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

newgold

New Gold, Rainy River Mine

Project Location
5967 Highway 11/71
PO Box 5, Emo
(Ontario) P0W 1E0

Prepared by:

Éric Lecomte, P.Eng.

InnovExplo Inc.
Val-d'Or (Québec)

Andrew Croal, P.Eng.
Michele Della Libera, P.Geo

New Gold Inc.
Toronto (Ontario)

Effective Date: March 28th 2022

Signature Date: March 31st 2022

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CERTIFICATE OF AUTHOR – Andrew Croal

I, Andrew Croal , (Professional Eng.) do hereby certify that:

1. I am employed by NewGold 181 Bay St, Suite 3320, Toronto, Ontario, Canada, M5J 2T3.
2. This certificate applies to the report entitled 'NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada' (the "Technical Report") with an effective date of March
3. 28th ,2022 and a signature date of March 31st, 2022. The Technical Report was prepared for New Gold.
4. I graduated with a Bachelor's degree in "Applied Science, Mining Engineering" from Queen's University (Kingston, Ontario) in 1983.
5. I am a Professional Enginer in the Province of Ontario (100131028 / 15 Dec 2008).
6. I have worked as a Mining Engineer for a total of 39 years since graduating from university. My expertise was acquired while working at large Open Pit mines in Canada, Papua New Guinea, South America and Africa.
7. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/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 QP for the purposes of NI 43-101.
8. I visited the Rainy River property many times as an employee of New Gold.
9. I am responsible for the overall coordination of the Technical Report and I share responsibility for sections 1, 5, 15,16, 18, 19 and 21.
10. I am not independent of the issuer.
11. I had prior involvement with the property that is the subject of this Technical Report.
12. I have read NI 43-101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 31th day of March 2022 in Toronto, Ontario, Canada.

Andrew Croal

Andrew Croal P.Eng. Ontario (100131028 / 15 Dec 2008)

NewGold

Andrew.Croal@newgold.com

LICENSED PROFESSIONAL ENGINEER
A. G. CROAL
100131028
PROVINCE OF ONTARIO

GOLDER
MEMBER OF WSP

CERTIFICATE OF QUALIFIED PERSON KENNETH BOCKING

I, Kenneth Bocking, state that:

(a) I am a Principal at:

Golder Associates Ltd.
6925 Century Avenue, Suite 100
Mississauga, Ontario, Canada, L5N 7K2

(b) This certificate applies to the technical report titled "NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada" (the "Technical Report") with an effective date of March 28, 2022 and a signature date of March 31, 2022. The Technical Report was prepared for New Gold company.

(c) I am a "qualified person" for the purposes of Nation Instrument 43-101 ("NI 43-101"). My qualifications as a qualified person are as follows. I am a graduate of the University of Saskatchewan with a bachelor's in civil engineering (1974) and an M.Sc. in Geotechnical Engineering (1978). I am a member in good standing of the Professional Engineers Ontario (Licence #4253654), and the Association of Professional Engineers and Geoscientists of Saskatchewan (Licence # 4131) and the Northwest Territories and Nunavut Association of Professional Engineers and Geoscientists (Licence # 400). My relevant experience after graduation and over 48 years for the purpose of the Technical Report includes consulting Geotechnical Engineering, specializing in mine waste management since 1988.

(d) My most recent personal inspection of each property described in the Technical Report occurred on October 20, 2020 and was for a duration of one day.

(e) I am responsible for Item(s) 16.2.3 and 18.9 of the Technical Report.

(f) I am independent of the issuer as described in section 1.5 of NI 43-101.

(g) My prior involvement with the property that is the subject of the Technical Report is as follows. I was responsible for the technical review of the detailed design of the open pit overburden slopes and the detailed design of the East Mine Rock Stockpile and the West Mine Rock Stockpile.

(h) I have read NI 43-101 and the parts of the Technical Report for which I am responsible has been prepared in compliance with NI 43-101; and

(i) At the effective date of the Technical Report, to the best of my knowledge, information, and belief, the parts of the Technical Report for which I am responsible, contain(s) all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Dated at Mississauga, Ontario this 29th day of March, 2022

Ken Bocking

Kenneth Bocking, P.Eng. (Ontario, Saskatchewan, NWT/Nunavut).

LICENSED PROFESSIONAL ENGINEER
K. A. BOCKING
29 March 2022
PROVINCE OF ONTARIO

CERTIFICATE OF AUTHOR – Michele Della Libera

I, Michele Della Libera, P.Geo. do hereby certify that:

1. I am employed by New Gold Inc,as Director, Exploration at: 181 Bay Street, Suite 3320, Toronto, Ontario, M5J 2T3
2. This certificate applies to the report entitled "NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada (the "Technical Report") with an effective date of March 28, 2022 and a signature date of March 31, 2022. The Technical Report was prepared for New Gold Inc. company.
3. I graduated with a Master's degree in Earth Sciences (Geology) from the University of Pisa, Italy in 1992.
4. I am registered as Professional Geoscientist in the Province of Ontario, Registration No. 2837.
5. I have worked as geologist for a total of twenty nine (29) years since graduating from university. My expertise was acquired while working as geologist practicing my profession continously for the last 27 years, with involvement in exploration projects from early stage to resource delineation phase as well as in active mining operations. I am experienced in precious and base metals exploration in a variety of geological settings and ore deposit types.
6. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/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 QP for the purposes of NI 43-101.
7. I visited the property on numerous occasions since July 2013, the most recent being January 25-27, 2022. In my role as Director, Exploration for New Gold Inc. I have had direct oversight of all exploration activities at Rainy River since July 2013.
8. I am the author and responsible for section 4-10, 23 as well as co-author of and share responsibility for section 11.
9. I am not independent of the Issuer applying all the tests set out in section 1.5 of NI 43-101.
10. I had prior involvement with the property that is the subject of the Technical Report as Director, Exploration.
11. I have read NI 43-101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 29 day of March 2022 in Toronto, Ontario, Canada.

Michele Della Libera

Michele Della Libera, P.Geo. (APGO Reg. # 2837)
New Gold Inc.
michele.dellalibera@newgold.com

PROFESSIONAL GEOSCIENTIST
MICHELE DELLA LIBERA (MD)
PRACTISING MEMBER
2837
ONTARIO

CERTIFICATE OF AUTHOR – Charles Gagnon, P.Eng

I, Charles Gagnon, (profession ing) do hereby certify that:

1. I am currently Owner and President with CGM with an office located at 1155, avenue des Érables, Québec. QC, G1R 2N4.
2. This certificate applies to the report entitled NI 43-101 Technical Report for the Rainy River Mine, Ontario Canada (the "Technical Report") with an effective date of March 28th, 2022 and a signature date of March 31st, 2022. The Technical Report was prepared for New Gold company.
3. I have graduated from Laval University with a B.Sc. in Mining Engineering in 2002, and from University Laval, Québec, Canada with a M.Sc. in 2005.
4. I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 130730).
5. I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, teaching (Mining engineering department, Laval University, Québec city), engineering and financial evaluations for 17 years, including (Eleonore (Goldcorp), Perseverance (Xstrata-Zinc), Bracemac-Mcleod (Glenncore)).
6. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/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 QP for the purposes of NI 43-101.
7. I have not visited the site property that is the subject of this report
8. I have participated in the preparation of the Technical Report and am responsible for the supervision or creation of the following sections and sub-sections: 16.3.8 and part of items 21 for capital and operating cost related to ventilation only.
9. I am independent of the issuer applying all the tests in section 1.5 of NI 43-101.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43-101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 30th of March 2022 Québec city, Québec, Canada.

Charles Gagnon P.Eng. OIQ, 130730

CGM

charles.gagnon@cgmexpert.com

INNOVEXPLO

CERTIFICATE OF AUTHOR – Éric Lecomte

I, Eric Lecomte, (P.Eng.) do hereby certify that:

1. I am employed by InnovExplo Inc. 560, 3rd Avenue, Val-d'Or, Quebec, J9P 1S4
2. This certificate applies to the report entitled "NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada" (the "Technical Report") with an effective date of March 28th, 2022 and a signature date of March 31st, 2022. The Technical Report was prepared for New Gold company.
3. I graduated from Laval University, Québec, Canada with a Bachelor's degree in Mining Engineering in 1998.
4. I am a Professional Engineer registered with the Ordre des ingénieurs du Québec (OIQ Licence: 122047).
5. I have worked as Mine Engineer for a total of twenty one (21) years since graduating from university. My expertise was acquired while working as a mining engineer. During these different years, I have occupied different positions both technical and operational related to mining engineering, and this, in underground operations as well as in open pit.
6. I have read the definition of a qualified person ("QP") set out in Regulation 43101/National Instrument 43101 ("NI 43101") and certify that by reason of my education, affiliation with a professional association (as defined in NI 43101) and past relevant work experience, I fulfill the requirements to be a QP for the purposes of NI 43101.
7. I visited the property one time on March 15, 2022 for the purpose of the Technical Report.
8. I am responsible for the overall supervision of the Technical Report and I am the co-author of and share responsibility for sections 1, 2, 3, 15,16, 21, 22, 24 and 27.
9. I am independent of the issuer applying all the tests in section 1.5 of NI 43101.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. I have read NI 43101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 29th day of March 2022 in Val-d'Or, Québec, Canada.

ing.

Eric Lecomte, P.Eng. (OIQ 122047)

InnovExplo

eric.lecomte@innovexplo.com

INGÉNIEUR
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29/03/2022

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AMC consultants
mine smarter

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, Ontario, Canada", with an effective date of 28 March 2022, (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 30 years since my graduation from university, the majority of which has been spent working in open pit gold mines performing operational roles, project evaluations and studies.

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 multiple times, the last visit being from 13-15 January 2020 for 3 days.

5 I am responsible for parts of Sections 15 and 16 related to open pit Mineral Reserves and mine planning aspects and related disclosure in Sections 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. I participated as a QP of the previous NI 43-101 Technical Report for the property which had an effective date of 12 March 2020.

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 : 28 March 2022
Signing Date : 31 March 2022

LICENSED PROFESSIONAL ENGINEER
F. J. McCANN
2022-03-31
PROVINCE OF ONTARIO

Francis J. McCann, P.Eng.
General Manager, Toronto / Principal Mining Engineer
AMC Mining Consultants (Canada) Ltd.

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AMC consultants
mine smarter

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, Ontario, Canada", with an effective date of 28 March 2022, (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, 11, 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 2020.

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: 28 March 2022
Signing Date: 31 March 2022

PROFESSIONAL PROVINCE OF D. NUSSIPAKYNOVA # 37412 BRITISH COLUMBIA GEOSCIENTIST

Dinara Nussipakynova, P.Geo.
Principal Geologist
AMC Mining Consultants (Canada) Ltd.

srk consulting

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CERTIFICATE OF AUTHOR – Edward Saunders

I, Edward Saunders, P.Eng., of Vancouver, British Columbia, do hereby certify that:

1 I am currently employed as a Principal Consultant, Mining Rock Mechanics with SRK Consulting (Canada) Inc., with an office at 22nd Floor, 1066 West Hastings Street, Vancouver, BC, V6E 3X2, Canada;

2 This certificate applies to the report entitled 'NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada' {the "Technical Report") with an effective date of March 28th, 2022 and a signature date of March 31st, 2022. The Technical Report was prepared for New Gold company;

3 I am a graduate of University of Canterbury, New Zealand in 2008 with a B.Sc. degree in Geological Sciences, and University of Canterbury, New Zealand in 2009 with a Post-Graduate diploma in Engineering Geology, and University of New South Wales, Australia in 2013 with a Masters of Engineering Science degree in Geotechnical Engineering.

4 I am a Professional Engineer in good standing of the Province of Ontario (Reg.# 100547510). I have 13-years' experience in:

- Geotechnical investigation, data processing and analytical calculations for project 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 deposits and environmental settings.

5 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.

6 I have visited the Rainy River Mine from on numerous occasions since February 2020. The most recent visit occurring in September 2021.

7 I am responsible for parts of 16, 25 and 26 of the Technical Report.

8 I am independent of the Issuer and related companies applying all of the tests in Section 1.5 of the NI 43-101;

9 I had prior involvement with the property that is the subject of the Technical Report.

10 I have read NI 43-101, and the Technical Report has been prepared in compliance with NI 43-101 and Form 43-101F1;

11 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;

Signed this 31 day of March 2022 in Vancouver, British Columbia, Canada.

LICENSED PROFESSIONAL ENGINEER
E. A. SAUNDERS
100547510
PROVINCE OF ONTARIO

Edward Saunders, P.Eng. (PEO Reg# 100547510)
Principal Consultant, Mining Rock Mechanics
SRK Consulting (Canada) Inc.

INNOVEXPLO

HALYARD

CERTIFICATE OF QUALIFIED PERSON – Justin Taylor, P.Eng

I, Justin Taylor, P.Eng, do hereby certify that:

1. I am a Founder, Director, Senior Project Manager and Engineer with Halyard Inc. for the last 9 years, with a business address at 212 King St W #501, Toronto, ON M5H 1K5 Canada.
2. This certificate applies to the report entitled 'NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada' {the "Technical Report") with an effective date of March 28th ,2022 and a signature date of March 31st, 2022. The Technical Report was prepared for New Gold company.
3. I am a graduate of the University of Pretoria in South Africa with degrees in Mechanical Engineering and Maintenance Engineering and a diploma in business administration. I obtained my undergraduate degree in 1999.
4. I am a member in good standing of the Professional Engineers of Ontario (membership number: 100140330).
5. I have practiced my profession in the mining industry continuously since graduation. My relevant experience includes over 20 years of engineering in project development specifically pertaining to minerals processing, materials handling and project management in the mining, processing and technical industries. Half of this experience is relevant to the Canadian environment whereas the balance is international.
6. I have read the definition of “qualified person” set out in National Instrument 43-101 Standards of Disclosure for Mineral Projects (“NI 43-101”) and certify that by virtue of my education, affiliation 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.
7. I am independent of the issuer, New Gold Inc., as defined in Section 1.5 of NI 43-101.
8. I am responsible for Section 17 and part of Section 21 and accept professional responsibility for those sections of the Technical Report.
9. Whilst I have not personally visited the Rainy River Mine site, key members of my team under my supervision who undertook significant work on this study visited the site on the 25th of May 2021.
10. I have not had prior involvement with the property that is the subject of the Technical Report.
11. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the portions of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the portions of the Technical Report for which I am responsible not misleading.
12. I have read NI 43-101 and Form 43-101F1, and the Technical Report has been prepared in accordance with NI 43-101 and Form 43-101F1.

