/t)
Longitudinal longhole stope	17,100	6,797	0.40
Lateral development (per round)	272	299	1.10

16.2.6 Underground infrastructure

16.2.6.1 Existing infrastructure

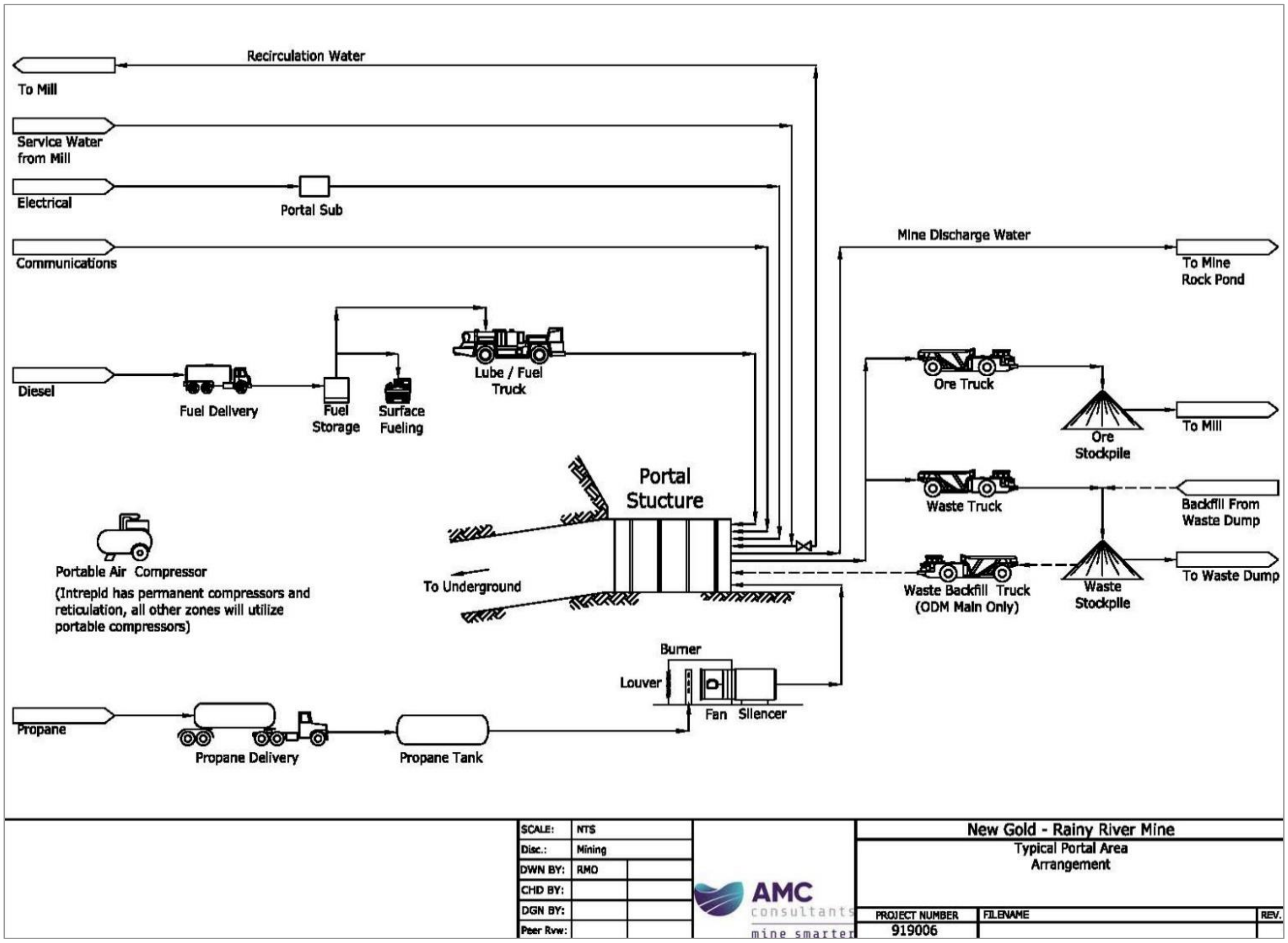
Several pieces of critical infrastructure are already installed for the underground mining operations; these include the following:

- A power line to the Intrepid Zone portal area.
- A DN150 insulated and heat-traced water line for process water.
- 2 x DN150 insulated and heat-traced discharge lines for mine water discharge.
- An office complex and workshop structure with air compressors.
- Ventilation fans and mine air heaters for the Intrepid Zone.
- A leaky feeder communication system for the Intrepid Zone.

16.2.6.2 Additional infrastructure

Additional infrastructure items required to facilitate mining of the ODM Main Zone, ODM East Zone, 17 East Upper Zone, and 433 Zone are outlined in the following sections. Due to the short mine life and isolation of each individual zone, emphasis has been placed on providing short-term, cost-effective solutions. Figure 16.20 shows a typical portal area schematic with the required infrastructure that has been costed.

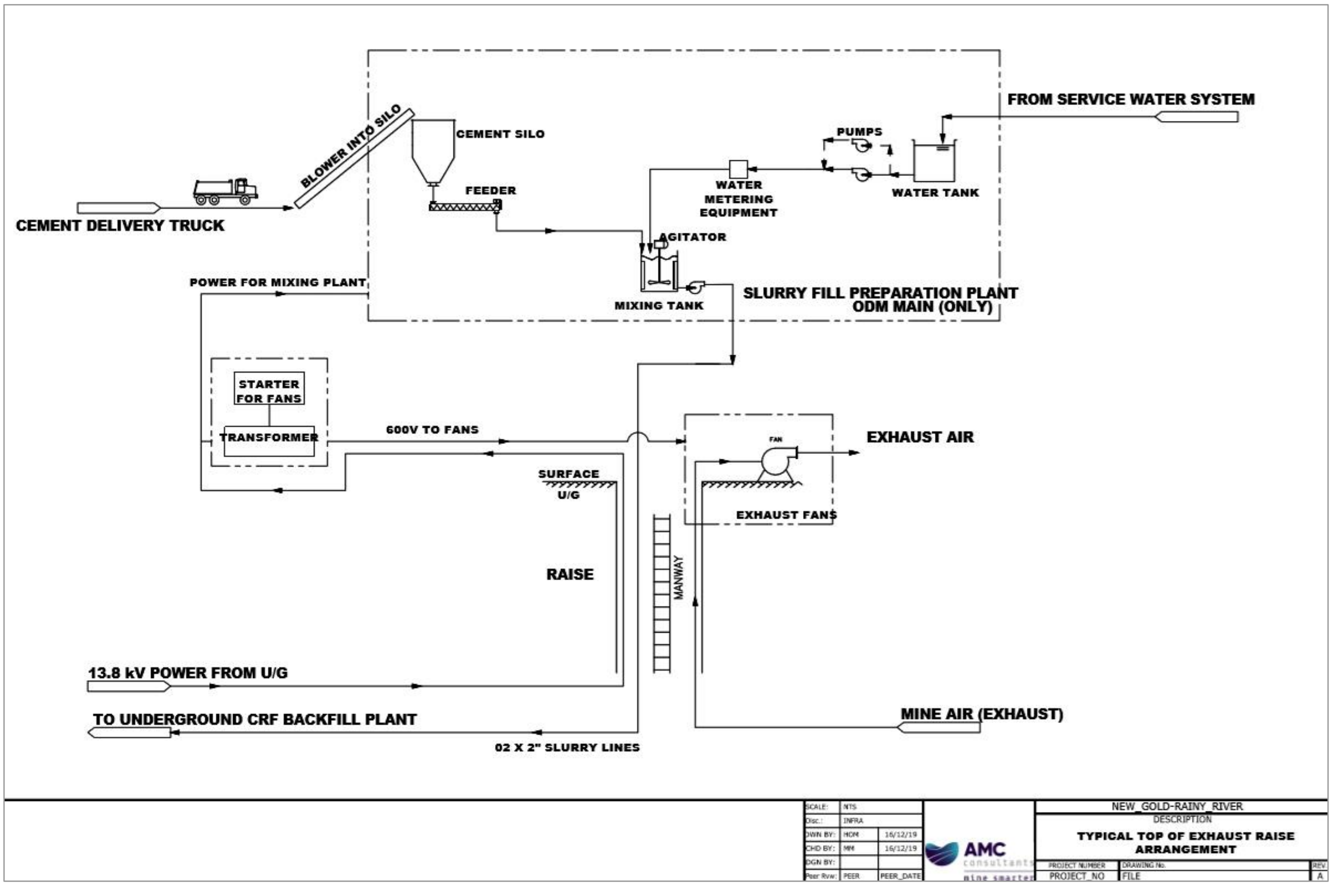
Figure 16.20 Schematic of typical portal area facilities



Source: AMC 2019.

Figure 16.21 shows a typical exhaust raise area schematic with the required infrastructure that has been costed. A cement slurry mixing plant is also shown which will apply to the ODM Main Zone only.

Figure 16.21 Schematic of typical top of exhaust raise area



Source: AMC 2019.

16.2.6.3 Ventilation

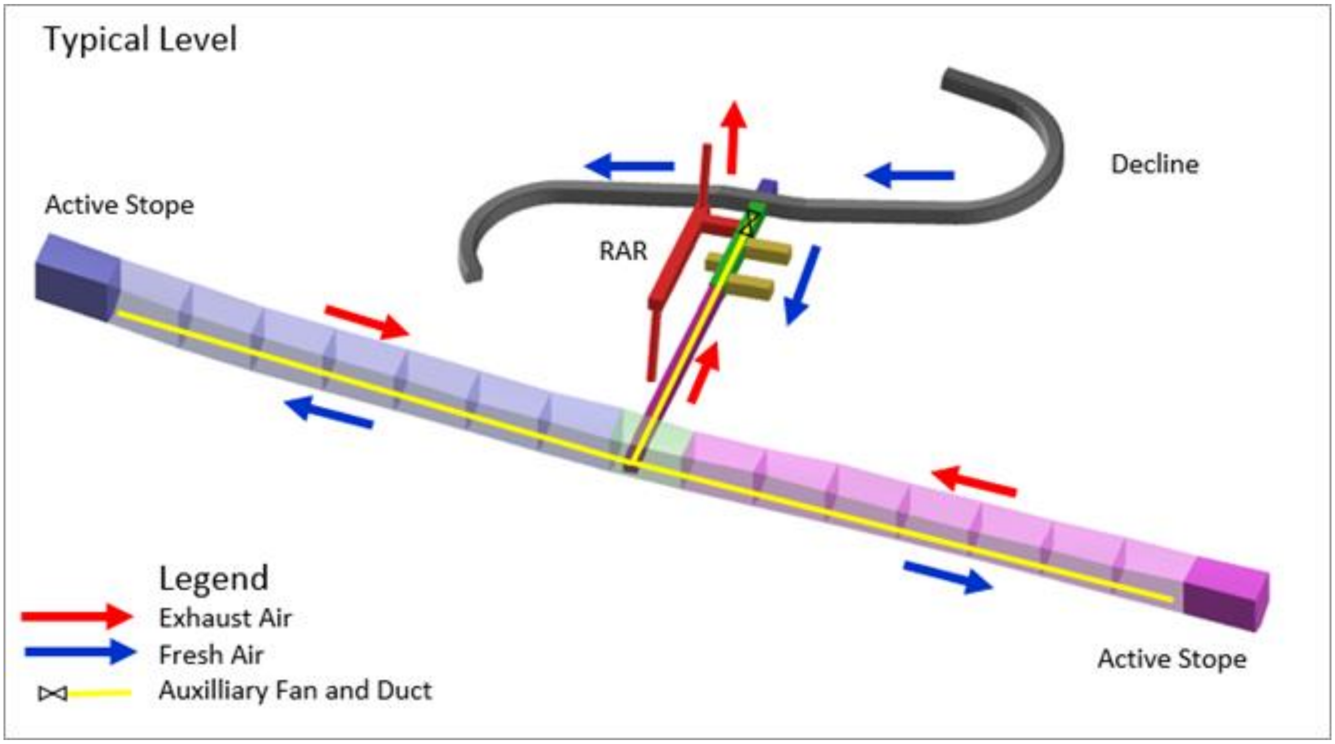
The function of the ventilation system is to dilute / remove airborne dust, diesel emissions and explosive gases, and to maintain temperatures at levels necessary to ensure safe production throughout the life of the mine. The ventilation system has been designed to meet the requirement of the Ontario Occupational Health and Safety Act, R.R.O. 1990, Regulation 854 – Mines and Mining Plants.

The design is based on an exhausting system configuration with the main surface fans located at each of the exhaust raises. Direct-fired mine air heating systems are located at each portal. Fans are also located at these intakes to ensure balance of airflow across the propane burners.

Fresh air enters each zone through the portal with fresh air being distributed through the ramps. Return air is exhausted from the mine through internal raises adjacent to each OZ.

Level distribution is designed such that fresh air will be sourced from ramp accesses via auxiliary fan and duct into each operating level. Contaminated air from development and production level activities returns to an internal return air raise (RAR). See Figure 16.22 for a typical level ventilation arrangement.

Figure 16.22 Typical level ventilation



Source: AMC 2019.

Air volume requirements are calculated to ensure safe production. The amount of air required is determined by the number of active workplaces; particularly to ensure dilution and removal of dust and noxious gases as well as diesel particulate matter.

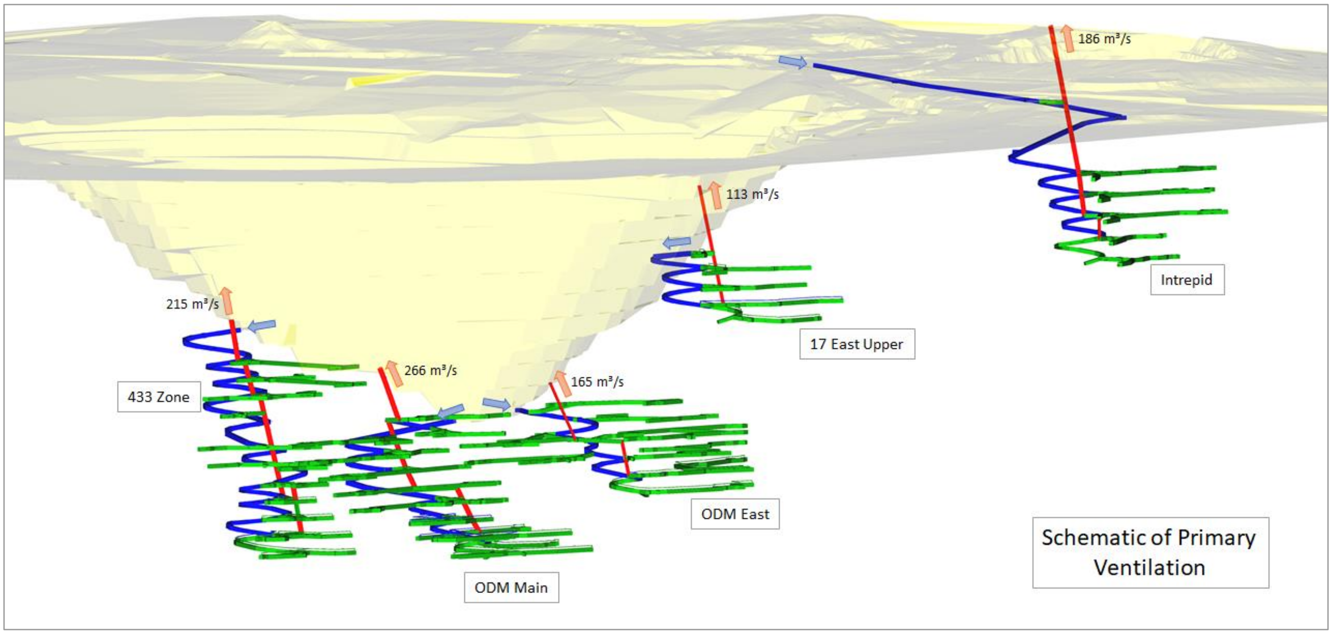
Based on the scheduled production and development activities, airflow allocations are summarized below in Table 16.20.

Table 16.20 Airflow allocation in m³/s

Zone	2020	2021	2022	2023	2024	2025	2026	2027	2028
ODM Main	-	-	-	-	-	165	249	219	151
ODM East	-	-	-	-	-	165	183	169	100
17 East Upper	-	-	113	104	-	-	-	-	-
433 Zone	-	-	-	-	126	199	116	-	-
Intrepid	34	-	-	186	177	100	-	-	-

Figure 16.23 shows an isometric view of the Rainy River ventilation system with maximum airflows for each zone indicated.

Figure 16.23 Schematic of primary ventilation



Source: AMC 2019.

16.2.6.4 Emergency preparedness

In development of the ventilation strategy for Rainy River, consideration has been given to the potential for mine emergencies. The following criteria have been established:

- In general, ramps will be in fresh air once developed.
- On almost all levels, escape can be either to a ramp or to the escape ladderway in the internal raises.
- In each ramp, escape may either be up the ramp or down the ramp to a safe area.
- Portable refuge chambers are required for flexibility of location at the most appropriate points in the mine.
- Whilst the primary means of communication will be by radio, a stench system will be in place for introduction of ethyl mercaptan into each portal in the event of fire.

There are a variety of incidents that will trigger the emergency response plan and / or evacuation plan. Such events may be fire, rock fall, injured personnel or major ventilation equipment breakdown.

If the primary egress (main ramps and portals) is unavailable, a secondary means of egress from the mine must be available to allow evacuation of all underground persons when it is safe to do so.

For each zone, a ladderway is installed in each of the raises located next to main ramps, with access through to the surface. The raises are sized to afford easy passageway. The surface exhaust construction will be such that personnel can exit the raise with no possibility of encountering the fan.

Personnel working underground are provided refuge by means of 12-person mobile self-sufficient rescue chambers. These will be independent of a compressed air supply, with appropriate provisions for safe refuge. They will be in areas where secondary egress is not, or has not yet been, established, and will be sited relative to the active working areas to be within the average walking pace duration of a personal self-rescuer device.

16.2.6.5 Backfill

Backfill will only be required for the ODM Main Zone. This zone will use primary and secondary sequencing of longitudinal longhole stopes. Depending on sequencing, the backfill will either be URF or CRF using underground development waste.

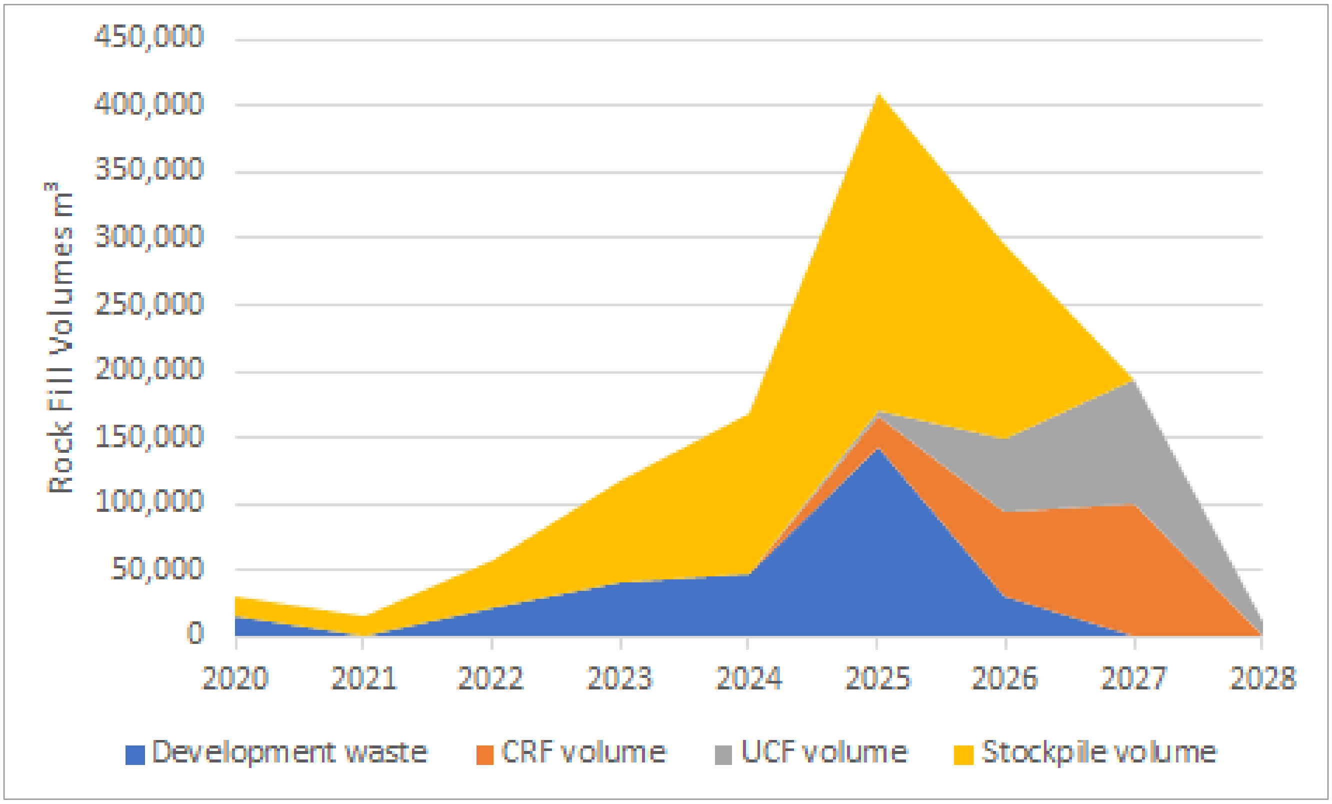
Primary sequenced stopes are projected to use CRF. This will provide optimum ground support and stability of the fill, which will facilitate mining of the secondary stopes. The secondary stopes will use URF for bulk filling.

Waste rock supply and demand

Development waste will be produced from all the underground zones and will be hauled to the surface stockpile, starting in 2020, for future backfill use. In 2025, when the first backfill is required for the ODM Main Zone, URF, and CRF will be produced from a combination of waste rock obtained directly from the zone but mostly from stockpiled surface waste.

Figure 16.24 charts the supply and demand of waste rock for underground backfill. The growth and depletion of the surface waste rock pile is also shown.

Figure 16.24 Supply and demand of development waste for URF & CRF



Source: AMC 2019.

Table 16.21 details the annual supply and demand of waste rock. The total waste rock requirement for the ODM Main Zone is 353,000 m^3 and the total supply of underground development waste is 295,000 m^3, resulting in a shortfall of 58,000 m^3. AMC concludes that there is sufficient development waste rock to supply all the CRF requirements, with the shortfall for URF purposes being easily made up from other waste rock stockpiled on surface.

Table 16.21 Rockfill schedule

ODM Main	2020	2021	2022	2023	2024	2025	2026	2027	2028	Totals
Development waste m^3	15,000	-	21,081	40,611	46,229	143,335	29,210	-	-	295,466
CRF volume m^3	-	-	-	-	-	23,292	64,855	99,940	-	188,087
UCF volume m^3	-	-	-	-	-	4,697	55,824	93,260	11,547	165,328
Stockpile volume m^3	15,000	15,000	36,081	76,692	122,921	238,267	146,797	(46,403)	(57,949)	(57,949)

Rockfill types and components

URF is made up of development rock that is dumped into an empty secondary stope with no additional material added to it. When the stope extraction and filling sequencing allows, development rock can be dumped straight into the stope being filled. URF can also be hauled from a mined-out stope used as waste rock storage or directly from surface storage.

CRF is a mixture of good quality, suitably sized waste rock, cement, and water. The strength of CRF largely depends on the waste rock quality and sizing, cement type and content, water cement ratio, and void ratio.

Waste rock

Good quality waste rock (with intact rock UCS greater than 50 MPa) can be sourced either from underground waste development or from surface waste stockpiles. The rock needs to be competent and not friable. Oxidized or weathered soft rock containing excessive clay minerals is not suitable for making CRF. Figure 16.25 shows an example of good quality development waste rock.

Figure 16.25 Good quality development waste rock

Source: AMC 2019.

The rock needs to be fresh and reasonably well graded. Generally, underground waste development blasting in competent ground conditions generates suitable particle sizing for making CRF. A well-graded waste rock with top size of 300 mm and enough fines less than 10 mm (30% by total weight) is required to obtain the optimum void ratio and high placed density. Oversize material greater than 300 mm should be limited to less than 5%.

It must be emphasized that CRF is not a fully engineered fill product, since the specifications of the main constituent i.e. development waste, are variable and cannot be closely controlled. The method of placement is also variable and may lead to significant undesirable segregation of the constituents. Consequently, a minimum FOS of 2.0 is recommended for CRF strength design.

For CRF applications, standard grade General Usage (GU) cement is usually acceptable.

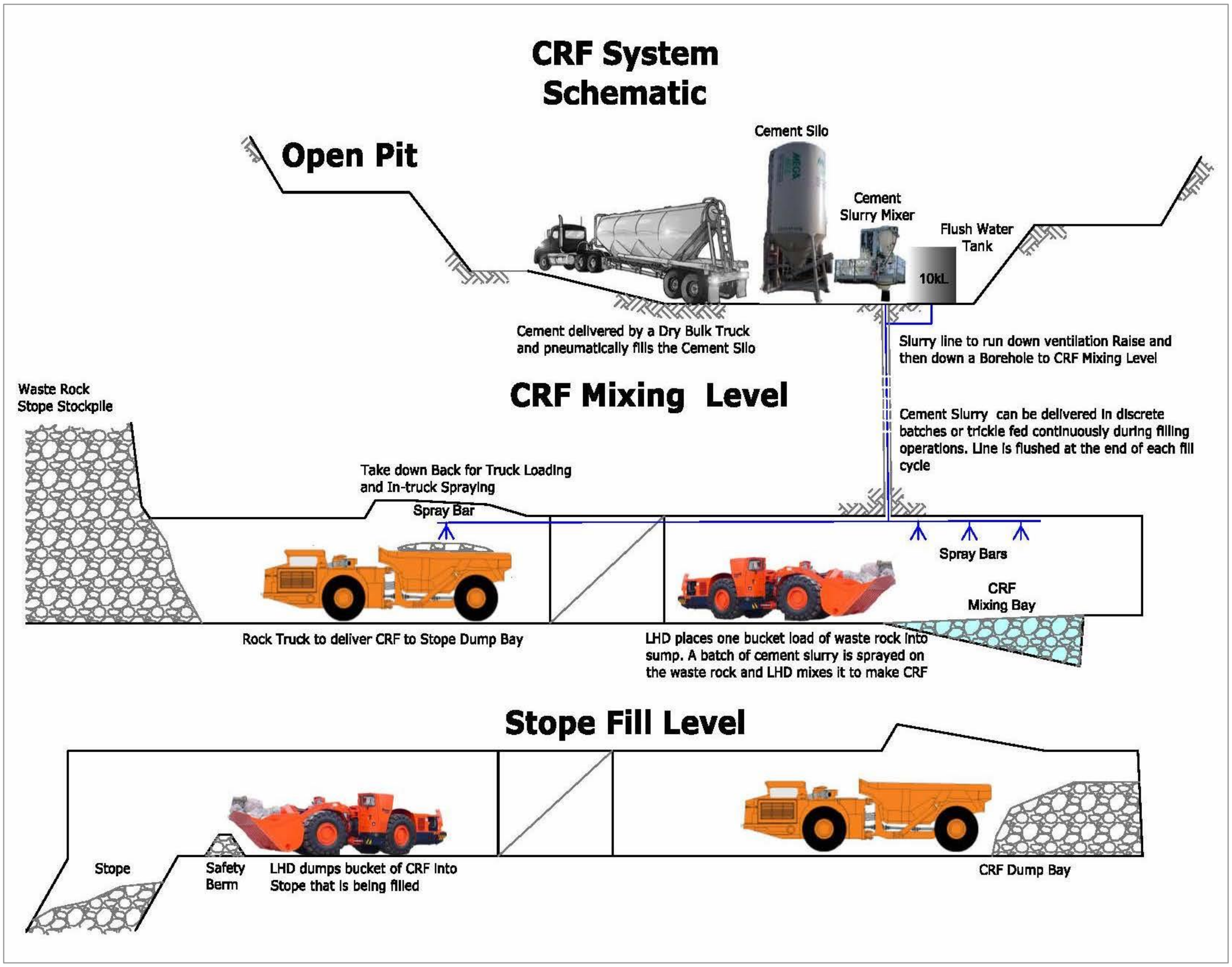
Good quality mix water is important for mine backfill. In particular, a pH of <6 or a high salt content and total dissolved solids greater than 15,000 ppm can be detrimental to the CRF strength.

Cemented rockfill system – general description

AMC recommends the following CRF system for Rainy River: URF to be placed directly into stopes as required; CRF to be prepared and placed using the plant / system described below.

Development waste rock will be stored underground in a temporary storage stope (silo) located at -30 RL in the ODM Main Zone. On demand, waste rock will be drawn from the stope drawpoint with a load-haul-dump (LHD) and either placed in a mixing bay sump to be dosed with cement slurry for mixing, or directly into a truck, with the rock then being sprayed with cement slurry. The CRF will then be transported to the nearby stope tip-point. Figure 16.26 shows a schematic of a representative CRF system.

Figure 16.26 Schematic of proposed CRF system



Source: AMC 2019.

Surface cement slurry operations

The cement batching plant will be located on surface, inside the pit near the ODM Main Zone portals and beside the egress-ventilation raise. For the cement slurry system, the key components are the silo, transfer augers, and weigh hopper.

Normal operation will be for the underground CRF station operator to call for a delivery of cement slurry. The batch will be prepared and dispatched by gravity down a 50 mm diameter (one operating, one standby) slurry line to the mixing station. Communications between the underground mixing station and the surface plant will primarily use the radio system, although direct telephone links may be practical. The surface mixing plant will operate continuously during cemented filling campaigns and will require lighting for night-time operations. A typical batching plant is shown below in Figure 16.27.

Figure 16.27 Example concrete batching plant

Source: AMC 2019.

Underground CRF mixing station

The underground CRF mixing station on -30 Level (L) will have two operating modes:

- Most frequently, trucks with development waste will be sprayed with a batch of cement slurry before proceeding to the filling area.
- In the second operating mode, a loader on -30 L will draw waste rock from the storage stope and place it in the sump area to be sprayed with cement slurry. The loader will mix the slurry and rock to form the CRF and load the truck to proceed to the filling area.

In the mixing station, a pipeline should be routed along the centerline of the back of the drive, with three cross pipelines over the area of the sump. These pipes are simple extension lines with narrow spray holes bored at frequent intervals to enable an even distribution of cement slurry spraying over the sump. A second pipeline and spray bars will be installed to enable direct spraying of cement slurry onto trucks.

The CRF mixing bay will consist of a short drive mined down grade to form a sump area in the floor. The back should be maintained at a constant elevation to enable the loader to place, load, and mix waste rock and slurry.

When operating the mixing bay, the slurry will be discharged over the sump area throughout mucking and mixing operations. The delivery rate will be adjusted to match the mixing rate of rock and to achieve a nominal 4% (or other specification) by dry weight of cement addition to the CRF. This will be determined by trials during commissioning to match surface batching operations with waste rock mixing.

The loader operator (or other person) will manage the handling of rock from stockpiles, mixing with the loader bucket in the sump and delivery of cement slurry batches. Each mixed CRF load will be loaded into the truck and then trammed to the stope tip point.

Direct spraying of cement slurry onto trucks will be operated in a similar manner with discrete batches of cement slurry prepared on surface and delivered directly into the trucks.

Underground URF and CRF operations

If rockfill operations are not in progress, development waste rock is loaded into conventional tipper trucks for haulage up the decline. If rockfill operations are in progress, the loaded trucks may be diverted to dump into the slurry mixing bay, to receive slurry sprayed directly onto the rock in the truck, or to the waste rock storage stope.

Where waste rock is dumped from above to a storage area, simple waste rock bunds will be required at the drawpoint level to eliminate the risk of fly rock. These can be constructed by pushing waste rock to close off the brow of drawpoints and will not be structural or load bearing.

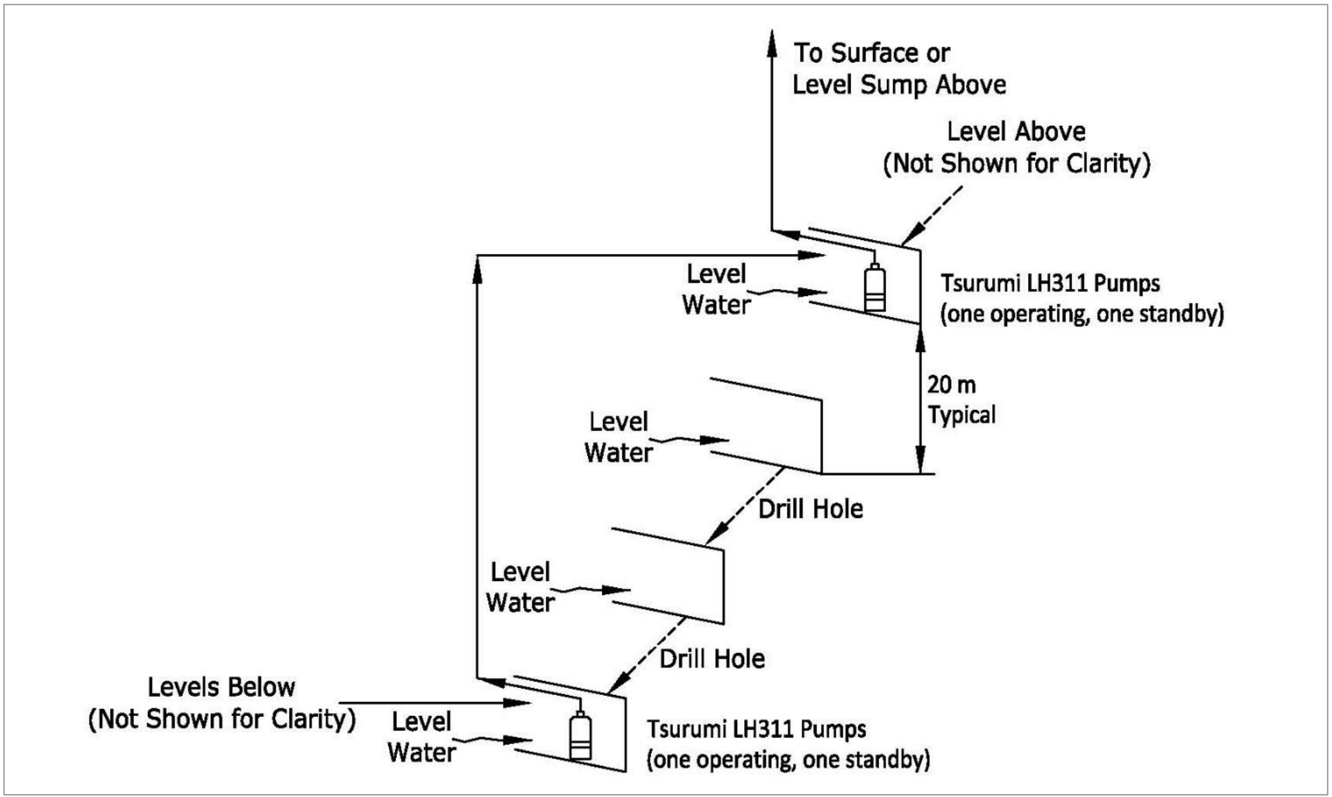
Tipping points will be protected with stop blocks. When the URF or CRF rill reaches the elevation of the tipping point, remote loader operations will be used to enable placement of fill into the whole volume of the stope by dumping and pushing over the stope edge.