Signed this 31st day of March 2022 Toronto, Ontario, Canada.

Signed this 31st day of March 2022 Toronto, Ontario, Canada.

Justin Taylor, P.Eng,
President, Halyard Inc.

LICENSED PROFESSIONAL ENGINEER
J. D. TAYLOR
100140330
2022-03-31
PROVINCE OF ONTARIO

CERTIFICATE OF AUTHOR – Mohammad Taghimohammadi

I, Mohammad Taghimohammadi, P.Eng., M.Sc., do hereby certify that:

1. I am Manager – Processing with New Gold Inc., 181 Bay Street, Suite 3320, Toronto, Ontario, M5J 2T3.
2. This certificate applies to the report entitled "NI 43-101 Technical Report for the Rainy
3. River Mine, Ontario, Canada") with an effective date of March 28th, 2022 and a signature date of March 30th, 2022. The Technical Report was prepared for New Gold Inc.
4. I graduated with a Bachelor's degree in Mining Engineering from Imam Khomeini International University (Qazvin, Iran) in 2004. I am a graduate of Amirkabir University of Technology (Tehran, Iran) in 2006 with a Master's degree in Mineral Processing.
5. I am registered as a Professional Engineer in the province of Ontario (PEO #100167579).
6. I have worked as a mineral processing engineer for a total of eighteen (18) years since graduating from university. My relevant experience for the purpose of the Technical Report is:
   - My position as Manager – Processing with New Gold Inc.
   - My position as Senior Plant Process Engineer with Rosebel Gold Mines, Suriname.
   - My position as Process Engineer with SGS Canada Inc.
   - My Position as Mineral Processing Engineer with Kahanroba Co., Iran.
7. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/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 QP for the purposes of NI 43-101.
8. I have visited the Rainy River Mine multiple times since 2020, most recently in March 2022.
9. I am responsible for Sections 13 and 17.
10. I am not independent of the issuer applying all the tests in section 1.5 of NI 43-101.
11. I had prior involvement with the property that is the subject of the Technical Report as an employee of New Gold Inc. since 2020.
12. I have read NI 43-101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
13. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 30th day of March 2022 in Toronto, Ontario, Canada.

Taghimohammadi

Mohammad Taghimohammadi, (P.Eng., M.Sc.) (PEO #100167579)

New Gold Inc.

Mohammad.Taghimohammadi@newgold.com

INNOVEXPLO

CERTIFICATE OF AUTHOR – Sitotaw Yirdaw-Zeleke

I, Sitotaw Yirdaw-Zeleke, P.Eng., do hereby certify that:

1. I am employed by New Gold Inc., Rainy River Mine,5967 Highway 11/71, P.O. Box 5, Emo Ontario, Canada, P0W 1E0.
2. This certificate applies to the report entitled "NI 43-101 Technical Report for the Rainy River Mine, Ontario, Canada" (the "Technical Report") with an effective date of March 28th, 2022 and a signature date of March 29th, 2022. The Technical Report was prepared for New Gold company.
3. I graduated with a Bachelor's degree in Civil Engineering (Hydraulic) from Arba Minch Universiy (Arba Minch, Ethiopia) in 2000, a Master degree in Water Resource Engineering from Free University of Brussels (Brussels, Belgium) in 2003, and a Ph.D. degree in Civil Engineering (Water Resources) form University of Manitoba (Winnipeg, Manitoba) in 2010.
4. I am a member of the Professional Engineers Ontario (PEO No. 100145990).
5. I have worked as a Profesional Engineer for a total of 14 years since graduating from university. My expertise was acquired while working as profesional engineer in Water Resources Engineering.
6. I have read the definition of a qualified person ("QP") set out in Regulation 43-101/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 QP for the purposes of NI 43-101.
7. I have been working on the Raniy River Mine since Januray 2020.
8. I am the author and responsible for section 20.
9. I am not independent of the Issuer applying all the tests set out in section 1.5 of NI 43 101.
10. I had prior involvement with the property that is the subject of the Technical Report as of 2020 as a Senior Water Resources Engineer.
11. I have read NI 43-101 and the items of the Technical Report for which I am responsible have been prepared in compliance with that instrument.
12. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make the Technical Report not misleading.

Signed this 29th day of March 2022 in Winnipeg, Manitoba, Canada.

Sitotaw Yirdaw-Zeleke, Ph.D., P.Eng. (PEO No. 100145990)
New Gold Inc.
Sitotaw.Yirdaw@newgold.com

LICENSED PROFESSIONAL ENGINEER
S.YIRDAW ZELEKE
100145990
29/03/2022
PROVINCE OF ONTARIO

ABBREVIATIONS & ACRONYMS

Abbreviations & Acronyms	Description
$	United States dollar
%	Percentage
°	Degree
°C	Degrees Celsius
$\boldsymbol{\mu}$	Poisson's ratio
µm	Micrometre
$\boldsymbol{\sigma_t}$	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
BBMWi	Bond ball mill work index
BBW	Beach below water
BC	British Columbia
BCR	Biochemical Reactor
BFA	Bench face angle
BRE	Brenna Formation
BWi	Bond work index
C	Gravity adjustment factor
c	Cohesion
C$	Canadian dollar
Capex	Capital expenditure
CaO	Calcium oxide

Abbreviations & Acronyms	Description
CCIC	Caracle Creek International Consulting Inc.
CDA	Canadian Dam Association
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

Abbreviations & Acronyms	Description
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
g	Gram
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

Abbreviations & Acronyms	Description
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
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

Abbreviations & Acronyms	Description
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
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

Abbreviations & Acronyms	Description
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
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
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

Abbreviations & Acronyms	Description
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
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

Abbreviations & Acronyms	Description
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

1 SUMMARY

New Gold Inc. (NG) commissioned Innovexplo Engineering Canada Inc. (Innovexplo) to update the previous NI 43-101. The main goal was to improve processing, production, cash flow and economics. Introducing the UG Main Zones (extension of the Open Pit with an underground production of 4,500tpd).

This Technical Report (Report) on the Rainy River Property (Property) 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 Technical Report will demonstrate the economical value of the extension of the actual Rainy River Open Pit through underground mining in the UG Main Zones. This NI 43-101 reflects the conversion of underground Mineral Resources to Mineral Reserves in this area.

Table 1.1 presents an overview of the mine schedule production, contained metals and indicated recovery (open pit, stockpile and UG included):

Table 1.1 – Production Schedule Overview

Year	Processed Ore	Contained Metal					
	Tonnes	Gold (g/t)	Gold (oz)	Gold (Recovery %)	Silver (g/t)	Silver (oz)	Silver (Recovery %)
2022	9,463,416	0.97	294,761	88.8%	2.4	726,742	57.9%
2023	9,855,000	0.97	306,749	88.4%	3.0	954,765	57.7%
2024	9,855,000	1.10	348,195	89.0%	2.9	922,174	57.4%
2025	9,855,000	1.14	360,803	89.1%	3.0	949,478	58.0%
2026	9,855,000	1.15	363,408	88.9%	3.0	946,209	57.5%
2027	9,855,000	1.15	365,232	88.9%	3.2	1,005,945	57.3%
2028	7,000,652	1.31	295,582	88.9%	3.6	802,173	58.0%
2029	1,643,071	3.10	163,874	94.9%	6.4	335,833	59.7%
2030	1,625,515	3.45	180,075	95.2%	4.3	223,783	58.6%
2031	1,212,232	3.07	119,611	94.9%	4.0	155,068	58.0%
Total	70,219,886	1.24	2,798,288	89.3%	3.1	7,022,169	57.7%

1.1 Introduction

This NI 43-101 Technical Report (Report) on the Rainy River Property (Property) located in north-western Ontario (ON) in Canada has been prepared by Innovexplo Mining Consultants (Canada) headquartered in Val-d'Or on behalf of New Gold Inc. (New Gold) headquartered in Toronto, Canada. It has been prepared to a standard which is in accordance with part of 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 12 March 2020 and filed on SEDAR on 27 March 2020, 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 Ontario (ON) and British Columbia (BC) and a mine in Mexico. New Gold owns 100% of the two Canadian operations, Rainy River and New Afton. The Cerro San Pedro Mine in Mexico is under reclamation and is also 100% owned by New Gold.

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

This Report was done by several QP according to their own field of expertise. Find in Table 1.2 the list of all QPs.

Table 1.2 – List of QPs

	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**	**Items of report**
QP	Mr E. Lecomte	Senior Mining Engineer	InnovExplo	No	15 March 2022	P.Eng. (Qc)	Items, 1, 2, 3, 15, 16, 21 22, 24, 27
QP	Mr A. Croal	Director Technical Services	New Gold Inc.	No	NA	P.Eng. (ON)	Parts of Items 1, 18 Infrastructure, 5, 19, and 21
QP	Mr F. McCann	General Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	Various, last visit 13-15 Jan 2020	P.Eng. (ON)	Parts of Items15 and 16 pertaining to open pit Mineral Reserves and mine planning aspects and related disclosure in Sections 1, 25, 26, and 27.
QP	Mr M. Della Libera	Director Exploration	New Gold Inc.	No	Various, last Jan 25-27, 2022	P.Geo. (ON)	Items 4 to 10, part of Item 11, Item 23
QP	Ms D. Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	11 Apr 2018	P.Geo. (BC)	Items 12 and 14, and related disclosure in Items 1,11, 25 ,26 and 27
QP	Mr K. Bocking	Principal	Golder Associates Ltd.	Yes	Various, last visit 20 Oct 2020	P.Eng. (ON)	Part of Items 16 and 18 (Soft rock aspects of OP and waste storage areas.)
QP	Mr E. Saunders	Senior Consultant, Mining Rock Mechanics	SRK Consulting (Canada) Inc.	Yes	Various, last visit 3-5 Feb 2020	P.Eng. (ON)	Item 16 geomechanics. (Hard rock aspects of OP)
QP	Mr A. Zerwer	Principal Geotechnical Engineer	BGC Engineering Inc.	Yes	Various, last visit 15-18 Nov 2021	P.Eng. (ON)	Part ofItem 18 tailings dam

	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**	**Items of report**
QP	Mr M. Taghimohammadi	Senior Processing Engineer	New Gold Inc.	No	NA	P.Eng. (ON)	Items 13 and 17
QP	Mr S. Yirdaw	Senior Environmental Engineer	New Gold Inc., Rainy River Mine	No	NA	P.Eng. (ON)	Item 20 Environment
QP	Mr J. Taylor	President	Halyard Inc	No	27-May-21	P.Eng. (ON)	17 small scale processing plant
QP	Mr C. Gagnon	Principal Ventilation Engineer	CGMexpert	No	NA	P.Eng. (Qc)	Part of 16

1.2 Property description and location

The Property comprises a portfolio of 209 patented mining rights, surface rights (SR), and Crown Lease properties. The Project Lands covering the mine area comprise 121 separate properties of which New Gold has the rights to the Surface and Minerals, with this area covering approximately 6,141 hectares (ha). There are also 1,157 unpatented claims and the total area covers approximately 36,657 ha. All unpatented claims are in good standing. The mine is in the townships of Fleming, Mather, Menary, Patullo, Potts, Richardson, Senn, Sifton, and Tait.

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, for 369 km. These access roads are sealed allowing year-round access.

1.3 Geology

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.

1.4 Mineralization

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.

1.5 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 2021. 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.

1.6 Mineral processing and 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 QP'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.

New Gold engaged Halyard to investigate the implications of significantly reducing the throughput of their Rainy River (RR) process plant and quantify a range of possible scenarios. The current plant capacity is rated at approximately 25 to 27 ktpd and the investigation considered the impacts of reconfiguring it to operate at 4 to 5 ktpd. This would accommodate ore from underground mining only when the ore from the open pit and waste ore stockpiles is depleted.

To address this scenario and arrive at an optimum "fit-for-purpose" solution, Halyard split the project into two phases:

- Phase 1 - Project definition and consideration of all alternatives, covering mainly comminution but also the downstream plant.
- Phase 2 – Cost estimation of the selected option to PFS level.

Phase 1 evaluated six possible options. Options 2a and b, as well as 3 were carried through to Phase 2 for further evaluation. The final recommended option is 2b, which involves minimal changes in that the existing SAG and ball mill are turned down and operated for six months per year, during the warmer months.

1.7 Mineral Resources Estimate

The Mineral Resource estimates for the Rainy River Mine are based on two block models. These are for the open pit and UG underground Main Zones and underground Intrepid Zones. The Main Zone was modelled and estimated by Mr Mauro Bassotti (formerly of New Gold), and the estimate for the Intrepid Zone 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 2021 is presented in Table 1.3. 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.

INNOVEXPLO

Table 1.3 – Mineral Resources as of 31 December 2021

Category	Tonnes & grade			Contained metal	
	Tonnes	Gold	Silver	Gold	Silver
	(t x '000)	(g/t)	(g/t)	(K oz)	(K oz)
Direct processing Mineral Resources					
Open pit					
Measured	570	1.61	3.0	30	55
Indicated	3,131	1.48	3.2	149	325
Sub-total open pit M + I	3,701	1.50	3.2	179	380
Inferred	481	0.98	2.5	15	38
Underground					
Measured	-	-	-	-	-
Indicated	14,014	2.99	7.6	1,348	3,422
Sub-total underground M + I	14,014	2.99	7.6	1,348	3,422
Inferred	1,593	3.30	2.7	169	141
Low grade Mineral Resources					
Open pit					
Measured	192	0.34	2.0	2	12
Indicated	1,268	0.34	1.9	14	80
Sub-total open pit M + I	1,460	0.34	2.0	16	92
Inferred	404	0.35	1.3	5	17
Total Mineral Resources					
Measured	762	1.29	2.7	32	67
Indicated	18,413	2.55	6.5	1,511	3,827
Total M + I Mineral Resources	19,175	2.50	6.3	1,543	3,894
Total Inferred Mineral Resources	2,478	2.37	2.5	189	196

Notes:

1. CIM Definition Standards (2014).
2. The Mineral Resources are stated exclusive of Mineral Reserves.
3. Mineral Resources were estimated using a long-term gold price of US$1,500 per troy oz and a long-term silver price of US$21 per troy oz. The exchange rate used was C$1.25: US$1 (C$1 = US$0.80).
4. Direct processing open pit Mineral Resources are reported at a gold equivalent (AuEq) cut-off grade 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 are reported at a gold equivalent cut-off of 0.30 g/t.
5. Gold equivalency for open pit was calculated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 90)].
6. Open pit assumptions include:
    a. Average gold and silver recoveries of 90% and 60%, respectively.
    b. Open pit Mineral Resources were constrained by a conceptual pit shell and exclude underground Mineral Reserves within the pit shell.
    c. Inferred open pit Mineral Resources include Inferred material from within the Mineral Reserve open pit.
7. Direct processing underground Mineral Resources are reported at a gold equivalent cut-off grade of 1.70 g/t.
8. Gold equivalency for underground was calculated as AuEq = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 95)].
9. 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.
    - Underground Mineral Resources were restricted by a vetting process that excluded clusters of blocks distal to the MSO Mineral Reserve shapes.

10. The Qualified Person for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC Mining Consultants (Canada) Ltd.
11. Totals may not add exactly due to rounding.
12. Tonnes and grades are in metric units.
13. All costs in US$ unless stated otherwise
14. Effective date of Mineral Resources is 31 December 2021.

Mineral Resources were reported from combined open pit and underground models that were 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 (ID2) 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.

1.8 Mineral Reserve Estimate

The Mineral Reserve estimates conform to CIM Definition Standards (2014) and only include Measured and Indicated Mineral Resources and do not include any inferred mineral resources. Mineral reserves are the estimated tonnage and grade of ore that is considered economically viable for extraction.

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 InnovExplo under the guidance of Mr Éric Lecomte, P.Eng., a mining engineer employed by InnovExplo. Mr Lecomte 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 1.4.

Table 1.4 – Rainy River Mineral Reserves Estimates

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	26,276	0.72	2.2	605	1,837
Probable	31,288	0.95	2.1	953	2,101
Sub-total open pit	57,563	0.84	2.1	1,558	3,938
Underground					
Proven	-	-	-	-	-
Probable	12,657	3.05	7.6	1,241	3,084
Sub-total underground	12,657	3.05	7.6	1,241	3,084
Total					
Proven	26,276	0.72	2.2	605	1,837
Probable	43,944	1.55	3.7	2,194	5,185
Total Mineral Reserves	70,220	1.24	3.1	2,799	7,022

Notes:

1. CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Reserves.
2. Mineral Reserves are estimated using a long-term gold price of US$1,400 per troy oz and a long-term silver price of US$19 per troy oz. The exchange rate used was 1:1.25 US$:C$.
3. 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) * 19 * 60)/ (1,400 * 90)].
4. Open pit assumptions include:
    - COGs applied to a regularized 10 m x 10 m x 10 m mine planning block model, which was generated from re blocking the original resource model. Modifying factors representing a potential dilution of 3.3 m and ore loss of 0.2 m were applied, including a factor of 0.89 applied against the gold grade in the East Lobe.
    - Metal recoveries are variable dependent on metal head grade. At Mineral Reserve COG, the gold recoveries are as follows:
        a. DPO
           CAP zone gold = 73.9%
           Non-CAP zone gold: ODM=83.7%, 433=92.0%, HS=85.4%
        b. LGO
           CAP zone gold = 73.1%
           Non-CAP zone gold: ODM=78.5%, 433=91.3%, HS=81.2%
        c. Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
5. Underground Mineral Reserves for UG Main are estimated at an AuEq COG of 1.74 g/t for Phase 1, AuEq COG of 2.25 g/t for Phase 2, and 0.83 g/t for development. Underground Mineral Reserves for Intrepid are estimated at an AuEq COG of 1.93 g/t.
6. Underground assumptions include:
    - In UGMain Zones and Intrepid, the hanging wall (HW) and footwall (FW) dilution of 0.6 m and 0.3 m, respectively, with total unplanned dilution of 14% approximately.
        a. Average mining recovery estimated as 95% for UG Main Zones and Intrepid.
        b. Average gold and silver mill recovery of 95% and 60%, respectively, for UG Main Zones and Intrepid
    - Cut-off value of CDN$84.24/t, CDN$75.69/t & CDN$98.05/t (Intrepid, UG Main Phase 1 & Phase 2, respectively), inclusive of costs for mining, processing, General and Administrative (G&A), refining & transport and royalties.

7. The qualified persons responsible for this item of the technical report are not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimates.
8. Effective date of Mineral Reserves is 31 December 2021.
9. Totals may not add exactly due to rounding.

1.9 Mining Methods

1.9.1 Open Pit

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 153,000 tpd of ore and waste and has an overall strip ratio of 2.32:1 (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).

Currently, there are recommendations to perform blast trials to evaluate potential back-break and bench-scale rock hazards through the IMV prior to excavation in the southwest design sectors. Based on these trials there may be requirements to modify the design recommendations to improve performance and safety around the planned Phase 4 southwest ramps.

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. As it is not always possible to separate the DPO from the LGO in the field resulting in a blending of the material types, the current mine plan includes an increased proportion of LGO stockpiles being rehandled and blended with DPO on an annual basis, to better reflect operational experience.This results in an open pit mine life extending to Q1-2025 with stockpile rehandling occurring in parallel with the underground operations through to Q4-2028 to fulfill available process plant capacity prior to the mill capacity being modified to only manage underground ore sources.

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 three locations: the East Mine rock stockpile (EMRS), the West Mine rock stockpile (WMRS) and the In-Pit rock stockpile (IPRS).

NAG requirements for the tailings management area (TMA) construction are capable of being fulfilled from in-pit mine production. However, NAG quantities being extracted from the mine after 2023 will be significantly reduced and it is recommended that New Gold review mitigating strategies to ensure sufficient quantities are available when required, 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. No future mining of the east outcrop (EOC) for NAG construction rock is included in the current mining schedule.

No further additional nor replacement open pit mine principal equipment fleet is considered for purchase during the remaining LOM plan.

1.9.2 Underground

The Underground Operations (UG Main Zones & Intrepid Zone) are designed as a mechanized ramp accessed mines that will use longitudinal long hole open stope techniques to exploit these underground Mineral Reserves. The location, size, shape, orientation (dip), and physical properties of the mineral deposit generally determine the selection of the appropriate mining method.

Level spacing is set at 25m. This has been evaluated as the best alternative between 20m and 30m level spacing to maximize profitability, while minimizing drill & blast challenges (mainly excessive deviation and dilution).

Mine development in the Underground Operations zones will employ numerous production fronts to maximize productivity and flexibility to reach the targeted 4,500 t/d rate. Two main long-hole mining methods will be employed: longitudinal retreat and transverse. The transverse stoping is only present in ODM Main zone, the widest zone of the mine, where stope's width exceeds 20.0m.

The Underground Resources consider the following items:

- 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 December 31, 2021.
- Underground Mineral Resources were restricted by a vetting process that excluded clusters of blocks distal to the MSO Mineral Reserve shapes.

The Underground Reserves consider the following items:

- In UG Main Zones and Intrepid, the hanging wall (HW) and footwall (FW) dilution of 0.6 m and 0.3 m, respectively, with total unplanned dilution of 14% approximately.
- Average mining recovery estimated as 95% for UG Main Zones and Intrepid.
- Average gold and silver mill recovery of 95% and 60%, respectively, for UG Main Zones and Intrepid
- Cut-off value of CDN$84.24/t, CDN$75.69/t & CDN$98.05/t (Intrepid, UG Main Phase 1 & Phase 2, respectively), inclusive of costs for mining, processing, General and Administrative (G&A), refining & transport and royalties.

The qualified persons responsible for this item of the technical report are not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimates.

All costs in US$ unless stated otherwise

1.10 Recovery Methods

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 P80 for the plant process design criteria. This study was based on the testwork data. A grind size P80 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 P80 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 the QP'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.

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. 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.

A new gold recovery formula for Non-CAP Zone was developed in October 2020. A multi-linear regression has been utilized to better represent gold recovery.

1.11 Project Infrastructure

Regarding general infrastructure, primary access roads, mine haul roads, truck shop, truck wash bay, fuel bays, explosive magazine and emulsion plant, warehousing, lubricant and fuel storage, principal buildings, assay lab, camp, ceremonial roundhouse, emergency power arrangements and communications facilities are all in place and appropriate to support ongoing mining operations.

The TMA and related water management structures are well described in Item 18. Tailing's deposition in TMA Cell 1 commenced in November 2017 with placement into TMA Cell 2 beginning in May 2018. Generally, the tailings deposition strategy is to establish tailings beaches upstream of the perimeter dams (i.e., TMA North Dam, TMA West Dam [Dams 4 and 5], and TMA South Dam), while maintaining a pond around the fixed reclaim located at TMA Cell 2. Since 2017, the dams have been constructed sequentially every year. The TMA is designed to provide sufficient containment for the projected tailings storage requirements and operational pond volume. The Enmvironmental Design Flood (EDF) is to be stored below the TMA emergency spillway invert level (also referred as the EDF Level or EDFL) and the TMA emergency spillway is designed to pass Inflow Design Flood (IDF). By 2025 the TMA is projected to have reached a crest elevation of 379.1m. The material quantities required for construction are well known, available, sufficient and the site teams are experienced in ongoing dam construction. TMA construction costs are well known and well managed. Construction costs for subsequent TMA storage to accommodate UG mined tonnage have been included in the Rainy River capital cost model and the UG cut-off grade calculations.

1.12 Market Studies

Project economics have been assessed using the following metal prices:

- Gold price = $1,400/oz
- Silver price = $19/oz

According to the London Bullion Market Association (LBMA), the average daily PM Fix gold price for 2021 was $1,799 per troy ounce. The three-year and five-year rolling average prices through the end of December 2021 are $1,651. and $1,496 per troy ounce, respectively.

According to LBMA, the average daily silver price for 2021 was $25.14 per troy ounce. The three year and five-year rolling average prices through the end of December 2021 are $20.71 and $18.98 per troy ounce, respectively.

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 296 k oz gold and 520 k oz silver over the period 2022 – 2028 and an annual average of 150 k oz of gold and 145 k oz of silver over the period 2029 to the end of the life of mine, for a total annual average of 252 k oz gold and 407 k oz of silver.

1.13 Environment and Permitting

1.13.1 Permitting and authorizations

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 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 in Item 20 of the report.

1.13.2 Closure plans

The Rainy River Closure Plan, dated 22 January 2015, was filed by the ENDM 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 ENDM 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. Once provided, it was filed by ENDM.

The Closure Plan has included consultation with agencies, the Aboriginal Community(s) and the public. These consultations will continue through to closure and beyond.

The cost estimate for implementing project closure in the Environmental Assessment (EA) was estimated to be $118M, and assumed third party implementation costs, no resale or scrap values, and that all materials will be treated as waste. The current financial assurance obligation / commitment is $104M based on current disturbance as of 31 December 2021.

1.14 Capital and Operating Costs

Capital and Operating costs for the Open Pit have been estimated by New Gold throughout their 2022 Budget and LOM planning process. Underground Capital and Operating cost were evaluated by Innovexplo.

Total LOM capital costs are estimated to total $718M as summarized in Table 21.1 of the report. This excludes $104M in funds identified for progressive and final closure.

1.14.1 Open Pit 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.

Total LOM capital costs are estimated to total $193M as summarized in Table 21.2 of Item 21.4 of the report. This excludes $104M in funds identified for progressive and final closure.

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 a replacement water truck, drill automation systems, dewatering pumps, etc.
- Capitalized / deferred stripping costs associated with the extraction of 43 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.

1.14.2 Underground Capital Costs

The underground LOM capital cost is estimated to total CDN$391M, inclusive of contingency, with CDN$65M in project capital and CDN$326M in sustaining capital, as summarized in Table 21.3.

The development cost and initial infrastructure costs for each zone is classified as project capital (non-sustaining) For simplification. Once stoping production is realized, all infrastructure cost and continued development is, thereafter, classified as sustaining capex.

1.15 Process Capital Costs

The process capital costs are estimated to total USD$1.3M in 2028, and relate to capital investment required to down-size the mill facility

1.16 Tailings Management Area and Infrastructure Capital Costs

Principal Tailings Management Area and Infrastructure capital costs include, but are not limited to the following principal items:

- TMA represents the expansion of the current tailings facility to accommodate the tailings generated from the processing of an additional 70 Mt of ore in the current mine plan via annual tailings dam raises.