16.2.6.6 Mine dewatering & water supply

Hydraulic conductivity analyses have arrived at an estimated mine-wide groundwater inflow of 18.7 litres per second (L/s). With the isolation of the individual zones, estimated inflow to each zone is estimated at around 3.74 L/s. The dewatering system for each zone must also accommodate service water. Total estimated capacity for the pumps is 4.2 L/s, with the pump design capacity at 5.6 L/s assuming a 75% duty cycle.

A permanent pumping station is not warranted for a mine with two to three years of life for each zone. A cascading discharge system, such as is commonly used during ramp development, is proposed for these zones. Mining level spacing is 20 m, with sumps being excavated at the entrance to each level to capture water from the level as well as ramp water. Drain holes will be drilled between levels such that water from three levels is captured in a single sump. Water from this sump will be pumped by two Tsurumi LH311 pumps (one operating, one standby) at each stage to the next stage (60 m vertical). An additional pump will be required to discharge from the face during development. Figure 16.28 Shows a schematic of the proposed dewatering system.

Figure 16.28 Generalized schematic of dewatering system



Source: AMC 2019.

A total of 25 pumps will be required for the entire mine, with some pumps being reused as one zone is mined out and others start. Table 16.22 outlines the number of pumps required for each zone.

Table 16.22 Pumps required by zone

Zone	Face pump	Pumps
ODM Main	1	6
ODM East	1	4
17 East Upper	1	4
433 Zone	1	10
Intrepid	1	8

DN100 DR11 high-density polyethylene (HDPE) pipe will be used for discharge water. The discharge line will be installed in the ramp. A cost comparison was undertaken comparing the benefits of moving the discharge line into the ventilation raise versus leaving it in place. Moving the discharge line to the ventilation raises once each raise is established is not cost effective as the operating life of the system is too short to allow the reduced operating costs to recover the capital cost of moving the line.

Once the water has reached surface, the existing in-pit pumps will deliver the water to the mine rock pond.

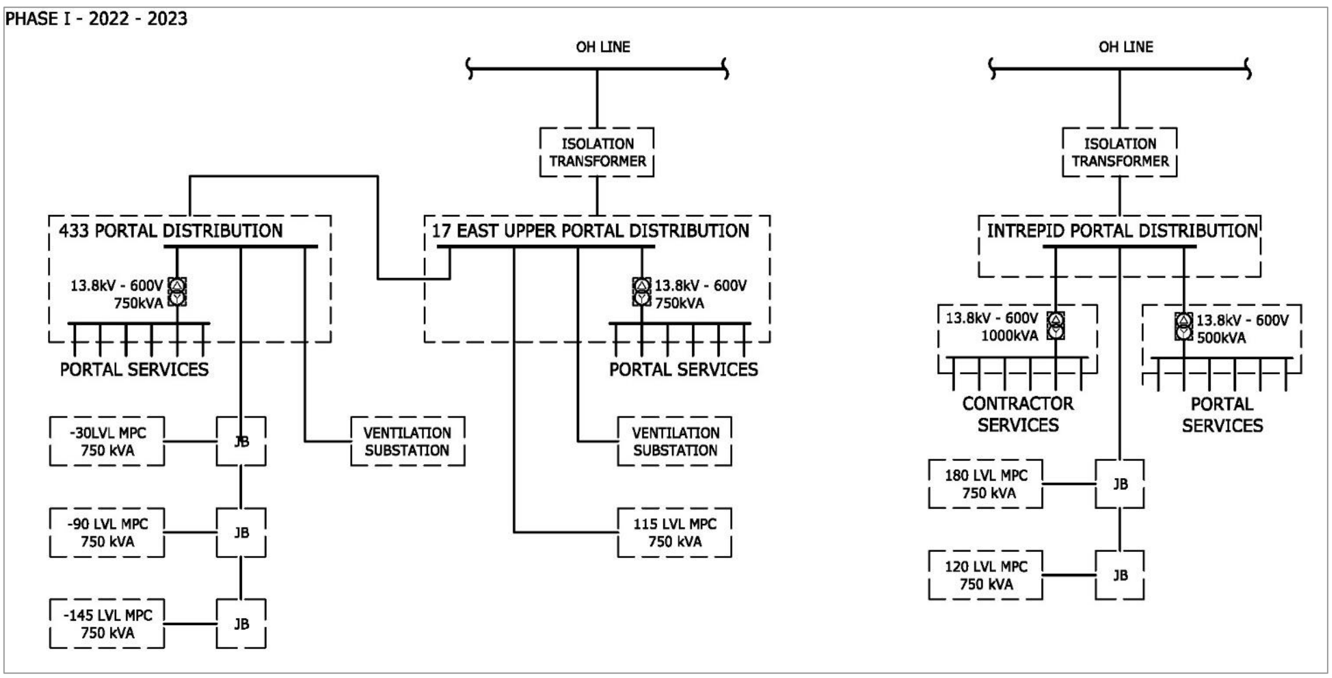
Service water is already available at the Intrepid Zone portal. To supply the remaining four portals with service water, a DN100 insulated and heat traced HDPE line will be required. This line will run into the pit along the existing haul road and will be protected by covering it with earth.

Each zone will be equipped with DN100 DR11 HDPE pipe in the ramp to supply water through the mine. Pressure reducing valves (PRVs) will be required at 60 m intervals vertically to maintain water pressure below 120 pounds per square inch (psi).

16.2.6.7 Electrical distribution system

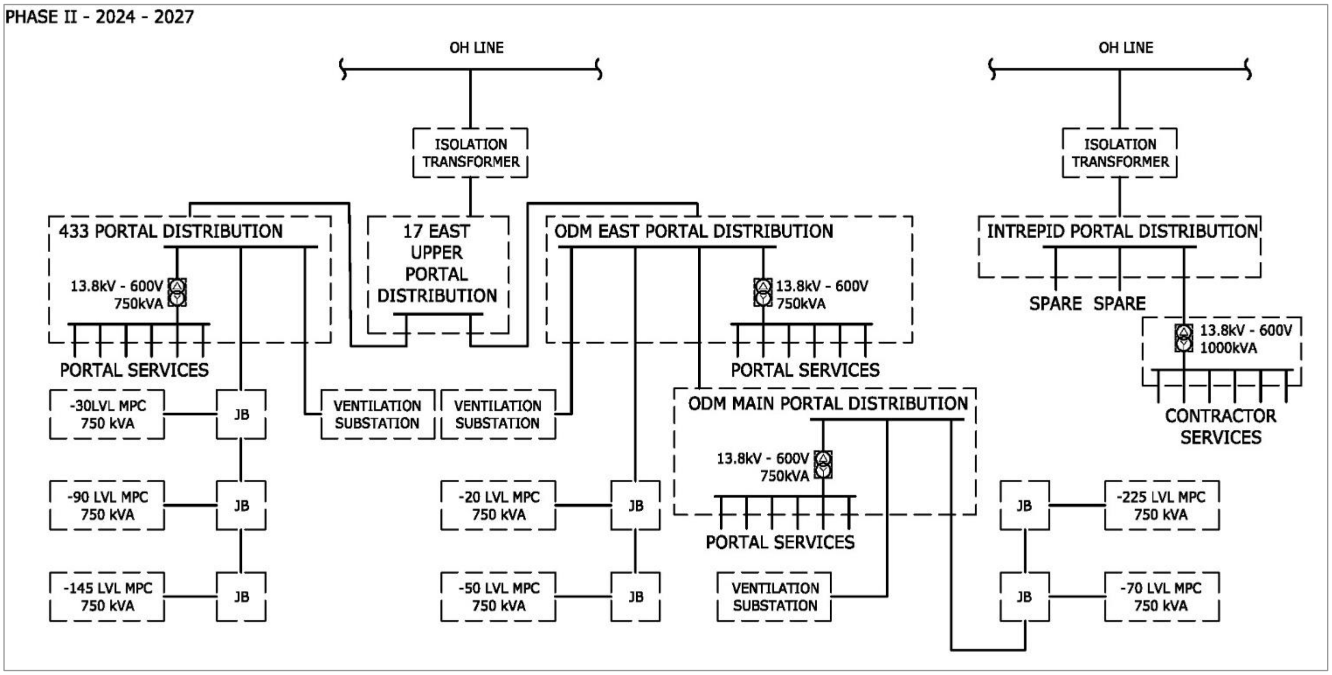
Electricity for the underground mine is to be provided from an existing 13.8 kilovolts (kV) overhead line running adjacent to the existing open pit. A grounding isolation 13.8-13.8 kV substation will be installed outside the pit rim and from there cabling will be run down the pit wall to the 17 East Upper Zone portal area, from which cabling will be daisy-chained to the three remaining portals (Phase I). Following the completion of the 17 East Upper Zone mine operation, the movable portal and ventilation substations will be relocated to the ODM East Zone portal and replaced with a 3-way junction box at the 17 East Upper Zone portal (Phase II). Power for vent raise fans will be delivered via 13.8 kV cabling run up the raise to a substation adjacent to the fan installation. The Intrepid Zone will continue to be supplied by the existing 13.8 kV feed. Block diagrams of the electrical distribution for the two phases are shown in Figure 16.29 and Figure 16.30.

Figure 16.29 Phase I electrical distribution block diagram



Source: AMC 2019.

Figure 16.30 Phase II electrical distribution block diagram



Source: AMC 2019.

A breaker will be provided to supply power to each of the five mining zones. Armored cables (3C #1 AWG) will be used to bring power to each portal. To protect the cables through the active pit, they will be covered by a steel half pipe of suitable size and covered with waste rock. The powerline for each portal will feed a 13.8 kV/600 volts (V) 750 kilovolt-ampere (kVA) mine load center (MLC) located at each portal and then daisy chained through each zone to additional underground MLCs. Each MLC will be located such that it can feed three or four active levels.

Each MLC will be equipped with appropriate breakers and plugs to accommodate up to three jumbo boxes, four 75 horsepower (hp) fans and four pumps. They will also be equipped with a 10 kVA 220 V lighting transformer for small loads. A total of 11 MLCs will be required for the life of the mine; however, two 500 kVA MLCs have already been procured, meaning a further nine will be required. The first lower capacity MLC is installed at the Intrepid portal, the second should be installed in another lower demand area. As with pumps, once mining is complete in a zone, the MLCs can be recovered and used elsewhere. Table 16.23 outlines the quantity of MLCs required for each zone.

Table 16.23 Mine load center (MLC) requirement by zone

Zone	MLC
ODM Main	3
ODM East	3
17 East Upper	2
433 Zone	4
Intrepid	3

The underground mine average electrical load is projected to increase to a maximum of approximately 2.8 MW in 2025, with an overall LOM average load of 1.6 MW. The system will be designed to handle the maximum peak loads, which are anticipated to be approximately 4.1 MW.

16.2.6.8 Communications

There are five ramps accessing the orebody - four from within the open pit and one outside the pit for the Intrepid Zone. The Intrepid Zone ramp has already been equipped with a leaky feeder and radio system that is connected to the surface communications network.

It is prudent to install the same system in all the other ramps at the ODM Main Zone, ODM East Zone, 17 East Upper Zone, and 433 Zone. The 17 East Upper Zone portal has been identified as the optimal location for the base-station / headend unit. There will be cables run from the base-station to the other portals based on the mining schedule. If the Intrepid Zone finishes mining prior to commencing mining of the 17 East Upper Zone, the equipment could be transferred.

The system has been configured to provide voice and data communication. There will be 75 handheld radios for all underground employees and 11 radios mounted on the mobile equipment. The system extends into the ramps and all level access drifts.

A Smart Blast system has been costed in the capital expenditure (capex). This system makes use of the leaky feeder.

As indicated above, there is currently a leaky feeder (Varis / Becker) system installed with some antenna cable at the Intrepid Zone portal. It is housed in a communications hut located next to the 1 mega volt amperes (MVA) transformer feeding underground and the portal workshop. There are also 20 handheld radios and 11 base units for equipment on site. Channels are programmed and include a patch channel to the surface radio system for emergencies. When costing the communications system, the existing equipment has been considered.

16.2.6.9 Compressed air

Modern mines typically operate with minimal air requirement; however, as all mining will be performed by a mine contractor and to facilitate the use of hand-held drills as well as pneumatic pumps, compressed air reticulation through all the zones is desired. There are two Ingersoll Rand R110i compressors installed near the Intrepid Zone portal workshop area. Each compressor is capable of 19.5 cubic metres per minute (m^3/min) at 8 Bar (690 cubic feet per minute (CFM) at 115 psi).

Reticulation through the zones will be via DN150 DR11 HDPE anti-static pipe. An insulated and heat-traced surface supply line from the Intrepid Zone portal to the other four zones will be required. The line will be buried in earth for protection while the pit is active. Insulation and heat trace are necessary as compressed air contains water vapor which will condense in the line as the air cools. In the winter this water could freeze, blocking the line if it is not heat-traced.

16.2.6.10 Maintenance facilities

The only maintenance facility will be the workshop structure already in place at the Intrepid Zone portal. This is a 24.7 m x 24.3 m fabric arched roof structure that should suffice for all maintenance activities throughout the life of the underground mine.

16.2.6.11 Explosives magazine

All explosives will be stored in the existing surface magazines located off a main access road in the vicinity of the open pit and brought to the underground as required each day.

16.2.6.12 Fuel and lube

A fueling station will be available near the workshop complex for haul trucks and LHDs that return to surface. Additional fueling stations will be made available utilizing double-walled tanks with self-contained pump units. One 5,000 litre (L) tank will be located at the 17 East Upper Zone portal and a second near the ODM East Zone portal. A fuel / lube truck will be utilized to fuel equipment that remains predominantly underground.

16.2.7 Mine equipment

The underground mobile mining equipment and labour will be provided by a contractor. The work-up of mining equipment and personnel required to execute the underground scope of work was used to determine various underground mine costs, such as: ventilation, fuel, electricity consumption, dry facilities, camp cost, maintenance facility and others.

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 in Table 16.24 is based on the peak mobile equipment numbers and types that are projected to be required to meet the development and production schedule for the LOM plan.

Projected personnel numbers are shown in Section 21.

Table 16.24 Peak underground mobile equipment requirements

Unit	**#**	**Maximum**
Jumbo	units	4
Longhole drill	units	3
Emulsion stope loader	units	4
Haul truck (40 tonne)	units	3
LHD (14 tonne)	units	4
Bolter	units	4
Pickup truck	units	5
Scissor lift	units	3
Boom truck	units	1
Blast pickup truck	units	1
Face charger	units	4
Grader	units	1
Forklift	units	1
Total	**units**	**38**

16.2.8 Mine development schedule

The Intrepid Zone decline will start advancing in Q2-2020 as an orebody investigation project. The total length of decline scheduled for 2020 will be 600 m. Underground development (not related to the Intrepid Zone orebody investigation project) is forecast to commence in Q3-2022, with the first ore from development obtained in the same quarter. The mine development will exclusively be undertaken by contractor crews throughout the LOM.

The mine development schedule is summarized in Table 16.25. Each zone will have a dedicated main decline crew, as well as separate development crews for level development (including stope undercut and / or overcuts).

The mine development schedule is based on a single heading advance of 120 m per month for the main decline. Level crews are projected to drive single headings at 75 m per month and multiple headings at 150 m per month. These are average development rates and are considered reasonably attainable with use of dedicated crews and equipment. As with all development, close monitoring of 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 for each zone be reviewed to include adequate time for drilling and data analysis before stope development and production commences.

Table 16.25 Mine development schedule

Year	Alimak (m)	Decline development (m)	Stope development (m)	Ore crosscut (m)	Waste crosscut (m)
2020		600			
2021					
2022	102	657	274	134	-
2023	231	860	591	646	104
2024	126	976	842	1,143	251
2025	393	3,003	2,311	3,215	968
2026	183	125	1,058	3,600	678
2027					
2028					
Total	**1,036**	**6,221**	**5,075**	**8,738**	**2,001**

Note: Totals may not compute exactly due to rounding.

16.2.9 Production schedule

Underground mine production is expected to commence in Q3-2022 and continue through to the beginning of Q1-2028. Mining begins in the 17 East Upper Zone and carries on with annual ore production from as many as four different zones in any given year over the LOM plan. The mine ore production is expected to increase from 2022 over a four-year period to year 2025, when a production level of approximately 3,100 tpd will be maintained for the remainder of the LOM.

The production build-up is relatively rapid and will be dependent on attaining the development rate advance described previously.

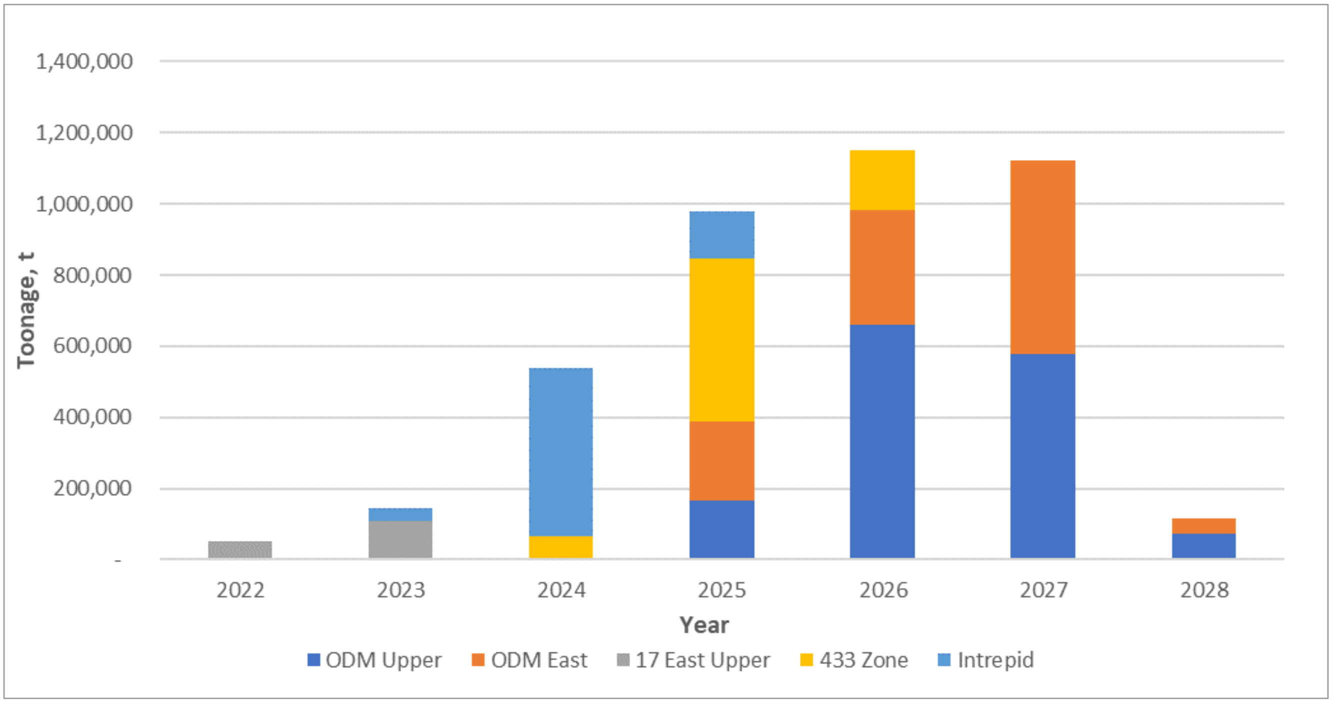
The underground LOM ore production schedule is shown in Table 16.26 and Figure 16.31.

Table 16.26 Underground ore production schedule

Year	Total t (000s)	ODM Main t (000s)	ODM East t (000s)	17E Upper t (000s)	433 Zone t (000s)	Intrepid t (000s)	Au (g/t)	Ag (g/t)	Au (koz)	Ag (koz)
2020	-	-	-	-	-	-	-	-	-	-
2021	-	-	-	-	-	-	-	-	-	-
2022	51	-	-	51	-	-	3.70	14.7	6	24
2023	143	-	-	108	-	35	2.99	18.8	14	86
2024	537	-	-	-	64	473	3.67	26.3	63	453
2025	978	166	222	-	458	133	4.26	6.8	134	215
2026	1,149	660	324	-	166	-	4.31	3.2	159	117
2027	1,121	577	544	-	-	-	4.35	3.5	157	124
2028	116	73	43	-	-	-	4.10	3.5	15	13
Total	**4,096**	**1,475**	**1,133**	**159**	**687**	**641**	**4.17**	**7.85**	**549**	**1,034**

Note: Totals may not compute exactly due to rounding.

Figure 16.31 Underground ore production profile



Source: AMC 2019.

16.3 Mine-to-mill schedule - all sources

Over the LOM, the open pit (including stockpile rehandle) and underground operations will feed to the mill a total of 77.6 Mt of ore grading 1.06 g/t gold and 2.5 g/t silver, totaling 2,636 koz of contained gold and 6,266 koz of contained silver. The mine-to-mill schedule is presented in Table 16.27.

Table 16.27 Mine-to-mill production schedule

Year	Tonnes (t x 000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Open pit to Mill					
2020	4,035	1.28	2.8	166	366
2021	4,685	0.97	2.3	147	347
2022	4,777	1.27	2.3	194	350
2023	4,760	1.21	2.4	186	370
2024	4,784	1.23	2.2	188	332
2025	1,086	1.28	2.1	45	72
2026	-	-	-	-	-
2027	-	-	-	-	-
2028	-	-	-	-	-
Total	**24,128**	**1.19**	**2.4**	**926**	**1,838**
Underground to Mill					
2020	-	-	-	-	-
2021	-	-	-	-	-
2022	51	3.70	14.7	6	24
2023	143	2.99	18.8	14	86
2024	537	3.67	26.3	63	453
2025	978	4.26	6.8	134	215
2026	1,149	4.31	3.2	159	117
2027	1,121	4.35	3.5	157	124
2028	116	4.10	3.5	15	13
Total	**4,096**	**4.17**	**7.9**	**549**	**1,034**
Stockpile to Mill					
2020	5,225	0.70	2.3	118	393
2021	4,920	1.06	2.8	168	441
2022	4,720	1.05	2.2	159	340
2023	4,618	1.11	2.3	165	338
2024	4,242	0.99	1.9	135	261
2025	7,410	0.85	1.8	202	439
2026	8,272	0.37	2.0	98	535
2027	8,318	0.37	2.0	98	538
2028	1,623	0.36	2.1	19	108
Total	**49,348**	**0.73**	**2.1**	**1,162**	**3,393**
Mill Feed					
2020	9,260	0.95	2.5	284	759
2021	9,605	1.02	2.6	315	788
2022	9,549	1.17	2.3	360	714
2023	9,521	1.19	2.6	364	795
2024	9,563	1.26	3.4	387	1,047
2025	9,474	1.25	2.4	380	726
2026	9,421	0.85	2.2	257	652
2027	9,439	0.84	2.2	255	662
2028	1,739	0.61	2.2	34	121
Total	**77,572**	**1.06**	**2.5**	**2,636**	**6,266**

Note: Totals may not compute exactly due to rounding.

17 Recovery methods

17.1 Process description

The Rainy River process plant commenced ore processing in September 2017 and commercial production in mid-October 2017. The process plant experienced initial operating and maintenance issues, but in 2019, Rainy River was consistently able to achieve daily plant throughputs in the 22,000 tpd - 24,000 tpd range. Throughput is programed to achieve approximately 25,300 tpd in 2020 with the LOM throughput averaging approximately 25,800 tpd.

The increased throughput has coarsened the grinding circuit P_{80} from the original design criteria P_{80} of 75 µm to a P_{80} of 90 µm to 110 µm. This has subsequently reduced gold recovery. Rainy River has determined that an increase in throughput at the expense of gold recovery is the most economically viable option.

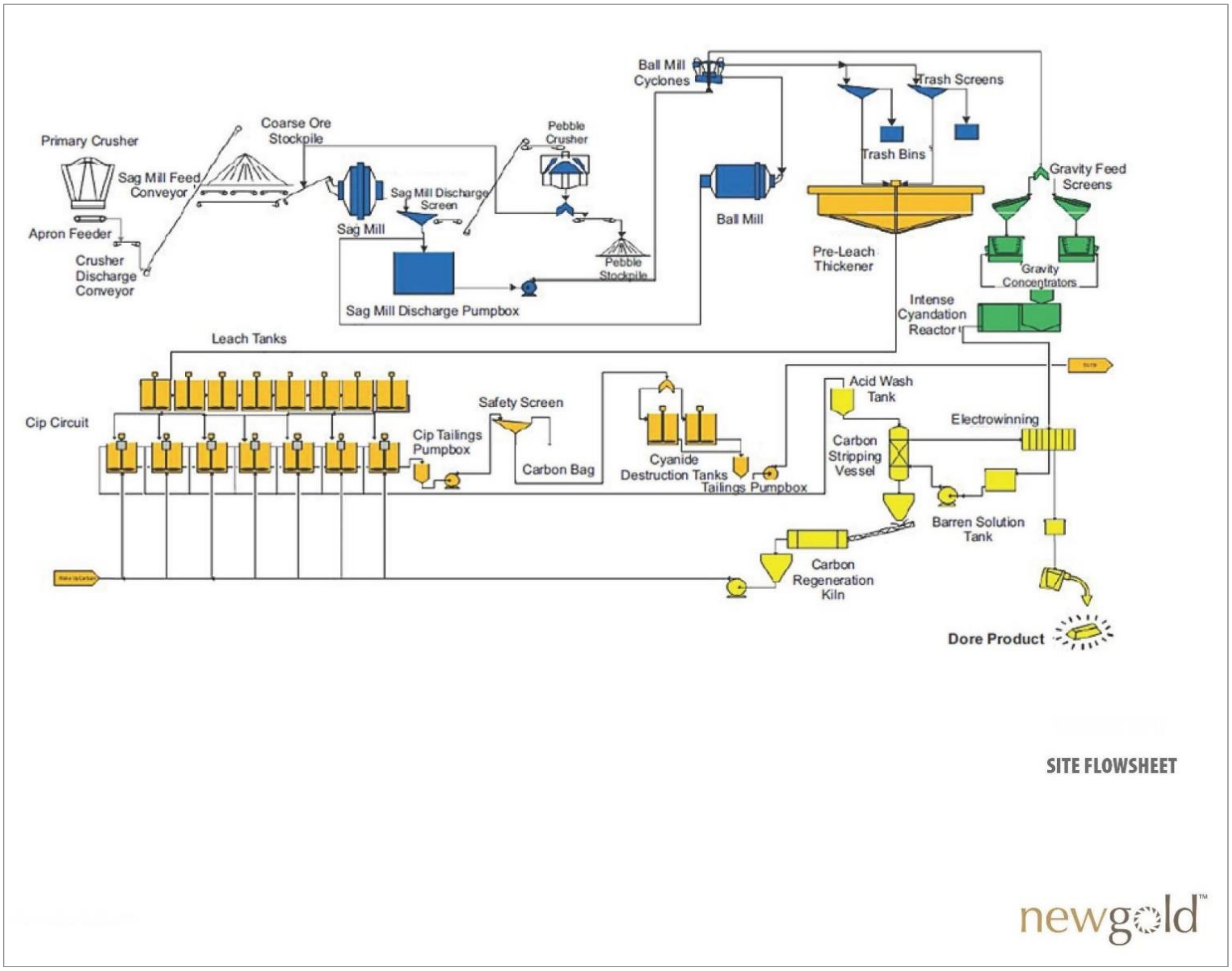
The future plant feed will include some underground ore, including upper zones of the Intrepid Zone ore and some material under / beside the projected open pit mine. The underground will be mined over a period from 2022 through Q1-2028. From Q2-2025 through Q1-2028, low grade stockpiles and underground ore are to be economically processed together.

Figure 17.1 illustrates the simplified flowsheet of the Rainy River process plant.

The process flowsheet consists of the following unit processes:

- Gyratory crusher.
- Coarse ore stockpile, discharged through draw pockets by apron feeders.
- SAG mill feed conveyor.
- SAG mill.
- Pebble crusher.
- Ball mill.
- Gravity concentration of cyclone feed slurry.
- Intensive cyanide leaching of the gravity concentrate using an Acacia reactor.
- Pre-leach thickener.
- Cyanide leaching.
- CIP circuit.
- Cyanide destruction using the sulphur dioxide-air process.
- Carbon stripping using the Zadra process.
- Electrowinning of the eluent and gravity concentrate leach solution.
- Casting of gold and silver doré bars (doré) in an induction furnace.

Figure 17.1 Site flowsheet



Source: New Gold 2019.

17.1.1 Ore delivery from the mine

Ore is delivered from the Rainy River open pit mine using 220 t class haul trucks. Primarily, LGO is trucked to stockpiles for rehandling and future production.

17.1.2 Crushing

The 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 crusher maintenance.

The crusher is designed to process 1,346 tph of ore with a F_{100} feed size of 1,050 mm, a F_{80} of 550 mm and an operating availability of 65%. The crusher operates with an open side setting of 120 mm to produce a P_{80} product size of about 120 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. The crusher discharge conveyor 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 crusher production rate and total ore processed. In addition, CV10 has a metal detector that shuts down the conveyor belt automatically, permitting the operators to extract the metal detected.

17.1.3 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. 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.

17.1.4 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 surveyed mill feed is a F_{80} of 46 mm and the discharge transfer P_{80} size is estimated to be 2.8 mm. The mill is currently operating at 58% of critical speed to achieve a production rate of 1,050 – 1,350 tph. The design operating power at the pinions is 12,580 kW, which is approximately 84% of the installed power. The mill currently has a grate discharge of 70 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 SAG mill currently operates with a 10% - 14% solids (v/v) ball charge made up with 130 mm balls and a total charge volume of 25% (v/v). The maximum design ball charge is 16% (v/v) with a maximum design mill fill volume of 30% (v/v).

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, pebble crusher (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 a design operating power draw of 235 kW. The crusher reduces the ore to an approximate 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 pebble crusher circuit assists in achieving the planned throughputs when the ore becomes harder. The pebble crusher circuit has been commissioned recently. The pebble crusher will not be operated unless the ore is sufficiently hard.

The SAG mill is operated with a slurry density of approximately 70% solids (w/w) and discharges into the cyclone feed pump box, where it is diluted and pumped to a cluster of 22 by 508 mm hydrocyclones for classification. The cyclone distribution header has 25 ports. A total of 22 ports are fitted with hydrocyclones – 3 ports are piped to the gravity concentration circuit feed distributor. The cyclone underflow feeds the ball mill, while the cyclone overflow reports to the trash screens.

17.1.5 Gravity concentration

Three 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 to the gravity screens. The underflow of the screens is directed to two 48 inch Knelson centrifugal concentrators for gravity gold recovery. The flow rate to each concentrator is approximately 300 tph, for a system 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, whilst the screen oversize flows to the gravity circuit launder and gravity flows to the cyclone feed pump box. The capacity of each centrifugal concentrator is 400 tph. The operating slurry feed density is 48% solids (w/w). Tailings from the Knelson concentrators combine with the screen oversize in the gravity circuit launder and flow by gravity to the cyclone feed pump box. Knelson concentrate flows by gravity to the Acacia intensive cyanide leach circuit.

Re-commissioning of the gravity concentration circuit was recently finalized in January 2020, and Rainy River has been able to generate concentrate in the Knelson concentrators and process a full cycle through the gravity circuit. In 2020, Rainy River plans to operate one Knelson concentrator for 2 weeks at the end of each quarter to remove the gold in the circulating load to solve any remaining process issues. After 2020, when gold and silver production increases, Rainy River will operate the circuit continuously to minimize the loads on the CIP and elution circuits.

17.1.6 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 VFDs. The typical mill feed has a F_{80} of 2,800 µm and the resultant product size is currently a P_{80} range of 90 µm - 110 µm. The mill operates at 75% of critical speed to achieve a production rate of 1,050 – 1,350 tph. The design operating power at the pinions is 12,360 kW, which is approximately 82% of the installed power of 15,000 kW. The slurry discharges from the mill through a trunnion magnet for steel removal and into the cyclone feed pump box.

The ball mill is operated with a target slurry density of 72% solids (w/w) and a circulating load of 300%. The maximum circulating load is projected to be 400%. The design ball charge is 32% (v/v) with a maximum design ball charge of 36% (v/v).

17.1.7 Intensive cyanide leaching of gravity concentrate

The Knelson concentrate 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 using leachate and a solution with 2.5% sodium cyanide and 1.5% sodium hydroxide 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 pump box for re-processing.

17.1.8 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 centrewell of a 45 m diameter by 3.3 m high pre-leach thickener.

The thickener underflow density is controlled to 50% - 55% solids (w/w) using density measurement and variable speed underflow pumps. The underflow slurry is pumped to the cyanide leach tanks.

The thickener overflow solution is pumped to the 17 m diameter by 9.1 m high process water tank.

17.1.9 Process water

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. The medium pressure process water pumps also feed the high-pressure process water distribution pump. The process water tank receives water from the pre-leach thickener overflow, process recirculation heat exchangers, cooling water return, the mine rock pond and the tailings reclaim pumps. Tailings reclaim water also reports to the pre-leach thickener feed tank and the tailings pump box.

17.1.10 Leaching and carbon in pulp

The thickener underflow slurry is adjusted to 50% - 55% solids (w/w) and pumped to the leach circuit. The leaching circuit consists of eight tanks in series which are 18.0 m in diameter 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. 1 is 22.7 m high and No. 8 is 19.2 m high. The first four tanks use oxygen for the leach reaction. The last four tanks have air injection to supply oxygen.

Leach tank No. 1 can be used for pre-aerating the slurry if required. The slurry overflows the pre-aeration tank to leach tank No. 2 where cyanide is added, and leaching continues through to 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. Gold in solution is absorbed onto the carbon. 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 m^3 for a total operating volume of 2,520 m^3 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 carbon stripping vessels. 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 pump the slurry from tank to tank.

The target carbon concentration is 20 t/tank. 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. 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 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 pump box, 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 pump box. The carbon fines recovered are loaded into bags.

Process controls in the leaching circuit include analyzers for both pH and cyanide concentration. Cyanide concentration is measured using the TAC 1000. There are primary and secondary slurry samplers on the CIP discharge following the carbon safety screen for analysis of the CIP tailings.

17.1.11 Carbon desorption and regeneration

The gold is desorbed from the carbon using the high pressure and temperature Zadra process. Two 10 t carbon stripping vessels are installed. The CIP carbon transfer batch size is 20 t. One strip vessel is operated at a time - while operating the first vessel, the second strip vessel is filled ready to be stripped. The stripping cycle includes 60 minutes to transfer carbon and 480 minutes to strip. The overlap time is approximately 240 minutes. Cooling of the carbon following stripping is for 60 minutes and carbon unloading time is for 60 minutes. The total stripping solution volume per batch is 450 m^3.

In the Zadra process, gold and silver are eluted from the carbon and recovered by electrowinning continuously. Eluent containing 1,500 ppm sodium cyanide and 2% (w/v) (weight in grams of solute / milliliters of solute) sodium hydroxide is pumped from the barren solution tank through heat exchangers, which heat the solution to 140°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 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.