The capital cost estimate is appropriate for process functions.

1.16.1 Operating cost

Operating costs have been estimated using first principal 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.

A summary of the estimated LOM operating costs is shown in Table 21.6, plus the LOM average, are shown in Table 21.7 in Item 21.

1.17 Economic Analysis

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

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 item of the Report.

2 INTRODUCTION

The Technical Report will demonstrate the economic of the extension of the actual Rainy River Open Pit through an underground operation (UG Main Zones) and the satellite underground Intrepid deposit. This NI 43-101 is done according to Standard of Disclosure for Mineral Projects.This Technical Report (Report) on the Rainy River Property (Property) located in north-western Ontario (ON) in Canada has been prepared by InnovExplo Mining Consultants (Canada) headquartered in Val-d'Or, 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 12 March 2020 and filed on SEDAR on 27 March 2020, titled "New Gold Inc. Technical Report on the Rainy River Mine, Ontario, Canada".

The QP for the underground mineral reserve estimates of Rainy River and Intrepid zones is Mr. Éric Lecomte, P.Eng. (InnovExplo). The names and details of all the persons who prepared, or who have assisted the Qualified Persons (QPs) in the preparation of this Report, are listed in Table 2.1.

This updated NI 43-101 Technical Report for New Gold Rainy River Operation is effective dated March 28, 2022.

INNOVEXPLO

Table 2.1 – List of QPs

	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	Items of report
QP	Mr E. Lecomte	Senior Mining Engineer	InnovExplo	No	15 March 2022	P.Eng. (Qc)	Items, 1, 2, 3, 15, 16, 21 22, 24, 27
QP	Mr A. Croal	Director Technical Services	New Gold Inc.	No	NA	P.Eng. (ON)	Parts of Items 1, 18 Infrastructure, 19, 5, and 21
QP	Mr F. McCann	General Manager / Principal Mining Engineer	AMC Mining Consultants (Canada) Ltd.	Yes	Various, last visit 13-15 Jan 2020	P.Eng. (ON)	Parts of Items15 and 16 pertaining to open pit Mineral Reserves and mine planning aspects and related disclosure in Sections 1, 25, 26, and 27.
QP	Mr M. Della Libera	Director Exploration	New Gold Inc.	No	Various, last Jan 25-27, 2022	P.Geo. (ON)	Items 4 to 10, part of Item 11, Item 23
QP	Ms D. Nussipakynova	Principal Geologist	AMC Mining Consultants (Canada) Ltd.	Yes	11 Apr 2018	P.Geo. (BC)	Items 12 and 14, and related disclosure in Items 1,11, 25 ,26 and 27
QP	Mr K. Bocking	Principal	Golder Associates Ltd.	Yes	Various, last visit 20 Oct 2020	P.Eng. (ON)	Part of Items 16 and 18 (Soft rock aspects of OP and waste storage areas.)
QP	Mr E. Saunders	Senior Consultant, Mining Rock Mechanics	SRK Consulting (Canada) Inc.	Yes	Various, last visit 3-5 Feb 2020	P.Eng. (ON)	Item 16 geomechanics. (Hard rock aspects of OP)
QP	Mr A. Zerwer	Principal Geotechnical Engineer	BGC Engineering Inc.	Yes	Various, last visit 15-18 Nov 2021	P.Eng. (ON)	Part ofItem 18 tailings dam

	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**	**Items of report**
QP	Mr M. Taghimohammadi	Senior Processing Engineer	New Gold Inc.	No	NA	P.Eng. (ON)	Items 13 and 17
QP	Mr S. Yirdaw	Senior Environmental Engineer	New Gold Inc., Rainy River Mine	No	NA	P.Eng. (ON)	Item 20 Environment
QP	Mr J. Taylor	President	Halyard Inc	No	27-May-21	P.Eng. (ON)	17 small scale processing plant
QP	Mr C. Gagnon	Principal Ventilation Engineer	CGMexpert	No	NA	P.Eng. (Qc)	Part of 16

Note: QP responsibility for ‘part’ items are governed by their respective areas of expertise.

An inspection of the property was carried out by all the QPs 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. Mr S. Yirdaw of New Gold, QP for Item 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$1.25

This report has an effective date of March 31, 2022.

2.1 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 12 March 2020, (2020 New Gold NI 43-101 Report).

Other reference material has been the 2022 Budget of the Rainy River Mine, prepared in 2021 by NG.

Parties who have supplied some information that was used for the development of this report include AMC Mining Consultants (Canada) Ltd. (AMC), BGC Engineering Inc. (BGC), Golder Associates Ltd. (Golder), SRK Consulting (Canada) Inc. (SRK), Orway Mineral Consultants Canada Limited (OMC), Halyard Inc, ASDR Consulting, Hydroresource, Machine Roger, C-Mac Mining, CGMexpert, and Howden Inc… Just to name a few.

Acknowledgement

InnovExplo Inc would like to acknowledge all the QPs for their contributions into this report and special dedication of Andrew Croal of New Gold to keep everyone on track.

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 Item 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 17 February 2022.

Extent of reliance:

- Full reliance.

Portion of Report to which disclaimer applies:

- Item 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 Item 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:

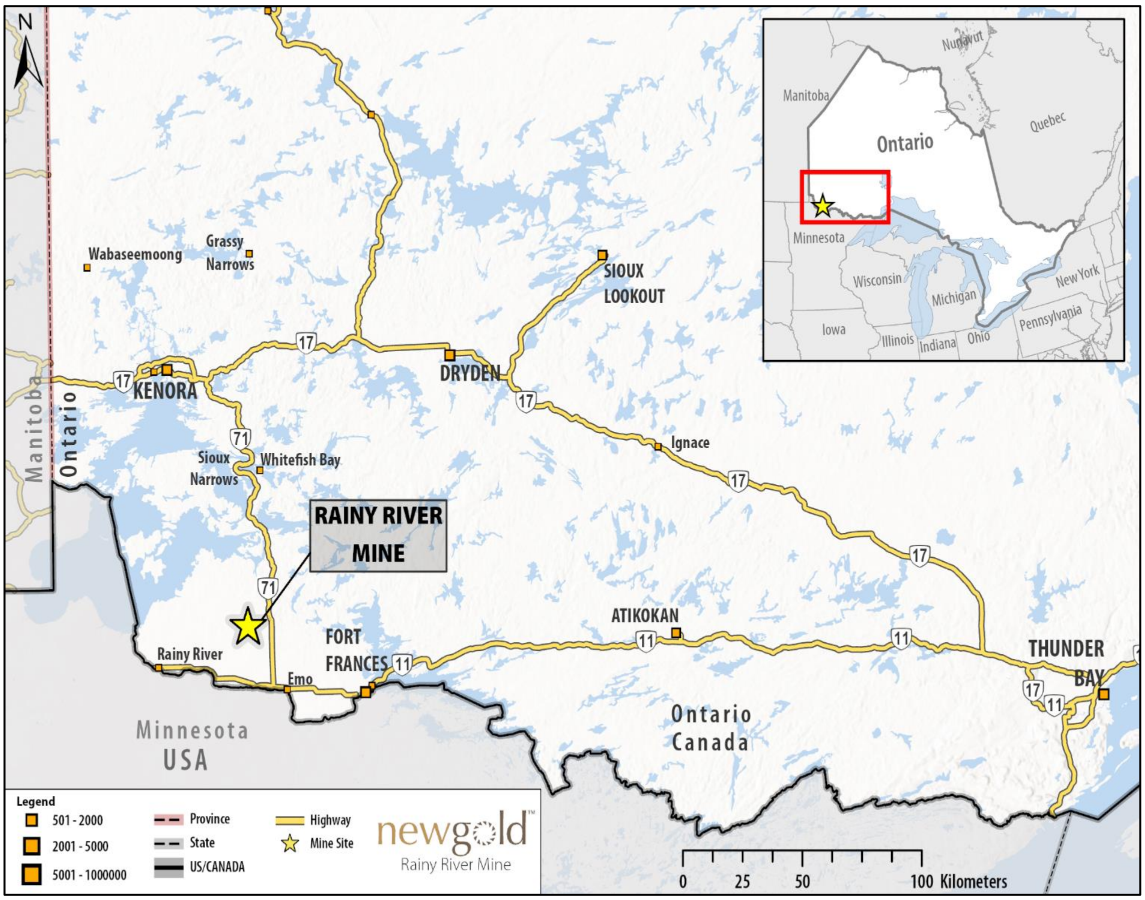
- Item 22 Economic Analysis.

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. It is situated 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 Introduction.

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).



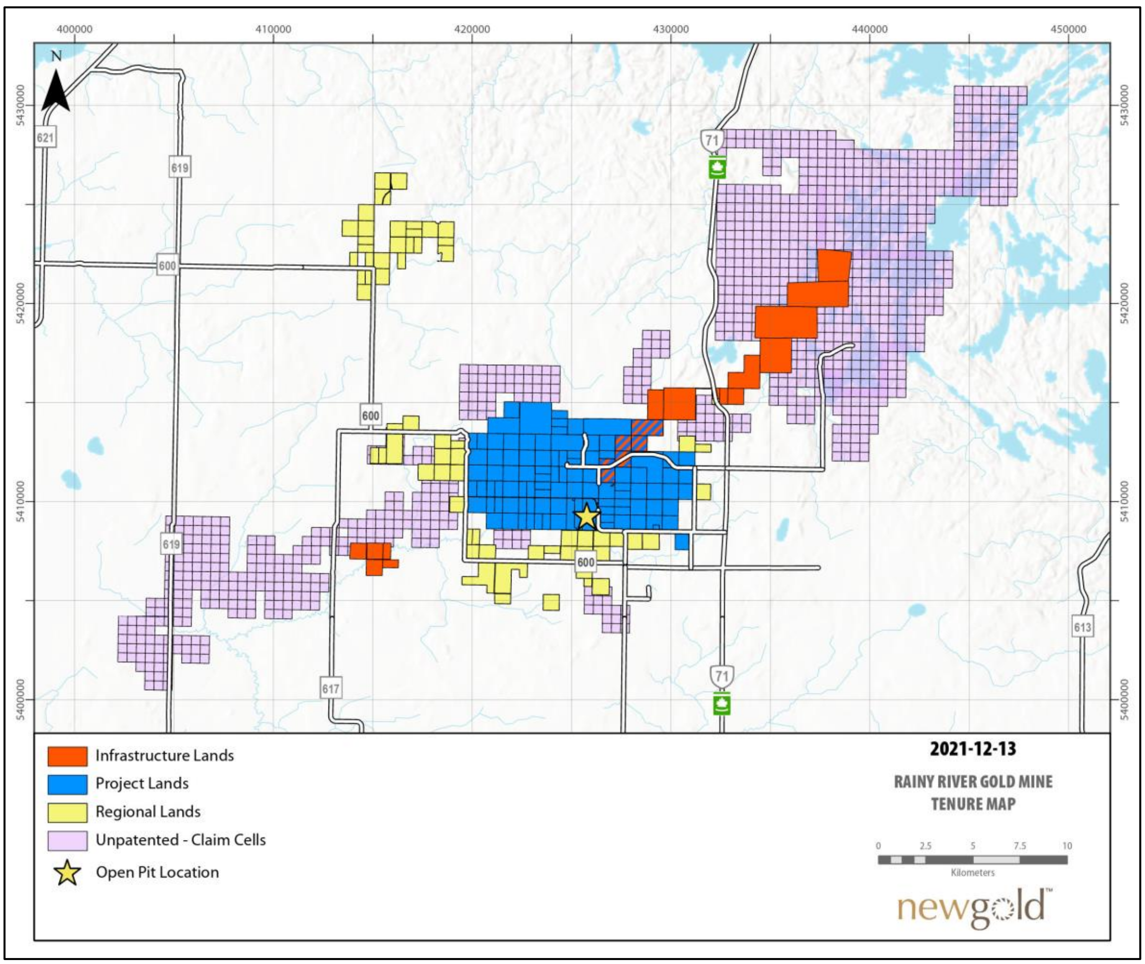
Source: New Gold January 2020.

Figure 4.1 – Site location

4.2 Land tenure

4.2.1 General

The Property comprises a portfolio of 209 patented mining rights, surface rights (SR), and Crown Lease properties. The Rainy River Project Lands (mine area) comprise 121 separate properties of which New Gold has the rights to the Surface and Minerals, and this area covers approximately 6,141 hectares (ha). Infrastructure Lands cover a further area of 2,800 ha, six of which overlap Project Lands, and Regional Lands cover 3,697 ha. New Gold's total land package covers an aggregate area of approximately 36,657 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.



Source: New Gold January 2022.

Figure 4.2 – Tenure map

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 Rainy River Project Lands as shown in Figure 4.2 are listed in Table 1.1 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

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
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
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	54.88

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
				Lease # 109626	
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	56042-0046	01: NG SR and MR	62.73
56042-0068	01: NG SR and MR	1.75			
Total hectares: 6,141.36					

The Infrastructure Lands as shown in Reliance on other experts are listed in Land tenure. 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. Land tenure 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., 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	56041-0215	01: NG SR and MR	10.09
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
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
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

PIN	Tenure type	Area (ha)	PIN	Tenure type	Area (ha)
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	56035-0042	01: NG SR and MR	64.80
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
Total hectares: 3,697.37					

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 Reliance on other experts in purple. There is a total of 1,157 such claims and these are all owned 100% by New Gold. They cover a total area of 24,437 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 The Rainy River Project Lands as shown in Figure 4.2 are listed in Table 1.1 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.

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Table 4.4 – Summary of unpatented land claims

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
181753	10 Apr 2018	2022-05-04	227102	10 Apr 2018	2022-05-04	118036	10 Apr 2018	2022-05-08
181752	10 Apr 2018	2022-05-04	231544	10 Apr 2018	2022-05-06	280269	10 Apr 2018	2022-05-15
168930	10 Apr 2018	2022-05-04	177676	10 Apr 2018	2022-05-06	204138	10 Apr 2018	2022-05-15
160280	10 Apr 2018	2022-05-04	158851	10 Apr 2018	2022-05-06	329434	10 Apr 2018	2022-05-15
121146	10 Apr 2018	2022-05-04	282246	10 Apr 2018	2022-05-06	270246	10 Apr 2018	2022-05-15
117789	10 Apr 2018	2022-05-04	269637	10 Apr 2018	2022-05-06	179729	10 Apr 2018	2022-05-15
101300	10 Apr 2018	2022-05-04	232992	10 Apr 2018	2022-05-06	121761	10 Apr 2018	2022-05-15
345286	10 Apr 2018	2022-05-04	166290	10 Apr 2018	2022-05-06	164234	10 Apr 2018	2022-05-16
170892	10 Apr 2018	2022-05-04	121145	10 Apr 2018	2022-05-06	145463	10 Apr 2018	2022-05-16
102699	10 Apr 2018	2022-05-04	101513	10 Apr 2018	2022-05-06	312710	10 Apr 2018	2022-05-16
265596	10 Apr 2018	2022-05-04	326883	10 Apr 2018	2022-05-06	279658	10 Apr 2018	2022-05-16
320943	10 Apr 2018	2022-05-04	314683	10 Apr 2018	2022-05-06	278142	10 Apr 2018	2022-05-16
284411	10 Apr 2018	2022-05-04	314682	10 Apr 2018	2022-05-06	230923	10 Apr 2018	2022-05-16
272327	10 Apr 2018	2022-05-04	280270	10 Apr 2018	2022-05-06	212121	10 Apr 2018	2022-05-16
264859	10 Apr 2018	2022-05-04	224260	10 Apr 2018	2022-05-06	203524	10 Apr 2018	2022-05-16
235669	10 Apr 2018	2022-05-04	212762	10 Apr 2018	2022-05-06	158210	10 Apr 2018	2022-05-16
217126	10 Apr 2018	2022-05-04	177678	10 Apr 2018	2022-05-06	128932	10 Apr 2018	2022-05-16
217126	10 Apr 2018	2022-05-04	177677	10 Apr 2018	2022-05-06	101958	10 Apr 2018	2022-05-16
168931	10 Apr 2018	2022-05-04	158853	10 Apr 2018	2022-05-06	197525	10 Apr 2018	2022-05-17
345288	10 Apr 2018	2022-05-04	158852	10 Apr 2018	2022-05-06	283586	10 Apr 2018	2022-05-17
345287	10 Apr 2018	2022-05-04	129578	10 Apr 2018	2022-05-06	270962	10 Apr 2018	2022-05-17
322396	10 Apr 2018	2022-05-04	102052	10 Apr 2018	2022-05-06	180430	10 Apr 2018	2022-05-17
274261	10 Apr 2018	2022-05-04	102051	10 Apr 2018	2022-05-06	180481	10 Apr 2018	2022-05-17
266294	10 Apr 2018	2022-05-04	218487	10 Apr 2018	2022-05-08	284268	10 Apr 2018	2022-05-17
266293	10 Apr 2018	2022-05-04	170310	10 Apr 2018	2022-05-08	235003	10 Apr 2018	2022-05-17
208343	10 Apr 2018	2022-05-04	344589	10 Apr 2018	2022-05-08	208264	10 Apr 2018	2022-05-25

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
328822	10 Apr 2018	2022-05-04	225841	10 Apr 2018	2022-05-08	183126	10 Apr 2018	2022-05-25
214227	10 Apr 2018	2022-05-04	125080	10 Apr 2018	2022-05-08	173075	10 Apr 2018	2022-05-25
195555	10 Apr 2018	2022-05-04	322309	10 Apr 2018	2022-05-08	173074	10 Apr 2018	2022-05-25
195554	10 Apr 2018	2022-05-04	226439	10 Apr 2018	2022-05-08	173073	10 Apr 2018	2022-05-25
344062	10 Apr 2018	2022-05-04	218488	10 Apr 2018	2022-05-08	125184	10 Apr 2018	2022-05-25
344061	10 Apr 2018	2022-05-04	218486	10 Apr 2018	2022-05-08	118151	10 Apr 2018	2022-05-25
344060	10 Apr 2018	2022-05-04	188556	10 Apr 2018	2022-05-08	102920	10 Apr 2018	2022-05-25
292441	10 Apr 2018	2022-05-04	188555	10 Apr 2018	2022-05-08	322974	10 Apr 2018	2022-05-25
285639	10 Apr 2018	2022-05-04	153667	10 Apr 2018	2022-05-08	286409	10 Apr 2018	2022-05-25
273556	10 Apr 2018	2022-05-04	118152	10 Apr 2018	2022-05-08	219163	10 Apr 2018	2022-05-25
207637	10 Apr 2018	2022-05-04	102833	10 Apr 2018	2022-05-08	170973	10 Apr 2018	2022-05-25
207636	10 Apr 2018	2022-05-04	102832	10 Apr 2018	2022-05-08	345289	10 Apr 2018	2022-05-25
182485	10 Apr 2018	2022-05-04	344590	10 Apr 2018	2022-05-08	286348	10 Apr 2018	2022-05-25
182484	10 Apr 2018	2022-05-04	321704	10 Apr 2018	2022-05-08	274262	10 Apr 2018	2022-05-25
182483	10 Apr 2018	2022-05-04	285662	10 Apr 2018	2022-05-08	266295	10 Apr 2018	2022-05-25
125058	10 Apr 2018	2022-05-04	273575	10 Apr 2018	2022-05-08	153747	10 Apr 2018	2022-05-25
125057	10 Apr 2018	2022-05-04	265628	10 Apr 2018	2022-05-08	153666	10 Apr 2018	2022-05-25
118014	10 Apr 2018	2022-05-04	225840	10 Apr 2018	2022-05-08	125783	10 Apr 2018	2022-05-25
118013	10 Apr 2018	2022-05-04	207661	10 Apr 2018	2022-05-08	125782	10 Apr 2018	2022-05-25
118012	10 Apr 2018	2022-05-04	118038	10 Apr 2018	2022-05-08	345359	10 Apr 2018	2022-05-25
118011	10 Apr 2018	2022-05-04	118037	10 Apr 2018	2022-05-08	345358	10 Apr 2018	2022-05-25
345341	10 Apr 2018	2022-05-25	344690	10 Apr 2018	2022-06-02	207632	10 Apr 2018	2022-06-02
322973	10 Apr 2018	2022-05-25	344591	10 Apr 2018	2022-06-02	169683	10 Apr 2018	2022-06-02
293738	10 Apr 2018	2022-05-25	285638	10 Apr 2018	2022-06-02	153046	10 Apr 2018	2022-06-02
293725	10 Apr 2018	2022-05-25	226440	10 Apr 2018	2022-06-02	153045	10 Apr 2018	2022-06-02
266845	10 Apr 2018	2022-05-25	225813	10 Apr 2018	2022-06-02	153044	10 Apr 2018	2022-06-02
266844	10 Apr 2018	2022-05-25	218374	10 Apr 2018	2022-06-02	118006	10 Apr 2018	2022-06-02

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
208397	10 Apr 2018	2022-05-25	170311	10 Apr 2018	2022-06-02	344689	10 Apr 2018	2022-06-02
206907	10 Apr 2018	2022-05-25	169682	10 Apr 2018	2022-06-02	273672	10 Apr 2018	2022-06-02
189210	10 Apr 2018	2022-05-25	153071	10 Apr 2018	2022-06-02	218490	10 Apr 2018	2022-06-02
126365	10 Apr 2018	2022-05-25	118153	10 Apr 2018	2022-06-02	208265	10 Apr 2018	2022-06-02
323602	10 Apr 2018	2022-05-25	292436	10 Apr 2018	2022-06-02	153668	10 Apr 2018	2022-06-02
287087	10 Apr 2018	2022-05-25	225814	10 Apr 2018	2022-06-02	118154	10 Apr 2018	2022-06-02
275006	10 Apr 2018	2022-05-25	182478	10 Apr 2018	2022-06-02	335470	10 Apr 2018	2022-06-02
267551	10 Apr 2018	2022-05-25	153047	10 Apr 2018	2022-06-02	335469	10 Apr 2018	2022-06-02
267530	10 Apr 2018	2022-05-25	293062	10 Apr 2018	2022-06-02	323643	10 Apr 2018	2022-06-02
227795	10 Apr 2018	2022-05-25	285763	10 Apr 2018	2022-06-02	294435	10 Apr 2018	2022-06-02
220352	10 Apr 2018	2022-05-25	273673	10 Apr 2018	2022-06-02	294434	10 Apr 2018	2022-06-02
208924	10 Apr 2018	2022-05-25	273671	10 Apr 2018	2022-06-02	275555	10 Apr 2018	2022-06-02
173856	10 Apr 2018	2022-05-25	266213	10 Apr 2018	2022-06-02	275554	10 Apr 2018	2022-06-02
173855	10 Apr 2018	2022-05-25	208266	10 Apr 2018	2022-06-02	171658	10 Apr 2018	2022-06-02
127049	10 Apr 2018	2022-05-25	344639	10 Apr 2018	2022-06-02	155023	10 Apr 2018	2022-06-02
127048	10 Apr 2018	2022-05-25	322256	10 Apr 2018	2022-06-02	127082	10 Apr 2018	2022-06-02
126526	10 Apr 2018	2022-05-25	285714	10 Apr 2018	2022-06-02	127081	10 Apr 2018	2022-06-02
126525	10 Apr 2018	2022-05-25	265672	10 Apr 2018	2022-06-02	103211	10 Apr 2018	2022-06-02
117295	10 Apr 2018	2022-05-25	207726	10 Apr 2018	2022-06-02	294501	10 Apr 2018	2022-06-02
344640	10 Apr 2018	2022-06-02	207725	10 Apr 2018	2022-06-02	267648	10 Apr 2018	2022-06-02
344059	10 Apr 2018	2022-06-02	207724	10 Apr 2018	2022-06-02	228399	10 Apr 2018	2022-06-02
322255	10 Apr 2018	2022-06-02	102777	10 Apr 2018	2022-06-02	220428	10 Apr 2018	2022-06-02
322254	10 Apr 2018	2022-06-02	344058	10 Apr 2018	2022-06-02	314099	10 Apr 2018	2022-06-13
285713	10 Apr 2018	2022-06-02	225815	10 Apr 2018	2022-06-02	174210	10 Apr 2018	2022-06-13
273622	10 Apr 2018	2022-06-02	218378	10 Apr 2018	2022-06-02	140174	10 Apr 2018	2022-06-13
226405	10 Apr 2018	2022-06-02	169687	10 Apr 2018	2022-06-02	107516	10 Apr 2018	2022-06-13
207723	10 Apr 2018	2022-06-02	169686	10 Apr 2018	2022-06-02	238958	10 Apr 2018	2022-06-13

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
188505	10 Apr 2018	2022-06-02	335449	10 Apr 2018	2022-06-02	202511	10 Apr 2018	2022-06-13
188504	10 Apr 2018	2022-06-02	335448	10 Apr 2018	2022-06-02	157834	10 Apr 2018	2022-06-13
125129	10 Apr 2018	2022-06-02	208940	10 Apr 2018	2022-06-02	137682	10 Apr 2018	2022-06-13
125128	10 Apr 2018	2022-06-02	189905	10 Apr 2018	2022-06-02	110923	10 Apr 2018	2022-06-13
265595	10 Apr 2018	2022-06-02	293140	10 Apr 2018	2022-06-02	341221	10 Apr 2018	2022-06-21
218377	10 Apr 2018	2022-06-02	274237	10 Apr 2018	2022-06-02	341220	10 Apr 2018	2022-06-21
218376	10 Apr 2018	2022-06-02	189140	10 Apr 2018	2022-06-02	289658	10 Apr 2018	2022-06-21
207635	10 Apr 2018	2022-06-02	125748	10 Apr 2018	2022-06-02	282274	10 Apr 2018	2022-06-21
182482	10 Apr 2018	2022-06-02	125747	10 Apr 2018	2022-06-02	282272	10 Apr 2018	2022-06-21
169688	10 Apr 2018	2022-06-02	102900	10 Apr 2018	2022-06-02	270177	10 Apr 2018	2022-06-21
293061	10 Apr 2018	2022-06-02	292435	10 Apr 2018	2022-06-02	262219	10 Apr 2018	2022-06-21
266212	10 Apr 2018	2022-06-02	273553	10 Apr 2018	2022-06-02	233020	10 Apr 2018	2022-06-21
208263	10 Apr 2018	2022-06-02	265591	10 Apr 2018	2022-06-02	233019	10 Apr 2018	2022-06-21
183125	10 Apr 2018	2022-06-02	265590	10 Apr 2018	2022-06-02	215013	10 Apr 2018	2022-06-21
154331	10 Apr 2018	2022-06-02	265589	10 Apr 2018	2022-06-02	179673	10 Apr 2018	2022-06-21
166325	10 Apr 2018	2022-06-21	273555	10 Apr 2018	2022-07-11	293063	10 Apr 2018	2022-07-11
121678	10 Apr 2018	2022-06-21	266214	10 Apr 2018	2022-07-11	273676	10 Apr 2018	2022-07-11
121677	10 Apr 2018	2022-06-21	218491	10 Apr 2018	2022-07-11	273675	10 Apr 2018	2022-07-11
292359	10 Apr 2018	2022-06-21	170312	10 Apr 2018	2022-07-11	273674	10 Apr 2018	2022-07-11
328857	10 Apr 2018	2022-06-21	153049	10 Apr 2018	2022-07-11	266215	10 Apr 2018	2022-07-11
328856	10 Apr 2018	2022-06-21	344057	10 Apr 2018	2022-07-11	208269	10 Apr 2018	2022-07-11
282273	10 Apr 2018	2022-06-21	292439	10 Apr 2018	2022-07-11	208268	10 Apr 2018	2022-07-11
215012	10 Apr 2018	2022-06-21	207634	10 Apr 2018	2022-07-11	208267	10 Apr 2018	2022-07-11
214255	10 Apr 2018	2022-06-21	188419	10 Apr 2018	2022-07-11	125187	10 Apr 2018	2022-07-11
214254	10 Apr 2018	2022-06-21	182481	10 Apr 2018	2022-07-11	125186	10 Apr 2018	2022-07-11
179672	10 Apr 2018	2022-06-21	102698	10 Apr 2018	2022-07-11	125185	10 Apr 2018	2022-07-11
116871	10 Apr 2018	2022-06-21	265594	10 Apr 2018	2022-07-11	118156	10 Apr 2018	2022-07-11