An acid wash tank, which was previously used for acid washing the carbon, has been decommissioned.

As the plant throughput ramps up, Rainy River does not expect that the CIP or elution circuits will be bottlenecks; especially if the gravity circuit is online (it is expected that 20% of the produced gold will come from the gravity circuit and bypass the elution circuit). The gold solution losses in the CIP circuit due to higher throughputs is anticipated to be minimal.

17.1.12 Carbon reactivation

The stripped carbon is reactivated in a horizontal electric rotary kiln operating at 750°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 kilogram per hour (kg/h) for a total of 12 tpd. The target is to regenerate approximately 60% of the carbon stripped.

17.1.13 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 though the electrowinning cells. There are three parallel trains of two 3.5 m^3 cells, with design flows of 44 cubic metres per hour (m^3/h) and 15 minutes retention time. 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 25 kg gold and silver doré.

17.1.14 Cyanide destruction

The slurry leaving the last CIP tank passes through a carbon safety screen to recover coarse carbon and then flows to the cyanide destruction circuit. The circuit consists of two 11.5 m diameter by 13.5 m high mixing tanks in series to provide a retention time of 1.5 hours. A mixture of copper sulphate, air-SO_2 gas and lime is added to destroy the cyanide. The system is designed to reduce the CN_{WAD} component of the tailings so that the total cyanide concentration (CN_{TOTAL}) in the tailings is less than 5 ppm.

17.1.15 Tailings and reclaim water system

17.1.15.1 Tailings management area

The detoxified slurry flows from the cyanide destruction circuit to the tailings pump box. The tailings slurry is then pumped by two 356 mm x 304 mm, 550 kW centrifugal pumps in series to the TMA. When depositing tailings along the North Dam, a booster pump station will be installed to allow for sustained flow rates to achieve 27,000 tpd nominal production. The booster pump station will be installed approximately halfway between the final tailings spigot location and the process plant.

Reclaim water is pumped from the TMA to the process water tanks and tailings pump box by two 1,350 m^3/h, 522 kW vertical turbine pumps, one operating and one spare. The reclaim water demand for the process facilities is 1,200 m^3/h.

Rainy River is permitted to operate at an average daily plant throughput of 27,000 tpd, averaged over a quarter. The peak daily limit is 32,400 tpd.

17.1.15.2 Water management pond

The water management pond (WMP) is designed to hold 5 Mm^3 of water, the water is to be of discharge quality in the event it needs to be sent to the surrounding environment. The WMP acts as a backup source of water for process plant supply if required. The pump house at the WMP is equipped with a 522 kW duty vertical turbine pump and a 223 kW standby vertical turbine pump.

17.1.15.3 Mine rock pond

The mine rock pond has a capacity of approximately 500,000 m^3 and receives water from open pit dewatering. Water from the mine rock pond is pumped to the reclaim water tank, the tailings pump box, the process water tank and the cyclone feed pump box. The mine rock pond is not available during the winter due to freezing. Water will be supplied from TMA reclaim water only during the winter months.

17.1.16 Reagents

Reagent dosing 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.

17.1.16.1 Sodium cyanide

Sodium cyanide is 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 the sodium cyanide is dissolved using the cyanide recirculation pumps. Air is then introduced to transfer all the solution from the ISO container 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 is then metered from the cyanide holding tank to the process using the cyanide feed pumps.

17.1.16.2 Lime

Quicklime (CaO) is 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 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 produce hydrated lime. The slaked lime is mixed to a density of 25% solids (w/w) and transferred to a 5.0 m diameter by 7.0 m high slaked lime holding tank. The lime is recirculated from the holding tank, through the plant and returned to the holding tank by the lime recirculation pumps. The lime is metered to the destination points, including the mill feed, pre-leach thickener, leach tanks, and cyanide destruction through lime delivery piping that tees off the main lime ring-main.

17.1.16.3 Caustic soda

Caustic soda (NaOH) is shipped by tanker truck as a 50% (w/v) solution. The solution is diluted with reclaim water in a 4.2 m diameter by 6.2 m high mixing tank to a concentration of 25% (w/v). The mixing tank is sized for one and a half 36 t tanker trucks.

17.1.16.4 Sulphur dioxide

Sulphur dioxide is delivered in liquid form by 28 t tanker trucks 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 ensures that no lines connected to the SO_2 system enter the process plant.

17.1.16.5 Copper sulphate

Copper sulphate ($CuSO_4.5H_2O$) is trucked as 95% dry crystal in 1,000 kg supersacks. The bags are mixed with reclaim water and dissolved to approximately 15% (w/v). The 2.4 m diameter by 3.1 m high mix tank has the capacity to mix two bags to the required concentration of 15% (w/v). The mix tank supplies a small 1.5 m diameter by 1.0 m holding tank or day tank for pumping.

17.1.16.6 Activated carbon

Natural coconut shell type activated carbon (typical dimensions 6 mesh x 12 mesh) is used in the CIP adsorption circuit. The carbon is trucked in 20 t shipments of 500 kg bulk bags. The new carbon is added to the attrition tank feed hopper and into the carbon attrition tank, where 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.

17.1.16.7 Antiscalant

Antiscalant is 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 is delivered in totes and stored inside the building.

17.1.16.8 Flocculant

Flocculant is delivered to the plant in 750 kg super sacks. The flocculant bags are stored in a cold storage facility. The bags are 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.

17.1.16.9 Sodium metabisulphite

Sodium metabisulphite is only used as a back-up reagent to sulphur dioxide, and thus far has not been used in the plant.

Sodium metabisulphite ($Na_2S_2O_5$) would be delivered as dry crystal in 1,000 kg super sacks by truck. The bags would be mixed to a 20% (w/v) solution in a 3 m diameter by 5 m high agitated tank. The solution would be pumped to the cyanide destruction circuit via a metering pump.

17.1.16.10 Reagent consumptions

The September 2019 Forecast developed by New Gold versus actual unit reagent consumptions for the process facilities for the months of October – December 2019, are presented in Table 17.1, along with the predicted LOM average reagent consumption. The forecasted values are based on historical consumption rates.

Table 17.1 Process plant reagent consumptions

Item	Units	Reagent consumptions				
		Forecast (Sep 2019)	Oct 2019	Nov 2019	Dec 2019	LOM average
Mill production	t	736,000	555,560	790,567	725.743	787,528
SAG media	kg/t	0.37	0.20	0.33	0.27	0.35
Ball mill media	kg/t	0.57	0.37	0.42	0.91	0.50
Flocculant	g/t	25	24	19	14	20
Lime	kg/t	0.80	0.55	0.55	0.65	0.70
Sodium cyanide	kg/t	0.24	0.20	0.17	0.18	0.20
Sulphur dioxide	kg/t	0.40	0.22	0.19	0.26	0.34
Copper sulphate	g/t	100	81	42	38	75
Activated carbon	g/t	40	29	15	27	30
Liquid oxygen	m^3/t	0.23	0.24	0.19	0.23	0.25
Sodium hydroxide	kg/oz	1.8	1.8	1.5	2.6	1.4
Antiscalant	kg/t	21	30	17	21	18

Many of the actual reagent consumptions, particularly sodium cyanide, sulphur dioxide, copper sulphate, activated carbon and lime are significantly lower than forecast and the predicted LOM average. New Gold has focused on optimizing reagent consumptions in Q4-2019 with an objective of reducing processing unit costs.

17.1.17 Auxiliary systems

17.1.17.1 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	Nominal pressure (psi)	Maximum pressure (psi)	Flowrate (Nm^3/h)
Primary crusher	Rotary screw	1	120	125	246
Leaching	Rotary screw	1	50	60	1,498
Plant air	Rotary screw	2	120	125	1,434
Cyanide destruction	Rotary screw	3	**50**	**60**	6,545

17.1.17.2 Oxygen plant

Oxygen is supplied to the first four cyanide leach tanks. Oxygen is supplied as a bulk liquid.

17.1.18 Control

Control of process equipment is done via a Delta V control system. Site possesses a KnowledgeScape expert system for advanced process control for the SAG mill. Site uses PARCview software for trending and data extraction.

17.1.19 Mill specific energy usage

Table 17.3 shows the specific energy usage for the SAG and ball mills for the 12-month period from 1 March 2019 to 29 February 2020.

Table 17.3 Mill energy usage from 1st March 2019 to 29th February 2020

Mill	Tonnage	Installed motor power (kW)	Operating hours	Total power draw (kWh)	Actual plant specific energy (kWh/t)	Design specific energy (kWh/t)*	Energy utilization (power drawn/installed motor power) (%)
SAG mill	8,204,373	15,000	7,261	77,971,405	9.5	15.8	72%
Ball mill	8,204,373	15,000	7,261	98,656,555	12.0	15.8	91%

Note: *These figures were based on an annual throughput of 951 tph at 8,000 hours of operation per annum.

The actual SAG mill specific energy usage for the 12-month period was 9.5 kWh/t against the design specific energy of 15.8 kWh/t. The actual energy utilization (power draw / installed power) for the 12-month period was 72%.

The actual ball mill specific energy usage for the 12-month period is 12.0 kWh/t against the design specific energy of 15.8 kWh/t. The actual energy utilization for the 12-month period was 91%.

The actual specific energies are below the design energies due to running much higher throughputs than the original design criteria.

17.1.20 Mineral processing plant performance and production statistics

The key operating parameters and performance indicators for the Rainy River processing plant for the months of October 2019 – December 2019 are presented in Table 17.4 together with the predicted LOM operating parameters and performance indicators.

Table 17.4 Rainy River processing plant operating parameters

Operating parameter description, units	Units	Forecast (Sep 2019)	Oct 2019	Nov 2019	Dec 2019	LOM average
Mill production	tpm	736,000	555,560	790,567	725.743	787,528
Ore milled (open pit + underground)	tpd	23,742	17,924	26,352	23,411	25,746
Crusher utilization	%	66.0	55.5	71.1	70.6	71.0
Mill utilization	%	86.0	71.3	87.2	87.5	89.8
Crusher production	tph	1,550	1,433	1,564	1,381	1,550
Mill production rate	tph	1,154	1,047	1,259	1,115	1,200
Process plant gold feed grade	g/t	0.84	0.87	0.88	0.85	1.06
Process plant silver feed grade	g/t	0.86	1.50	2.26	1.48	2.51
Target grind size P_{80}	µm	90	74	113	104	104
Gravity gold recovery	%	N/A	N/A	N/A	N/A	20
Leach gold recovery	%	87.3	92.2	88.6	91.5	86.1
Overall gold recovery	%	87.3	92.2	88.6	91.5	88.9
Gravity silver recovery	%	N/A	N/A	N/A	N/A	N/A
Leach silver recovery	%	53.9	60.6	60.0	60.2	57.3
Overall silver recovery	%	53.0	60.6	60.0	60.2	57.3

Rainy River's current focus is to operate at higher throughputs than design which produces a coarser grind size P_{80} range of between 90 µm to 110 µm compared to the design criteria grind size P_{80} of 75 µm. This subsequently reduces gold recovery. Rainy River has determined that an increase in throughput at the expense of gold recovery is the most economically viable option.

17.2 Plant debottlenecking and expansion projects

The following sections summarize the projects planned for the increase in plant throughput capacity.

17.2.1 Pre-leach thickener

Settling rates in the pre-leach thickener have been identified as a plant bottleneck. When the plant experiences excessive grinding circuit throughputs, fine solids tend to report to the thickener overflow launder. To rectify this issue, Rainy River will trial a polymer slicing unit and an alternative flocculant supply system with the goal of improving the dissolution of flocculant and subsequently increasing flocculant flowrates. This bottleneck principally occurs during winter months when the solution is at a lower temperature. This modification has been budgeted for 2020.

17.2.2 Crushed ore stockpile

Dust from the crushed ore stockpile has been identified as an environmental and health concern. Solutions to the dust problem include water spray bars for dust suppression during the summer and a calcium chloride solution distributed by spray bars in winter. Rainy River expects these measures should largely remove the dust issue.

17.2.3 Tailings pumping

The tailings pumps are a bottleneck when depositing along the North Dam of the TMA. A project is currently ongoing which will implement a booster pump station that will sustain 27,000 tpd nominal throughput rates when depositing at the North Dam. Detailed engineering and procurement are currently in progress. Rainy River expects to install the booster pump station in the 2nd quarter of 2020. Commissioning will commence after installation.

17.3 OMC process plant review and audit

OMC completed an audit of the Rainy River process plant in April 2019. OMC used the comminution data that was collected from the audit for creating a JKSimMet model. The purposes of the JKSimMet model were to forecast the process plant throughput based on comminution testwork data and to simulate different comminution circuit flowsheet configurations.

17.3.1 OMC comminution simulations

OMC completed simulations of the grinding circuit using two comminution modelling software packages. The objectives of the simulations were to determine strategies to increase SAG mill throughput and to grind as fine as possible.

Simulations were completed with both the JKSimMet software and OMC Power Model to provide a range of throughput forecasting. Average predicted results were calculated as the average of the two models.

OMC completed a survey of the comminution circuit on 11 April 2019. The survey data was used as a basis for constructing a baseline JKSimMet model. Survey inputs such as A x b values, ball mill work index and mill load were used for constructing and calibrating the baseline JKSimMet model. The circuit was then mass balanced on mass of solids, water and particle size distribution using JKSimMet software.

The baseline model had the following inputs which were based on the survey data:

- Total SAG mill load of 26.2%.
- SAG mill oversize screen grate aperture of 70 mm.
- A x b value of 46.8.
- CWi of 15.9 kWh/t.
- BWi of 13.6 kWh/t.
- SAG mill ball charge of 10%.

Additional simulations were completed where the process variables were sequentially manipulated to observe the effect on SAG mill throughput and the ball mill cyclone overflow product size. A total of eight simulations were completed in SAB mode (SAG mill / ball mill) and another seven simulations were completed in SABC mode (SAG mill / ball mill / pebble crusher).

The simulations evaluated the effect of:

- Increasing SAG mill speed.
- Varying SAG mill ball charge.
- Varying SAG mill ball size.
- Varying SAG mill grate configurations:
  - 70 mm (existing grate at time of survey).
  - 32 mm (new alterative to promote more grinding within the SAG mill in SAB mode).
  - 64.1 mm (new grate for SAB / SABC operation).
- Increase SAG mill ball charge.
- Reducing ball mill media size to 50 mm to improve ball mill grinding efficiency.
- Variable cyclone feed percent solids (constant pressure).
- Grate opening area.

Five ore hardness values were considered:

- Survey ore
- 50th percentile ODM ore
- 80th percentile ODM ore
- 50th percentile CAP ore
- 80th percentile CAP ore

Simulations were capped to operating conditions of 75% ball mill critical (Nc), a ball mill ball size of 50 mm (with the exception of the baseline survey) and 31% ball mill charge (with the exception of the baseline survey). The cyclone feed percent solids were manipulated between 42% to 62% solids (w/w) to maintain a constant cyclone pressure of ~98 kilopascal (kPa) and the ball mill feed chute water was adjusted to provide a ball mill discharge density in the range of ~72% (same as survey).

A summary of the simulations is shown in Table 17.5 and Table 17.6.

Table 17.5 Summaries of comminution simulations using SAB configuration

Circuit configuration	**SAB**	**SAB**	**SAB**	**SAB**	**SAB**	**SAB**	**SAB**	**SAB**	**SAB**
Ore description	Survey	Survey	Survey	Survey	ODM 80th	ODM 50th	ODM 50th	CAP 80th	ODM 50th
Simulation description	Survey 1 model fit	32 mm grate size	64.1 mm grate size	75% Nc, 10% BC	ODM 80th	ODM 50th	0% BC	CAP 80th	ODM 50th
Grate size (mm)	70.0	32.0	64.1	64.1	64.1	64.1	64.1	64.1	64.1
Grate opening area (%)	6.09%	6.09%	7.56%	7.56%	7.56%	7.56%	7.56%	7.56%	7.56%
SAG mill ball charge (% v/v)	15.6%	7.3%	8.5%	10.0%	10.0%	10.0%	**0.0%**	10.0%	10.0%
SAG mill ball speed (% of Nc)	58.4%	72.0%	72.5%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%
SAG mill ball size (mm)	130	100	130	130	130	130	130	130	N/A
Ball mill ball charge (% v/v)	29.5%	31.0%	31.0%	31.0%	31.0%	31.0%	31.0%	31.0%	31.0%
Ball mill speed (% of Nc)	74.8%	74.8%	74.8%	74.8%	74.8%	74.8%	74.8%	74.8%	74.8%
Ball mill ball size (mm)	65	50	50	50	50	50	50	50	50
A x b	46.8	46.8	46.8	46.8	26.6	32.7	32.7	20.3	22
CWi (kWh/t)	15.9	15.9	15.9	15.9	23.0	23.0	24.0	14.3	14.3
BBMWi (kWh/t)	13.6	13.6	13.6	13.6	14.7	13.8	13.8	15.5	14.8
Avg. process plant throughput (tph)	1,078	1,223	1,223	1,676	1,147	1,315	542	947	1,066
Avg. cyclone overflow P_{80} (µm)	79	61	64	128	66	77	21	55	57
Avg. SAG mill power (kWh/t)	8.7	7.6	8	6.5	9.6	8.4	12.7	11.6	10.9
Avg. ball mill power (kWh/t)	11.4	11.5	11.5	8.1	12.4	10.8	25.8	13.7	13.0
Avg. total ball mill power (kWh/t)	20.1	19.1	19.5	14.9	22.0	19.2	38.5	25.3	23.9

Table 17.6 Summaries of comminution simulations using SABC configuration

Circuit configuration	SABC	SABC	SABC	SABC	SABC	SABC	SABC	SABC
Ore description	Survey	Survey	ODM 80th	ODM 50th	CAP 80th	CAP 50th	ODM 80th	CAP 80th
Simulation description	27 kt/d	75% Nc, 10% BC	ODM 80th	ODM 50th	CAP 80th	CAP 50th	New optimized grate with 10% OA	New optimized grate with 10% OA
Grate size (mm)	64.1	64.1	64.1	64.1	64.1	64.1	64.1	64.1
SAG mill ball charge (v/v)	8.5%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%	10.0%
SAG mill ball speed (% of Nc)	72.5%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%	75.0%
Grate opening area (%)	7.56%	7.56%	7.56%	7.56%	7.56%	7.56%	10.00%	10.00%
A x b	46.8	46.8	26.6	32.7	20.3	22.0	26.6	20.3
BBMWi (kWh/t)	13.6	13.6	14.7	13.8	15.5	14.8	14.7	15.5
Avg. process plant throughput (tph)	1,223	1,673	1,218	1,397	994	1,056	1,280	1,067
Avg. cyclone overflow P_{80} (µm)	73	131	77	91	55	57	92	71
Avg. SAG mill power (kWh/t)	7.5	6.3	9.0	7.8	10.9	10.3	8.5	10.1
Avg. ball mill power (kWh/t)	11.5	8.5	11.6	10.1	14.2	13.4	11.0	13.2
Avg. pebble crusher power (kWh/t)	0.29	0.22	0.29	0.26	0.33	0.3	0.3	0.35
Avg. total ball mill power (kWh/t)	19.3	15.0	20.9	18.2	25.4	24.0	19.8	23.7

The following observations were made from the simulation work:

- Simulations for 1,233 tph in SAB mode considered two SAG mill grate configurations. The first considers grates with 32 mm wide slots and open area (OA) of 6.09% (same as existing 70 mm grate). The second simulation considered grates with 64.1 mm width slots and 7.56% OA. Both scenarios considered running the SAG mill at higher mill speed and reduced ball charge. The smaller 32 mm grates allow a reduction in SAG mill ball size to enhance abrasion grinding within the mill. The use of 32 mm grates is not amenable for future pebble crushing. This would pose a risk to the project if the detailed mine plan and geometallurgical forecast is not available to identify when the conversion to larger grates would be required for operation in SABC mode.
- Simulation of the smaller 100 mm balls in the SAG mill with 64.1 mm grates was not attempted due to the scaling between the base case and subsequent simulations at higher mill speed and reduced ball charge. It is recommended to perform onsite empirical trial of smaller SAG mill media size.
- Higher variability is observed between the 50th and 80th percentiles of the ODM Zone ore compared to the CAP Zone ore. The simulation results indicate a +15% increase in average SAG mill throughput for 50th percentile ODM Zone ore and +6% throughput increase for 50th percentile CAP Zone ore, compared to their 80th percentile counterparts.
- Based on the available power in the circuit, increasing the overall milling rate coarsens the cyclone overflow product. Simulations on soft survey ore yielded the highest cyclone overflow P_{80}, namely 123 to 133 µm in SAB mode or 125 to 137 µm in SABC mode. SABC simulations on more competent ODM Zone and CAP Zone ores yielded cyclone overflow P_{80}s of 67 to 81 µm, which is similar to current plant grind size.
- In SAB mode, the predicted average SAG mill throughput is 1,675 tph for survey ore with an average final product P_{80} of 128 µm. The survey ore is comparable to the 50th percentile of historic ODM Zone testwork and it is considerably softer than the ore properties from the plant design. For this soft ore, there is no gain in SAG mill throughput in SABC mode.
- The predicted average SAG mill throughput for 80th percentile ODM Zone ore in SAB mode is 1,147 tph with an average final product P_{80} of 66 µm. In SABC mode, the average SAG mill throughput increases to 1,218 tph and the average final product P_{80} is 77 µm.
- The predicted average SAG mill throughput for 80th percentile CAP Zone ore in SAB mode is 947 tph with an average final product P_{80} of 55 µm. In SABC mode, the average SAG mill throughput increases to 994 tph and the average final product P_{80} is 55 µm.
- Simulations in SABC mode were also conducted with 64.1 mm grate with optimized grate area of 10% OA to evaluate the effect of increasing OA on pebble extraction and SAG mill throughput. The results indicate roughly a +62% increase in pebble extraction from 122 tph to 198 tph for ODM Zone ore and from 116 to 189 tph for CAP Zone ore. This leads to a predicted increase in average SAG mill throughput in the range of +5% to +7% for ODM Zone and CAP Zone ores respectively. Plant trials with 64.1 mm grate aperture with 10% OA should be considered only after the pebble crusher circuit is fully operational.
- When operating the SAG mill at high speed and low ball charge with 64.1 mm grates, it may be difficult in practice to maintain the 26.2% total SAG mill load which was used in the simulations. The SAG mill speed may need to be adjusted to hold up the charge inside the mill if the grind is too coarse.
- The predicted average SAG mill throughput for 50th percentile ODM ore in AB mode (i.e. SAG mill with 0% steel) is 542 tph with an average final product P_{80} of 21 µm.
- All simulations were completed with ball mill discharge density varying from ~70.4% to 71.8% solids, which is similar to the survey. Given the simulated circulating load, the calculated mill superficial velocity inside the ball mill is ≤0.25 m/s, which is reasonable. Both JKSimMet and OMC modelling indicate a reduction in circulating load with smaller grinding media inside the

mill. In general, it is recommended to operate the ball mill with 75% - 78% solids and to charge the mill with as much steel media as possible to draw full power. The simulations were completed with 31% ball charge of 50 mm balls. If the circulating load increases above predicted levels, it is recommended to remove water addition at the ball mill feed chute to increase the discharge density. This in turn will reduce the superficial velocity to help retain the balls inside the mill.

- The current cyclone configuration has spigots with an inside diameter (Du) of 120 mm and vortex finders (VF) with an inside diameter (Do) of 190 mm. This configuration provides reasonable classification performance; thus, no changes were recommended. The current Du / Do ratio is 0.64. This ratio is reasonable to provide spray discharge at the cyclone underflow. Water recovery to the cyclone underflow was 22% for the survey and 23% for the circuit simulations, which is also a characteristic of good cyclone performance.

AMC has reviewed the OMC report and concurs with the report's observations, conclusions and recommendations. These are included below with specific recommendations listed in Section 26. Note, Rainy River has yet to take advantage of these recommendations but intend to implement them in future years.

Based on the historical operating data, the stability of the grinding circuit can be improved but is operating reasonably well. The coefficient of variance (st. dev. / average) was 11% for the SAG mill fresh feed throughput and 13% for the final product P_{80}. A well-run grinding circuit typically has coefficients of variance of approximately 10% or less for both throughput and grind size. The SAG mill load control is stable with the use of the current expert system logic and much of the variability may be associated with the ore variability.

The SAG mill is currently operated at a low speed (~56% Nc) and should be run at higher speeds. Simulations were completed with SAG mill speeds of 72.5% to 75% of the Nc.

The SAG motor power utilization is relatively low at 69%, as motor power utilization should be in the range of 80% to 95%. SAG mill power draw can be increased by operating at 9.3 revolutions per minute (rpm) to 10 rpm (72% - 80% Nc) if required.

The ball mill motor power utilization was reasonable at 90%, as motor power utilization should be in the range of 90% to 100%.

Rainy River should increase the ball mill charge to 30% - 32% and optimize ball mill speed to produce better power utilization in the ball mill. Although there is 5% - 10% available power in the ball mill, an increase in overall milling rate will likely be at the cost of coarsening the cyclone overflow product, unless additional SAG mill power is utilized, or ball mill circuit grinding efficiency can be improved.

The actual grinding circuit availability at 85% is considered to be low, as the design grinding circuit availability is 92%. Rainy River should complete a thorough review to identify the causes of downtime causes, which would assist in creating preventative maintenance action items which would reduce the likelihood of these downturn events reoccurring.

Site personnel do not perform routine mill grind outs for steel ball charge measurement or crash stops for total load measurement in the SAG mill or ball mill, nor do they have a formal database of historic measurements. Rainy River should create a database which details SAG mill crash stops and ball mill grind-out measurements to provide information for grinding circuit performance reviews and audits.

Rainy River should investigate equipment and sensor types which can provide real-time data for advanced process control purposes to help improve circuit stability. These include conveyor cameras for particle size analysis (e.g. Split Engineering or KnowledgeScape), mill vibration sensors for mill load control (e.g. MillSlicer or MillsScanner) and FloCEP cyclone sensors for roping detection. Rainy River should review each technology to determine if they can become part of the mill optimization strategy.

The SAG mill bearing pressure is currently used to measure the SAG mill running load mass. Real-time control of the SAG mill load should be used instead of the SAG mill bearing pressure, using a combination of bearing pressure control and vibration or acoustic sensor logic. An installed vibration sensor would allow for additional control strategies that could improve grinding efficiency and extend liner life.

The metallurgy group should work with the mining group to log daily / weekly SAG mill feed ore blends and major ore types.

Rainy River should obtain a copy of MolyCop Tools© software, to estimate media wear for forecasting SAG mill ball makeup.

Rainy River should program the distributed control system (DCS) with equipment sensors that record operating hour totalizers. This would assist with maintenance programs and calibrations. For example, installing a cyclone operating hour totalizer that would indicate when cyclones should be opened and closed.

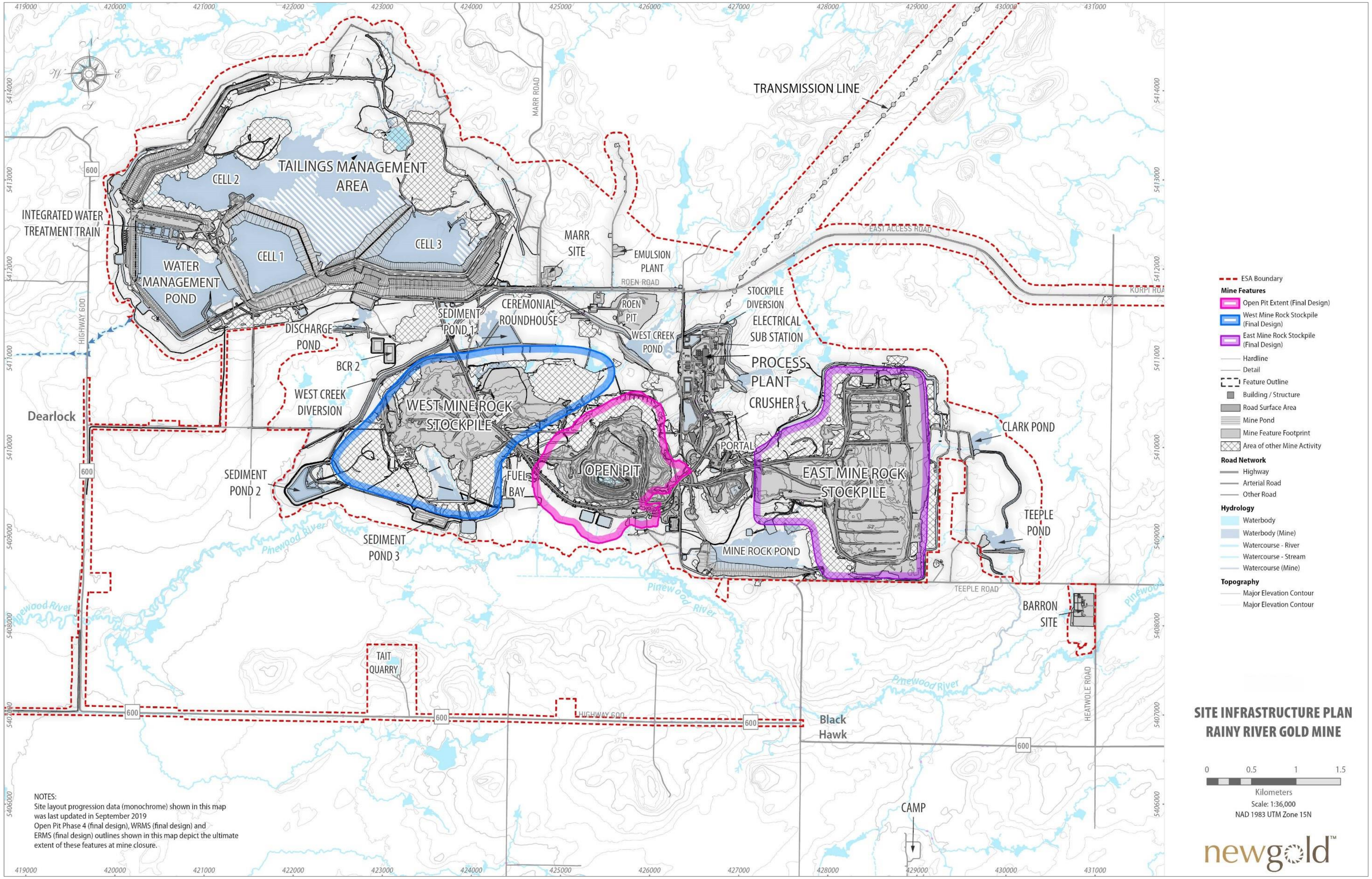
Rainy River should perform routine preventative maintenance and calibration checks on the sump level sensors, cyclone flowmeters and cyclone density meters.

Rainy River should perform monthly checks of the SAG mill feed conveyor weightometer and the pebble crusher return conveyor weightometer.

18 Project infrastructure

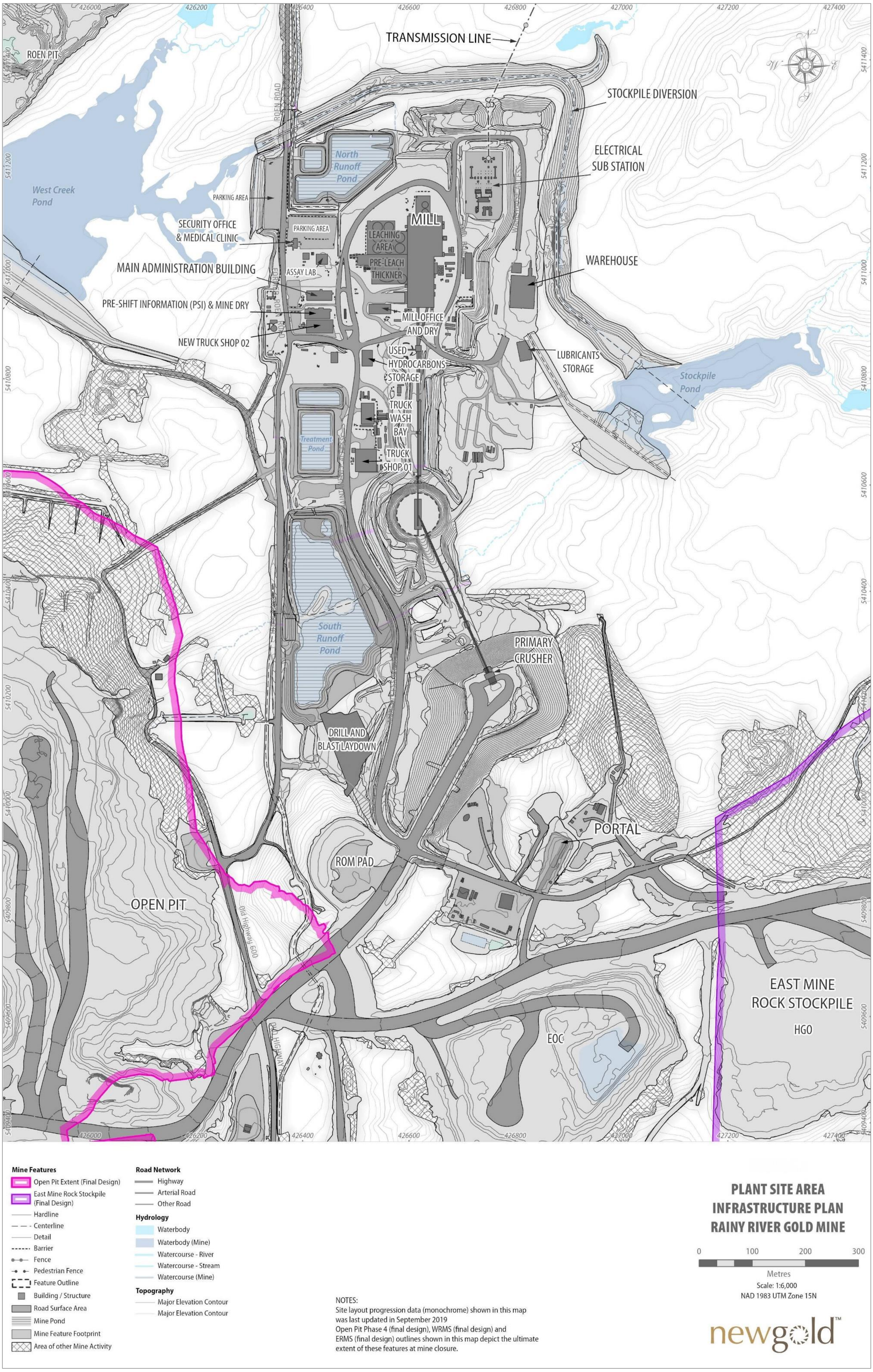
This section summarizes the principal project infrastructure. Figure 18.1 provides a general site plan indicating principal project infrastructure and Figure 18.2 presents a detailed site plan of principal infrastructure located in the vicinity of the process plant.

Figure 18.1 General site plan



Source: New Gold 2020.

Figure 18.2 Detailed site plan



Source: New Gold 2020.

18.1 Primary 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 Haul Road 13. A branch to the north provides access to the explosive magazine and the emulsion plant.