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
328221	10 Apr 2018	2022-06-26	335422	10 Apr 2018	2022-07-11	118155	10 Apr 2018	2022-07-11
194969	10 Apr 2018	2022-06-26	335421	10 Apr 2018	2022-07-11	102834	10 Apr 2018	2022-07-11
230295	10 Apr 2018	2022-06-26	267529	10 Apr 2018	2022-07-11	321680	10 Apr 2018	2022-07-11
181707	10 Apr 2018	2022-06-26	227781	10 Apr 2018	2022-07-11	292440	10 Apr 2018	2022-07-11
167652	10 Apr 2018	2022-06-26	227780	10 Apr 2018	2022-07-11	292438	10 Apr 2018	2022-07-11
314077	10 Apr 2018	2022-06-26	189890	10 Apr 2018	2022-07-11	273554	10 Apr 2018	2022-07-11
277487	10 Apr 2018	2022-06-26	171613	10 Apr 2018	2022-07-11	218375	10 Apr 2018	2022-07-11
223522	10 Apr 2018	2022-06-26	154991	10 Apr 2018	2022-07-11	207633	10 Apr 2018	2022-07-11
203385	10 Apr 2018	2022-06-26	154990	10 Apr 2018	2022-07-11	182480	10 Apr 2018	2022-07-11
117119	10 Apr 2018	2022-06-26	117294	10 Apr 2018	2022-07-11	153048	10 Apr 2018	2022-07-11
100482	10 Apr 2018	2022-06-26	117293	10 Apr 2018	2022-07-11	125056	10 Apr 2018	2022-07-11
261588	10 Apr 2018	2022-06-26	103175	10 Apr 2018	2022-07-11	118010	10 Apr 2018	2022-07-11
194970	10 Apr 2018	2022-06-26	345266	10 Apr 2018	2022-07-11	118009	10 Apr 2018	2022-07-11
314076	10 Apr 2018	2022-06-26	345265	10 Apr 2018	2022-07-11	118008	10 Apr 2018	2022-07-11
223521	10 Apr 2018	2022-06-26	293143	10 Apr 2018	2022-07-11	102697	10 Apr 2018	2022-07-11
291689	10 Apr 2018	2022-06-26	274241	10 Apr 2018	2022-07-11	292442	10 Apr 2018	2022-07-11
168893	10 Apr 2018	2022-06-26	226516	10 Apr 2018	2022-07-11	265597	10 Apr 2018	2022-07-11
117749	10 Apr 2018	2022-06-26	219074	10 Apr 2018	2022-07-11	182487	10 Apr 2018	2022-07-11
101262	10 Apr 2018	2022-06-26	208328	10 Apr 2018	2022-07-11	296316	10 Apr 2018	2022-07-16
326809	10 Apr 2018	2022-06-30	208327	10 Apr 2018	2022-07-11	258930	10 Apr 2018	2022-07-16
312775	10 Apr 2018	2022-06-30	208326	10 Apr 2018	2022-07-11	230274	10 Apr 2018	2022-07-16
278201	10 Apr 2018	2022-06-30	189142	10 Apr 2018	2022-07-11	222990	10 Apr 2018	2022-07-16
177619	10 Apr 2018	2022-06-30	183181	10 Apr 2018	2022-07-11	222989	10 Apr 2018	2022-07-16
164298	10 Apr 2018	2022-06-30	170374	10 Apr 2018	2022-07-11	203360	10 Apr 2018	2022-07-16
116008	10 Apr 2018	2022-06-30	153722	10 Apr 2018	2022-07-11	143453	10 Apr 2018	2022-07-16
294396	10 Apr 2018	2022-07-11	125751	10 Apr 2018	2022-07-11	128259	10 Apr 2018	2022-07-16
274240	10 Apr 2018	2022-07-11	125750	10 Apr 2018	2022-07-11	174458	10 Apr 2018	2022-08-04

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
226517	10 Apr 2018	2022-07-11	125749	10 Apr 2018	2022-07-11	170225	10 Apr 2018	2022-08-04
189141	10 Apr 2018	2022-07-11	102901	10 Apr 2018	2022-07-11	102723	10 Apr 2018	2022-08-04
183182	10 Apr 2018	2022-07-11	335471	10 Apr 2018	2022-07-11	273576	10 Apr 2018	2022-08-04
173841	10 Apr 2018	2022-07-11	323644	10 Apr 2018	2022-07-11	218396	10 Apr 2018	2022-08-04
154992	10 Apr 2018	2022-07-11	275556	10 Apr 2018	2022-07-11	335401	10 Apr 2018	2022-08-04
125752	10 Apr 2018	2022-07-11	227829	10 Apr 2018	2022-07-11	292456	10 Apr 2018	2022-08-04
265593	10 Apr 2018	2022-07-11	173878	10 Apr 2018	2022-07-11	273574	10 Apr 2018	2022-08-04
169685	10 Apr 2018	2022-07-11	127083	10 Apr 2018	2022-07-11	117397	10 Apr 2018	2022-08-04
118007	10 Apr 2018	2022-07-11	322310	10 Apr 2018	2022-07-11	323074	10 Apr 2018	2022-08-04
266993	10 Apr 2018	2022-08-04	232195	10 Apr 2018	2022-10-15	214904	10 Apr 2018	2022-10-26
266992	10 Apr 2018	2022-08-04	224909	10 Apr 2018	2022-10-15	196266	10 Apr 2018	2022-10-26
266991	10 Apr 2018	2022-08-04	213495	10 Apr 2018	2022-10-15	166967	10 Apr 2018	2022-10-26
154963	10 Apr 2018	2022-08-04	194851	10 Apr 2018	2022-10-15	290298	10 Apr 2018	2022-10-26
265629	10 Apr 2018	2022-08-04	194225	10 Apr 2018	2022-10-15	290297	10 Apr 2018	2022-10-26
218398	10 Apr 2018	2022-08-04	178349	10 Apr 2018	2022-10-15	289633	10 Apr 2018	2022-10-26
218397	10 Apr 2018	2022-08-04	178324	10 Apr 2018	2022-10-15	233682	10 Apr 2018	2022-10-26
170226	10 Apr 2018	2022-08-04	165576	10 Apr 2018	2022-10-15	180312	10 Apr 2018	2022-10-26
125083	10 Apr 2018	2022-08-04	165575	10 Apr 2018	2022-10-15	101682	10 Apr 2018	2022-10-26
125082	10 Apr 2018	2022-08-04	165574	10 Apr 2018	2022-10-15	289635	10 Apr 2018	2022-10-26
612706	14 Sept 2020	2022-09-14	159487	10 Apr 2018	2022-10-15	282256	10 Apr 2018	2022-10-26
326783	10 Apr 2018	2022-09-25	159471	10 Apr 2018	2022-10-15	216309	10 Apr 2018	2022-10-26
312756	10 Apr 2018	2022-09-25	120317	10 Apr 2018	2022-10-15	214999	10 Apr 2018	2022-10-26
312755	10 Apr 2018	2022-09-25	120316	10 Apr 2018	2022-10-15	160805	10 Apr 2018	2022-10-26
296992	10 Apr 2018	2022-09-25	116551	10 Apr 2018	2022-10-15	100995	10 Apr 2018	2022-10-26
230963	10 Apr 2018	2022-09-25	341350	10 Apr 2018	2022-10-26	289632	10 Apr 2018	2022-10-26
204064	10 Apr 2018	2022-09-25	329514	10 Apr 2018	2022-10-26	262194	10 Apr 2018	2022-10-26
206991	10 Apr 2018	2022-10-13	282921	10 Apr 2018	2022-10-26	116852	10 Apr 2018	2022-10-26

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
173749	10 Apr 2018	2022-10-13	282920	10 Apr 2018	2022-10-26	101520	10 Apr 2018	2022-10-26
103072	10 Apr 2018	2022-10-13	166941	10 Apr 2018	2022-10-26	233659	10 Apr 2018	2022-10-26
103071	10 Apr 2018	2022-10-13	342630	10 Apr 2018	2022-10-26	180310	10 Apr 2018	2022-10-26
278093	10 Apr 2018	2022-10-15	330208	10 Apr 2018	2022-10-26	179795	10 Apr 2018	2022-10-26
230883	10 Apr 2018	2022-10-15	291018	10 Apr 2018	2022-10-26	101680	10 Apr 2018	2022-10-26
224179	10 Apr 2018	2022-10-15	271578	10 Apr 2018	2022-10-26	341351	10 Apr 2018	2022-10-26
101917	10 Apr 2018	2022-10-15	197549	10 Apr 2018	2022-10-26	282922	10 Apr 2018	2022-10-26
314797	10 Apr 2018	2022-10-15	167638	10 Apr 2018	2022-10-26	270315	10 Apr 2018	2022-10-26
298203	10 Apr 2018	2022-10-15	151631	10 Apr 2018	2022-10-26	262865	10 Apr 2018	2022-10-26
214127	10 Apr 2018	2022-10-15	341932	10 Apr 2018	2022-10-26	262864	10 Apr 2018	2022-10-26
101425	10 Apr 2018	2022-10-15	329597	10 Apr 2018	2022-10-26	233656	10 Apr 2018	2022-10-26
100839	10 Apr 2018	2022-10-15	329538	10 Apr 2018	2022-10-26	233655	10 Apr 2018	2022-10-26
230884	10 Apr 2018	2022-10-15	291076	10 Apr 2018	2022-10-26	215633	10 Apr 2018	2022-10-26
164191	10 Apr 2018	2022-10-15	234246	10 Apr 2018	2022-10-26	215632	10 Apr 2018	2022-10-26
289621	10 Apr 2018	2022-10-15	205627	10 Apr 2018	2022-10-26	196213	10 Apr 2018	2022-10-26
282249	10 Apr 2018	2022-10-15	180368	10 Apr 2018	2022-10-26	166942	10 Apr 2018	2022-10-26
282248	10 Apr 2018	2022-10-15	328860	10 Apr 2018	2022-10-26	160948	10 Apr 2018	2022-10-26
281017	10 Apr 2018	2022-10-15	282276	10 Apr 2018	2022-10-26	160947	10 Apr 2018	2022-10-26
232297	10 Apr 2018	2022-10-15	215015	10 Apr 2018	2022-10-26	160946	10 Apr 2018	2022-10-26
214229	10 Apr 2018	2022-10-15	214258	10 Apr 2018	2022-10-26	101678	10 Apr 2018	2022-10-26
214228	10 Apr 2018	2022-10-15	214257	10 Apr 2018	2022-10-26	342631	10 Apr 2018	2022-10-26
213491	10 Apr 2018	2022-10-15	179674	10 Apr 2018	2022-10-26	283695	10 Apr 2018	2022-10-26
194849	10 Apr 2018	2022-10-15	160828	10 Apr 2018	2022-10-26	283694	10 Apr 2018	2022-10-26
159581	10 Apr 2018	2022-10-15	121684	10 Apr 2018	2022-10-26	205583	10 Apr 2018	2022-10-26
327520	10 Apr 2018	2022-10-15	101550	10 Apr 2018	2022-10-26	197630	10 Apr 2018	2022-10-26
314799	10 Apr 2018	2022-10-15	270343	10 Apr 2018	2022-10-26	168222	10 Apr 2018	2022-10-26
314798	10 Apr 2018	2022-10-15	161485	10 Apr 2018	2022-10-26	151632	10 Apr 2018	2022-10-26

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
298224	10 Apr 2018	2022-10-15	161483	10 Apr 2018	2022-10-26	101087	10 Apr 2018	2022-10-26
281019	10 Apr 2018	2022-10-15	270894	10 Apr 2018	2022-10-26	330834	10 Apr 2018	2022-10-26
280893	10 Apr 2018	2022-10-15	215657	10 Apr 2018	2022-10-26	330833	10 Apr 2018	2022-10-26
291075	10 Apr 2018	2022-10-26	262196	10 Apr 2018	2022-10-26	115963	10 Apr 2018	2022-11-27
271660	10 Apr 2018	2022-10-26	262195	10 Apr 2018	2022-10-26	326764	10 Apr 2018	2022-11-27
271659	10 Apr 2018	2022-10-26	214998	10 Apr 2018	2022-10-26	312744	10 Apr 2018	2022-11-27
271658	10 Apr 2018	2022-10-26	179653	10 Apr 2018	2022-10-26	312743	10 Apr 2018	2022-11-27
205628	10 Apr 2018	2022-10-26	179652	10 Apr 2018	2022-10-26	296979	10 Apr 2018	2022-11-27
151686	10 Apr 2018	2022-10-26	166299	10 Apr 2018	2022-10-26	279679	10 Apr 2018	2022-11-27
341224	10 Apr 2018	2022-10-26	160807	10 Apr 2018	2022-10-26	278171	10 Apr 2018	2022-11-27
328862	10 Apr 2018	2022-10-26	160806	10 Apr 2018	2022-10-26	223675	10 Apr 2018	2022-11-27
328861	10 Apr 2018	2022-10-26	116853	10 Apr 2018	2022-10-26	212147	10 Apr 2018	2022-11-27
262220	10 Apr 2018	2022-10-26	101522	10 Apr 2018	2022-10-26	164259	10 Apr 2018	2022-11-27
214259	10 Apr 2018	2022-10-26	101521	10 Apr 2018	2022-10-26	128962	10 Apr 2018	2022-11-27
180367	10 Apr 2018	2022-10-26	329519	10 Apr 2018	2022-10-26	128961	10 Apr 2018	2022-11-27
121685	10 Apr 2018	2022-10-26	270319	10 Apr 2018	2022-10-26	115964	10 Apr 2018	2022-11-27
116873	10 Apr 2018	2022-10-26	262866	10 Apr 2018	2022-10-26	281565	10 Apr 2018	2022-11-27
341911	10 Apr 2018	2022-10-26	233660	10 Apr 2018	2022-10-26	178957	10 Apr 2018	2022-11-27
341888	10 Apr 2018	2022-10-26	180311	10 Apr 2018	2022-10-26	535473	10 Apr 2018	2022-11-28
329540	10 Apr 2018	2022-10-26	160950	10 Apr 2018	2022-10-26	535472	10 Apr 2018	2022-11-28
290300	10 Apr 2018	2022-10-26	160949	10 Apr 2018	2022-10-26	310722	10 Apr 2018	2022-12-02
282955	10 Apr 2018	2022-10-26	101681	10 Apr 2018	2022-10-26	286903	10 Apr 2018	2022-12-02
262891	10 Apr 2018	2022-10-26	272960	10 Apr 2018	2022-10-26	274758	10 Apr 2018	2022-12-02
215659	10 Apr 2018	2022-10-26	225726	10 Apr 2018	2022-10-26	274757	10 Apr 2018	2022-12-02
214932	10 Apr 2018	2022-10-26	182368	10 Apr 2018	2022-10-26	171439	10 Apr 2018	2022-12-02
196253	10 Apr 2018	2022-10-26	169580	10 Apr 2018	2022-10-26	154885	10 Apr 2018	2022-12-02
196234	10 Apr 2018	2022-10-26	169579	10 Apr 2018	2022-10-26	125605	10 Apr 2018	2022-12-02

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
180333	10 Apr 2018	2022-10-26	169578	10 Apr 2018	2022-10-26	125604	10 Apr 2018	2022-12-02
180332	10 Apr 2018	2022-10-26	124451	10 Apr 2018	2022-10-26	115763	10 Apr 2018	2022-12-02
166968	10 Apr 2018	2022-10-26	117904	10 Apr 2018	2022-10-26	314674	10 Apr 2018	2022-12-19
161502	10 Apr 2018	2022-10-26	117903	10 Apr 2018	2022-10-26	314657	10 Apr 2018	2022-12-19
161484	10 Apr 2018	2022-10-26	102588	10 Apr 2018	2022-10-26	268216	10 Apr 2018	2022-12-19
329596	10 Apr 2018	2022-10-26	539565	10 Apr 2018	2022-10-26	204136	10 Apr 2018	2022-12-19
329595	10 Apr 2018	2022-10-26	323479	10 Apr 2018	2022-10-27	177653	10 Apr 2018	2022-12-19
234244	10 Apr 2018	2022-10-26	323478	10 Apr 2018	2022-10-27	177652	10 Apr 2018	2022-12-19
215714	10 Apr 2018	2022-10-26	274789	10 Apr 2018	2022-10-27	177651	10 Apr 2018	2022-12-19
116219	10 Apr 2018	2022-10-26	274788	10 Apr 2018	2022-10-27	164854	10 Apr 2018	2022-12-19
116218	10 Apr 2018	2022-10-26	227627	10 Apr 2018	2022-10-27	164832	10 Apr 2018	2022-12-19
233681	10 Apr 2018	2022-10-26	227626	10 Apr 2018	2022-10-27	116058	10 Apr 2018	2022-12-19
215658	10 Apr 2018	2022-10-26	227625	10 Apr 2018	2022-10-27	340573	10 Apr 2018	2022-12-19
180331	10 Apr 2018	2022-10-26	171464	10 Apr 2018	2022-10-27	298930	10 Apr 2018	2022-12-19
122359	10 Apr 2018	2022-10-26	125635	10 Apr 2018	2022-10-27	281645	10 Apr 2018	2022-12-19
122358	10 Apr 2018	2022-10-26	115792	10 Apr 2018	2022-10-27	269556	10 Apr 2018	2022-12-19
285018	10 Apr 2018	2022-10-26	115791	10 Apr 2018	2022-10-27	194959	10 Apr 2018	2022-12-19
272961	10 Apr 2018	2022-10-26	100496	10 Apr 2018	2022-10-27	179030	10 Apr 2018	2022-12-19
217764	10 Apr 2018	2022-10-26	262844	10 Apr 2018	2022-11-22	166206	10 Apr 2018	2022-12-19
117907	10 Apr 2018	2022-10-26	326881	10 Apr 2018	2022-11-22	121027	10 Apr 2018	2022-12-19
328831	10 Apr 2018	2022-10-26	314677	10 Apr 2018	2022-11-22	538594	8 Jan 2019	2023-01-08
289634	10 Apr 2018	2022-10-26	314676	10 Apr 2018	2022-11-22	538593	8 Jan 2019	2023-01-08
282257	10 Apr 2018	2022-10-26	268221	10 Apr 2018	2022-11-22	538592	8 Jan 2019	2023-01-08
282255	10 Apr 2018	2022-10-26	158847	10 Apr 2018	2022-11-22	538591	8 Jan 2019	2023-01-08
269648	10 Apr 2018	2022-10-26	123757	10 Apr 2018	2023-01-28	341909	10 Apr 2018	2023-02-13
538590	8 Jan 2019	2023-01-08	123756	10 Apr 2018	2023-01-28	262907	10 Apr 2018	2023-02-13
538589	8 Jan 2019	2023-01-08	123755	10 Apr 2018	2023-01-28	215686	10 Apr 2018	2023-02-13

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
538588	8 Jan 2019	2023-01-08	342573	10 Apr 2018	2023-01-28	204882	10 Apr 2018	2023-02-13
538587	8 Jan 2019	2023-01-08	342572	10 Apr 2018	2023-01-28	284970	10 Apr 2018	2023-02-13
538586	8 Jan 2019	2023-01-08	271013	10 Apr 2018	2023-01-28	270878	10 Apr 2018	2023-02-13
538585	8 Jan 2019	2023-01-08	167653	10 Apr 2018	2023-01-28	264921	10 Apr 2018	2023-02-13
538584	8 Jan 2019	2023-01-08	330207	10 Apr 2018	2023-01-28	215690	10 Apr 2018	2023-02-13
538583	8 Jan 2019	2023-01-08	330189	10 Apr 2018	2023-01-28	161506	10 Apr 2018	2023-02-13
538582	8 Jan 2019	2023-01-08	263551	10 Apr 2018	2023-01-28	122386	10 Apr 2018	2023-02-13
538581	8 Jan 2019	2023-01-08	263550	10 Apr 2018	2023-01-28	329576	10 Apr 2018	2023-02-13
538580	8 Jan 2019	2023-01-08	205008	10 Apr 2018	2023-01-28	290325	10 Apr 2018	2023-02-13
538579	8 Jan 2019	2023-01-08	205007	10 Apr 2018	2023-01-28	166990	10 Apr 2018	2023-02-13
538578	8 Jan 2019	2023-01-08	205006	10 Apr 2018	2023-01-28	330231	10 Apr 2018	2023-02-13
538577	8 Jan 2019	2023-01-08	180471	10 Apr 2018	2023-01-28	204968	10 Apr 2018	2023-02-13
538576	8 Jan 2019	2023-01-08	180470	10 Apr 2018	2023-01-28	161581	10 Apr 2018	2023-02-13
145402	10 Apr 2018	2023-01-09	337118	10 Apr 2018	2023-01-28	101040	10 Apr 2018	2023-02-13
296857	10 Apr 2018	2023-01-11	337117	10 Apr 2018	2023-01-28	215784	10 Apr 2018	2023-02-13
285689	10 Apr 2018	2023-01-11	249632	10 Apr 2018	2023-01-28	204969	10 Apr 2018	2023-02-13
226386	10 Apr 2018	2023-01-11	241632	10 Apr 2018	2023-01-28	342008	10 Apr 2018	2023-02-13
207699	10 Apr 2018	2023-01-11	241631	10 Apr 2018	2023-01-28	330170	10 Apr 2018	2023-02-13
207698	10 Apr 2018	2023-01-11	212246	10 Apr 2018	2023-01-28	283587	10 Apr 2018	2023-02-13
188484	10 Apr 2018	2023-01-11	146947	10 Apr 2018	2023-01-28	215787	10 Apr 2018	2023-02-13
188483	10 Apr 2018	2023-01-11	130237	10 Apr 2018	2023-01-28	215786	10 Apr 2018	2023-02-13
125113	10 Apr 2018	2023-01-11	130236	10 Apr 2018	2023-01-28	341910	10 Apr 2018	2023-02-13
102758	10 Apr 2018	2023-01-11	297524	10 Apr 2018	2023-01-28	329563	10 Apr 2018	2023-02-13
279029	10 Apr 2018	2023-01-11	230984	10 Apr 2018	2023-01-28	270872	10 Apr 2018	2023-02-13
277502	10 Apr 2018	2023-01-11	230983	10 Apr 2018	2023-01-28	270871	10 Apr 2018	2023-02-13
230302	10 Apr 2018	2023-01-11	212192	10 Apr 2018	2023-01-28	262908	10 Apr 2018	2023-02-13
230301	10 Apr 2018	2023-01-11	212191	10 Apr 2018	2023-01-28	234219	10 Apr 2018	2023-02-13

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
157580	10 Apr 2018	2023-01-11	204092	10 Apr 2018	2023-01-28	215685	10 Apr 2018	2023-02-13
117133	10 Apr 2018	2023-01-11	164297	10 Apr 2018	2023-01-28	215684	10 Apr 2018	2023-02-13
100490	10 Apr 2018	2023-01-11	158783	10 Apr 2018	2023-01-28	204884	10 Apr 2018	2023-02-13
100489	10 Apr 2018	2023-01-11	102013	10 Apr 2018	2023-01-28	204883	10 Apr 2018	2023-02-13
342571	10 Apr 2018	2023-01-28	329433	10 Apr 2018	2023-01-28	161501	10 Apr 2018	2023-02-13
330940	10 Apr 2018	2023-01-28	270244	10 Apr 2018	2023-01-28	116204	10 Apr 2018	2023-02-13
330217	10 Apr 2018	2023-01-28	233589	10 Apr 2018	2023-01-28	343974	10 Apr 2018	2023-02-13
234372	10 Apr 2018	2023-01-28	233588	10 Apr 2018	2023-01-28	292360	10 Apr 2018	2023-02-13
217070	10 Apr 2018	2023-01-28	215065	10 Apr 2018	2023-01-28	272956	10 Apr 2018	2023-02-13
217069	10 Apr 2018	2023-01-28	121759	10 Apr 2018	2023-01-28	264986	10 Apr 2018	2023-02-13
101019	10 Apr 2018	2023-01-28	121758	10 Apr 2018	2023-01-28	206368	10 Apr 2018	2023-02-13
263558	10 Apr 2018	2023-01-28	228398	10 Apr 2018	2023-01-28	206367	10 Apr 2018	2023-02-13
180479	10 Apr 2018	2023-01-28	209063	10 Apr 2018	2023-01-28	329575	10 Apr 2018	2023-02-13
326808	10 Apr 2018	2023-01-28	190572	10 Apr 2018	2023-01-28	329574	10 Apr 2018	2023-02-13
158782	10 Apr 2018	2023-01-28	172298	10 Apr 2018	2023-01-28	270879	10 Apr 2018	2023-02-13
264293	10 Apr 2018	2023-01-28	117466	10 Apr 2018	2023-01-28	270877	10 Apr 2018	2023-02-13
168873	10 Apr 2018	2023-01-28	117465	10 Apr 2018	2023-01-28	270876	10 Apr 2018	2023-02-13
152272	10 Apr 2018	2023-01-28	117464	10 Apr 2018	2023-01-28	262915	10 Apr 2018	2023-02-13
234225	10 Apr 2018	2023-02-13	156137	10 Apr 2018	2023-03-01	282386	10 Apr 2018	2023-11-22
180352	10 Apr 2018	2023-02-13	144034	10 Apr 2018	2023-03-01	233559	10 Apr 2018	2023-11-22
166989	10 Apr 2018	2023-02-13	114878	10 Apr 2018	2023-03-01	225616	10 Apr 2018	2023-11-22
166988	10 Apr 2018	2023-02-13	314078	10 Apr 2018	2023-03-01	280268	10 Apr 2018	2023-11-22
161505	10 Apr 2018	2023-02-13	211476	10 Apr 2018	2023-03-01	268219	10 Apr 2018	2023-11-22
122388	10 Apr 2018	2023-02-13	203387	10 Apr 2018	2023-03-01	177672	10 Apr 2018	2023-11-22
122387	10 Apr 2018	2023-02-13	288873	10 Apr 2018	2023-03-03	297585	10 Apr 2018	2023-11-22
342583	10 Apr 2018	2023-02-13	268220	10 Apr 2018	2023-03-03	268218	10 Apr 2018	2023-11-22
290980	10 Apr 2018	2023-02-13	212758	10 Apr 2018	2023-03-03	268217	10 Apr 2018	2023-11-22