On the south side of the TMA a single lane light vehicle service road runs parallel to the tailings and reclaim water pipelines. This road ties into double-lane service roads along the south and west sides of the WMP and ultimately continues through to the north-west, north, north-east, and east of the TMA.

Haul trucks and other heavy equipment access the TMA via haul roads primarily constructed within the buttress of the dam. These haul roads are modified annually with each dam raise.

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.

18.2 Mine haul roads

The mine haul roads provide connectivity of the open pit to the overburden and waste rock dumps as well as ore stockpiles; connect the open pit to the crusher pad and pertinent mine facilities (truck shop, truck wash, fuel farm, etc.); and connect the open pit to the TMA to provide access for the haulage and placement of dam construction materials as required.

18.3 Principal mine & maintenance operation facilities

The principal mine and maintenance facilities include the truck shop, truck wash, fuel bay and explosives storage and mixing facilities.

18.3.1 Truck shop

Truck Shop 1 is a 1,350 metres squared (m^2) heated and insulated fabric covered steel structure building with interlocking mat flooring that includes two service bays and additional space to house a mobile service crane. Truck Shop 1 is located in the plant site area to the west of the conveyor system and south of the truck wash.

Truck Shop 2 will be a 1,500 m^2 heated and insulated fabric covered steel structure building with a concrete floor that includes three service bays and is currently being constructed south of the existing Mine Dry. The new shop will provide additional service bay capacity to support the ongoing maintenance of the mine fleet and will include a 50 t crane, compressed air and lubricant distribution systems. Truck Shop 2 is scheduled to be completed by the end of Q2-2020.

18.3.2 Truck wash bay

The 330 m^2 truck wash is located adjacent to Truck Shop 1 on the north side. The truck wash can accommodate a single Komatsu 830E mine haul truck with the box up and includes a pressure wash system and an oil / water separation system. The truck wash system has mud settling basins for oil and grease removal and a water filtration system for continuous recycling of wash water.

18.3.3 Fuel bays

The mine operations fuel bay is located west of the open pit along Haul Road 5. The fuel bay consists of two 75,000 L double walled storage tanks and is currently being expanded by an additional two 75,000 L double wall storage tanks. The total future storage capacity of 300,000 L of diesel fuel will provide mine operations with approximately two days of production storage.

The light vehicle fuel station is located east of the plant site at the corner of Marr Road and Roen Road. This installation consists of four double walled storage tanks including one 26,000 L gasoline, one 50,000 L clear diesel and two 75,000 L dyed diesel tanks. This fuel station provides service to light vehicles, buses as well as contractor fueling requirements.

18.3.4 Explosive magazine and emulsion plant

The explosive magazine and emulsion plant are located on a dedicated road to the north of the Roen 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.

18.4 Warehousing and storage

18.4.1 Warehouse

A new warehouse currently under construction will be a 2,800 m^2 facility located at the upper laydown to the east of the process plant.

Located to the south of the warehouse, the warehouse office is being constructed to house the supply chain department. The building will consist of 11 offices, a single meeting room as well as a kitchen and bathroom facilities.

Construction of the warehouse and office is scheduled to be completed in Q1-2020. Materials currently stored at the Fort Frances leased warehouse will be relocated and the lease terminated. Materials stored at the Barron site warehouse will be relocated and this warehouse will be re-purposed to a yet-to-be-defined use.

18.4.2 Lubricant storage building

Located to the south of the warehouse, a 650 m^2 fabric structure (uninsulated and unheated, but with passive ventilation) is being constructed to warehouse new lubricants. The construction of this storage building is scheduled to be complete by the end of Q1-2020.

18.4.3 Hydrocarbon storage building

Located to the west of Truck Shop 2, a 260 m^2 fabric structure (uninsulated and unheated, but with passive ventilation) is being constructed for temporary storage of used hydrocarbons and will include a double walled waste oil tank. The construction of this storage building is scheduled to be complete by the end of Q2-2020.

18.5 Principal offices and buildings

18.5.1 Security office and medical clinic

The security office and medical clinic building houses security and medical staff. An ambulance and fire truck are parked in an adjacent building.

The medical clinic is staffed by a Nurse Practitioner and the clinic is equipped with life support and resuscitation units.

18.5.2 Main administration building

The main administration building is located south of the security / medical building and assay lab in the mill site area. The main administration building houses site management, technical and administrative staff, including Health & Safety, Environmental, Finance, Human Resources, Capital Projects, Mine Operations, Mill Operations, Mobile Maintenance, and Site Services.

18.5.3 Mine dry

The mine dry is located to the south of the main administration building and includes a dry area to support mine operations staff as well as a single meeting room.

18.5.4 Mill office and dry

A consolidated building is currently being constructed to house both a mill office and dry near the south west corner of the process building. The building will consist of 13 offices, a single meeting room as well as kitchen and hygiene facilities for office staff in addition to a dry area to support mill operations staff. The construction of the mill office and dry is scheduled to be completed during Q1-2020.

18.5.5 Parking area

Parking is provided adjacent to mill building, with capacity for 150 vehicles and two buses.

18.5.6 Assay lab

The assay lab is located adjacent to the main administration building. The lab is designed to process 200 mine blasthole 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 analysers for carbon and sulphur analyses.
- Wet chemical lab for solution samples.
- Environmental lab.
- Two offices, a lunchroom, and hygiene facilities.

18.5.7 Camp

A camp facility located on Atkinson Road was purchased by Rainy River in 2019 and is currently in the process of undergoing renovations. There exist ten dormitories with a capacity of 376 rooms and the ability to house up to a maximum of 376 people (single person rooms). Dormitories are classified as Private (118 rooms), Semi-Private (83 rooms), and Jack & Jill (175 rooms). Parking is available adjacent to the camp.

Recreational facilities at the camp include a gymnasium, TV room, pool tables, library, and a commissary store. Internet Wi-Fi is available to all rooms.

A dining facility is available for breakfast and dinner services. Lunches are required to be packed and taken during breakfast and dinner hours of operation.

18.5.8 Ceremonial roundhouse

A ceremonial roundhouse is currently under construction to provide a place for gatherings and traditional indigenous ceremonies. It is being constructed on the south side of Roen Road and west of Roen Pit. Construction is expected to be completed in Q1 2020.

18.6 Electric 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 north-east 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.

18.6.1 Emergency power

There are two emergency generators, both generating 600 V, then transformed to 13.8 kV, to connect to the main substation bus. During a power outage, total generator loading is monitored at the main substation, while critical loads are monitored by Operations. Critical loads include 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.

18.6.2 Communication

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

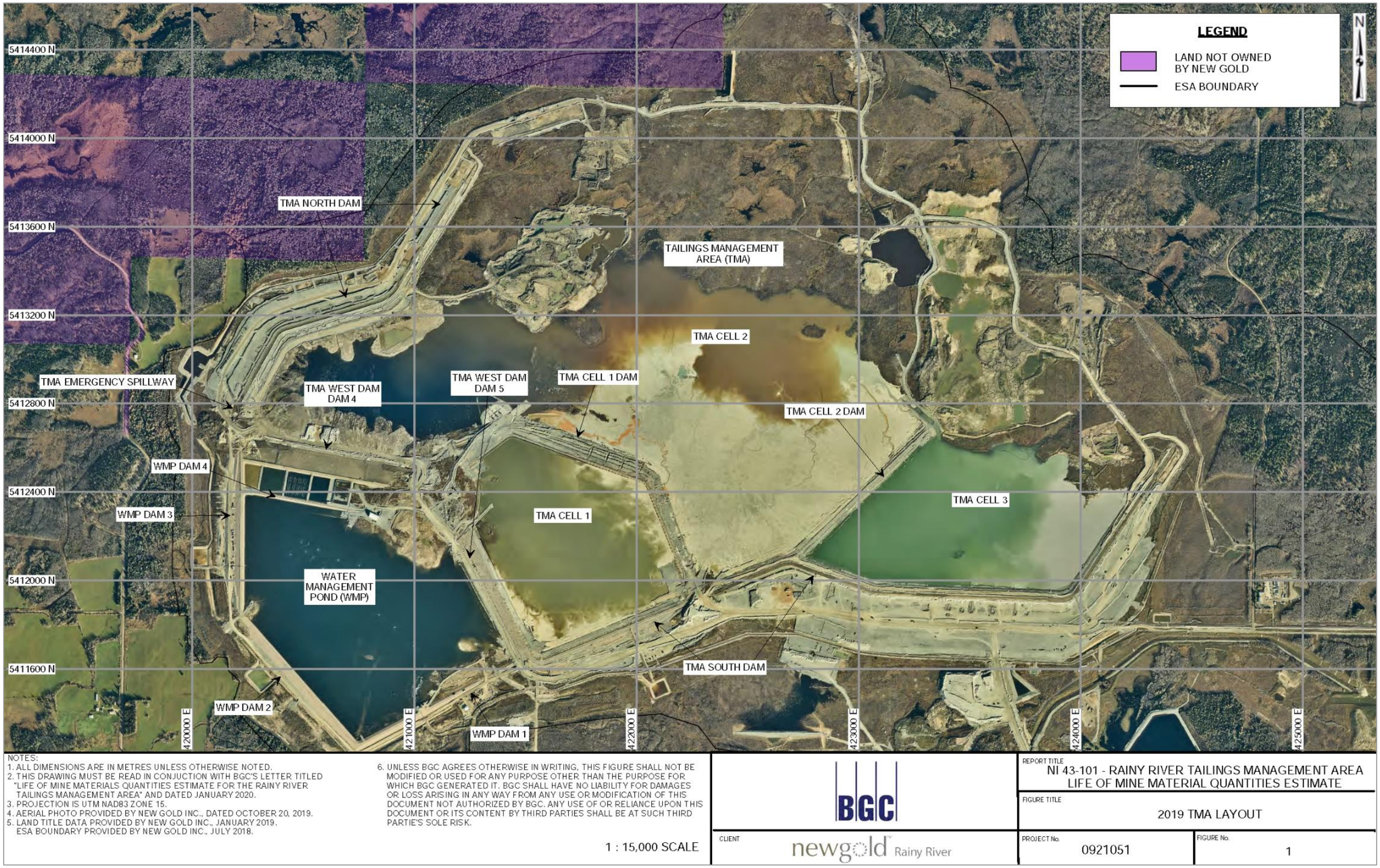
- Telemetry, data acquisition, and control between the process plant and exterior process equipment.
- Computer network between all departments.
- Local telephone lines.
- Computer network for maintenance on all electrical equipment.
- Fire detection.
- Video surveillance and access control systems.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

18.7 Tailings management area

18.7.1 Background

The TMA is divided into three independent cells for tailings deposition: TMA Cell 1, TMA Cell 2, and TMA Cell 3 (Figure 18.3) with a combined footprint area of approximately 550 ha. Containment for the TMA is provided by perimeter impoundment dams; the TMA North Dam along the north-west side, the TMA West Dam (Dams 4 and 5) along the west side, and the TMA South Dam along the south side. Naturally occurring high topography provides containment along the north and north-east sides of the facility. Internal impoundment dams provide separation between the internal cells with the TMA Cell 1 Dam situated between TMA Cell 1 and TMA Cell 2, and the TMA Cell 2 Dam located between TMA Cell 2 and TMA Cell 3 (Figure 18.3).

Figure 18.3 2019 TMA layout



Source: BGC 2020.

The TMA Cell 1 Dam and the TMA Cell 2 Dam were constructed to their ultimate dam crest elevations of 371.5 m and 366.5 m and will eventually be buried by tailings as the Rainy River TMA converts into a single tailings containment cell.

The WMP, located adjacent to the TMA, is a part of the water treatment system and stores treated water from the TMA and supplies water to the mill. The WMP is separated from the TMA by the TMA West Dam (comprising Dam 4 and Dam 5), and the remaining perimeter of the impoundment consists of WMP Dam 1, WMP Dam 2, WMP Dam 3, and WMP Dam 4. WMP Dams (1, 2, 3, and 4) were constructed to their ultimate dam crest elevation of 371.5 m.

The TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam will be raised incrementally during the mine life to provide sufficient storage for tailings. The TMA West Dam (Dam 4) was constructed to the starter dam elevation of 366.5 m by July 2017 (AMEC, 31 October 2017). The TMA West Dam (Dam 5) and a portion of the TMA South Dam (from approximate station (Sta.) 0+000 m to 0+800 m) were constructed to their Stage 2 crest elevation of 371.5 m by September 2017 (AMEC, 6 December 2017). Stage 2 construction of the TMA North Dam and TMA West Dam (Dam 4) to a crest elevation of 371.5 m and the TMA South Dam to a crest elevation of 369.0 m was completed by December 2019. Stage 2 construction of the TMA South Dam to a crest elevation of 371.5 m is planned to be completed in 2020, which will provide a containment elevation of 371.5 m with 0.4 m of freeboard for tailings.

Tailings deposition in TMA Cell 1 commenced in Q4 of 2017 and was filled around April 2018. The TMA Cell 2 Dam was commissioned in May 2018 and tailings were deposited in TMA Cell 2 until May 2019. Tailings deposition into TMA Cell 3 began in May 2019 and is ongoing. Generally, the tailings deposition strategy is to establish tailings beaches upstream of the perimeter dams (i.e. TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam).

18.7.2 Design

LOM material quantities for the Rainy River TMA were estimated using:

- 2019 Tailings Management Plan Update and Dam Raise Schedule (BGC, 25 January 2020).
- Stability modelling for the Rainy River TMA ultimate dam height.

The following sections describe the key assumptions and results of the tailings management planning, the ultimate dam stability modelling, and the material quantities estimate.

18.7.2.1 Tailings management planning

The tailings management plan used historical tonnage records from mill start-up on 9 August 2017 to 21 December 2019 as provided by New Gold (23 January 2019; 10 September 2019; and 22 December 2019). Forecasted tailings production tonnages for the LOM are based on the updated LOM plan provided by M.A. Okane Consultants Inc. (OKC) on 22 December 2019. LOM cumulative tailings tonnage provided to BGC was estimated to be 91.6 Mt. BGC estimated an average dry settled density of 1.35 t/m^3, a beach above water (BAW) slope = 0.35%, and beach below water (BBW) slope = 0.8%. In addition, BGC assumed the 99^{th} percentile pond to be contained below the spillway invert elevation.

Table 18.1 provides a summary of the dam raising schedule based on the tailings deposition modeling completed by BGC (25 January 2020). The tailings management plan indicates that the current fixed reclaim would be decommissioned and replaced with a barge reclaim system. Material quantities for the decommissioning of the fixed reclaim system and the commissioning of the barge system were not considered in the material quantities estimate.

Table 18.1 Summary of tailings deposition plan and dam raise schedule

Year	Dam crest elevation (m)	Raise height (m)	Spillway elevation (m)	Dams to be raised
2020	371.5	2.5	369.7	TMA South Dam
2021	373.6	2.1	371.8	TMA Perimeter Dams [1]
2022	375.1	1.5	373.3	TMA Perimeter Dams
2023	376.6	1.5	374.8	TMA Perimeter Dams
2024	377.9	1.3	376.1	TMA Perimeter Dams
2025	379.0	1.1	377.2	TMA Perimeter Dams

Notes: [1] TMA perimeter dams include the TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam.

18.7.3 Ultimate dam stability

Stability modelling was run for the ultimate TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam completed to a crest elevation of 379.0 m. Stability modelling was not performed for interim raises of the dams. The TMA ultimate dams were designed to satisfy the static, pseudo-static, and post-earthquake stability criteria consistent with the Stage 2 raise designs. The stability criteria (Table 18.2), loading conditions, and analysis methodologies are consistent with BGC, 21 December 2018; BGC, 16 May 2019; and BGC, 11 January 2020. For downstream stability, the maximum upstream tailings elevation (377.2 m) was assumed as documented on the Tailings Deposition Plan and Dam Raising Schedule (BGC, 25 January 2020).

Table 18.2 Minimum factor of safety

Loading Condition	Factor of Safety (min)
End of construction (EOC)	1.5
Pseudo-static	1.0
Post-earthquake	1.1

18.7.3.1 Stability sections modeled

The required downstream slope geometry is defined for different 'design zones' in the TMA based on subsurface conditions and observed foundation behaviour. Design zones of the TMA South Dam and TMA West Dam (Dam 4) used for stability modelling were consistent with those defined for the Stage 2 raise design. Only design zones 3 and 4 of the TMA North Dam were modelled; the results for zone 3 were used for zones 1 and 2, and the results for zone 4 were used for zone 5. This approach was followed due to the similarity of the subsurface conditions between the design zones and the feasibility level of this assessment. For the TMA West Dam (Dam 5), the critical section defined as a part of the TMA Cell 1 design was used for the stability assessment.

For each design zone, the downstream buttress geometry was determined to satisfy the stability criteria (see Section 18.7.3.4). A detailed discussion on the design methodology of the Stage 2 raise, on which the ultimate dam design is based, is described in the Stage 2 design reports (BGC, 21 December 2018; BGC, 16 May 2019; and BGC, 11 January 2020). A summary of the design zones and the stratigraphic conditions is presented below.

In general, the following stratigraphic foundation units were defined in stability analyses (listed from ground surface downwards):

- Peat.
- Brenna Formation (BRE) – glaciolacustrine clay and silt (BRE Silt and Clay).
- BRE – glaciolacustrine silty clay (BRE Silty Clay).
- Upper Whitemouth Lake till (Upper WML) – silty clay till.

- Lower Whitemouth Lake till (Lower WML) – silty clay till.
- Wylie Formation (WYL) – glaciolacustrine silt and clay.
- Whiteshell Till (WST) – granular till.

18.7.3.2 Geotechnical parameters

Geotechnical parameters are consistent with the Stage 2 Raise design (BGC, 21 December 2018; BGC, 16 May 2019; and BGC, 11 January 2020).

18.7.3.3 Porewater pressure conditions

Porewater pressure (PWP) assumptions for cohesive soils and fills (i.e., BRE, WML, WYL, clay core) were based on PWP data trends as measured from VWPs during construction of the TMA dams. Due to the low permeability and compressibility of the cohesive foundation soils and fills, construction-induced excess PWP is generated in response to the applied load from fill placement. The in situ PWP for cohesive soils includes the hydrostatic PWP and construction-induced excess PWP.

PWP conditions for cohesive and coarse-grained soils and fills assumed for the ultimate design of the TMA South Dam, TMA West Dam (Dam 4 and Dam 5), and TMA North Dam considered the following:

- The WML 'base case' piezometric line was updated based on VWP readings measured up to December 2019. These represent the PWP conditions after completion of the 2019 Stage 2 Raise.
- The WML incremental PWP ratio ($\overline{B}$), assumed for the increase of construction-induced excess PWP due to new buttress fill placement, was updated to $\overline{B} = 0.5$ for a total applied stress ≤ 130 kPa, $\overline{B} = 0.9$ for a total applied stress > 130 kPa, and $\overline{B} = 1$ below the dam crest (Table 18.3). These $\overline{B}$ values include considerations for 6 months of pore pressure dissipation between successive dam raises.
- BGC has assumed that the BRE Silty Clay has the same PWP conditions as the WML. This basis was adopted as the stratigraphic contact between the WML and overlying BRE Silty Clay is challenging to differentiate. This update only affects TMA South Dam design zone 3 where BRE Silty Clay is encountered.

Table 18.3 Summary of design $\overline{B}$ for the ultimate dam stability modelling

Stress condition	PWP incremental ratio ($\overline{B}$)
Downstream buttress applied total vertical stress ≤ 130 kPa (Approximately ≤ 6 m of fill)	0.5
Downstream buttress applied total vertical stress > 130 kPa (Approximately > 6 m of fill)	0.9
Below dam crest	1

Based on discussions with New Gold, BGC understands that buttresses may not encroach on the West Creek Diversion Channel at the toe of the TMA South Dam, or Biochemical Reactor 1 (BCR1) at the toe of the TMA West Dam (Dam 4). Wick Drains are proposed to be used to decrease the foundation pore pressure response and thus limit the required extents of buttress fills in these areas.

18.7.3.4 Results

This section summarizes the stability results for the ultimate dam stability assessment. The critical loading case was determined to be the EOC loading condition with effective stress analysis (ESA) strength parameters (residual shear strength for BRE Silty Clay, WML Silty Clay, and WYL). The location of the critical slip surfaces is controlled by the WML, due to the relatively weak shear strength and the development of relatively high construction-induced excess PWPs within this stratum. Stability results are summarized in Table 18.4 for the following loading conditions: EOC, pseudo-static, and post-earthquake.

Table 18.4 Summary of stability results

Loading condition	Construction configuration	Slope analyzed	Strength condition	Factor of safety results											
				TMA South Dam							TMA West Dam		TMA North Dam		WMP
				Design Zone 1	Design Zone 2	Design Zone 3 (without wick drains)	Design Zone 3 (with wick drains)	Design Zone 4 (without wick drains)	Design Zone 4 (with wick drains)	Design Zone 5	Dam 4	Dam 5	Design Zone 3	Design Zone 4	Dam 4
End of Construction	TMA South Dam, TMA West Dam 4, TMA West Dam 5, TMA North Dam at 379.0 m, WMP Dam 4 at 371.5 m	US [2]	ESA [4]	4.5	2.4	5.3	N/A [6]	4.2	N/A [6]	2.2	4.9	4.7	2.0	1.9	N/A [7]
			USA [5]	4.5	2.7	5.4	N/A [6]	4.5	N/A [6]	3.0	4.8	4.7	2.5	3.0	N/A [7]
		DS [3]	ESA	1.5	1.5	1.5	1.5	1.5	1.5	1.5	1.6	1.5	1.5	1.5	1.5
			USA	1.8	1.9	1.7	1.7	1.7	1.7	1.9	1.9	1.8	1.9	1.9	2.0
Pseudo-Static [1]	Refer to EOC Loading Condition	US	USA	2.1	1.7	1.9	N/A [6]	1.7	N/A [6]	2.6	2.4	2.5	1.6	2.2	N/A [7]
		DS	USA	1.2	1.2	1.1	1.1	1.1	1.2	1.2	1.2	1.1	1.2	1.3	1.6
Post-Earthquake	Refer to EOC Loading Condition	US	USA	3.3	4.0	3.7	N/A [6]	3.6	N/A [6]	3.8	3.5	2.8	2.3	1.9	N/A [7]
		DS	USA	1.5	1.6	1.5	1.4	1.4	1.5	1.6	1.5	1.3	1.4	1.3	2.0

Notes:
[1] A seismic coefficient (kh) equal to 0.02 was assumed for Pseudo-Static stability analyses (BGC, 21 December 2018).
[2] Upstream
[3] Downstream
[4] Effective stress analysis
[5] Undrained stress analysis
[6] N/A stands for not applicable. Wick drains do not affect upstream stability, so upstream stability results for TMA South Dam design Zones 3 and 4 are only listed once.
[7] N/A stands for not applicable. Upstream stability of WMP Dam 4 was not analyzed.

18.7.4 Material quantities

18.7.4.1 Assumptions

The LOM material quantities were estimated based on the following general assumptions:

- The quantities presented herein were estimated using neat line, straight measures, and compacted in place. Allowances for contingencies, over building, access ramps, or wastage (other than winter damage wastage) were not considered.
- The dam raise schedule shown in Table 18.1 and an ultimate dam crest elevation of 379.0 m.
- The TMA West Dam (Dam 4) cross sectional design was adjusted to reduce the Zone 1 core to half the original design width.
- Dental concrete and slush grout quantities were estimated based on a ratio of the actual dental concrete and slush grout quantities to actual bedrock cleaning area for 2019 construction at the TMA West Dam (Dam 4) and TMA North Dam.
- The estimated material quantities were based on simplified geometries for the ultimate dam configuration, do not consider stability analyses for the intermediate construction stages, and do not account for foundation consolidation settlement.
- Filter blankets were assumed to be fully constructed in 2021.
- For new footprint areas with limited site investigation data, ground conditions were assumed to be the same as the adjacent dam design zones and excavation depths were assumed based on 2019 construction experience.

18.7.4.2 LOM quantities

Table 18.5 provides a summary of the LOM quantities for the following structures:

- TMA South Dam
- TMA West Dam (Dams 4 and 5)
- TMA North Dam
- WMP Dam 4
- North Dam Seepage Collection System
- South Dam Seepage Collection System
- TMA Emergency Spillway

Table 18.5 Rainy River TMA LOM quantities

Material Type	Total	Units	Year					
			2020	2021	2022	2023	2024	2025
Zone 1	662,700	m^3	64,600	167,400	119,600	119,500	103,800	87,800
Zone 2	2,556,000	m^3	153,400	672,800	480,500	480,500	416,400	352,400
Zone 2A	212,190	m^3	18,590	54,200	38,700	38,700	33,600	28,400
Zone 3	8,007,500	m^3	517,500	2,234,000	1,460,000	1,460,000	1,265,300	1,070,700
Zone 3A	348,610	m^3	29,610	89,400	63,800	63,800	55,300	46,700
Zone 4	114,120	m^3	10,220	29,100	20,800	20,800	17,900	15,300
Zone 4A	101,700	m^3	7,500	94,200	-	-	-	-
Zone 5	215,400	m^3	15,700	128,100	19,900	17,400	18,400	15,900
Zone 6	-	m^3	-	-	-	-	-	-
Zone 7	56,900	m^3	800	54,980	280	280	280	280
Zone 10	11,850	m^3	1,500	2,070	2,070	2,070	2,070	2,070
Common Excavation	423,550	m^3	16,000	288,650	62,000	21,900	19,000	16,000
Bedrock Cleaning	8,710	m^2	1,100	2,060	1,480	1,480	1,360	1,230
Dental Concrete	1,200	m^3	495	198	141	141	122	103
Slush Grout	7,670	m^2	980	1,870	1,340	1,340	1,160	980
HDPE Geomembrane	1,780	m^2	530	250	250	250	250	250
Non-Woven Geotextile	63,200	m^2	1,400	60,030	480	450	430	410
Cementitious Composite Mat	660	m^2	210	90	90	90	90	90
Wick Drains	650,400	m	-	650,400	-	-	-	-

Note: The quantities presented herein are estimates only based on near line, straight measures, and compacted in place. Allowances for contingencies, over building, access ramps, or wastage (other than winter damage wastage) are not considered.

18.7.5 Discussion

To contain the proposed LOM tailings and water storage volumes, the Rainy River TMA perimeter dams will need to be raised to an ultimate elevation of 379.0 m. Preliminary stability modelling showed that structures and constraints downstream of the Rainy River TMA would be impacted by the proposed buttress extents. New Gold indicated that impacts to BCR1 and the West Creek Diversion Channel would need to be avoided, but that other downstream impacts were acceptable.

To limit impacts to the West Creek Diversion Channel, BGC assumed that wick drains would be installed to support more rapid dissipation of excess PWPs. To meet these requirements, the TMA South Dam design Zone 3 requires a 340 m long by 55 m wide area of wick drains that is contiguous with a 220 m long by 77 m wide area of wick drains in TMA South Dam design Zone 4. Both areas will require that wick drains be installed at a 1.5 m triangular spacing pre-drilled through existing rockfill.

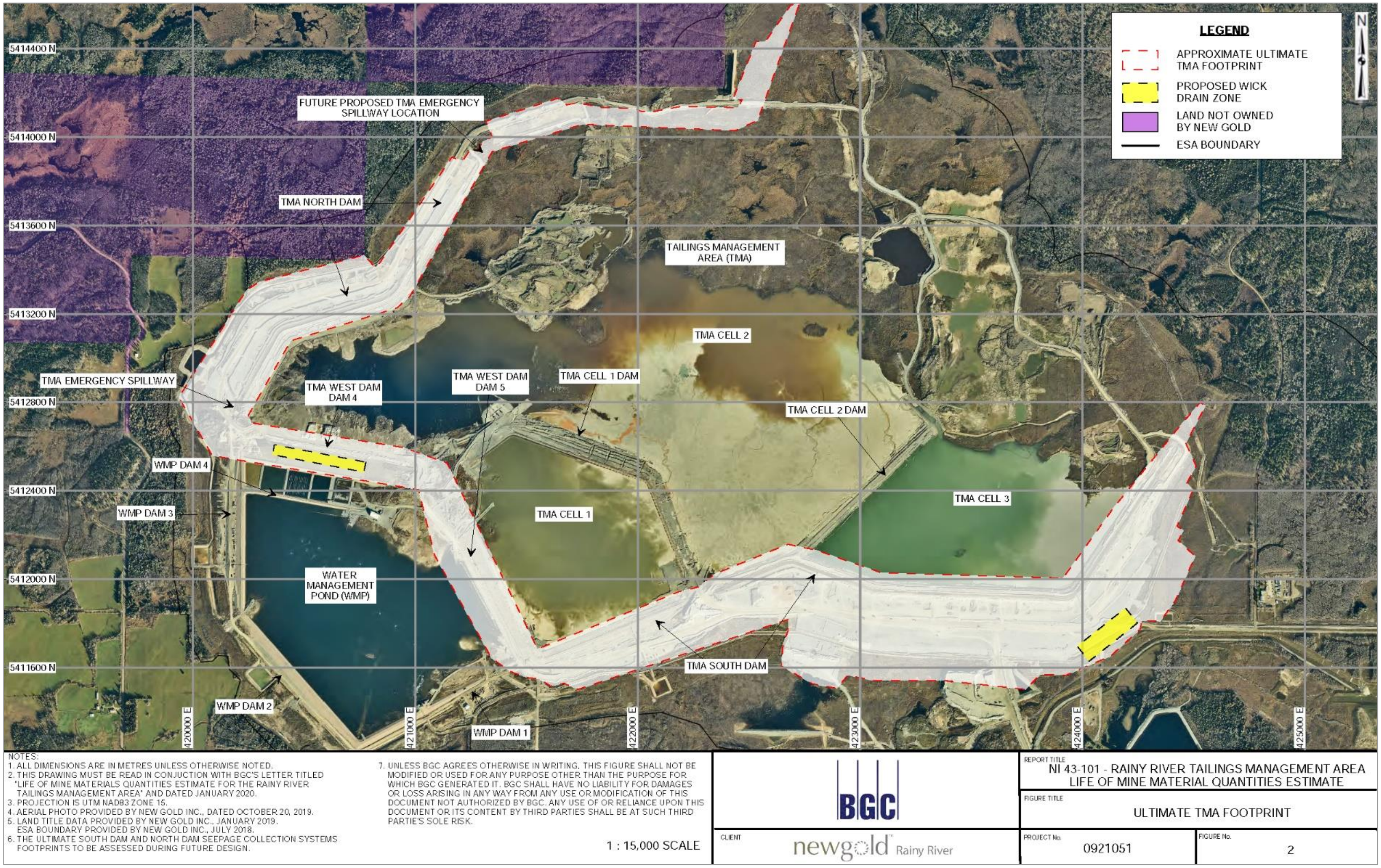
To limit impacts to BCR1, BGC assumed a modification to the cross-sectional design of the TMA West Dam (Dam 4) and installation of wick drains within the dam footprint. A 460 m long by 40 m wide area of wick drains installed at a 1.5 m triangular spacing will be required. The design modifications to the TMA West Dam (Dam 4) cross-section requires a reduction in the width of the Zone 1 core. This reduction in core width results in a limited ability to raise the TMA West Dam (Dam 4) to an elevation above 379.0 m if future mine plans require it.

The ultimate Rainy River TMA dam footprint is shown on Figure 18.4. Downstream structures and boundaries impacted by the ultimate TMA are shown on Figure 18.5. The following structures and boundaries will be impacted by the ultimate TMA:

- TMA North and TMA South Dam seepage collection systems.
- Environmental site assessment boundary and property boundary adjacent to the TMA North Dam.

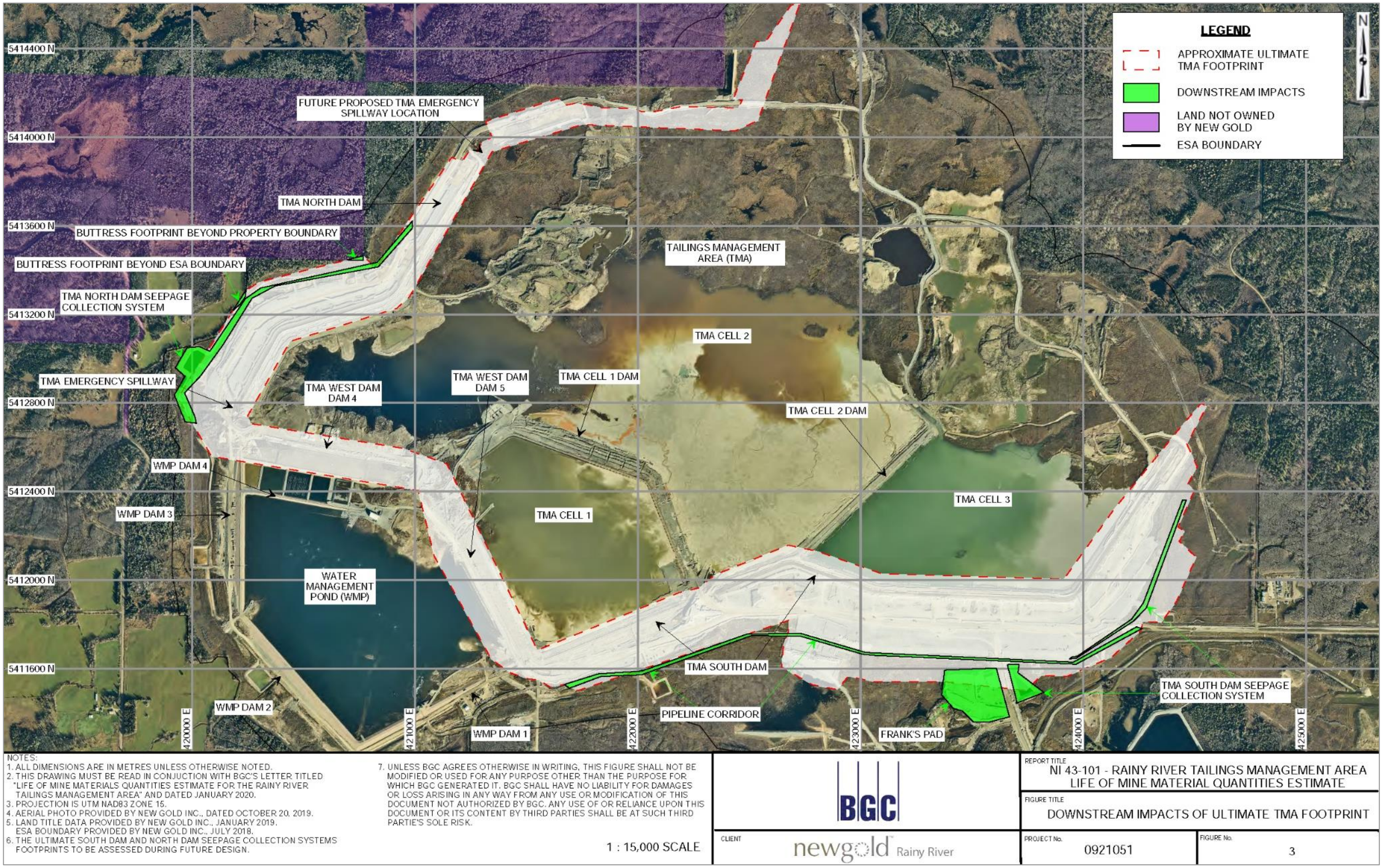
Material quantities for the reconstruction of the TMA North Dam and TMA South Dam seepage collection systems are provided in this document.

Figure 18.4 Ultimate TMA footprint



Source: BGC 2020.

Figure 18.5 Downstream impacts of ultimate tailings management footprint



Source: BGC 2020.

18.7.6 Uncertainties

The assessment of the LOM material quantities carries the following limitations:

- Stability modelling was performed only for the ultimate dam case, and annualized costs are based on a weighted average to dam raise height. Required annual quantities may differ from those presented.
- Consolidation modelling of the Rainy River tailings has not been completed, and projections of long-term tailings densities are uncertain.
- Long-term PWP dissipation assumptions were based on limited field scale and lab testing data and actual pore pressure dissipation may differ from the dissipations assumed in this document.
- The LOM material quantities were estimated without consideration of the closure design for the facility.
- The estimated volumes do not account for foundation consolidation settlement.

18.8 Integrated water treatment train

Discharge to the Pinewood River is currently targeted to a minimum 1:1 receiver to final effluent mixing ratio. The Pinewood River is required to have surpassed a minimum flow of 10,000 m^3/day before site water discharge begins for the year. Discharged water is also required to meet water quality guidelines in order to minimize or avoid impacts to the receiving environment. The total annual volume discharged through the treatment system is predicted to be between approximately 2.07 and 2.12 Mm^3 (Contango 2019). In order to meet both the discharge rate and quality requirements, Contango Strategies Ltd. (Contango) designed an integrated water treatment train that consists of a water treatment plant (WTP), a nitrification cell, and two (2) BCRs. The Lime WTP, Nitrification Cell, and first BCR #1 have been built and commissioned. They were operational from September to early November of 2019 and performed above expectations. The second BCR (BCR #2) was constructed in the winter of 2019 and will be commissioned in the spring of 2020.

Key sources of water being treated by this system are:

- Water from TMA and sources that pump to the TMA including sediment ponds 1, 2, and 3.
- Surface runoff and seepage from the TMA.
- Surface runoff and seepage from the TMA that have reported to the water discharge pond.

18.9 Mine rock and overburden stockpiles

Storage of mine rock (waste rock and LGO) and overburden waste is provided at two locations, the EMRS and the WMRS as shown in Figure 18.1.

The available stockpile storage capacity provided in addition to the August 2019 end-of month surveyed stockpile surface is summarized as follows:

- EMRS total storage volume – 70.9 Mm^3.
  - EMRS overburden storage volume – 11.1 Mm^3.
  - EMRS mine rock storage volume – 59.8 Mm^3.
    - About 24% (14.4 Mm^3) of the mine rock storage volume may be used for additional overburden storage, as required.
- WMRS total storage volume – 27.6 Mm^3.

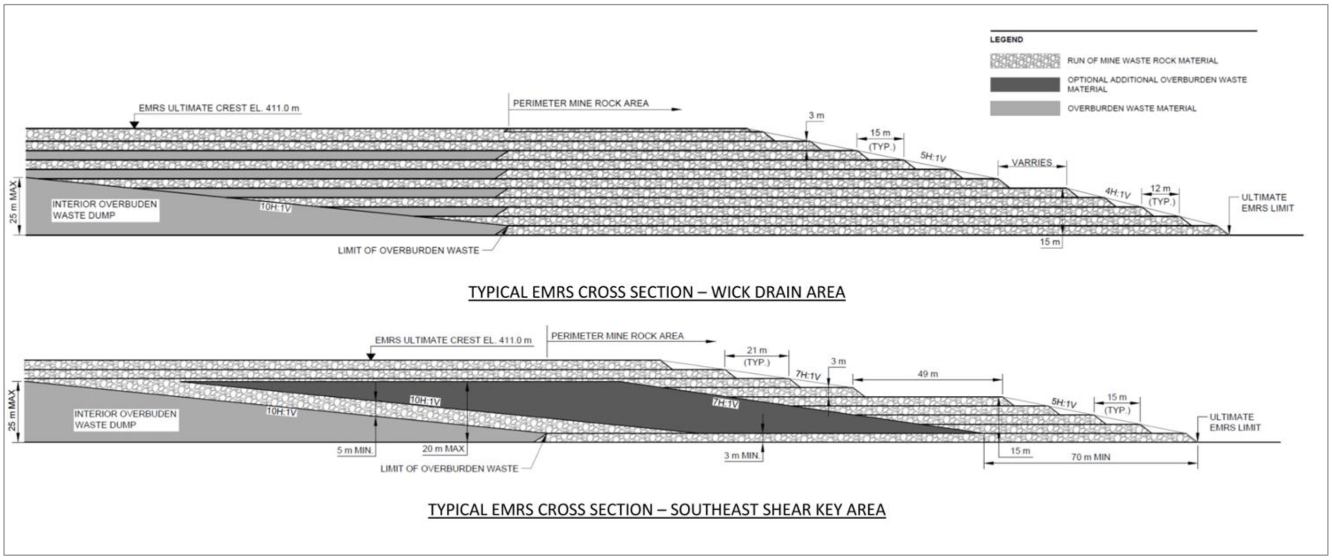
18.9.1 East Mine rock stockpile (EMRS)

Detailed design of the EMRS has been completed to accommodate a combination of overburden waste and PAG mine rock. The mine rock includes both waste rockfill and LGO. The low-grade ore stockpile (LGOS) is located at the west end of the EMRS. Overburden waste is currently stored internally in an overburden waste dump. The internal dump accommodates up to 25 m height of overburden. Additional overburden capacity is available through placement of 3 m thick lifts of overburden alternating with lifts of waste rock above the internal dump and within the south-east shear key area. Waste rock will be placed either internally above the overburden dump, or around the perimeter of the EMRS where it will serve to buttress the internal overburden dump.

The EMRS stockpile typical section is provided in Figure 18.6. The EMRS design crest elevation varies, from a minimum elevation of 402 m at the east and the west, to a maximum elevation of 411 m at the north and the south. The EMRS design perimeter slopes vary from 4H:1V (Horizontal: Vertical) to 5H:1V below 15 m stockpile height, and from 5H:1V to 7H:1V above 15 m stockpile height. Benches are typically 3 m in height and of similar width below and above the 15 m height bench. The internal overburden dump has a 10H:1V internal slope, and a maximum height of 25 m. Due to the presence of historically placed overburden fill near the EMRS perimeter at the south end of the LGO area, a 200 m wide intermediate bench is required at elevation 392 m. The design of the LGOS is currently being updated.

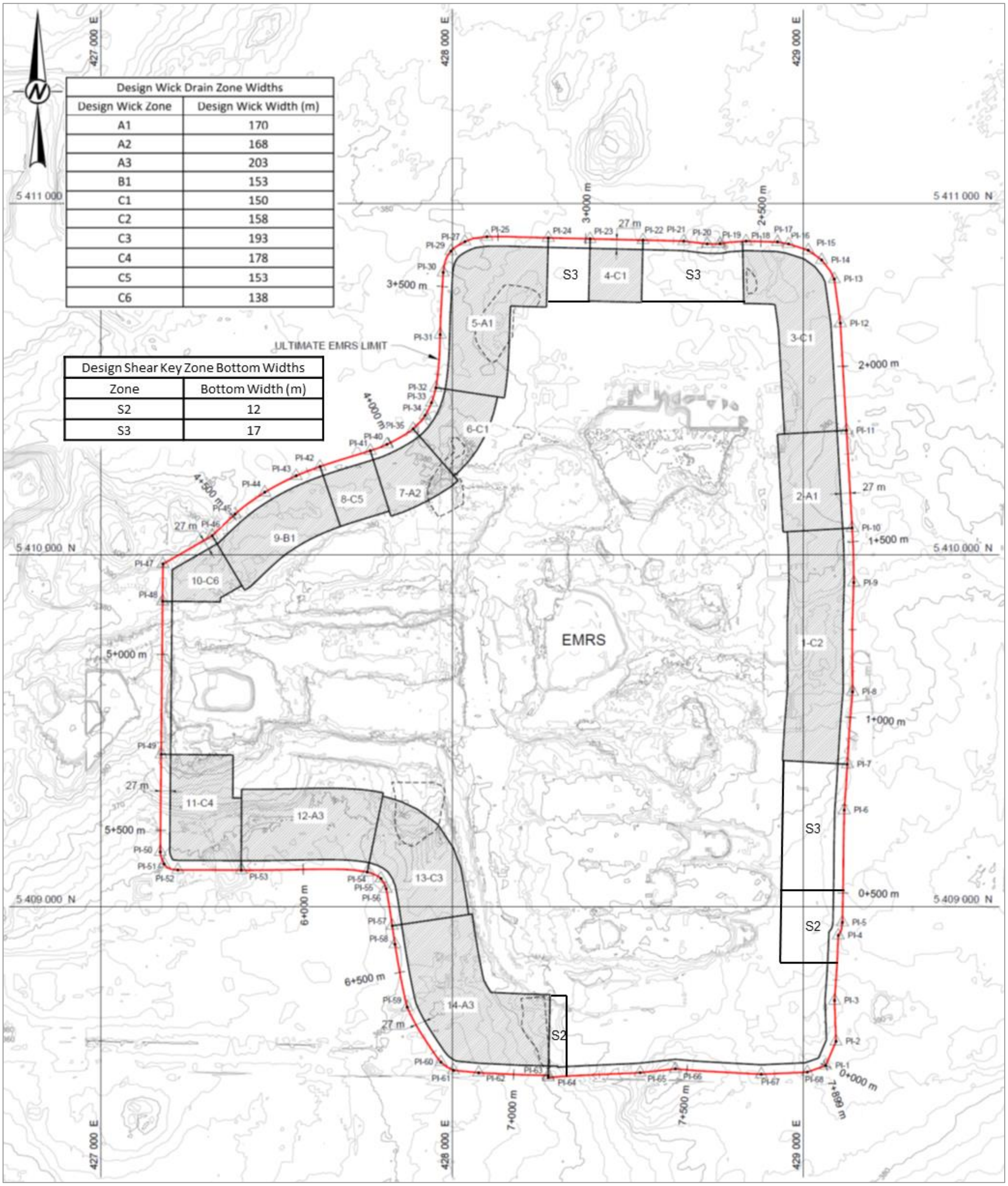
Ground improvement measures have been or are currently being implemented to improve the shearing resistance of the foundation of the EMRS perimeter. EMRS ground improvement measures include: constructing waste rock shear keys where foundation clay thicknesses are between 3 m to 8 m, and installing wick drains at 2 m spacing as shown in Figure 18.7, where more than 8 m of foundation clay is present. No ground improvement measures were required where the clay thickness is less than 3 m. The wick drain area width varies from 138 m to 203 m. A controlled rate of stockpile raising is required within the perimeter area to allow time for the dissipation of excess PWPs due to loading, and the associated consolidation strength gain. A maximum rate of raise of 9 m/year within the EMRS perimeter has been considered in the design.

Figure 18.6 EMRS typical cross sections



Source: Golder 2020.

Figure 18.7 EMRS wick drain layout plan



Design Wick Drain Zone Widths	
Design Wick Zone	Design Wick Width (m)
A1	170
A2	168
A3	203
B1	153
C1	150
C2	158
C3	193
C4	178
C5	153
C6	138

Design Shear Key Zone Bottom Widths	
Zone	Bottom Width (m)
S2	12
S3	17

Source: Golder 2020.

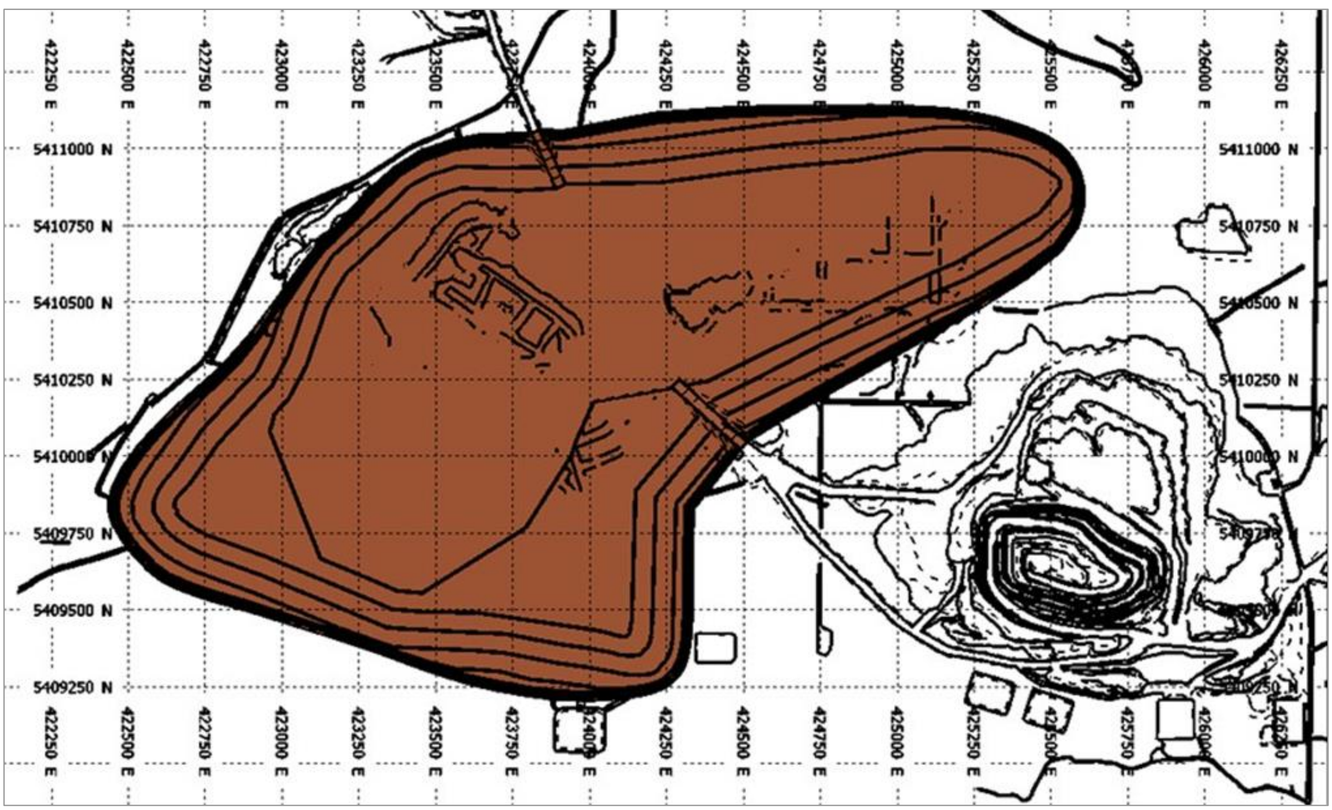
18.9.2 West Mine rock stockpile (WMRS)

A preliminary design of the WMRS has been completed to assess the maximum stockpile height and geometry which may be obtained without the requirement for foundation improvement measures, such as wick drain installation or construction of shear keys, (which have been implemented at the EMRS). The WMRS provides storage for a combination of NAG waste rock and overburden waste. The design provides storage for overburden waste internally in an overburden waste dump, with waste rock to be stockpiled around the perimeter, where it will function to buttress the overburden waste.

The WMRS geometry plan is provided in Figure 18.8. The WMRS has been designed to a maximum height of 20 m. The internal overburden dump will have a maximum height of 18 m. It will be maintained a minimum 180 m setback from the ultimate WMRS perimeter, and it will be constructed with 10H:1V side slopes. Waste rock will be stockpiled within this setback area and will form the exterior WMRS side slope geometry. The waste rock perimeter slope with vary from a 4H:1V slope for pile heights below 9 m. For pile heights above 9 m, the side slopes will vary between 13H:1V to 13.5H:1V, with a stabilizing bench provided at 9 m height with a width varying from 81 m to 86 m.

The design has been carried out for two typical design sections, considering a maximum foundation clay thickness of 26 m and 32 m. Further optimization will be carried out at the detailed design stage for specific areas where the foundation clay thickness is less than 26 m. Optimization is expected to somewhat increase the nominal capacity of the WMRS.

Figure 18.8 West mine rock stockpile plan view



Source: Golder 2020.

19 Market studies and contracts

19.1 Metal prices

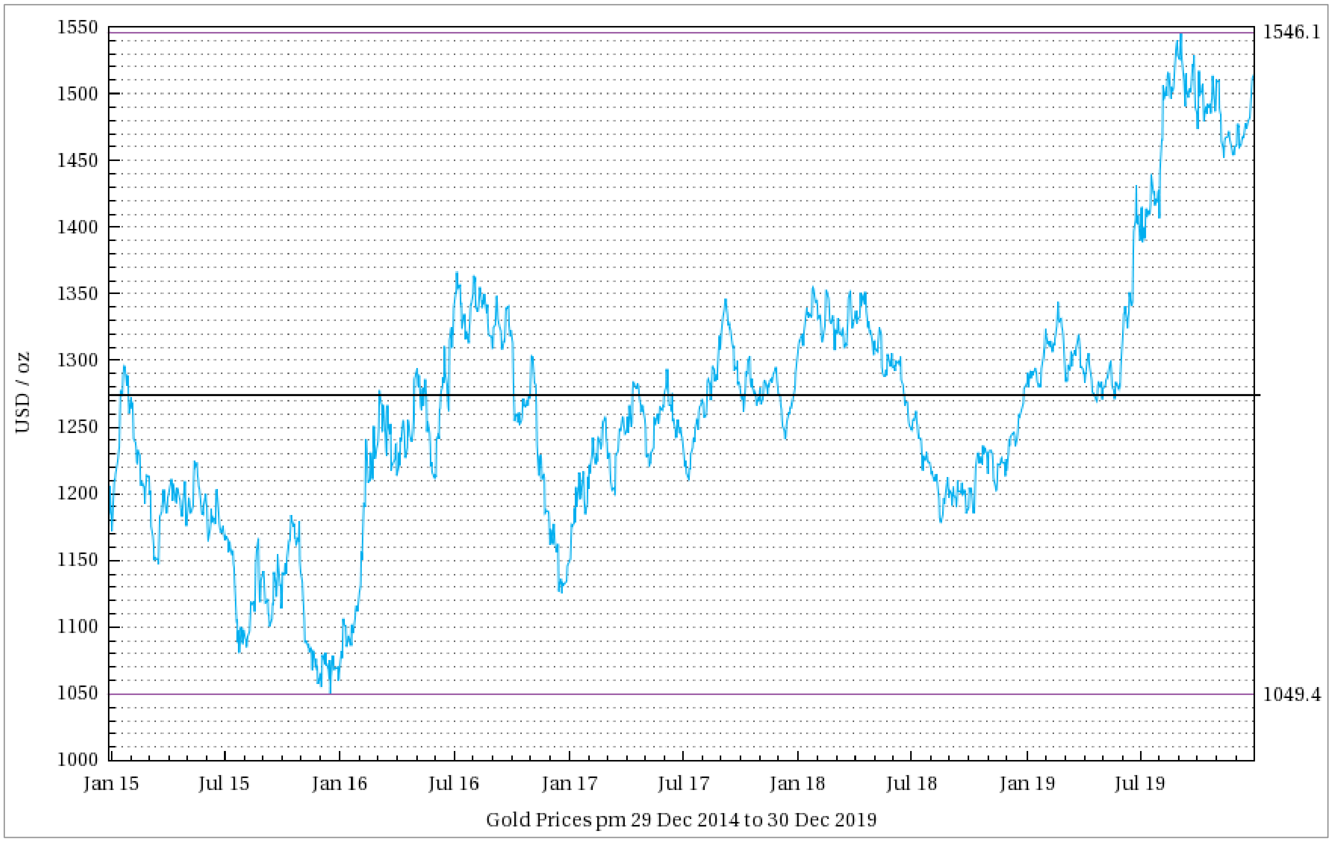
Project economics have been assessed using the following metal prices provided by New Gold:

- Gold price = $1,275.00/oz
- Silver price = $17.00/oz

AMC has reviewed these metal prices taking into consideration current market and recent historical prices, values used in other recent projects, and forecasts in the public domain.

According to the London Bullion Market Association (LBMA), the average daily PM Fix gold price for 2019 was $1,393 per troy ounce. The three-year and five-year rolling average prices through the end of December 2019 are $1,306. and $1,266 per troy ounce, respectively. The volatility of the gold price over these periods can be seen illustrated in Figure 19.1, and shows the low of $1,049 and the high of $1,546 during the five-year period.

Figure 19.1 LBMA PM Fix gold price (daily)



Source: lbma.org.uk 2020.

According to LBMA, the average daily silver price for 2019 was $16.21 per troy ounce. The three-year and five-year rolling average prices through the end of December 2019 are $16.32 and $16.35 per troy ounce, respectively.

The volatility of the silver price over these periods can be seen illustrated in Figure 19.2, and shows the low of $13.58 and the high of $20.71 during the five year period.

Figure 19.2 LBMA silver price (daily)

Source: lbma.org.uk 2020.

Based on the undertaken review, AMC considers the metal prices selected by New Gold to be reasonable, particularly given the recent appreciation in the gold metal price.

19.2 Markets

Gold and silver markets are mature global markets with reputable refiners located throughout the world.

Gold output from the Rainy River Mine operation is in the form of doré containing approximately 40% gold and 60% silver on average. Silver credits are received from the Refiner. The doré is shipped to either Asahi Refining Canada Ltd. in Brampton, ON or to the Royal Canadian Mint in Ottawa, ON. Transportation of the doré to either refinery is contracted out by the respective refineries. Responsibility for the doré changes hands at the gold room gate upon signed acceptance by the Refiner or its Transport Provider.

The mill at Rainy River is expected to produce an annual average of 284,000 ounces of gold and 435,000 ounces of silver in the form of doré, over the mine life.

To date, no penalty element charges have been incurred in the refining of the doré and none are currently expected to be incurred in the future.

The QP responsible for Section 19 has reviewed the refining contract payment conditions and finds that the terms, rates and charges are within industry norms and supports the assumptions presented in this Report.

19.3 Contracts

19.3.1 Gold price option contracts

In the second quarter of 2019, New Gold entered into gold price option contracts to cover approximately 50% of the company's consolidated gold production by purchasing put options at an average strike price of $1,300 per ounce and selling call options at an average strike price of $1,355 per ounce for 72,000 ounces of gold production between January 2020 and June 2020. Further, New Gold entered into gold price option contracts by purchasing put options at an average strike price of $1,300 per ounce and selling call options at an average strike price of $1,415 per ounce for 96,000 ounces of gold production between July 2020 and December 2020.

New Gold entered into these contracts at no premium and therefore incurred no investment costs upon initiation.

19.3.2 Metal streaming contracts

In 2015, New Gold entered into a $175M streaming transaction with Royal, a wholly-owned subsidiary of Royal Gold. Under the terms of the agreement, New Gold will deliver to Royal Gold 6.5% of gold production from Rainy River up to a total of 230,000 ounces of gold and then 3.25% of the mine's gold production thereafter. New Gold will also deliver to Royal Gold 60% of the mine's silver production to a maximum of 3.1 million ounces and then 30% of silver production thereafter.

In addition to the upfront deposit, Royal Gold will pay 25% of the average spot gold or silver price at the time each ounce of gold or silver is delivered under the stream. The difference between the spot price of metal and the cash received from Royal Gold will reduce the $175M deposit over the life of the mine. Upon expiry of the 40-year term of the agreement (which may be extended in certain circumstances), any balance of the $175M upfront deposit remaining unpaid will be refunded to Royal Gold.

19.3.3 Other contracts

As of 31 December 2019, the main contracts involved with the mine are:

- Refining: Asahi Refining Canada Ltd. and Royal Canadian Mint
- Electricity: Independent Electricity System Operator
- Fuel supply: Imperial Oil Ltd.
- TMA construction: Anokiigamig / Sigfusson-Northern
- Tire supply: Michelin North America, Inc.
- Explosive supply: Dyno Nobel Canada Inc.
- Lubricant supply / support: Anokiigamig / Petro
- Sodium cyanide supply: Chemours
- TMA engineering: BGC Engineering Ltd.
- Camp catering / housekeeping: In process of award

New Gold and Rainy River have policies and procedures in place for the letting of contracts. These are awarded based on pricing, supplier competencies and their ability to address where applicable, New Gold's commitments with respect to Aboriginal groups regarding business, employment, and other opportunities relating to the operation of the Rainy River Mine.

The QP is of the opinion that the contracts for Rainy River are competitive and are within industry norms.

20 Environmental studies, permitting, and social or community impact

20.1 Introduction

New Gold is committed to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against New Gold's Health, Safety, Environment, and Corporate Social Responsibility Policy approved by the Board of Directors on 25 July 2018.

The Environmental Department is adequately staffed, and has accountabilities including water resource management. New Gold conducts ambient air quality, surface water, and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

From 2018 to 2019, the Rainy River Mine has recorded eleven non-compliance related issues associated with an unauthorized effluent discharge, surface water quality exceedance, air quality particulate matter exceeding permit limits and noise level threshold exceedances. New Gold has reported all non-compliances to the appropriate regulatory agencies. No charges or fines were levied.

20.2 Environmental studies

New Gold is committed to complying with various permits, licenses, authorizations, approvals, and assessments to avoid and / or mitigate environmental impacts associated with the Rainy River Mine activities.

The following outlines past studies and ongoing monitoring that is programmed to continue during operations.

20.2.1 Meteorology and air quality

Climate information is suppled and correlated with the Environment Canada climate station at Barwick, ON and an on-site meteorological station located 5 km south-east of the process mill. The on-site station provides real-time site wind speed, wind direction, temperature, relative humidity, and precipitation data.

Air quality at the mine site is generally influenced by offsite meteorological conditions and by volatile organic emissions from insects, vegetation, and natural fires. The greatest impact on air quality is increased particulate matter generated from vehicle traffic, with less significant impacts from other site generated activities. Background air quality data for particulate matter is reported from two on site ambient air monitoring stations. Installation of a third air monitoring station located north of the TMA was recommended by Trinity Consultants. Installation is planned for fall of 2020.

20.2.2 Acoustics

Noise measurement studies were performed during the summers of 2018 and 2019 at locations in and surrounding the mine site. Significant noise sources included equipment associated with construction activities at and near the TMA. As of the end of 2019, construction of major infrastructure on the site has been completed. Future construction activities are principally limited to tailings dam raises. Noise measurement studies will continue to ensure that activities do not exceed applicable regulatory sound level criteria.

20.2.3 Geochemistry

As part of the environmental approvals process, New Gold was required to prepare and implement a geochemical monitoring plan to meet permit requirements. The purpose of the plan is to; assess the potential acid generating conditions of all mine rock materials extracted during the mine life, and to ensure proper segregation and management of these materials as per best industry practices for metal leaching / acid rock drainage sampling and characterizations. Since 2017 geochemical monitoring data has been collected and assessed as per requirements defined in the Geochemical Monitoring Plan (Wood 2016). As of the end of 2019, over 65,000 samples of mine rock materials from the open pit have been collected and analyzed. New Gold continues to meet all geochemical monitoring requirements stipulated under permitting conditions.

The Independent Technical Review Board, comprised of external consultants who report to New Gold corporate management, reviews updates on mine rock geochemistry and acid rock drainage studies to determine the effects on water quality and closure planning.

20.2.4 Hydrogeology

Under the conditions of the Environmental Compliance Approval permit updates to the hydrological model (i.e., the underground flow model) are to occur every 3 years during mine operations. Wood PLC (Wood) provided the first update in the 2017 monitoring report (Wood 2018). New Gold is currently constructing a transient groundwater model that will be reported on in the fall of 2020. A 1-D model is currently being constructed and this will be reported in early 2020 along with a design-based memorandum outlining the construction of the transient model.

Based on the 2018 assessment of groundwater level data by Wood, some private wells are now located within the south-eastern edge of the updated zone of influence, with one present within the existing drawdown cone. Site wide monitoring wells and surface water level stations will continue to monitor groundwater seepage rates and confirm any changes to the predicted drawdown cone from dewatering of the open pit.

20.2.5 Surface water

Sixteen surface water monitoring stations are located both upstream and downstream of current plant and mine facilities, positioned accordingly along the Pinewood River and major tributaries, to evaluate impact of the operations on local drainage systems. Comparisons of current and historical surface water sampling results with applicable permit benchmark limits and provincial objectives show that water quality is generally good. Parameter concentrations are generally below standards for the protection of aquatic life, except for iron and phosphorus, which commonly exceeded permitted limits. As well, pH, aluminium, cadmium, copper, cobalt, uranium, vanadium, zinc, and zirconium occasionally exceed permitted limits. Wood reported that site effluent discharge monitoring results were within final effluent limits and all acute toxicity tests results registered with passing grades.

Surface water quality is proactively monitored within the TMA, WMP, water discharge ponds, and sediment ponds for corporate due diligence and management purposes, but not for effluent quality and comparison to effluent limits, as there are no discharges.

20.2.6 Groundwater

Groundwater monitoring is regularly completed by site personnel at 45 monitoring wells and 3 VWPs arrays. Groundwater level measurements and field chemical parameters are manually recorded. Continuous groundwater level measurements using transducers are recorded for 15 monitoring wells as per permit requirements. Groundwater water chemistry sampling is completed 3 times per year as required by permit conditions. Water samples are analyzed for a complete suite of parameters and compared with permit limits. The 2018 groundwater quality monitoring results are very similar to 2016 baseline results, indicating minimal change in conditions. The 2019 results are not yet reported. Results from neighbouring private wells showed generally good water quality, with occasional exceedances of some parameters.

20.2.7 Aquatic resources

As part of the environmental monitoring program, annual performance monitoring for constructed fish habitat and fish tissue monitoring activities is conducted as outlined in Federal Regulations, Fishery Act Offset Plan Authorizations and Environmental Compliance Approval permit conditions.

Constructed fish habitat monitoring is comprised of fish community surveys, fish habitat surveys, and associated reporting. Fish community and fish habitat surveys were conducted at the West Creek Diversion and Clark Creek Diversion. Minnow Environmental Inc. (Minnow) conducted the study and reported excellent fish habitat for all pond and diversion channels. Fish abundance was high in most of the watercourses, except the Stockpile Pond and diversion channel, which encountered low water levels during April to October.

Fish tissue monitoring is comprised of two components, a large-bodied and small-bodied fish tissue survey.

The objective of the large-bodied fish tissue quality monitoring is to characterize concentrations of contaminants of potential concern in tissues of two sentinel sport fish species, northern pike and walleye, collected downstream of historical effluent discharge. Data collected by Minnow in 2018 indicate that the mine activities have not influenced concentrations of metals in large bodied sentinel fish species.

The objective of the small-bodied fish tissue survey is to quantify mercury concentrations in a single fish species. Fish tissue samples will be collected upstream, midstream and downstream of effluent discharge points. Data collected will be used to determine if mine activities influence small bodied fish.

20.2.8 Vegetation studies

Closure activities and reclamation require revegetation of all disturbed areas. In 2017, New Gold constructed two test stockpiles made from PAG rock, overlain by an engineer designed cover, as per the 2015 Closure Plan. The western stockpile was identified as a suitable location to establish a vegetation trial program. A field trial was designed and implemented in 2017. Planting of vegetation was completed in fall of 2019. Monitoring of the vegetation trial plot will continue until a mature vegetation community is established.

20.2.9 Wildlife

Bird monitoring studies executed by Wood suggests that the operations have not had an adverse effect on several of the most commonly occurring bird species. Data collected suggests that most birds are not avoiding areas associated with mine activities, other than where habitats have been directly impacted. Mine related construction activities appear to provide increased habitats for some birds. Some forest bird species may have been impacted by the mine activities and moved further away from mine activities to establish breeding territories. Some grassland and open country bird species show population increases. This increase may be attributed to grasslands habitat established by New Gold for species at risk habitat compensation.

In 2018 a wildlife exclusion fence was constructed around the TMA. The fence was designed to prevent access by wildlife and reptiles into the ponds. New Gold is responsible for monitoring the fence perimeter on a monthly basis and reporting any wildlife casualties to federal regulators.

20.2.10 Species at risk and critical habitat

The Species at Risk known to occur on the site are listed in Table 20.1. Until 2019, New Gold worked with the Ministry of Natural Resources and Forestry (MNRF) to meet all permitting requirements related to the Ontario Endangered Species Act (ESA). In 2019, the Ministry of Environment, Conservation and Parks (MECP) became the regulatory agency responsible for enforcing the Act and all permits issued under the Act.

A condition of the ESA permit required New Gold to establish overall benefit lands for two bird species (Bobolink and Eastern Whip-poor-will) to compensate for the effects of habitat loss by construction of the mine site. New Gold is responsible for management of these lands as grassland habitat. Condition of the ESA permit defines the requirements to satisfy the primary objectives of the monitoring program. These are: (a) quantifying any adverse effects to these species and (b) confirming that the overall benefit lands are providing compensatory habitats. Wood completed a comprehensive monitoring program in 2018. Results indicated there was a decrease in the number of bird species counted in areas south-east and west of the mine site boundary. However, the overall benefits lands established by New Gold were functioning as intended.

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

New Gold continues to work with the MECP to satisfying the terms and conditions of the ESA permit related to the Eastern Whip-poor-will habitant management plan.

20.2.11 Traditional knowledge and Traditional Land Use (social license)

Traditional Knowledge (TK) and Traditional Land Use (TLU) sessions were held with several Indigenous groups to discuss the inclusion of native species and traditional medicine plant species in closure plan vegetation studies. At the request of Indigenous groups, wild rice was planted in two water diversion ponds in 2017.

New Gold has undertaken a joint water quality monitoring and reporting program with the area First Nations. The program is funded by New Gold and employs First Nations environmental monitors as an integral part of the sitewide environmental management program.

20.2.12 Cultural heritage

During 2018, a stage 4 archaeological study was conducted on two inventoried and registered sites located within the boundary of the mine site infrastructure. Both sites were fully excavated and documented as per provincial archaeological assessment requirements. Preliminary reporting met the provincial standards and guidelines, resulting in the sites holding no further cultural heritage value or interest. Final reports documenting the mitigation of the sites will be available in 2020.

20.2.13 Overall environmental sensitivities

Increase in regulatory required monitoring and reporting during mine operations phase requires trained and competent staffing.

Guidance for meeting ESA permit conditions for approval of Eastern Whip-poor-will habitat management plan may be at risk with the change of provincial regulators responsible for enforcement of Ontario Species at Risk Act.

Contingency planning for loss of staff and cross-training of job responsibilities is being implemented and tracked.

Delays in obtaining approvals for amendments to permits, authorizations, and closure plan changes is substantial and may affect the ability to remain compliant.

20.3 Project permitting

The mine has received all the permits and authorizations needed to construct major infrastructure and operate. Active permits and authorizations are listed in Table 20.2.

Table 20.2 Permit list

Title	Permit type
Aggregate Dewatering Out Crop 3 and Roen Pit	Permit to Take Water
Aggregate Dewatering - Tait Quarry	Permit to Take Water
Mine Dewatering	Permit to Take Water
SAR Eastern Whip-poor-will and Bobolink	Endangered Species Act Permit
Air and Noise	Environmental Compliance Approval
Sewage Works	Environmental Compliance Approval
Fisheries Act 35(2)(b) Authorization (Offset Plan)	Authorization
Effluent Mixing Structure & Hydrology Gauge	Work Permit – Letter of authority
Aggregate Resources – Tait Quarry	Aggregate Resources License
Aggregate Resources – Laydown 4 Quarry	Aggregate Resources License
Fish Collection Permits	Authorization
Wildlife Scientific Collectors Authorization	Authorization
Authorization for Wildlife Interference	Authorization
Nuclear Substance and Radiation Device	Nuclear Radiation License
Electricity Wholesaler	License
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
Occupancy	Municipal Permit

New Gold has completed the first phase of development of an Environmental Management System (EMS) that will manage permits, licenses, and environmental commitments at the Mine.

20.4 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. As of December 2019, the mine work force was 22% Indigenous.

Engagement of neighbours, Indigenous communities, local municipalities and employees remains a priority for New Gold. Tours of the site and facilities are provided to the general public, business partners, school groups, neighbours, Indigenous community members and to families of employees. New Gold annually distributes two newsletters throughout the local communities.

New Gold is committed to providing opportunities to Indigenous communities through various existing partnerships with Indigenous groups. The company continues to engage through participation and implementation committee meetings, site visits, business development assistance, and participation in community events.

20.5 Mine closure

The Rainy River Closure Plan, dated 22 January 2015, was filed by the MENDM on 23 February 2015. A Comprehensive Closure Plan Amendment was prepared in support of the Rainy River Project transition to early production. It was submitted to the MENDM in October of 2017 for comments. Further Comprehensive Closure Plan Amendment comments were received from MENDM, MNRF, and MECP on 21 August 2018. In December 2019, New Gold continued the consultation process and submitted responses to a second round of comments received from government agencies.

The Closure Plan has included consultation with agencies, the Aboriginal Community(s) and the public. These consultations will continue through to closure and beyond.

A groundwater monitoring network, developed in 2015 and 2016, will continue to be used to monitor conditions through operations phases and into reclamation and closure. Additional environmental monitoring and water management programs will be established near the end of the operations phase and continue into closure.

The cost estimate for implementing project closure in the Environmental Assessment (EA) was estimated to be $107M, 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 has prepared and submitted an update(s) to the reclamation costs, with the Comprehensive Closure Plan Amendment. The cost update(s) reflects new knowledge, progressive reclamation, project transition changes related to early production, as well as any changes to costing due to inflation. The financial assurance provided to MENDM 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 financial assurance obligation / commitment is $76M based on current disturbance as at 31 December 2019.

21 Capital and operating costs

Capital and operating costs have been estimated by New Gold throughout their 2020 Budget and LOM planning process and have been reviewed by AMC. All costs presented in this section are presented in constant Q1-2020 US$, with no inflation or escalation factors considered. Where applicable a foreign exchange rate of C$:US$ of 1.3 was utilized.

21.1 Capital costs

Capital costs have been estimated based on existing work contracts, manufacturer / provider quotes or recent actual construction / installation costs. Where none of the preceding were available, budgetary estimates were made by New Gold based on experience.

21.1.1 Summary

Total LOM capital costs are estimated to total $642M as summarized in Table 21.1. This excludes $107M in funds identified for progressive and final closure. Details for each category follow in this section.

Table 21.1 Capital costs summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Open pit	$(000)	60,909	62,758	94,536	53,945	22,246	506	5,584	606	-	301,090
Underground	$(000)	3,846	-	16,599	16,881	19,579	52,185	10,857	333	-	120,281
Process and TMA	$(000)	35,483	37,152	29,991	31,159	30,260	21,721	-	-	-	185,765
Infrastructure + other	$(000)	22,708	6,637	1,923	1,923	1,923	-	-	-	-	35,115
Grand total	**$(000)**	**122,946**	**106,547**	**143,050**	**103,908**	**74,008**	**74,411**	**16,441**	**938**	**-**	**642,250**
Project capital	$(000)	3,846	-	10,072	3,390	11,149	27,695	-	-	-	56,152
Sustaining capital	$(000)	119,100	106,547	132,978	100,518	62,859	46,716	16,441	938	-	586,098
Grand total	**$(000)**	**122,946**	**106,547**	**143,050**	**103,908**	**74,008**	**74,411**	**16,441**	**938**	**-**	**642,250**
Reclamation / closure [1]	$(000)	1,178	3,051	3,036	3,044	3,028	2,974	3,013	4,655	82,953	106,932

Note: Totals may not compute exactly due to rounding.
[1] The 2028 amount includes $66M of final closure and post-closure costs to be expended after 2028.

21.1.2 Open pit capital cost estimate

The open pit capital cost is estimated to total $301M as summarized in Table 21.2.

Table 21.2 Open pit capital costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Mine equipment	$(000)	738	-	-	-	-	-	-	-	-	738
Parts & components	$(000)	21,624	26,716	27,798	19,109	22,079	506	5,584	606	-	124,020
Mobile maintenance	$(000)	2,498	1,535	-	-	-	-	-	-	-	4,033
Stripping	$(000)	32,843	29,027	62,652	34,720	-	-	-	-	-	159,242
Overburden	$(000)	1,371	4,777	3,650	-	-	-	-	-	-	9,799
Other	$(000)	1,835	704	437	116	167	-	-	-	-	3,259
Grand total	**$(000)**	**60,909**	**62,758**	**94,536**	**53,945**	**22,246**	**506**	**5,584**	**606**	**-**	**301,090**

Note: Totals may not compute exactly due to rounding.

Principal open pit capital costs include, but are not limited to the following principal items:

- Principal parts and component repairs and replacements that are contemplated for sustaining capital including: engines, wheel motors, large compressors, buckets, under-carriages, etc.
- Mobile maintenance capital for new and / or replacement equipment including, but not limited to: 12-tonne telehandlers, vacuum / steam truck, open pit fuel / lube truck, etc.
- Capitalized / deferred stripping costs associated with the extraction of 68 Mt of waste.
- Overburden costs to profile current and future excavated slopes in overburden to the required design criteria.

The capital cost estimate is considered to be appropriate for the open pit operation.

21.1.3 Underground capital cost estimate

The underground LOM capital cost is estimated to total $120M, inclusive of contingency, with $56M in project capital and $64M in sustaining capital, as summarized in Table 21.3.

The project capital is composed of mobile equipment, stationary equipment, infrastructure, DDH grouting program, and portal construction. The sustaining capital is composed of sustaining cost for stationary equipment, infrastructure, capital development, owner's costs, and indirect project costs.

First and second year infrastructure cost is classified as project capital (non-sustaining). For simplification, when ore is realized, all infrastructure cost is, thereafter, classified as sustaining capex.

The development cost for each zone in the first year is classified as project capital (non-sustaining). Thereafter, development after the first year is sustaining capex.

Capital spending of $3.8M for the Intrepid Zone decline has been brought forward from 2023 to 2020 for the provision of an orebody investigation project. This capital spending is classified as project capital for the year 2020. This early spending will provide options for the company to revise, improve and update the mine plan.

Table 21.3 Underground capital costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Mobile equipment	$(000)	-	-	167	84	84	84	-	-	-	418
Stationary equipment and infrastructure	$(000)	-	-	6,466	2,756	2,794	6,322	93	17	-	18,448
Stationary equipment and infrastructure - sustaining costs	$(000)	-	-	45	166	250	465	543	271	-	1,739
Capital development	$(000)	3,846	-	6,527	11,139	13,326	37,634	8,992	-	-	81,463
DDH grouting program	$(000)	-	-	100	100	100	100	-	-	-	400
Portal construction	$(000)	-	-	629	-	515	1,031	-	-	-	2,175
Owner's costs	$(000)	-	-	56	-	-	-	-	-	-	56
Indirects	$(000)	-	-	683	284	307	679	9	2	-	1,963
Contingency	$(000)	-	-	1,928	2,353	2,204	5,871	1,220	43	-	13,618
Grand total	**$(000)**	**3,846**	**-**	**16,599**	**16,881**	**19,579**	**52,185**	**10,857**	**333**	**-**	**120,281**
Project capital	$(000)	3,846	-	10,072	3,390	11,149	27,695	-	-	-	56,152
Sustaining	$(000)	-	-	6,527	13,491	8,430	24,490	10,857	333	-	64,129
Grand total	**$(000)**	**3,846**	**-**	**16,599**	**16,881**	**19,579**	**52,185**	**10,857**	**333**	**-**	**120,281**

Note: Totals may not compute exactly due to rounding.

The capital cost estimate has been prepared based upon quotations for mobile and stationary equipment, and for contracted development throughout the mine life. Selected costs from previous 2013 and 2016 estimates that were not requoted, such as explosives, cement, and wages, were escalated for inflation.

Mobile equipment purchases are minimal and consist of five underground pickup trucks for the mine technical department. There is no cost for rebuilds and replacement for these pickup trucks throughout the LOM, due to the short mine life and their low utilization by the mine technical group. The contractor will provide all the necessary equipment and maintenance for all development and production work and will be responsible for operation and maintenance of all underground infrastructure. Contractor unit prices are all-inclusive, except for power, diesel fuel, explosives, and camp accommodation supplied by New Gold.

Stationary equipment and infrastructure includes primary exhaust fans, portal fans and heaters, regulators, refuge chambers, stench system, auxiliary fans, electrical infrastructure within 100 m of each portal, underground electrical power distribution, dewatering pumps, service water, portal structures, fuel stations, highwall preparation, communication systems, backfill plant and pipes, remote mucking stands, survey equipment, and office supplies and computers.

Capital development includes the main decline ramps, level accesses, remuck bays, sumps, electrical bays, ventilation raises, and ventilation accesses. The capital mine development unit rate costs are summarized in Table 21.4. The capital lateral development unit rate is based on a 5 m x 5 m drift. In some sections where larger drifts are required for ventilation purposes, an equivalent length was used to calculate the total lateral development cost. The capital raise development is based on Alimak raising with dimensions of 3 m x 3 m. In some sections where larger raises are required for ventilation purposes, an equivalent length was used to calculate the total vertical development cost. The raising cost includes set-up, drilling, blasting and mucking, ground supporting, steel manway supply, manway installation, teardown, fuel and explosives. Stope development and ore drift development are considered operating costs.

Table 21.4 Capital development costs

Description	$/m*
Capital lateral development	6,287
Capital raise development	6,730

Notes:
*Average LOM cost per development metre.
*Unit rates are all-inclusive (includes fuel and explosives).

The indirect costs include costs for freight, spares, and commissioning. The owner's costs include field engineering work.

The contingency allowance is based upon:

- 15% of the costs for mobile equipment, mobile equipment replacement, and rebuilds.
- 15% of the fixed equipment costs.
- 15% of the fixed equipment sustaining costs.
- 12.5% of contractor underground costs related to the capital development of the lateral drifts and vertical raises.
- 7.5% of construction and contractor costs related to the portal construction, and owner's cost.
- 7.5% of indirect costs.

The overall contingency in the estimate is 12.8% of the capital costs.

Comparing with the NI 43-101 Technical Report from 25 July 2018, the LOM underground capital cost has decreased by 63%. This is due to several factors as described below:

- Mobile equipment along with rebuilds and replacements have been significantly reduced, since the contractor will be bringing in its own equipment for the underground operations throughout the LOM. Those costs are captured in the contractor's unit rates.
- Reduction in the stationary equipment and infrastructure costs along with related sustaining cost after generation of the new mine plan, which has a reduction in development and ore, thus a shorter mine life.
- Overall reduction in capital development cost due to the reduction of capital development metres in the new mine plan.
  - Reduction in lateral capital development metres from 32.4 km to 11.3 km.
  - Reduction in vertical development metres from 2.4 km to 1.0 km.
- Capitalized operating cost is removed, since all operating expenditures are covered in the operating costs.
- The Intrepid Zone portal has already been established and the decline ramp has been driven for 166 m.
- The underground shop has been removed due to the short mine life. All maintenance work will be done on surface.

The capital cost comparison summary is shown in Table 21.5.

Table 21.5 Underground capital costs comparison (2018 vs 2020)

Description	Unit	July 2018 Tech Report [1]	Q1 2020 Tech Report
Mobile equipment [2]	$(000)	29,145	418
-Rebuild / replace	$(000)	31,771	-
Stationary equipment	$(000)	26,845	-
Stationary equipment and infrastructure [3]	$(000)	-	18,448
Stationary equipment and infrastructure - sustaining costs	$(000)	17,001	1,739
Capital development	$(000)	106,008	81,463
Capitalized operating cost	$(000)	7,580	-
Infrastructure	$(000)	4,958	-
Shop, CRF and services	$(000)	5,156	-
DDH grouting program [4]	$(000)	-	400
Portal [5]	$(000)	48,839	-
Portal construction	$(000)	-	2,175
Owner’s costs	$(000)	13,158	56
Indirects	$(000)	7,959	1,963
Contingency	$(000)	28,753	13,618
Grand total	**$(000)**	**327,172**	**120,281**
Project capital	$(000)	56,423	56,152
Sustaining	$(000)	270,750	64,129
Grand total	**$(000)**	**327,172**	**120,281**

Notes: Totals may not compute exactly due to rounding.
[1] Costs from the 2018 Technical Report have been converted into US dollars for comparison purposes (exchange rate of US$1 = C$1.30).
[2] Reduction in mobile equipment due to contractor mining now being employed.
[3] Stationary equipment and infrastructure are grouped together in this Q1-2020 Technical Report.
[4] DDH grouting program was previously under ‘Infrastructure’ in the 2018 Technical Report and it is now in its own category in this Q1-2020 Technical Report.
[5] Portal cost included a portion of the costs from mobile equipment, stationary equipment, capital development, owner’s costs, project indirect costs, and contingency in the 2018 Technical Report.

21.1.4 Process and tailings management area capital cost estimate

The process and tailings management area capital costs are estimated to total $186M as summarized in Table 21.6.

Table 21.6 Process and tailings management area sustaining capital costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Process equipment	$(000)	1,849	462	-	-	-	-	-	-	-	2,311
Parts & components	$(000)	350	-	-	-	-	-	-	-	-	350
Mobile maintenance	$(000)	-	-	-	-	-	-	-	-	-	-
TMA	$(000)	29,735	36,340	29,901	31,004	30,260	21,721	-	-	-	178,960
Other	$(000)	3,549	350	90	155	-	-	-	-	-	4,144
Grand total	**$(000)**	**35,483**	**37,152**	**29,991**	**31,159**	**30,260**	**21,721**	**-**	**-**	**-**	**185,765**

Note: Totals may not compute exactly due to rounding.

Principal process capital costs include, but are not limited to the following principal items:

- Process equipment required for planned mill upgrades. Principal upgrades include two 700 hp VFD pumps for the reclaim water system, online vibration monitoring systems for the mill and the installation of a self-cleaning magnet and chute for CV10 principally to monitor ores for potential metal coming out of the underground.
- Parts and components represent the purchase of a spare pinon that can be utilized for either the SAG or ball mill.
- TMA represents the expansion of the current tailings facility to accommodate the tailings generated from the processing of an additional 78 Mt of ore in the current mine plan.
- Other capital items include the construction of a new mill dry as well as various smaller items.

The capital cost estimate is considered to be appropriate for process functions.

21.1.5 Infrastructure and other capital cost estimate

The infrastructure and other capital costs are estimated to total $35M, as summarized in Table 21.7.

Table 21.7 Infrastructure and other capital cost estimate

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Maintenance facilities	$(000)	7,015	1,240	-	-	-	-	-	-	-	8,254
Waste storage facilities	$(000)	12,344	2,820	-	-	-	-	-	-	-	15,164
Surface facilities	$(000)	2,537	-	-	-	-	-	-	-	-	2,537
Miscellaneous	$(000)	399	154	-	-	-	-	-	-	-	552
Other	$(000)	413	2,424	1,923	1,923	1,923	-	-	-	-	8,607
Grand total	**$(000)**	**22,708**	**6,637**	**1,923**	**1,923**	**1,923**	**-**	**-**	**-**	**-**	**35,115**

Note: Totals may not compute exactly due to rounding.

Principal infrastructure and other capital costs include, but are not limited to the following principal items:

- Maintenance facilities include the construction of the following installations: Truck shop 2, optimization of the wash bay, hydro-carbon management building, laydown 7 enhancements, lubricant storage building, and mine operations fuel bay. In addition, it includes the purchase of a 50 t crane for installation in the existing truck shop 1.
- Waste storage facilities costs related to access roads, overburden removal and wick drain installations.
- Surface facilities represent the costs related to renovations of the current camp accommodations and the construction of a new warehouse.
- Other represents principally an allowance of $1,923,000/year from 2021 through 2024 for miscellaneous capital items which have not been identified in the current overall site budget.

The capital cost estimate is considered to be appropriate for necessary infrastructure.

21.1.6 Reclamation / closure

The reclamation / final closure capital cost is estimated to total $107M over the mine life and the current financial assurance is $76M based on current disturbance as at 31 December 2019. The annual expenditures are presented in Table 21.8.

Table 21.8 Reclamation / closure capital costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028+	LOM total
Progressive reclamation	$(000)	1,178	3,051	3,036	3,044	3,028	2,974	3,013	4,655		23,979
Final closure [1]	$(000)									82,953	82,953
Grand total	**$(000)**	**1,178**	**3,051**	**3,036**	**3,044**	**3,028**	**2,974**	**3,013**	**4,655**	**82,953**	**106,932**

[1] The 2028 amount includes $66M of final closure and post-closure costs to be expended after 2028.

The capital cost estimate is considered to be appropriate for reclamation and closure.

21.2 Operating costs

Operating costs have been estimated using first principle estimates, where applicable, based upon the annual mine production schedule, equipment availability, utilization and equipment productivities. Principal reagent costs and contractor rates utilized have been based on current contract prices and agreements where available.

21.2.1 Summary

A summary of the estimated LOM operating costs is shown in Table 21.9. Estimated unit operating costs, plus the LOM average, are shown in Table 21.10.

Table 21.9 Operating cost summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Open pit	$(000)	113,445	108,811	66,554	95,214	88,850	29,725	23,448	21,286	3,589	550,922
Underground	$(000)	-	-	4,146	12,061	27,094	55,861	62,162	40,816	5,188	207,328
Process	$(000)	69,435	71,322	70,139	70,316	69,680	68,612	68,306	68,173	9,028	565,010
G&A	$(000)	32,430	33,273	29,037	28,547	25,920	21,290	21,076	19,547	2,222	213,342
Grand total	**$(000)**	**215,310**	**213,406**	**169,875**	**206,138**	**211,544**	**175,489**	**174,992**	**149,822**	**20,026**	**1,536,602**

Note: Totals may not compute exactly due to rounding.

Table 21.10 Unit operating cost summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg.
Open pit	$/t moved	2.36	2.27	2.24	2.54	2.73	3.21	2.83	2.55	2.33	2.47
Open pit	$/t mined	2.74	2.57	2.84	2.90	3.14	15.40	-	-	-	3.23
Underground	$/t ore mined	-	-	80.75	84.30	50.45	57.11	54.08	36.42	44.72	50.62
Mining	$/t milled	12.25	11.33	7.40	11.27	12.12	9.03	9.09	6.58	5.05	9.77
Process	$/t milled	7.50	7.43	7.34	7.39	7.29	7.24	7.25	7.22	5.19	7.28
G&A	$/t milled	3.50	3.46	3.04	3.00	2.71	2.25	2.24	2.07	1.28	2.75
Grand total	**$/t milled**	**23.25**	**22.22**	**17.79**	**21.65**	**22.12**	**18.52**	**18.57**	**15.87**	**11.52**	**19.81**

Note: Totals may not compute exactly due to rounding.

21.2.2 Mine operating costs

21.2.2.1 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 21.11. Estimated open pit unit operating costs, plus the LOM average, are shown in Table 21.12. Costs were estimated by New Gold and are based on 2020 budget / LOM cost estimates.

Table 21.11 Open pit operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Administration	$(000)	5,122	4,922	3,118	3,989	3,664	1,107	689	689	158	23,459
Maintenance	$(000)	2,545	2,530	1,566	2,209	2,094	1,177	1,105	952	10	14,188
Engineering	$(000)	2,073	1,852	1,210	1,717	1,451	283	-	-	-	8,585
Geology	$(000)	4,607	4,522	2,728	3,756	3,432	405	-	-	-	19,450
Drilling	$(000)	13,863	12,162	7,248	9,355	7,708	331	-	-	-	50,666
Blasting	$(000)	12,870	12,270	8,142	11,353	9,498	2,110	-	-	-	56,242
Loading	$(000)	10,608	11,492	7,524	10,623	9,371	3,759	3,409	3,383	387	60,555
Hauling	$(000)	28,876	26,833	14,096	22,747	21,026	5,729	5,162	5,597	850	130,918
Floors / roads / dumps	$(000)	953	745	471	674	961	531	15	15	-	4,366
Dewatering	$(000)	1,678	1,842	1,175	1,799	2,566	999	923	923	216	12,121
Support services	$(000)	14,466	14,059	9,107	12,519	10,224	3,371	2,307	1,573	284	67,911
Mobile maint.	$(000)	14,939	14,745	9,573	13,628	16,236	9,605	9,519	7,836	1,683	97,765
Light vehicles	$(000)	846	837	595	845	620	318	318	318	-	4,696
Grand total [1]	**$(000)**	**113,445**	**108,811**	**66,554**	**95,214**	**88,850**	**29,725**	**23,448**	**21,286**	**3,589**	**550,922**

Notes:
- [1] Includes maintenance costs but excludes sustaining capital for parts and component repair and / or replacement.
- Totals may not compute exactly due to rounding.

Table 21.12 Open pit unit operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg.
Administration	$/t moved	0.11	0.10	0.11	0.11	0.11	0.12	0.08	0.08	0.10	0.11
Maintenance	$/t moved	0.05	0.05	0.05	0.06	0.06	0.13	0.13	0.11	0.01	0.06
Engineering	$/t moved	0.04	0.04	0.04	0.05	0.04	0.03	-	-	-	0.04
Geology	$/t moved	0.10	0.09	0.09	0.10	0.11	0.04	-	-	-	0.09
Drilling	$/t moved	0.29	0.25	0.24	0.25	0.24	0.04	-	-	-	0.23
Blasting	$/t moved	0.27	0.26	0.27	0.30	0.29	0.23	-	-	-	0.25
Loading	$/t moved	0.22	0.24	0.25	0.28	0.29	0.41	0.41	0.41	0.25	0.27
Hauling	$/t moved	0.60	0.56	0.48	0.61	0.65	0.62	0.62	0.67	0.55	0.59
Roads	$/t moved	0.02	0.02	0.02	0.02	0.03	0.06	0.00	0.00	-	0.02
Dewatering	$/t moved	0.03	0.04	0.04	0.05	0.08	0.11	0.11	0.11	0.14	0.05
Support services	$/t moved	0.30	0.29	0.31	0.33	0.31	0.36	0.28	0.19	0.18	0.30
Mobile maint.	$/t moved	0.31	0.31	0.32	0.36	0.50	1.04	1.15	0.94	1.10	0.44
Light vehicles	$/t moved	0.02	0.02	0.02	0.02	0.02	0.03	0.04	0.04	-	0.02
Grand total [1]	**$/t moved**	**2.36**	**2.27**	**2.24**	**2.54**	**2.73**	**3.21**	**2.83**	**2.55**	**2.33**	**2.47**
	$/t mined	**2.74**	**2.57**	**2.84**	**2.90**	**3.14**	**15.40**	**-**	**-**	**-**	**3.23**

Notes:
- [1] Includes maintenance costs but excludes sustaining capital for parts and component repair and / or replacement.
- Material moved includes open pit stockpile reclaim and in-pit rehandle.
- Totals may not compute exactly due to rounding.

21.2.2.2 Underground operating costs

The underground mining costs were estimated from contractor quotations for the mine development and operation, considering the planned activities and estimated productivities and costs. Consumable costs for fuel, power, and backfill cement, which are supplied by the owner, are included in the cost estimate. The LOM underground cost is estimated to be $207M, or an average of $50.62 per tonne of ore mined. The underground costs exclude rehandling of the ore on surface to feed the ore to the mill, since this is covered in the open pit operating costs. A contingency of 10% has been applied to the operating cost for all the contractor's activities. A summary of the estimated LOM operating costs is shown in Table 21.13. Estimated unit operating costs, plus the LOM average, are shown in Table 21.14.

Table 21.13 Underground operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Development	$(000)	-	-	796	4,408	8,176	24,458	24,992	-	-	62,830
LH stoping	$(000)	-	-	770	2,219	10,569	16,793	20,287	27,203	2,835	80,677
Backfill	$(000)	-	-	-	-	-	920	3,966	6,349	379	11,614
Mine general	$(000)	-	-	1,533	4,131	6,540	10,475	9,944	4,989	1,419	39,031
Mine maint.	$(000)	-	-	-	-	-	169	169	169	28	536
Camp & Other	$(000)	-	-	1,049	1,308	1,807	3,053	2,808	2,106	526	12,659
Total UG	**$(000)**	**-**	**-**	**4,146**	**12,061**	**27,094**	**55,861**	**62,162**	**40,816**	**5,188**	**207,328**

Note: Totals may not compute exactly due to rounding.

Table 21.14 Underground unit operating costs

Description	Unit*	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg
Development	$/t	-	-	15.51	30.81	15.22	25.00	21.74	-	-	15.34
LH stoping	$/t	-	-	15.00	15.51	19.68	17.17	17.65	24.28	24.44	19.70
Backfill	$/t	-	-	-	-	-	0.94	3.45	5.67	3.27	2.84
Mine general	$/t	-	-	29.81	28.83	12.18	10.70	8.65	4.45	12.23	9.53
Mine maint.	$/t	-	-	-	-	-	0.17	0.15	0.15	0.24	0.13
Camp & Other	$/t	-	-	20.43	9.15	3.37	3.12	2.44	1.88	4.54	3.09
Total UG	**$/t**	**-**	**-**	**80.75**	**84.30**	**50.45**	**57.11**	**54.08**	**36.42**	**44.72**	**50.62**

Notes:
- *Unit rate is $ per tonne of ore mined.
- Totals may not compute exactly due to rounding.

Development cost includes waste level development for stope access and ore drift development for longhole stopes.

Longhole stoping cost includes all costs associated with stope drilling, blasting, loading, hauling, ground support, and services.

Backfill costs, which include cement cost and labour cost, are based on URF and CRF requirements.

Mine general costs include power, heating, support equipment, technical services labour, technical services consumables, mine technical software, geology development face sampling, mine general consumables, garbage waste removal, personal protective equipment, 5% freight cost on consumables, and definition drilling. The rest of the mining labour cost is captured in the contractor unit rates, which have been allocated into the cost for development, longhole stoping, and backfilling.

Mine maintenance cost only includes labour cost for the mine technical maintenance department, which consists of one mine maintenance / electrical superintendent who will oversee the contractor's maintenance group. The rest of the maintenance cost is captured in the contractor unit rates, which have been allocated into the cost for development, longhole stoping and backfilling.

Camp cost consists of room and board for the underground personnel throughout the underground period. Other costs consist of carbon taxes.

Compared to the NI 43-101 Technical Report from 25 July 2018, the overall LOM underground mining operating cost has decreased significantly, both in total and on a unit basis. Below is a summary of the changes for the underground mining operating cost:

- Previously, the underground was operated by a contractor in the initial years and then the operation was undertaken by the owner. The new mine plan is contractor operated throughout the LOM. Therefore, most of the mining labour and maintenance costs have been reallocated into development, longhole stoping and backfilling.
- Slight increase in diesel unit cost from $0.71/L to $0.73/L.
- Slight increase in propane unit cost from $0.42/L to $0.45/L.
- Large reduction in power unit cost from $0.069/KWh to $0.024/KWh.
- Large reduction in stope development metres from 6.1 km to 2.0 km.
- Large reduction in ore drift development metres from 19.3 km to 8.7 km.
- Large reduction in ore mined from 9.0 Mt to 4.1 Mt, which also results in a reduction of definition drilling required.
- Large reduction in backfill required from 1.6 Mm^3 to 0.35 Mm^3.
- Surface rehandle cost has been removed from the underground cost centre and is now covered in the open pit rehandle cost centre.
- Large reduction in mine life from 14.2 years to 5.7 years, which reduces the mine general and mine maintenance costs.
- Reduction in average haulage distance for each zone, since mining takes place closer to the surface and each zone has a portal.

21.2.3 Process and tailings management area operating costs

The process and tailings management area operating costs were estimated from first principles considering the planned activities and estimated productivities, reagent consumptions and costs. A breakdown of the costs is shown in Table 21.15. Estimated process unit operating costs, plus the LOM average, are shown in Table 21.16. Costs were estimated by New Gold and are based on 2020 budget / LOM cost estimates.

Table 21.15 Process and tailings management area operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Administration	$(000)	11,659	12,275	12,239	12,216	12,181	11,921	11,880	11,737	2,240	98,349
Surface maint.	$(000)	92	92	92	92	92	92	92	92	-	738
Lab	$(000)	2,625	2,608	2,608	2,608	2,608	1,754	1,754	1,754	268	18,589
Metallurgical services	$(000)	102	102	102	102	102	102	102	71	-	782
Refinery	$(000)	542	542	544	544	544	544	544	544	-	4,349
Tailing disposal	$(000)	185	185	185	185	185	185	185	185	-	1,477
Cyanide destruction	$(000)	3,774	4,150	3,884	3,874	3,889	3,855	3,836	3,842	661	31,765
Reclaim piping	$(000)	765	789	785	784	786	780	777	778	120	6,365
Water treatment	$(000)	1,271	894	894	894	894	894	894	894	-	7,528
Comminution	$(000)	25,945	27,062	26,282	26,484	25,760	26,105	26,052	26,070	2,764	212,522
Concentration	$(000)	148	443	443	443	443	443	443	443	-	3,249
Dewatering	$(000)	623	623	623	623	623	623	623	623	-	4,985
Stripping and regeneration	$(000)	834	862	863	896	1,009	882	779	781	58	6,964
Mobile equip.	$(000)	12,121	11,632	11,573	11,573	11,573	11,509	11,463	11,463	1,585	94,493
Hydromet leach	$(000)	8,749	9,063	9,020	8,999	8,991	8,923	8,883	8,896	1,331	72,855
Grand total	**$(000)**	**69,435**	**71,322**	**70,139**	**70,316**	**69,680**	**68,612**	**68,306**	**68,173**	**9,028**	**565,010**

Note: Totals may not compute exactly due to rounding.

Table 21.16 Process and tailings management area unit operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg.
Administration	$/t milled	1.26	1.28	1.28	1.28	1.27	1.26	1.26	1.24	1.29	1.27
Surface maint.	$/t milled	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	-	0.01
Lab	$/t milled	0.28	0.27	0.27	0.27	0.27	0.19	0.19	0.19	0.15	0.24
Metallurgical services	$/t milled	0.01	0.01	0.01	0.01	0.01	0.01	0.01	0.01	-	0.01
Refinery	$/t milled	0.06	0.06	0.06	0.06	0.06	0.06	0.06	0.06	-	0.06
Tailing disposal	$/t milled	0.02	0.02	0.02	0.02	0.02	0.02	0.02	0.02	-	0.02
Cyanide destruction	$/t milled	0.41	0.43	0.41	0.41	0.41	0.41	0.41	0.41	0.38	0.41
Reclaim piping	$/t milled	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.07	0.08
Water treatment	$/t milled	0.14	0.09	0.09	0.09	0.09	0.09	0.09	0.09	-	0.10
Comminution	$/t milled	2.80	2.82	2.75	2.78	2.69	2.76	2.77	2.76	1.59	2.74
Concentration	$/t milled	0.02	0.05	0.05	0.05	0.05	0.05	0.05	0.05	-	0.04
Dewatering	$/t milled	0.07	0.06	0.07	0.07	0.07	0.07	0.07	0.07	-	0.06
Stripping and regeneration	$/t milled	0.09	0.09	0.09	0.09	0.11	0.09	0.08	0.08	0.03	0.09
Mobile equip.	$/t milled	1.31	1.21	1.21	1.22	1.21	1.21	1.22	1.21	0.91	1.22
Hydromet leach	$/t milled	0.94	0.94	0.94	0.95	0.94	0.94	0.94	0.94	0.77	0.94
Grand total	**$/t milled**	**7.50**	**7.43**	**7.34**	**7.39**	**7.29**	**7.24**	**7.25**	**7.22**	**5.19**	**7.28**

Note: Totals may not compute exactly due to rounding.

Overall, the process and tailings management area operating costs appear reasonable and in-line with expectations for the LOM.

21.2.4 General & administrative operating costs

The G&A operating costs were estimated from first principles considering the planned activities and associated costs. A breakdown of the costs is shown in Table 21.17. Estimated G&A unit operating costs, plus the LOM average, are shown in Table 21.18. Costs were estimated by New Gold and are based on 2020 budget / LOM cost estimates.

Table 21.17 G&A operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM total
Administration	$(000)	2,181	2,181	2,235	2,235	1,850	1,400	1,363	1,284	126	14,855
Community	$(000)	3,525	2,777	2,029	2,029	1,832	1,650	1,650	1,481	92	17,063
Environment	$(000)	4,067	3,839	3,913	3,582	3,153	2,709	2,347	2,206	132	25,949
Human resources	$(000)	2,643	2,549	2,310	2,298	2,358	1,791	1,580	1,511	131	17,170
Employee transportation	$(000)	1,760	1,910	1,899	1,899	1,899	1,366	1,319	1,319	281	13,651
Information technology	$(000)	1,230	1,229	1,229	1,229	845	810	800	786	38	8,196
Land maintenance	$(000)	737	762	736	736	736	736	736	719	-	5,900
Safety & security	$(000)	2,494	2,685	2,493	2,493	2,143	1,893	1,463	1,459	93	17,216
Services	$(000)	2,038	2,038	2,038	2,038	2,038	1,462	1,371	1,051	194	14,268
Warehousing and purchasing	$(000)	4,688	5,900	3,210	3,059	2,738	1,882	1,630	1,531	214	24,851
Accounting	$(000)	3,369	3,455	3,513	3,572	3,246	3,104	3,093	2,988	123	26,462
Temporary accommodation	$(000)	461	341	341	341	341	341	341	328	21	2,858
Camp administration	$(000)	3,239	3,607	3,090	3,035	2,740	2,146	3,384	2,883	781	24,904
Grand total	**$(000)**	**32,430**	**33,273**	**29,037**	**28,547**	**25,920**	**21,290**	**21,076**	**19,547**	**2,222**	**213,342**

Note: Totals may not compute exactly due to rounding.

Table 21.18 G&A unit operating costs

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	LOM avg.
Administration	$/t milled	0.24	0.23	0.23	0.23	0.19	0.15	0.14	0.14	0.07	0.19
Community	$/t milled	0.38	0.29	0.21	0.21	0.19	0.17	0.18	0.16	0.05	0.22
Environment	$/t milled	0.44	0.40	0.41	0.38	0.33	0.29	0.25	0.23	0.08	0.33
Human resources	$/t milled	0.29	0.27	0.24	0.24	0.25	0.19	0.17	0.16	0.08	0.22
Employee transportation	$/t milled	0.19	0.20	0.20	0.20	0.20	0.14	0.14	0.14	0.16	0.18
Information technology	$/t milled	0.13	0.13	0.13	0.13	0.09	0.09	0.08	0.08	0.02	0.11
Land maintenance	$/t milled	0.08	0.08	0.08	0.08	0.08	0.08	0.08	0.08	-	0.08
Safety & security	$/t milled	0.27	0.28	0.26	0.26	0.22	0.20	0.16	0.15	0.05	0.22
Services	$/t milled	0.22	0.21	0.21	0.21	0.21	0.15	0.15	0.11	0.11	0.18
Warehousing and purchasing	$/t milled	0.51	0.61	0.34	0.32	0.29	0.20	0.17	0.16	0.12	0.32
Accounting	$/t milled	0.36	0.36	0.37	0.38	0.34	0.33	0.33	0.32	0.07	0.34
Temporary accommodation	$/t milled	0.05	0.04	0.04	0.04	0.04	0.04	0.04	0.03	0.01	0.04
Camp administration	$/t milled	0.35	0.38	0.32	0.32	0.29	0.23	0.36	0.31	0.45	0.32
Grand total	**$/t milled**	**3.50**	**3.46**	**3.04**	**3.00**	**2.71**	**2.25**	**2.24**	**2.07**	**1.28**	**2.75**

Note: Totals may not compute exactly due to rounding.

Overall, the G&A operating costs appear reasonable and in-line with expectations for the LOM.

21.2.5 Manpower

All manpower rosters were sourced from New Gold's 2020 Budget / LOM. All manpower levels are reported as full-time equivalent (FTE). A manpower summary is provided in Table 21.19.

Table 21.19 Manpower summary

Description	2020	2021	2022	2023	2024	2025	2026	2027	2028
Open pit	493	493	488	486	344	159	116	99	97
Underground	-	-	-	-	3	14	14	14	14
Process	161	160	160	160	160	145	145	145	145
G&A	133	133	133	131	117	82	62	56	52
Grand total	**787**	**786**	**781**	**777**	**624**	**399**	**337**	**314**	**307**

Note: Figures may not add as based upon FTE. Contractor manpower is not included in the proceeding.

Contractors are utilized in various positions at site that are not fulfilled by direct New Gold employees (security, projects / construction, blasting contractor, camp administration, etc.). Contractor numbers on site are estimated to be approximately 200/year through 2025, dropping thereafter as open pit mining ceases. In addition to the previous, during underground operations, it is estimated that up to 160 contractors will be employed in the underground operations.

21.2.5.1 Open pit manpower

The projected manpower levels for the open pit operation are shown in Table 21.20.

Table 21.20 Open pit manpower

Description	2020	2021	2022	2023	2024	2025	2026	2027	2028
Administration	33	33	31	31	24	9	8	6	4
Engineering	16	16	17	17	8	3	-	-	-
Geology	19	19	20	20	18	4	-	-	-
Operations	272	272	268	266	162	62	27	27	27
Maintenance	153	153	153	153	133	81	81	66	66
Grand total	**493**	**493**	**488**	**486**	**344**	**159**	**116**	**99**	**97**

Note: Figures may not add as based upon FTE.

Open pit manpower remains relatively consistent throughout the 2020 – 2023 time period, reducing throughout 2024 and 2025 as the pit nears the end of its operation and transitions into a stockpile re-handle operation.

The projected manpower levels for the open pit operation are considered to be appropriate for the project.

21.2.5.2 Underground manpower

Throughout the underground LOM, a contractor will be performing all underground development and production work. The owner will have its own technical and management staff that will oversee and assist the underground operations. The work schedule is based upon operating personnel working 12-hour shifts on a two-weeks-on / two-weeks-off rotation. Technical and management personnel will work a five-days-on / two-days-off schedule. During the period 2020 to 2024, the existing open pit technical and management staff will take on underground duties alongside their open pit activities. It is assumed that the open pit workload will diminish as the pit gets deeper, with the staff being able to take on the additional workload as the underground mine starts to develop. In 2025, they will fully transition into underground operations. There will be some personnel operational readiness / capability risk during the transitioning process; therefore, a training and hiring cost allowance has been made.

Camp cost for the underground contractors is covered in the underground mining cost estimates. Flight cost to and from site for the owner's staff is covered in the G&A allowance. Most of the owner's staff will, however, reside locally near the mine site. Flight cost to and from site for the underground contractor is included in the contractor rates.

The projected manpower levels for the underground mine for the LOM period (2020 to 2028) are shown in Table 21.21.

Table 21.21 Underground manpower

Description	2020*	2021*	2022*	2023*	2024*	2025	2026	2027	2028
Owner personnel									
Mine manager / superintendent	-	-	-	-	-	1	1	1	1
Maint. and elect. superintendent	-	-	-	-	-	1	1	1	1
Chief mine engineer	-	-	-	-	1	1	1	1	1
Senior mine engineer	-	-	-	-	-	1	1	1	1
Mine engineers	-	-	-	-	1	2	2	2	2
Surveyors	-	-	-	-	0.7	3	3	3	3
Mine geologists	-	-	-	-	-	3	3	3	3
Ground control technician	-	-	-	-	-	1	1	1	1
Ground control engineer	-	-	-	-	0.3	1	1	1	1
Owner staff sub-total	**-**	**-**	**-**	**-**	**3**	**14**	**14**	**14**	**14**
Contractor personnel									
Mine operations									
Mine shift supervisors	4	-	4	4	4	8	8	8	8
Jumbo operators	4	-	4	8	8	16	12	4	4
Longhole drill operators	4	-	4	4	4	8	8	8	4
Scoop operators	2	-	2	2	8	16	16	16	4
Truck drivers	2	-	2	2	8	12	12	12	4
Trainer (equip and safety)	1	-	1	1	1	1	1	1	1
Diamond drillers	2	-	2	2	2	2	2	2	-
Blasters	6	-	6	6	12	24	18	12	12
Bolters and ground support	4	-	4	8	8	16	12	4	4
Grader operator	1	-	1	1	1	1	1	1	1
General labourers	3	-	3	4	8	8	8	8	4
Services	4	-	4	4	8	12	12	4	4
Mine operations sub-total	**37**	**-**	**37**	**46**	**72**	**124**	**110**	**80**	**50**
Mine maintenance		-							
UG warehouse person	2	-	2	2	2	2	2	2	2
Mechanical foreman	1	-	1	1	1	1	1	1	1
Lead hand mechanic	1	-	1	1	1	1	1	1	1
Welders	1	-	1	1	3	3	3	1	1
Mechanics	8	-	8	8	8	16	16	12	12
Electrical foreman	1	-	1	1	1	1	1	1	1
Electrician	4	-	4	8	8	8	8	4	4
Labourers	1	-	1	4	4	4	4	4	4
Mine maint. sub-total	**19**	**-**	**19**	**26**	**28**	**36**	**36**	**26**	**26**
Contractor sub-total	**56**	**-**	**56**	**72**	**100**	**160**	**146**	**106**	**76**
Grand total including owner personnel	**56**	**-**	**56**	**72**	**103**	**174**	**160**	**120**	**90**

Note: *Mine technical staff covered by open pit personnel from 2020 – 2024.

21.2.5.3 Process manpower

The projected manpower levels for the process plant operation are shown in Table 21.22.

Table 21.22 Process manpower

Description	2020	2021	2022	2023	2024	2025	2026	2027	2028
Administration	28	27	27	27	27	27	27	27	27
Metallurgy	4	4	4	4	4	4	4	4	4
Lab	31	31	31	31	31	16	16	16	16
Operations	53	53	53	53	53	53	53	53	53
Maintenance	45	45	45	45	45	45	45	45	45
Total process	**161**	**160**	**160**	**160**	**160**	**145**	**145**	**145**	**145**

Process manpower remains steady throughout the 2020 – 2024 time period, dropping for the 2025 – 2028 period principally as lab staff are decreased due to the reduction in daily sample requirements as open pit mining ceases (blasthole samples).

The projected manpower levels for the process plant operation are considered to be appropriate for the project.

21.2.5.4 G&A manpower

The projected manpower levels for the G&A areas are shown in Table 21.23.

Table 21.23 G&A manpower 2020 – 2027

Description	2020	2021	2022	2023	2024	2025	2026	2027	2028
Administration	2	2	2	2	2	1	1	1	1
Community	6	6	6	6	4	3	3	3	3
Environment	13	13	13	12	9	7	5	5	5
Project team	12	12	12	12	12	12	-	-	-
Human resources	10	10	10	10	10	6	5	5	5
Employee transportation	5	5	5	5	5	5	4	4	4
Information technology	4	4	4	4	2	2	2	2	2
Safety and security	8	8	8	8	6	4	4	4	4
Services	29	29	29	29	29	20	20	15	15
Warehousing and purchasing	24	24	24	23	21	11	8	8	8
Accounting	10	10	10	10	7	5	5	4	4
Temporary accommodation	2	2	2	2	2	2	2	2	1
Camp administration	1	1	1	1	1	1	-	-	-
Exploration	7	7	7	7	7	4	4	4	-
Grand total	**133**	**133**	**133**	**131**	**117**	**82**	**62**	**56**	**52**

Note: Figures may not add up as based upon FTE.

The projected manpower levels for the G&A function are considered to be appropriate for the project.

22 Economic analysis

22.1 Introduction

A financial analysis for Rainy River was developed by New Gold using a discounted cash flow approach on a pre-tax and after-tax basis. The NPV was calculated from the cash flow generated by the project based on a discount rate of 5%. A sensitivity analysis was performed for the pre-tax base case to assess the impact of variations of the capital costs, operating costs, exchange rate, and the gold metal selling price. The internal rate of return (IRR) for Rainy River is not discussed as it would be misleading due to the fact that the project has been principally constructed and has been in commercial operations since October 2017.

22.2 Methods, assumptions, and basis

The economic analysis was performed using the following assumptions and basis:

- The financial analysis was performed on Proven and Probable Mineral Reserves as outlined in this report for the open pit and underground mines.
- The LOM NPV was determined on a pre-tax and after-tax basis with discounting to the start of 2020, which marks the first year in the current LOM.
- Annual cash flows used for NPV calculations are assumed to be realized at year-end.
- Base case gold and silver metal selling prices are $1,300/oz and $16/oz, respectively for this economic analysis.
- The exchange rate has been assumed to be 1.3 C$:US$.
- All costs and sales are presented in constant Q1-2020 US$, with no inflation or escalation factors considered.
- All gold and silver sales are assumed to be in the same period as produced. Re-circulating plant metal load is assumed to be produced and sold in the last period of operation.
- All related payments and disbursements incurred prior to year-end 2019 are considered as sunk costs.
- Details of capital and operating costs are provided in Section 21 of this report.
- Cash flows shown are post payment of royalties and metal streaming agreements.
- Progressive and final closure costs are included in the period incurred, with post closure costs reported in 2032, the final year of closure prior to entering post-closure.
- The financial analysis includes working capital.
- After-tax results and royalty payments were provided by New Gold. AMC has not verified this work.
- After-tax figures assume a combined income tax rate of 25% with 2.7% corporate minimum tax, a mining tax of 10% of taxable mining profits over $500,000 and an allocation of corporate tax attributes among New Gold's operations.

The general assumptions used for this financial model are summarized in Table 22.1 and Table 22.2.

Table 22.1 Financial model criteria and production summary

Description	Unit	Value
Gold price	$/oz	1,300
Silver price	$/oz	16
Ore milled [1]	Mt	78
Open pit waste mined [2]	Mt	171
Open pit strip ratio (waste:ore)	-	2.53
Gold grade	g/t	1.06
Silver grade	g/t	2.51
Gold recovery	%	88.9
Silver recovery	%	57.3
Gold recovered	koz	2,343
Silver recovered	koz	3,591
Gold sold [3]	koz	2,354
Silver sold	koz	3,591
Exchange rate	C$:US$	1.30
Discount rate	%	5.0
Non-sustaining capital cost	$M	56
Sustaining capital cost	$M	586
LOM	years	8.25

Notes:
[1] Includes open pit, underground and stockpiles.
[2] Including capitalized waste.
[3] Includes 11 koz of gold within the recirculating load of the plant that is only realized as production at the end of processing.

Table 22.2 Financial model operating costs and cash costs over the LOM

Description	LOM total ($M)	Cost of production ($/oz gold produced)
Mining (including stockpile)	758	324
Processing	565	241
General & administration	213	91
Refining and transportation	7	3
Royalty payments	25	11
Inventory movement	23	10
Sub-total costs	1,592	679
Silver sales	(57)	(25)
Total costs net silver	1,535	655
Sustaining capital	586	250
Other sustaining costs	139	60
All-in sustaining cash costs	2,260	964

Note: Totals may not compute exactly due to rounding.

22.3 Royalties

The annual royalty costs were calculated by New Gold and are based on the open pit and underground mine production profiles presented in this report, along with the terms of the individual royalty agreements. Royalty costs are based on a gold price of $1,300 per ounce of gold and $16 per ounce of silver. Over the remaining LOM, approximately $25.3M in royalties are projected to be paid based on the base case metal prices and LOM assumptions, with $2.7M NPI/NSR (net profit interest / Net Smelter Return) and $22.6M in other royalties.

22.4 Metal streaming

The annual metal streaming costs associated with the Royal Gold agreement were calculated by New Gold and are based on the open pit and underground mine production profiles presented in this report, along with the terms of the streaming agreement. Streaming costs are based on a gold price of $1,300 per ounce of gold and $16 per ounce of silver. Over the remaining LOM, approximately $175.0M in net streaming benefits are projected to be paid based on the base case metal prices and LOM assumptions.

22.5 Salvage value

Total equipment salvage value was estimated by New Gold to be $41M and is credited partially during 2026 after the open pit is depleted, and in 2028 at the end of the LOM. In 2026, the open pit mining equipment not being used for stockpile reclamation will be sold for a total estimated salvage value of $19M. In 2028 at the end of the LOM, the remaining mine and process plant equipment will be sold for an estimated salvage value of $22M. Major equipment includes the SAG and ball mills (including motors), primary crusher, pebble crusher, and apron feeders, along with the balance of the plant equipment.

22.6 Taxation

The tax rate calculations and taxation assumptions used by New Gold in the after-tax NPV calculations are summarized below.

The federal government imposes income tax on mining income at the same rate that applies to other types of income. The federal rate applicable to resource profits is 15%. Ontario's taxation of the resource sector is generally harmonized to the federal system. The provincial corporate income tax rate applicable to mining income is 10%. A combined rate of 25% is used in the model to compute the federal and provincial tax liability in respect of the Rainy River Mine. In addition, in the earlier years of the project, the Ontario Corporate Minimum Tax has been computed at a rate of 2.7%. All deductions and rates are based on currently enacted legislation. In addition, Ontario's Mining Tax Act is levied at a rate of 10% on annual taxable profits in excess of $500,000.

The federal and provincial tax legislation provides a number of deductions, allowances and credits that are specifically available to taxpayers engaged in qualifying mining activities. The most notable of these deductions are Canadian Exploration Expenditures (CEE), Canadian Development Expenses (CDE), and capital costs eligible for Class 41 of the capital cost allowance system. Because these deductions and allowances are only available when incurred, a high-level assumption was made with regard to the allocation of expenditures between the three categories in the LOM model.

Similarly, the Ontario Mining Tax Act provides a number of deductions in arriving at taxable profits, the key ones being an allowance for Exploration and Development Expenditures, Depreciation Allowance and Processing Allowance. Since these deductions depend on when the expenses are incurred and the degree of processing that occurs in Canada, a number of high-level assumptions are made with regards to the allocation and timing of expenditures for the depreciation allowance calculation. A general assumption regarding the processing allowance has been made and it has been assumed that Rainy River will qualify for a minimum 15% processing allowance.

The Rainy River Mine, New Afton Mine and Blackwater Project are all held by New Gold. New Gold is intending to utilize its current and projected tax attributes, with the overall goal of maximizing the profitability of New Gold's Canadian operations as a whole, as opposed to any one operation or project. Part of New Gold's growth strategy, including the acquisition of Rainy River Resources in

2013, has been to build its business in jurisdictions where it already has an established presence. One of the many benefits of this approach is that it enables the company to manage its business in a tax-efficient manner. New Gold plans to utilize a portion of the tax attributes that have been and will continue to be generated in the company's Canadian head office. Currently, the company's primary objective is to maximize the cash flow generation of its New Afton Mine in Kamloops, BC, with any remaining tax attributes planned to be used to maximize Rainy River's future after-tax cash flow. As New Gold's focus in such allocation will be to maximize the company's overall profitability rather than that of any one operation or project, this will remain a dynamic process.

At the end of 2019, on a combined basis, the New Gold corporate entity had the tax attributes provided in Table 22.3.

Table 22.3 New Gold corporate tax attributes (end 2019)

Description	Amount $(000s)	Expiry
ITC receivable	65,965	2020 – 2039
Realized capital losses	60,207	n/a
Financing costs	7,587	n/a
CEE balance	1,132,305	n/a
CDE balance	94,343	n/a
Undepreciated capital cost	1,012,239	n/a

22.7 Financial analysis summary

A financial analysis for Rainy River was developed by New Gold using a discounted cash flow approach on a pre-tax and after-tax basis. The NPV was calculated from the cash flow generated by the project based on a discount rate of 5%. The pre-tax base case financial model resulted in an undiscounted cash flow of $559M with an NPV of $426M at a discount rate of 5%. On an after-tax basis, the base case financial model resulted in an undiscounted cash flow of $557M with an NPV of $424M at a discount rate of 5%. The minimal change in value between pre-tax and after-tax cash flows is a result of the utilization of current and projected tax attributes as presented in Section 22.6.

A summary of the base case financial analysis is presented in Table 22.4.

Table 22.4 Financial analysis summary

Description	Unit	2020	2021	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	2032	LOM total
Mine plan (Open Pit & Underground)															
Ore mined	t (000)	13,232	15,452	10,384	12,067	15,537	2,585	1,149	1,121	116	-	-	-	-	71,643
Waste mined [1]	t (000)	42,052	39,772	40,873	34,636	13,502	791	136	-	-	-	-	-	-	171,762
Total mined	t (000)	55,284	55,224	51,257	46,703	29,039	3,376	1,285	1,121	116	-	-	-	-	243,405
Rehandle [2]	t (000)	6,647	6,276	6,228	4,760	4,784	8,315	9,434	9,453	1,653	-	-	-	-	57,550
Total moved	t (000)	61,931	61,500	57,486	51,463	33,823	11,691	10,719	10,573	1,769	-	-	-	-	300,955
Process plan															
Ore processed	t (000)	9,260	9,605	9,549	9,521	9,563	9,474	9,421	9,439	1,739	-	-	-	-	77,572
Gold grade	g/t	0.95	1.02	1.17	1.19	1.26	1.25	0.85	0.84	0.61	-	-	-	-	1.06
Silver grade	g/t	2.55	2.55	2.33	2.60	3.41	2.38	2.15	2.18	2.16	-	-	-	-	2.51
Gold recovery [3]	%	87.8%	88.3%	89.5%	90.4%	90.7%	90.2%	86.6%	86.6%	83.2%	0.0%	0.0%	0.0%	0.0%	88.9%
Silver recovery [3]	%	57.1%	56.9%	57.0%	57.2%	59.2%	57.6%	56.3%	56.4%	56.3%	0.0%	0.0%	0.0%	0.0%	57.3%
Gold produced	koz	249	278	322	329	351	343	222	220	28	-	-	-	-	2,343
Silver produced	koz	433	449	407	455	620	418	367	373	68	-	-	-	-	3,591
Gold sold	koz	249	278	322	329	351	343	222	220	39	-	-	-	-	2,354
Silver sold	koz	433	449	407	455	620	418	367	373	68	-	-	-	-	3,591
Financial analysis															
Metal price															
Gold	$/oz	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	1,300	-	-	-	-	1,300
Silver	$/oz	16	16	16	16	16	16	16	16	16	-	-	-	-	16
Revenue															
Gold	$(000s)	323,911	361,933	418,377	428,150	456,266	445,543	289,045	286,494	50,250	-	-	-	-	3,059,969
Silver	$(000s)	6,933	7,177	6,513	7,276	9,919	6,690	5,877	5,975	1,091	-	-	-	-	57,451
Gross revenue	$(000s)	330,844	369,110	424,890	435,426	466,185	452,234	294,922	292,469	51,340	-	-	-	-	3,117,419
Operating costs															
Mining	$(000s)	113,445	108,811	70,699	107,275	115,944	85,586	85,610	62,103	8,776	-	-	-	-	758,249
Processing	$(000s)	69,435	71,322	70,139	70,316	69,680	68,612	68,306	68,173	9,028	-	-	-	-	565,010
G&A site	$(000s)	32,430	33,273	29,037	28,547	25,920	21,290	21,076	19,547	2,222	-	-	-	-	213,342
Royalties	$(000s)	2,400	2,444	2,636	2,809	4,340	4,770	2,714	2,692	472	-	-	-	-	25,277
Refining and freight	$(000s)	783	840	880	925	1,074	921	695	696	196	-	-	-	-	7,009
Other expenses															
Exploration	$(000s)	2,126	2,126	2,054	1,970	1,970	1,317	1,317	1,317	-	-	-	-	-	14,196
Reclamation [4]	$(000s)	1,178	3,051	3,036	3,044	3,028	2,974	3,013	4,655	17,365	17,365	17,365	17,365	13,494	106,932
Other (incl. working capital)	$(000s)	5,287	(3,698)	(65)	(474)	5,360	7,992	816	121	12,702	2,031	-	-	-	30,073
Capital															
Sustaining	$(000s)	119,100	106,547	132,964	100,518	62,859	46,716	16,441	910	-	-	-	-	-	586,056
Non-Sustaining	$(000s)	3,846	-	10,072	3,390	11,149	27,695	-	-	-	-	-	-	-	56,152
Other investing cash flow															
Other investing activities	$(000s)	13,502	26	(8,486)	5,099	3,367	(730)	(10,446)	1,915	(22,262)	-	-	-	-	(18,014)
Financing activities															
Metal streaming (Royal)	$(000s)	18,910	20,874	23,327	24,147	26,707	24,731	16,735	16,655	2,941	-	-	-	-	175,026
Capital leases	$(000s)	9,978	9,657	9,540	8,332	1,142	-	-	-	-	-	-	-	-	38,649
Cash flow															
Cash flow (pre-tax)	$(000s)	(61,577)	13,836	79,058	79,530	133,645	160,359	88,644	113,686	19,901	(19,396)	(17,365)	(17,365)	(13,494)	559,463
Taxes	$(000s)	205	-	-	663	0	1,005	453	0	-	-	-	-	-	2,326
Cash flow (after-tax)	$(000s)	(61,782)	13,836	79,058	78,867	133,645	159,354	88,191	113,686	19,901	(19,396)	(17,365)	(17,365)	(13,494)	557,137
Discounted cash flow															
Pre-tax @ 5%	$(000s)	(58,645)	12,550	68,293	65,430	104,714	119,662	62,998	76,947	12,828	(11,908)	(10,153)	(9,669)	(7,156)	425,892
After-tax @ 5%	$(000s)	(58,840)	12,550	68,293	64,884	104,714	118,912	62,676	76,947	12,828	(11,908)	(10,153)	(9,669)	(7,156)	424,080

Note: Totals may not compute exactly due to rounding.
[1] Waste totals include both operating and capitalized / deferred waste tonnages.
[2] Includes all rehandle sources (in-pit rehandle, and open pit and underground stockpile rehandle).
[3] Metal recoveries in the cash flow reflect the impact of process circuit inventory movements between periods. Process plant circulating load assumed to be recovered at end-of-LOM in 2028.
[4] Post-closure reclamation monitoring costs have been discounted and included in year 2032.

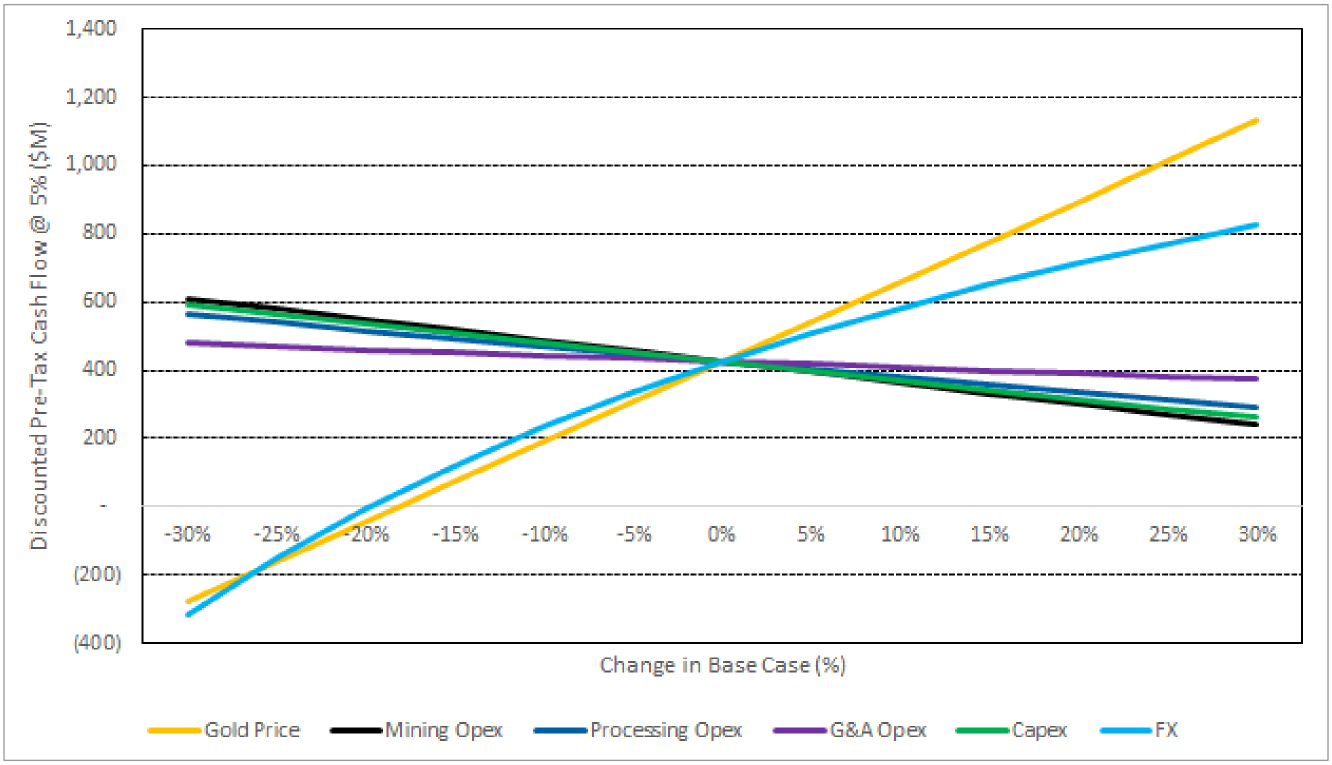
22.8 Sensitivity analysis

A financial sensitivity analysis was conducted on the pre-tax LOM base case cash flow NPV for variations in gold metal selling price, foreign exchange rate and operational and capital expenditures. The results are presented in Table 22.5 and illustrated in Figure 22.1. Due to the complexity of New Gold's financial model, only the key drivers are varied in the sensitivity analysis, with other costs in the model maintained constant per the base case (i.e. royalty payments, refining and freight, etc.). The impact of this does not change the overall interpretation of the sensitivity analysis.

Table 22.5 Sensitivity analysis

Description	Sensitivities pre-tax NPV @ 5% ($M)						
	-30%	**-20%**	**-10%**	**0%**	**10%**	**20%**	**30%**
Gold price	(280)	(44)	191	426	661	896	1,131
Exchange rate	(314)	(6)	234	426	583	714	824
Mining costs	612	550	488	426	364	302	240
Processing costs	563	517	471	426	380	335	289
G&A costs	479	461	443	426	408	391	373
Capital costs	590	536	481	426	583	714	824

Figure 22.1 Sensitivity analysis



Source: AMC 2020.

From the proceeding Table and Figure, it can be appreciated that although increases and / or decreases in operating and capital costs have material impacts on the resultant projected cash flow, the movements in gold price and the C$:US$ exchange rate have the most significant impacts on the economics of the project. The impact of the C$:US$ exchange rate is significant as New Gold estimates that 95% of its costs (operating and capital expenses) are based in C$. Thus, as the US$

appreciates against the C$, significant improvements in cash flow are realized as revenues from gold sales occur in US$, with the opposite effect occurring as the US$ depreciates against the C$.

22.9 Conclusion

AMC has reviewed New Gold's LOM financial model and has performed an economic analysis of the Rainy River Mine using this model adjusted for Mineral Reserve reporting metal prices declared in this report of $1,275 per ounce gold and $17 per ounce silver. The pre-tax Mineral Reserve price financial model resulted in an undiscounted cash flow of $505M with an NPV of $382M at a discount rate of 5%. On an after-tax basis, the Mineral Reserve financial model resulted in an undiscounted cash flow of $504M with an NPV of $381M at a discount rate of 5%. The minimal change in value between pre-tax and after-tax cash flows is a result of the utilization of current and projected tax attributes.

Due to the complexity of New Gold's financial model, only the key drivers are varied in the preceding analysis, with other costs in the model maintained constant per the base case (i.e. royalty payments, refining and freight, etc.). The impact of this does not change the overall interpretation of the analysis.

AMC 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

There is no additional information or explanation necessary to make the technical report understandable and not misleading.

25 Interpretation and conclusions

The QPs offer the following interpretations and conclusions:

25.1 Geology

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

25.1.1 Quality Assurance/Quality Control

Drilling programs completed on the Property between 2005 and 2017 have included QA/QC monitoring programs which have incorporated the insertion of CRMs, blanks, and duplicates into the sample streams. In general, the QA/QC sample insertion rates used at Rainy River fall below the general accepted industry standards.

The performance of several CRMs currently in use by New Gold show good precision but poor accuracy. New Gold believes that this is an issue with the CRMs and not a function of lab performance. The CRMs used by previous operators have not adequately covered the COG of the open pit Mineral Resource. Overall performance of one of the assay labs was inadequate. This was recognized and remedial action taken.

Between 2005 and 2011, blank material was sourced from a local granite. Analytical results indicate that this material contained low levels of gold. Blank material was switched to a coarse marble in 2011, and results from this date onwards are considered acceptable and suggest that no systematic contamination occurred throughout the analytical process.

Duplicate sample results show suboptimal performance which is a probable result of the heterogenous nature of the mineralization.

Umpire samples show no bias and indicate that the primary lab currently in use is performing accurately.

Despite the concerns highlighted above, the QP does not consider these issues to be material to the global, long term Mineral Resource estimate. There is however no guarantee that there are no material impacts on the local scale. Overall, the QP considers the assay database to be acceptable for Mineral Resource estimation.

25.1.2 Data verification and Mineral Resources

The Mineral Resource database is sufficiently reliable for grade modelling and Mineral Resource estimation from the checks carried out by the QP. Reconciliation is carried out monthly and on a global basis the comparisons are good.

The geology block model has not been updated for some years and the interpretation should be revisited to include any new interpretation gained though mining of the deposit. There is also some new data that should be included.

The data handling and estimating has been done in a fair manner and the modifying factors including cut-offs applied are reasonable.

Measured and Indicated Mineral Resources are estimated to total 23.1 Mt at grades of 2.57 g/t Au and 6.9 g/t Ag, containing 1,914 koz of gold and 5,120 koz of silver. Inferred Mineral Resources are estimated to total 3.5 Mt at grades of 1.77 g/t Au and 2.4 g/t Ag, containing 198 koz of gold and

268 koz of silver. The Mineral Resources are exclusive of Mineral Reserves. Mineral Resources are not Mineral Reserves and have not demonstrated economic viability.

The block model has been performing adequately and presents low risk to the project. The opportunity for growth of Mineral Resources on the deposit are mainly price and cost driven. Otherwise exploration elsewhere on the property presents the next best opportunity for further mine life.

25.2 Mining and Mineral Reserves

25.2.1 Open pit mining and Mineral Reserves

Open pit Proven and Probable Mineral Reserves, including stockpile, total 73.5 Mt grading 0.88 g/t gold and 2.2 g/t silver, containing 2,087 koz and 5,231 koz of gold and silver, respectively.

The reconciliation of the mine planning model to 2019 production indicates that the mine planning model is performing well within the normal range of variability (±10%). Overall, Mineral Reserve estimates for the period are within 10% of production results for tonnes and grade.

The current pit design and resulting Mineral Reserve is robust over a range of metal prices under the current design criteria which has been developed on a mine planning resource model employing representative modifying factors, updated metallurgical recovery curves and geotechnical slope design criteria as well as actual costs developed with reasonable productivity improvements included.

The open pit overburden design slopes satisfy the required FOS under very short-term, short-term, long-term and seismic conditions. The design requires placement of a toe berm / slope buttress shortly following the completion of excavation. The construction of toe berm / slope buttress allows excavation of steeper slope cuts and reduces the waste dump size. This in turn reduces the rock haulage and wick drains construction cost. Timely placement of the rockfill toe berm is critical; delays in rockfill placement may result in slope instability. Control of the surface water is important to good performance, stability, and to decrease the need for maintenance of overburden pit slopes.

The open pit hard rock pit slope stability and resulting design is defined by:

- The orientation of the regional south-southwest dipping foliation structures (North Wall).
- The kinematic stability related to the major joint sets (all pit walls).

With consideration to the above, the revised design recommendations were predominantly driven by the bench-berm stability assessments with shallower BFAs adopted to reduce the undercutting potential from planar sliding and wedge intersections.

Approximately 238 Mt of material is scheduled to be extracted from the open pit using conventional truck and shovel mining methods. Open pit mining is executed by a fleet of 220 t payload haul trucks combined with diesel powered hydraulic excavators and large FELs as primary loading units.

LGO mined and stockpiled is critical to the success of the underground mine and needs to be managed strategically to ensure sufficient quantities are available after the completion of the open pit mine to support the underground mine plan.

Mine equipment requirements were developed from the annual mine production schedule, based on equipment availability, utilization, and equipment productivities. Reasonable productivity improvements were included based on recent successes, but these need to be achieved to realize planned mining rates in peak years. No additional or replacement capex is required for the principal mining fleet within the current mine plan except for the leasing of a CAT 994HL in 2020 for the remaining LOM. Fleet size and age are suitable to execute the proposed mine plan.

The principal fleet is capable of the productivity requirements to execute the LOM plan. There exists an area in Phase 3 and Phase 4 on the south wall where, for a relatively short distance, minimum mining widths need to be employed (based on the size of the largest loading unit operating with double-sided loading). The vertical advance of mine development is within industry norms for large-scale gold mines, albeit at the upper range, with peak yearly vertical advance ranging from 9 to 12 benches per year in a single phase. Rainy River has not mined at these rates of vertical advance in the past and will need to adjust their short- to mid-range planning to manage this quantity of bench turn-over, which is achievable with best-in-class planning practices.

NAG quantities for TMA construction are available from in-pit mining at an average recovery rate of 65% over the LOM. No mining of the EOC is included in the current mine plan. Should the NAG material not present itself as identified in the mine planning resource model or should the expected recovery rate be less than anticipated, additional mining capacity will be required.

25.2.2 Underground mining and Mineral Reserves

The underground Probable Mineral Reserves are estimated to be 4.1 Mt grading 4.17 g/t Au and 7.85 g/t Ag. The Mineral Reserves include factors for COG, dilution, minimum mining width, and mining extraction.

The underground Mineral Reserves are stated at a COG of 2.2 g/t gold equivalent, which is higher than the initially calculated breakeven COG of 1.7 g/t gold equivalent. The use of the higher COG generates an underground production schedule that terminates when the open pit processing is complete. Checking of the COG against final estimated costs for this NI 43-101 report indicates a full-cost value of 2.4 g/t gold equivalent (1.8 g/t gold equivalent breakeven without sustaining capital). It is noted that less than 1% of the MSO stopes tonnage lies between 2.2 g/t gold equivalent and 2.5 g/t gold equivalent and, as such, the cut-off of 2.2 g/t gold equivalent is considered appropriate and robust for the Rainy River underground, as the Mineral Reserve is not sensitive to moderate changes in COG.

At $1,275 per ounce gold price, only 25% of current underground Mineral Resources are converted to Mineral Reserves. This low ratio for underground ore highlights current economic drivers and the dependence of the underground mine operation on the concurrent delivery rate of open pit material to the mill. In a higher gold price environment and subject to realized costs, the proportion of underground Mineral Resources converted to Mineral Reserves could substantially increase.

Access to the underground is via five portals (i.e. ODM Main Zone, ODE East Zone, 17 East Upper Zone, 433 Zone, and Intrepid Zone). The underground will be a mechanized operation utilizing uphole stoping for areas less than 15 m wide and downhole stopes (ODM Main Zone only) with CRF and / or URF for areas wider than 15 m.

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

The overall rock mass quality in terms of RQD for the underground is classified as “Fair” to predominantly “Good”, with RQD typically ranging from 90% to 100% throughout all stoping domains.

There is an opportunity to start mining the Intrepid Zone at an earlier date. The Intrepid Zone decline will start advancing in Q2-2020 as an orebody investigation project. Once this development is completed, diamond drilling for the Intrepid Zone will begin. The Intrepid Zone decline is currently planned to recommence in June 2023, with stoping to begin in December 2023. However, start dates could potentially be earlier, depending on results of the orebody investigation project and further economic assessment. A portion of the Intrepid Zone Mineral Resources has been converted to Mineral Reserves and is already in the mine plan, and the orebody investigation project could potentially identify more that is economically viable.

To achieve the underground mining schedule, some key activities will need to be closely monitored. Of these activities, 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. However, 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. Geotechnical analyses and modelling support the planned open stope designs. Modelling should be calibrated with additional geotechnical data from underground operations. In some non-entry areas, large HW exposures will exist, supported by relatively small pillars (as designed). Any failures of the HW or pillars in these locations could generate additional dilution and / or loss of ore.

25.3 Process and metallurgy

Rainy River's current focus is to operate at higher throughputs than the original plant design throughput. This produces a coarser grind size P_{80} than the design criteria grind P_{80} of 75 µm, which reduces gold recovery. Rainy River has determined that an increase in throughput at the expense of gold recovery is the most economically viable option.

Settling rates in the pre-leach thickener have been a process plant bottleneck. Rainy River will install a polymer slicing unit and an alternative flocculant supply system with the objective of improving settling rates and de-bottlenecking the thickener.

The tailings pumps are a process plant bottleneck when depositing along the North Dam of the TMA. Rainy River will implement a booster pump station in the 2nd quarter of 2020 that will allow plant throughputs of 27,000 tpd to be sustained when depositing at the North Dam.

New Gold has been able to reduce reagent consumptions compared to the forecast and LOM average values. This has subsequently reduced operating costs. These reagents include cyanide, sulphur dioxide, copper sulphate, carbon, and lime.

25.4 Tailings management area

To contain the proposed LOM tailings and water storage volumes, the Rainy River TMA perimeter dams will need to be raised to an ultimate elevation of 379.0 m. Preliminary stability modelling shows that structures and constraints downstream of the Rainy River TMA would be impacted by the proposed buttress extents. New Gold indicated that impacts to BCR1 and the West Creek Diversion Channel would need to be avoided, but that other downstream impacts were acceptable. Acceptable impacts require the purchase of minimal additional lands and an amendment to the permit, granted under the Endangered Species Act, which currently limits the extension of the TMA to the north-west.

25.5 Mine rock and overburden stockpiles

The design of the slope on the south side of the low grade ore stockpile will be updated shortly. This is expected to slightly reduce the nominal capacity of the EMRS, but also to slightly reduce the number of wick drains required in the area. Some of the mine rock that is produced in the open pit will be directly placed as buttresses against the open pit overburden slopes. This will reduce the storage capacity required for mine rock in the EMRS and WMRS.

The design of the WMRS slopes was conceptual and based on consideration of only two cross-sections. More detailed design could optimize the slope design and potentially increase the storage capacity of the WMRS should this be required in the future.

A geotechnical instrumentation system is being installed in the foundation of the EMRS and WMRS. Piezometer readings supplemented by engineering analysis may indicate that the placement of mine rock can be accelerated in certain areas, particularly in the initial lifts.

25.6 Environmental, social, community, and reclamation / closure

New Gold remains committed to environment, social and community resources and relations in and around the Rainy River Mine. This commitment is mandated and assessed against New Gold's Health, Safety, Environment and Corporate Social Responsibility Policy approved by the Board of Directors on 25 July 2018.

At the time of this review, the Mine Environmental Department was adequately staffed, and had increased accountabilities with addition of water resource management. New Gold conducts ambient air quality, surface and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

The Mine environmental budget for 2020 was reviewed and appears to be reasonable to accommodate the number of regulatory reporting and study commitments, and flexible enough to respond to changes in operating conditions.

New Gold continues to develop the EMS. Phase 1 was completed during 2019. Phase 2 is scheduled for implementation in 2020.

25.7 Risks

25.7.1 Open pit mining

- Mine production rates include reasonable improvements that are included in the LOM and the ability to meet the mine plan requires that these improvements are successfully implemented and maintained. Likewise, the implementation of best-in-practices short- to mid-term planning procedures will need to be implemented to achieve the planned production and vertical advance rate in Phases 3 and 4. Should these planned improvements not occur, mine operating unit costs are at risk to increase as deferred waste stripping is envisioned which could additionally put at risk ore production profiles commencing in approximately 2022. Close monitoring of compliance to mine plan will allow early detection and possible corrective actions to be taken. A visit by Technical Services planning engineers to similar operations with higher vertical advance rates will allow best-practices and knowledge to be transferred to Rainy River operations.
- The prediction of NAG availability for TMA construction is based on current knowledge projected into the future, however, there is a level of risk involved until Phases 3 and 4 start mining in hard-rock and actual geology can be observed and reconciled. Technical Service engineers and geologists should commence NAG reconciliation and geological mapping of areas as they are exposed to project NAG availability. If negative reconciliation is encountered,

consider use of RC grade control drilling applied to NAG to assist in identifying any short- to mid-term NAG deficits (or excess) and improve planning for TMA construction materials. A deficit of NAG for TMA construction would require mining from the EOC which would either require additional mining resources to be acquired or would reduce resources available to execute the current mine plan, both of which would result in increased costs.

25.7.2 Open pit geotechnical considerations – hard rock

Based on the findings of the SRK study, the following risks should be considered:

- The design bench configuration is reliant on best practice blasting to successfully implement the slope recommendations. The kinematic stability work, and the current slope performance, indicates the bench slopes will continue to be susceptible to wedge and planar sliding mechanisms. These mechanisms can contribute to back-break and rock fall issues with improper implementation and there is a risk that design adjustments will be required to reduce stability risks during operations.
- The design configuration of the shallower North Walls (Domain II, III, and IV) that are orientated parallel to foliation are reliant on successful implementation of stab-hole blasting techniques. Should the foliation fabric be irregular in bench scale orientation or too massive to initiate fracturing along continuous structures (at the bench scale) there may be a risk a hang-up will occur that will require additional effort to scale and excavate.
- The slope stability analyses indicate that groundwater has limited influence on overall stability conditions, except for Section 3.5 along the South Wall (FOS of 1.2). Although assessed to be stable, there is a risk that the draw-down will be less positive that the GW2 case modelled and that a lower design FOS may exist (i.e. for a failure related to adversely orientated joints and / or fault structures).
- Several fault-on-joint and fault-on-fault intersections have been identified at the inter-ramp scale. Under the 90% water filling case FOS and / or probability of failure conditions are less than the minimum design criteria. Although, the recommended designs are based on the bench scale stability work, additional study is needed to evaluate the draw-down and groundwater pressures immediately behind the slope face for stability risks related to wedge intersections along identified or other unknown structures. Possible wedges intersect at the inter-ramp scale and should adverse conditions exist, there are related stability risks.
- The 3D structural geology model was developed with the geotechnical and exploration drillhole data and mapping of the existing slope faces. There may be stability risks associated with possible adversely orientated fault structures that were not intercepted by the drillholes or observed in the current pit slopes.
- The pit slopes excavated within the ODM Shear Zone may be susceptible to shallow surface water infiltration conveyed from the north. Inadequate implementation of surface water management could result in infiltration and elevated pore water pressures leading to a slabbing or bench-scale planar sliding instability within weaker strength and intensely foliated rocks.

25.7.3 Underground mining

- There will be some personnel operational readiness / capability risk during the transitioning process from open pit to underground.
- There is some risk that backfill operating costs may be higher if the ratio of CRF / URF is ultimately greater than estimated.
- 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. However, multiple stoping areas operating concurrently will provide significant flexibility.

- Areas where the dip is less than 55° may suffer additional ore loss and / or dilution, or higher costs to recover all the ore in the stope designs. In some non-entry areas, large HW exposures will exist, supported by relatively small pillars. Any failures of the HW or pillars in these locations could generate additional dilution and / or loss of ore.
- Dependence of underground production economic viability on concurrent processing of open pit low grade stockpile material means that inadequate management of such material could jeopardize underground Mineral Reserves.

25.7.4 Tailings management area

The following risks may increase the overall LOM material quantities:

- Adjustments to the TMA West Dam (Dam 4) cross-section to limit potential impacts to BCR1 result in a maximum constructible height of the TMA West Dam (Dam 4) of 379.0 m. Additional raises above 379.0 m may require extensive rework of the TMA West Dam (Dam 4) or may not be feasible.
- The water balance model used as the basis for the Tailings Management Plan includes assumptions of water storage in the pit as well as discharge of water from the WMRS sediment ponds. If water cannot be stored in the pit, or cannot be discharged from the sediment ponds, additional storage may be required in the Rainy River TMA.
- Wick drains may not perform as successfully as assumed at the TMA West Dam (Dam 4) or TMA South Dam adjacent to the West Creek Diversion Channel which may require additional buttressing, additional wick drain installation, or alternative ground improvement approaches.

25.7.5 Other

- There is some risk that the power cost may be higher upon renewing the contract with hydro once the current contract expires at the end of 2024.

25.8 Opportunities

25.8.1 Open pit mining

- Improved early ore definition through the current implementation of RC grade control should allow improved mine planning decisions to be made such that dilution and ore loss can minimized, coupled with improved operational execution in the field. This would lead to better quality ore being sent to the process plant.
- There is a possible opportunity to reduce ore rehandle through an investigation into the root cause behind the elevated quantities of rehandle. A reduction in rehandle quantities could lead to a subsequent reduction in mine operating costs.
- In an elevated metal price environment (compared to LOM assumptions), there is an opportunity to reduce the low grade ore COG and increase material quantities that may be possibly stored in long-term stockpiles. Should this be possible, this would provide additional flexibility to the operation to minimize the impact of any ore production shortfalls and could subsequently allow the underground operation to extend its operational life.

25.8.2 Open pit geotechnical considerations – hard rock

Based on the findings of the SRK study, the following opportunities are considered:

- The bench slope designs are controlled by wedge and planar sliding mechanisms. Should the kinematic controls be less influential on stability there may be opportunities to increase the design BFA, and resulting IRA, in some sectors. This will require successful wall control blasting techniques.

- Should the rock mass condition within the FLS rocks along the North Walls (Domain II, II, and IV) have less continuous foliation structures and a massive rock fabric, there may be opportunities to form steeper design BFA’s that will under-cut foliation with acceptable stability conditions. This will require blast trialing, ongoing geotechnical evaluation and a robust monitoring system.

25.8.3 Underground mining

- There will be an opportunity to save costs on labour for some personnel during the transitioning process from open pit to underground.
- There will be an opportunity to reduce underground backfilling cost by examining the contractor’s labour to perform backfilling activities.
- In a higher gold price environment and subject to realized costs, the proportion of underground Mineral Resources converted to Mineral Reserves could substantially increase.
- There is an opportunity to start mining the Intrepid Zone at an earlier date, depending on results of the orebody investigation project and further economic assessment. A portion of the Intrepid Mineral Resources is already converted to Mineral Reserves and included in the mine plan, and the orebody investigation project could potentially identify more that is economically viable.

25.8.4 Tailings management area

The following opportunities may allow for a reduction in the overall LOM material quantities:

- Additional site investigation at the boundary of TMA South Dam design Zones 3 and 4 may allow for a reduction in wick drain quantities required to prevent impacts to the West Creek Diversion Channel.
- Quantities of Dental Concrete may be reduced through the increased use of form works.
- Feasibility level studies of alternative ground improvement methods may reduce the estimated costs associated with TMA South Dam design Zones 3 and 4.
- Realignment of the TMA North Dam may allow for a reduction in the dam length and may increase the tailings storage capacity. Realignment of the TMA North Dam would require the purchase of property adjacent to the Rainy River TMA.
- Refining assumptions related to PWP response in areas with and without wick drains resulting from dam construction may allow the determination of preloading strategies to reduce the required wick drain quantities.

26 Recommendations

Unless indicated with a corresponding monetary value, recommendations are currently budgeted within New Gold's current LOM plan or are assumed to be able to be undertaken as an everyday task within the corresponding department at site without additional cost.

The QPs offer the following recommendations:

26.1 Geology and Mineral Resources

26.1.1 Geology

- Ensure that the New Gold internal protocol, that requires a 5% insertion rate for CRMs, blanks, and umpire samples, is achieved.
- An additional CRM that covers the COG of the open pit should be acquired.
- If a CRM shows consistent bias at multiple laboratories, this issue needs to be understood and resolved or a new CRM should be obtained. If it is not practical to discard a large CRM inventory, then internal calculation of the CRM expected value and standard deviation would be appropriate. The rationale should be documented.
- Recalculate standard deviations for low-grade Geostats samples based on New Gold data and use these as a measure of performance instead of those indicated on the certificate. These should be used in concert with a recalculated expected value.
- Consider adding the HoleID to the QA/QC sample database as a cross check to ensure QA/QC samples relate to the dataset and the time period in question. AMC makes this recommendation to minimize future investigative work.
- Send any potential new blank material to an analytical lab to ensure the material is below analytical detection with respect to any minerals of economic interest.
- Lower the blank failure limit to 3x detection limit.
- Further investigative work be completed to assess pulp duplicate performance. Such as, applying screen fire assay analyses to a subset of samples in order to better understand the size distribution of gold particles.

26.1.2 Mineral Resource

- AMC recommends that reconciliation of the model, short term controls and production should be done on a rolling 3-month basis and presented graphically, enabling trends to be reviewed.
- Classification of the Mineral Resource should be revisited, smoothing the outlines and removing the spotted dog effect.
- Consider updating the geological model with new thinking which has been collected from mapping and observation in the pit as well as any drilling carried out since production commenced.
- Investigate the high grades of the domain ODM 114 to the west of the open pit. Re-model and re-estimate domain ODM 114.
- Ensure use of a certified lab for assaying of any new in-pit diamond drilling that may be in the block model.

26.2 Mining and Mineral Reserves

26.2.1 Open pit

- Modifying factors in the development of the mine planning resource model should be reviewed periodically when new mining areas are exposed, and additional reconciliation information is gathered to continue validating the model performance.
- Monitor the growth of LGO stockpiles during the LOM to ensure correct strategic decisions are made regarding its use as the LGO stockpile is an integral part of the post-open pit underground mine plan success.
- Investigate developing a very low-grade ore (VLGO) stockpile which may be able to take advantage of higher long-term metal prices and offset any shortage of LGO ore availability if encountered during the mine plan.
- Implement results of reverse circulation grade control results in the development of a short- to mid-term GC model to be used for planning. Consider implementation of reverse circulation grade control for NAG identification during Phase 3 and Phase 4 stripping should negative NAG reconciliation be encountered.
- Minimize mining pit bottoms during peak snowmelt and rainy seasons when water inflows to the pit are at their peak. Attempt to keep pit bottoms during these periods as sumps for water collection and pumping.
- Develop a root-cause evaluation behind the elevated stockpile rehandle rate to reduce this quantity without losing project value.
- Maintain continued focus on improving equipment productivities, utilizations, and availabilities to meet LOM plan requirements.
- Arrange a visit by Technical Services planning engineers to similar operations with higher vertical advance rates to allow best-practices and knowledge to be transferred to Rainy River operations. (Estimate: $10,000).
- The open pit overburden slopes should be instrumented to monitor the performance of the slopes, both prior to and following the placement of the toe berm / slope buttress. VWPs have been proposed to monitor the phreatic surface within the clay units and the excess PWP due to the berm / slope buttress construction. Slope inclinometers have also been proposed to monitor the deformations close to the slope toe. (Estimate $200,000).
- Investigate the implementation of a two-ramp approach through Phase 2/3 to reduce consequences of an instability location above or below critical accesses.
- Utilize a 3D modelling software package (i.e. Leapfrog™) to compile and review existing and future data collected for ongoing geotechnical evaluation.
- Carry out bench face mapping to measure joint and foliation orientation as it is exposed. The foliation orientations carry increased importance for the short-term FW design and performance.
- Carry out a structural geology review in 2020 to map new faults and revise the 3D model interpretation for use in geotechnical stability assessment and design.
- Conduct a pit water management review to understand the surface water, shallow and deep bedrock system. The review should include all the monitoring data, water storage infrastructure, flow data to support an appropriate sump and pumping strategy.
- Implement a monitoring system that includes inspections, prism and radar systems. A prism system should be initially set-up on the existing Phase 1 Pit slopes with prisms systematically installed on new benches as excavated. Determine locations for total station locations and determine the accuracy of the system for consideration in trigger level allocation.
- Install at least two new VWP's in 2020. The installations should be located closer to the planned Phase 2 Pit slopes to monitor and evaluate near-slope groundwater conditions.
- Conduct an inventory of inactive and active monitoring well locations and carry out rehabilitation where required. Incorporate all active monitoring installations, including the 2019 SRK installations, into a single database.

- Carry out regular review of groundwater monitoring data, including excavation, rainfall and nearby water storage influences. Evaluate the data against the pit slope stability assumptions.
- Trial blasting approaches, including the stab-hole approach, for achieving the north wall BFAs for Domains II, III, and IV.
- Develop new blast designs based on the recommended slope design criteria with input from geotechnical.
- Conduct post-blast inspections of new cuts to evaluate the performance and determine the factors that contribute. Provide geotechnical input into the blast designs.
- Consideration to a third-party consultant to review and provide input into the wall control and production blasting (estimate $25,000).
- Develop a workflow for design, QA/QC and performance evaluation for the blast trials that will be carried out along the North Wall.

26.2.2 Underground

- Review the development and mining schedule for each zone to allow adequate time for definition drilling and data analysis before stope development and production commence.
- Undertake close monitoring of mine development and rapid implementation of remedial actions in the event of development advance shortfalls.
- Perform CMSs as part of the production records and reconciliation of production to the Mineral Reserve estimates.
- Install ground control monitoring systems for analysis of the HW and pillar stability in open stope areas ($30,000).
- Review the stability of the overall stope HW for open stopes, including assessment of rib pillar dimensions as local mining knowledge increases.
- Calibrate numerical models for stability of stopes and pillars based on latest geotechnical and CMS data. Investigate stope stability against mine design and sequence using the calibrated model.
- As underground mining progresses and operational experience is gained, reassess separation distance and pillar stability between pit walls and underground stopes.
- As development and mining proceeds in each zone, confirm or adjust ground support plans based on actual conditions encountered.
- Install a microseismic monitoring system for underground ($100,000).
- Should the development schedule allow, consider relocating and reusing pumping, ventilation and electrical equipment such as pumps, fans and switchgear from one portal / ramp area in the next.
- Continually test to verify field performance of backfill binder content.
- As development and mining proceeds in each zone, consider all opportunities for optimized backfill, e.g. in some areas where CRF is currently planned, it may be practicable to use URF instead.
- Where CRF is required, finalize delivery system details in time to fit with production requirements.
- Proceed, as planned for 2020, with the initial Intrepid Zone development. At that time consider an earlier and / or expanded extraction of Intrepid Zone material considering diamond drilling results; cost; overall processing plant grades, throughput and recoveries; the metal price environment; and company strategy.
- As development and mining proceeds for both open pit and underground mining, regularly reconsider all potentially economic mineralization in the context of grades; production rates; cost; overall processing plant grades, throughput and recoveries; the metal price environment; and company strategy.

- Relative to the low conversion ratio of underground Mineral Resources to Mineral Reserves, undertake regular reassessment of what is currently considered marginal material in the context of potential economic benefit of adjusted ratios of open pit and underground mined quantities over time.
- Should mining progress and economic conditions justify an earlier expansion to depth of the Intrepid Zone, reconsider mine planning options for earlier access to other zones and possible optimization of key infrastructure such as ventilation.
- Constantly review all underground mining costs and potential opportunities for improvement.
- Investigate all options, including that of process plant sizing, for an economically viable operation handling underground ore only.

26.3 Metallurgy and Processing

The comminution circuit recommendations are from the OMC grinding circuit audit report and are supported by AMC.

- Increase the SAG mill speed from ~56% Nc to a range of 73% Nc to 75% Nc.
- Increase the SAG motor power utilization from 69% to a range of 80% - 95%. SAG mill power draw can be increased by operating at 9 rpm to 10 rpm (72% – 80% Nc) if required.
- Increase the ball mill steel charge to 30% - 32% to allow better power utilization in the ball mill. Ball mill speed will also need to be optimized.
- Conduct a thorough review to identify the root causes of grinding circuit downtime causes, which would assist in creating preventative maintenance action items which would reduce the likelihood of these downturn event reoccurring.
- Perform routine mill grind outs for steel ball charge measurement or crash stops for total load measurement in the SAG mill and ball mill.
- Create a database which includes details of SAG mill crash stops and ball mill grind-out measurements in order to provide useful information for auditing and grinding circuit performance reviews.
- Investigate equipment and sensor types which can provide real-time data for advanced process control purposes. This will help improve circuit stability. These include conveyor cameras for particle size analysis (Split Engineering and KnowledgeScape), mill vibration sensors for mill load control (MillSlicer and MillsScanner) and FloCEP cyclone sensors for roping detection. (Estimate $300,000).
- Implement real-time control of the SAG mill load, using a combination of bearing pressure control and vibration or acoustic sensor logic. This would replace using the SAG mill bearing pressure for controlling the SAG mill load.
- Introduce more interaction between the metallurgy group and the mining group to log daily / weekly SAG feed ore blends and major ore types.
- Obtain MolyCop Tools© software, for forecasting SAG mill media makeup.
- Program the DCS with equipment sensor operating hour totalizers to assist with maintenance program and calibration.
- Perform routine preventative maintenance and calibration checks on the sump level sensors, cyclone flowmeters and cyclone density meters.
- Perform monthly checks of the SAG feed conveyor weightometer and the pebble return conveyor weightometer.

26.4 Infrastructure and other

- Monitor metal price fluctuation and trends and adapt the LOM plan as required to maximize value.

- Instrumentation readings should be carried out at the specified frequencies to monitor the mine rock and overburden stockpiles performance. In case the amount of waste rock and overburden increases significantly resulting in rate of raise much faster than those assumed in the design, the engineer should be notified to verify the performance of the wick drains.

26.5 Environmental

- Provide training of employees to reduce reliance on consultants for routine work.
- Continue with development and implementation of the EMS.
- Continue consultation and coordination with regulators for permit changes and amendments.
- Plan and start interim reclamation where possible to reduce post mining obligations.
- Continue with revegetation test plots to determine what species and composition works best for stockpiles and disturbed areas.
- Develop a water resources team to provide technical support in order to mitigate risk to operations from water resource uncertainty.

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28 QP Certificates

CERTIFICATE OF AUTHOR

I, Ken Bocking, P.Eng., of Oakville, Ontario, do hereby certify that:

1 I am currently employed as a Principal with Golder Associates Ltd., with an office at 6925 Century Avenue, Suite #100, Mississauga, Ontario L5N 7K2.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of University of Saskatchewan in Saskatoon, Canada (Master of Science in 1978). I am a member in good standing of the Professional Engineers Ontario (License #4253654), the Association of Professional Engineers & Geoscientists of Saskatchewan (License #4131), and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License #400). I have experience in geotechnical engineering and mine waste management.

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.

4 I have visited the Rainy River Project on three occasions, most recently on 11 February 2020 for 1 day.

5 I am responsible for parts of Sections 1, 16, 18, 25, and 26 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report; which comprises technical review of the design of the open pit overburden slopes and design of the stabilization for the mine rock stockpiles.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 16 March 2020

Original signed and sealed by

Ken Bocking, P.Eng.

Principal

Golder Associates Ltd.

CERTIFICATE OF AUTHOR

I, Twila Griffith, P.Geo., of Emo, Ontario, do hereby certify that:

1 I am currently employed as a Senior Environmental Specialist with New Gold Inc., Rainy River Mine, with an office at 5967 Highway 11/71, PO Box 5, Emo, Ontario P0W 1E0.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of the University of Alberta in Edmonton, Canada (Bachelor of Science in 1988 and Master of Science in 1994). I am a member in good standing of the Association of Professional Geoscientists Ontario (License #3218) and the Alberta Professional Engineers and Geoscientists Association (License #50524). I have experience in environmental monitoring, permitting, and compliance.

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.

4 I have worked at the Rainy River Mine in the role of Senior Environmental Specialist from 25 April 2016 to present.

5 I am responsible for Section 20 and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am not independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report as an employee of New Gold Inc. at the Rainy River Mine since 25 April 2016.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 18 March 2020

Original signed and sealed by

Twila Griffith, P.Geo.
Senior Environmental Specialist
New Gold Inc., Rainy River Mine

CERTIFICATE OF AUTHOR

I, Francis J. McCann, P.Eng., of Oakville, Ontario, do hereby certify that:

1 I am currently employed as a General Manager, Toronto / Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 200, 140 Yonge Street, Toronto, Ontario M5C 1X6.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of Queen's University in Kingston, Canada (Bachelor of Science in Applied Sciences - Mining Engineering, in 1992). I am a member in good standing of Professional Engineers Ontario (License #90395393), and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have worked as a Mining Engineer for a total of 28 years since my graduation from university, the majority of which has been spent working in open pit gold mines performing evaluations and studies through operational roles.

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.

4 I have visited the Rainy River Project multiple times, the last visit being from 13-15 January 2020 for 3 days.

5 I am responsible for Sections 2, 3, 4, 5, 19, 22, and 24 and parts of 1, 15, 16, 18, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report. I participated during 2019 in strategic planning and pit / phase design for Rainy River.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 18 March 2020

Original signed and sealed by

Francis J. McCann, P.Eng.

General Manager, Toronto / Principal Mining Engineer

AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Andrew D. R. Millar, MAusIMM, CP (Metallurgy), of Brisbane, Queensland, do hereby certify that:

1 I am currently employed as a Principal Metallurgist with AMC Consultants Pty Ltd, with an office at Level 21, 179 Turbot Street, Brisbane, QLD 4000.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of University of Queensland in Brisbane, Australia (Bachelor of Engineering in Minerals Process in 2001). I am a member in good standing of the Australasian Institute of Mining and Metallurgy (License #329290), and a member of the Registered Professional Engineers of Queensland. I have experience in gold operations.

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.

4 I have visited the Rainy River Project from 13 – 15 August 2019 for 3 days.

5 I am responsible for Sections 13 and 17 and parts of 1, 21, 25, 26 and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had no prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 16 March 2020

Original signed by

Andrew D. R. Millar, MAusIMM, CP (Metallurgy)
Principal Metallurgist
AMC Consultants Pty Ltd

CERTIFICATE OF AUTHOR

I, Mo Molavi, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Director / Mining Services Manager / Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate from Laurentian University in Sudbury, Canada (Bachelor of Engineering in 1979) and McGill University of Montreal, Canada (Master of Engineering in Rock Mechanics and Mining Methods in 1987). I am a registered member in good standing of the Association of Professional Engineers and Geoscientists of Saskatchewan (License #5646), the Association of Engineers and Geoscientists of British Columbia (License #37594), and a Member of the Canadian Institute of Mining, Metallurgy and Petroleum. I have worked as a Mining Engineer for a total of 40 years since my graduation from university and have relevant experience in project management, feasibility studies, and technical report preparations for mining projects.

  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.

4 I have not visited the Rainy River Project.

5 I am responsible for parts of Sections 1 and 16 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property as a QP for previous NI 43-101 Technical Report in 2014.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 19 March 2020

Original signed and sealed by

Mo Molavi, P.Eng.

Director / Mining Services Manager / Principal Mining Engineer

AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Dinara Nussipakynova, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Geologist with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of Kazakh National Polytechnic University (Bachelor of Science and Master of Science in Geology in 1987). I am a member in good standing of the Association of 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 33 years since my graduation from university. My experience is principally in Mineral Resource estimation, database management, and geological interpretation.

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.

4 I have visited the Rainy River Project site on 11 April 2018.

5 I am responsible for Sections 12 and 14, and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property as a QP for previous NI 43-101 Technical Report in 2018.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 18 March 2020

Original signed and sealed by

Dinara Nussipakynova, P.Geo.

Principal Geologist

AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Adrienne A. Ross, P.Geo., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Geology Manager / Principal Geologist with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of the University of Alberta in Edmonton, Canada (Bachelor of Science (Hons) in Geology in 1991) and the University of Western Australia in Perth, Australia (Ph.D. in Geology). I am a registered member in good standing of the Association of Engineers and Geoscientists of British Columbia (License #37418) and the Association of Professional Engineers and Geoscientists of Alberta (Reg. #52751). I have practiced my profession for a total of 26 years since my graduation and have relevant experience in precious and base metal deposits.

   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.

4 I have not visited the Rainy River Project.

5 I am responsible for Sections 6 to 11 and 23, and parts of 1, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report; Reviewed the Intrepid model at a high-level in 2016 as part of AMC's Rainy River Mine Plan Update.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 17 March 2020

Original signed and sealed by

Adrienne A. Ross, P.Geo.
Geology Manager / Principal Geologist
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Edward Saunders, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Senior Consultant, Mining Rock Mechanics with SRK Consulting (Canada) Inc., with an office at 22nd Floor, 1066 West Hastings Street, Vancouver, British Columbia V6E 3X2, Canada.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of the University of Canterbury in New Zealand (Bachelor of Science in Geological Sciences in 2008 and Post-Graduate diploma in Engineering Geology in 2009) and the University of New South Wales in Australia (Master of Engineering Science in Geotechnical Engineering in 2013).

I am a Professional Engineer in good standing of the Province of British Columbia (Reg.#46438) and Ontario (Reg.#100547510). The have experience in:

- Geotechnical investigation, data processing and analytical calculations for greenfield studies and for operational open pit projects in Canada and internationally.
- Geotechnical pit slope stability assessment and design implementation for operational mines located in various geological settings.
- Inspections / audits for the evaluation of pit slope stability performance at existing mine operations, both for regulatory and internal governance.

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.

4 I have visited the Rainy River Mine approximately five times during 2019 and 2020, with the most recent visit occurring on 3-5 February 2020 for 3 days.

5 I am responsible for parts of 1, 16, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have not had prior involvement with the property that is the subject of the Technical Report.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 17 March 2020

Original signed and sealed by

Edward Saunders, P.Eng.

Senior Consultant, Mining Rock Mechanics

SRK Consulting (Canada) Inc.

CERTIFICATE OF AUTHOR

I, Herbert A. Smith, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Senior Principal Mining Engineer with AMC Mining Consultants (Canada) Ltd., with an office at Suite 202, 200 Granville Street, Vancouver, British Columbia V6C 1S4.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of the University of Newcastle Upon Tyne, England (Bachelor of Science in Mining Engineering in 1972 and Master of Science in Rock Mechanics and Excavation Engineering in 1983). I am a registered member in good standing of the Engineers and Geoscientists of British Columbia (License #32378), the Association of Professional Engineers Ontario (License #100017396), the Association of Professional Engineers and Geoscientists of Alberta (License #31494), and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (License #L4413). I have worked as a Mining Engineer for a total of 42 years since my graduation and have relevant experience in underground mining, feasibility studies, and technical report preparation for mining projects.

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.

4 I have not visited the Rainy River Project.

5 I am responsible for parts of Sections 1, 15, 16, 21, 25, 26, and 27 of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property as a QP for previous NI 43-101 Technical Report in 2018.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, this Technical Report contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 19 March 2020

Original signed and sealed by

Herbert A. Smith, P.Eng.
Senior Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

CERTIFICATE OF AUTHOR

I, Andre Zerwer, P.Eng., of Sudbury, Ontario, do hereby certify that:

1 I am currently employed as a Principal Geotechnical Engineer with BGC Engineering Inc., with an office at Unit 265, Rainbow Centre, 40 Elm St., Sudbury, Ontario P3C 1S8.

2 This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine in Ontario, Canada", with an effective date of 12 March 2020, (the "Technical Report") prepared for New Gold Inc. ("the Issuer").

3 I am a graduate of the University of Waterloo in Waterloo, Canada (Bachelor of Science in Geological Engineering in 1992, Master of Science in Earth Science in 1995, and Doctor of Philosophy in Civil Engineering in 2000). I am a member in good standing of the Association of Professional Engineers Ontario (License #100055967). I have worked as a geotechnical engineer for 19 years. My relevant experience for the purpose of this technical report is in the design and construction of mining dams for tailings facilities. I am currently the Engineer of Record for the Tailings and Water Management Dams at the Rainy River Mine.

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.

4 I have visited the Rainy River Mine on many occasions since April 2018. I completed an inspection of the Tailings Management Area and Water Management Dams in June of 2019 and visited the site most recently on 25 to 28 November 2019 to review dam construction completed in 2019.

5 I am responsible for parts of Sections 1, 18, 25, 26, and 27of the Technical Report.

6 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101.

7 I have had prior involvement with the property that is the subject of the Technical Report. I completed an independent review on the geotechnical aspects of the Rainy River Feasibility Study when the property was owned by Rainy River Resources Ltd. My observations were provided in a letter titled "Rainy River Feasibility Study – Independent Review, Plant Site, Tailings Storage, Overburden and Waste Rock Stockpiles, Overburden Pit Slopes, Geotechnical Recommendations", issued 11 February 2013.

8 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1.

9 As of the effective date of the Technical Report and the date of this certificate, to the best of my knowledge, information and belief, the information in the sections of the technical report for which I am responsible contains all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Effective Date: 12 March 2020
Signing Date: 18 March 2020

Original signed and sealed by

Andre Zerwer, P.Eng.
Principal Geotechnical Engineer
BGC Engineering Inc.

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