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
263585	10 Apr 2018	2023-02-13	202707	10 Apr 2018	2023-03-03	260197	10 Apr 2018	2023-11-22
234900	10 Apr 2018	2023-02-13	126238	10 Apr 2018	2023-03-03	224258	10 Apr 2018	2023-11-22
216369	10 Apr 2018	2023-02-13	287549	10 Apr 2018	2023-03-03	212757	10 Apr 2018	2023-11-22
197596	10 Apr 2018	2023-02-13	213515	10 Apr 2018	2023-03-03	326768	10 Apr 2018	2023-11-22
168190	10 Apr 2018	2023-02-13	313383	10 Apr 2018	2023-03-03	326767	10 Apr 2018	2023-11-22
151591	10 Apr 2018	2023-02-13	295630	10 Apr 2018	2023-03-03	296982	10 Apr 2018	2023-11-22
343919	10 Apr 2018	2023-02-13	229585	10 Apr 2018	2023-03-03	278173	10 Apr 2018	2023-11-22
321009	10 Apr 2018	2023-02-13	229584	10 Apr 2018	2023-03-03	224178	10 Apr 2018	2023-11-22
272901	10 Apr 2018	2023-02-13	202708	10 Apr 2018	2023-03-03	204051	10 Apr 2018	2023-11-22
264922	10 Apr 2018	2023-02-13	144783	10 Apr 2018	2023-03-03	158238	10 Apr 2018	2023-11-22
217694	10 Apr 2018	2023-02-13	296983	10 Apr 2018	2023-03-03	128963	10 Apr 2018	2023-11-22
169012	10 Apr 2018	2023-02-13	279682	10 Apr 2018	2023-03-03	115966	10 Apr 2018	2023-11-22
263505	10 Apr 2018	2023-02-13	278174	10 Apr 2018	2023-03-03	101980	10 Apr 2018	2023-11-22
234316	10 Apr 2018	2023-02-13	128964	10 Apr 2018	2023-03-03	326138	10 Apr 2018	2023-11-22
215785	10 Apr 2018	2023-02-13	294288	10 Apr 2018	2023-03-03	279040	10 Apr 2018	2023-11-22
204970	10 Apr 2018	2023-02-13	287550	10 Apr 2018	2023-03-03	277516	10 Apr 2018	2023-11-22
197526	10 Apr 2018	2023-02-13	209412	10 Apr 2018	2023-03-03	277515	10 Apr 2018	2023-11-22
180429	10 Apr 2018	2023-02-13	142755	10 Apr 2018	2023-03-03	277514	10 Apr 2018	2023-11-22
161583	10 Apr 2018	2023-02-13	116846	10 Apr 2018	2023-03-03	223549	10 Apr 2018	2023-11-22
161582	10 Apr 2018	2023-02-13	339966	10 Apr 2018	2023-03-03	223548	10 Apr 2018	2023-11-22
122483	10 Apr 2018	2023-02-13	159596	10 Apr 2018	2023-03-03	203410	10 Apr 2018	2023-11-22
101818	10 Apr 2018	2023-02-13	120435	10 Apr 2018	2023-03-03	203409	10 Apr 2018	2023-11-22
277533	10 Apr 2018	2023-02-20	120434	10 Apr 2018	2023-03-03	203408	10 Apr 2018	2023-11-22
223567	10 Apr 2018	2023-02-20	290446	10 Apr 2018	2023-03-13	163622	10 Apr 2018	2023-11-22
296873	10 Apr 2018	2023-02-20	283635	10 Apr 2018	2023-03-13	117150	10 Apr 2018	2023-11-22
163633	10 Apr 2018	2023-02-20	167651	10 Apr 2018	2023-03-13	282949	10 Apr 2018	2023-11-22
117167	10 Apr 2018	2023-02-20	161642	10 Apr 2018	2023-03-13	233680	10 Apr 2018	2023-11-22

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
117166	10 Apr 2018	2023-02-20	259586	10 Apr 2018	2023-05-16	214929	10 Apr 2018	2023-11-22
277475	10 Apr 2018	2023-03-01	120457	10 Apr 2018	2023-11-22	166964	10 Apr 2018	2023-11-22
128264	10 Apr 2018	2023-03-01	102048	10 Apr 2018	2023-11-22	161478	10 Apr 2018	2023-11-22
294897	10 Apr 2018	2023-03-01	312747	10 Apr 2018	2023-11-22	161477	10 Apr 2018	2023-11-22
314106	10 Apr 2018	2023-03-01	224177	10 Apr 2018	2023-11-22	101701	10 Apr 2018	2023-11-22
211513	10 Apr 2018	2023-03-01	158239	10 Apr 2018	2023-11-22	341325	10 Apr 2018	2023-11-22
101846	10 Apr 2018	2023-03-01	326139	10 Apr 2018	2023-11-22	282387	10 Apr 2018	2023-11-22
294896	10 Apr 2018	2023-03-01	145347	10 Apr 2018	2023-11-22	270293	10 Apr 2018	2023-11-22
288151	10 Apr 2018	2023-03-01	279549	10 Apr 2018	2023-11-22	270292	10 Apr 2018	2023-11-22
276047	10 Apr 2018	2023-03-01	223550	10 Apr 2018	2023-11-22	196185	10 Apr 2018	2023-11-22
257542	10 Apr 2018	2023-03-01	145346	10 Apr 2018	2023-11-22	179766	10 Apr 2018	2023-11-22
162157	10 Apr 2018	2023-03-01	128307	10 Apr 2018	2023-11-22	101647	10 Apr 2018	2023-11-22
101646	10 Apr 2018	2023-11-22	200785	10 Apr 2018	2023-12-02	118243	10 Apr 2018	2023-12-02
340574	10 Apr 2018	2023-11-22	196214	10 Apr 2018	2023-12-02	118242	10 Apr 2018	2023-12-02
328213	10 Apr 2018	2023-11-22	180313	10 Apr 2018	2023-12-02	335764	10 Apr 2018	2023-12-02
298931	10 Apr 2018	2023-11-22	171526	10 Apr 2018	2023-12-02	335763	10 Apr 2018	2023-12-02
281647	10 Apr 2018	2023-11-22	116173	10 Apr 2018	2023-12-02	248334	10 Apr 2018	2023-12-02
281646	10 Apr 2018	2023-11-22	100559	10 Apr 2018	2023-12-02	248333	10 Apr 2018	2023-12-02
232905	10 Apr 2018	2023-11-22	274846	10 Apr 2018	2023-12-02	240838	10 Apr 2018	2023-12-02
225617	10 Apr 2018	2023-11-22	267413	10 Apr 2018	2023-12-02	229466	10 Apr 2018	2023-12-02
194960	10 Apr 2018	2023-11-22	220896	10 Apr 2018	2023-12-02	192182	10 Apr 2018	2023-12-02
160185	10 Apr 2018	2023-11-22	200786	10 Apr 2018	2023-12-02	145634	10 Apr 2018	2023-12-02
116749	10 Apr 2018	2023-11-22	142699	10 Apr 2018	2023-12-02	128132	10 Apr 2018	2023-12-02
116748	10 Apr 2018	2023-11-22	341356	10 Apr 2018	2023-12-02	277522	10 Apr 2018	2024-01-26
101427	10 Apr 2018	2023-11-22	341354	10 Apr 2018	2023-12-02	259488	10 Apr 2018	2024-01-26
101426	10 Apr 2018	2023-11-22	329522	10 Apr 2018	2023-12-02	203420	10 Apr 2018	2024-01-26
320908	10 Apr 2018	2023-11-22	329521	10 Apr 2018	2023-12-02	117159	10 Apr 2018	2024-01-26

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
284377	10 Apr 2018	2023-11-22	329520	10 Apr 2018	2023-12-02	343305	10 Apr 2018	2024-01-26
284376	10 Apr 2018	2023-11-22	270341	10 Apr 2018	2023-12-02	270336	10 Apr 2018	2024-01-26
181717	10 Apr 2018	2023-11-22	270320	10 Apr 2018	2023-12-02	122352	10 Apr 2018	2024-01-26
123787	10 Apr 2018	2023-11-22	262867	10 Apr 2018	2023-12-02	217094	10 Apr 2018	2024-01-26
117757	10 Apr 2018	2023-11-22	233661	10 Apr 2018	2023-12-02	206230	10 Apr 2018	2024-01-26
101271	10 Apr 2018	2023-11-22	214926	10 Apr 2018	2023-12-02	326142	10 Apr 2018	2024-01-26
259592	10 Apr 2018	2023-11-27	214905	10 Apr 2018	2023-12-02	314101	10 Apr 2018	2024-01-26
158217	10 Apr 2018	2023-11-27	166947	10 Apr 2018	2023-12-02	314100	10 Apr 2018	2024-01-26
158216	10 Apr 2018	2023-11-27	166946	10 Apr 2018	2023-12-02	296866	10 Apr 2018	2024-01-26
115945	10 Apr 2018	2023-11-27	166945	10 Apr 2018	2023-12-02	279552	10 Apr 2018	2024-01-26
230947	10 Apr 2018	2023-11-27	122333	10 Apr 2018	2023-12-02	277523	10 Apr 2018	2024-01-26
230946	10 Apr 2018	2023-11-27	116192	10 Apr 2018	2023-12-02	223559	10 Apr 2018	2024-01-26
223676	10 Apr 2018	2023-11-27	323538	10 Apr 2018	2023-12-02	211499	10 Apr 2018	2024-01-26
340688	10 Apr 2018	2023-11-27	274845	10 Apr 2018	2023-12-02	203419	10 Apr 2018	2024-01-26
282247	10 Apr 2018	2023-11-27	274844	10 Apr 2018	2023-12-02	163627	10 Apr 2018	2024-01-26
269638	10 Apr 2018	2023-11-27	227684	10 Apr 2018	2023-12-02	157596	10 Apr 2018	2024-01-26
232993	10 Apr 2018	2023-11-27	208823	10 Apr 2018	2023-12-02	145358	10 Apr 2018	2024-01-26
214990	10 Apr 2018	2023-11-27	171529	10 Apr 2018	2023-12-02	128314	10 Apr 2018	2024-01-26
214989	10 Apr 2018	2023-11-27	171528	10 Apr 2018	2023-12-02	117160	10 Apr 2018	2024-01-26
214226	10 Apr 2018	2023-11-27	171527	10 Apr 2018	2023-12-02	117158	10 Apr 2018	2024-01-26
179644	10 Apr 2018	2023-11-27	100560	10 Apr 2018	2023-12-02	320899	10 Apr 2018	2024-01-26
343290	10 Apr 2018	2023-11-27	345304	10 Apr 2018	2023-12-02	282940	10 Apr 2018	2024-01-26
205708	10 Apr 2018	2023-11-27	345303	10 Apr 2018	2023-12-02	270335	10 Apr 2018	2024-01-26
198301	10 Apr 2018	2023-11-27	345302	10 Apr 2018	2023-12-02	284378	10 Apr 2018	2024-01-26
181696	10 Apr 2018	2023-11-27	322915	10 Apr 2018	2023-12-02	217093	10 Apr 2018	2024-01-26
152280	10 Apr 2018	2023-11-27	286365	10 Apr 2018	2023-12-02	344935	10 Apr 2018	2024-01-26
123767	10 Apr 2018	2023-11-27	274275	10 Apr 2018	2023-12-02	306216	10 Apr 2018	2024-01-26

Tenure number	Issue date	Anniversary	Tenure number	Issue Date	Anniversary	Tenure number	Issue Date	Anniversary
314675	10 Apr 2018	2023-12-02	274274	10 Apr 2018	2023-12-02	294051	10 Apr 2018	2024-01-26
227685	10 Apr 2018	2023-12-02	227057	10 Apr 2018	2023-12-02	286030	10 Apr 2018	2024-01-26
177670	10 Apr 2018	2023-12-02	227056	10 Apr 2018	2023-12-02	227400	10 Apr 2018	2024-01-26
341355	10 Apr 2018	2023-12-02	173093	10 Apr 2018	2023-12-02	218105	10 Apr 2018	2024-01-26
294224	10 Apr 2018	2023-12-02	170905	10 Apr 2018	2023-12-02	138229	10 Apr 2018	2024-01-26
274843	10 Apr 2018	2023-12-02	125802	10 Apr 2018	2023-12-02	108292	10 Apr 2018	2024-01-26
204068	10 Apr 2018	2024-04-19						
312759	10 Apr 2018	2024-04-19						
296996	10 Apr 2018	2024-04-19						
212190	10 Apr 2018	2024-04-19						
204069	10 Apr 2018	2024-04-19						
158251	10 Apr 2018	2024-04-19						
158250	10 Apr 2018	2024-04-19						
101996	10 Apr 2018	2024-04-19						
101995	10 Apr 2018	2024-04-19						
Total hectares: 24,437								

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.

4.3 Royalty and streaming agreements

Royal Gold Inc. (Royal Gold) through its wholly owned subsidiary RGLD Gold AG (Royal) entered 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 Item 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 favor of certain First Nations.

4.4 Environmental, permits, and other factors

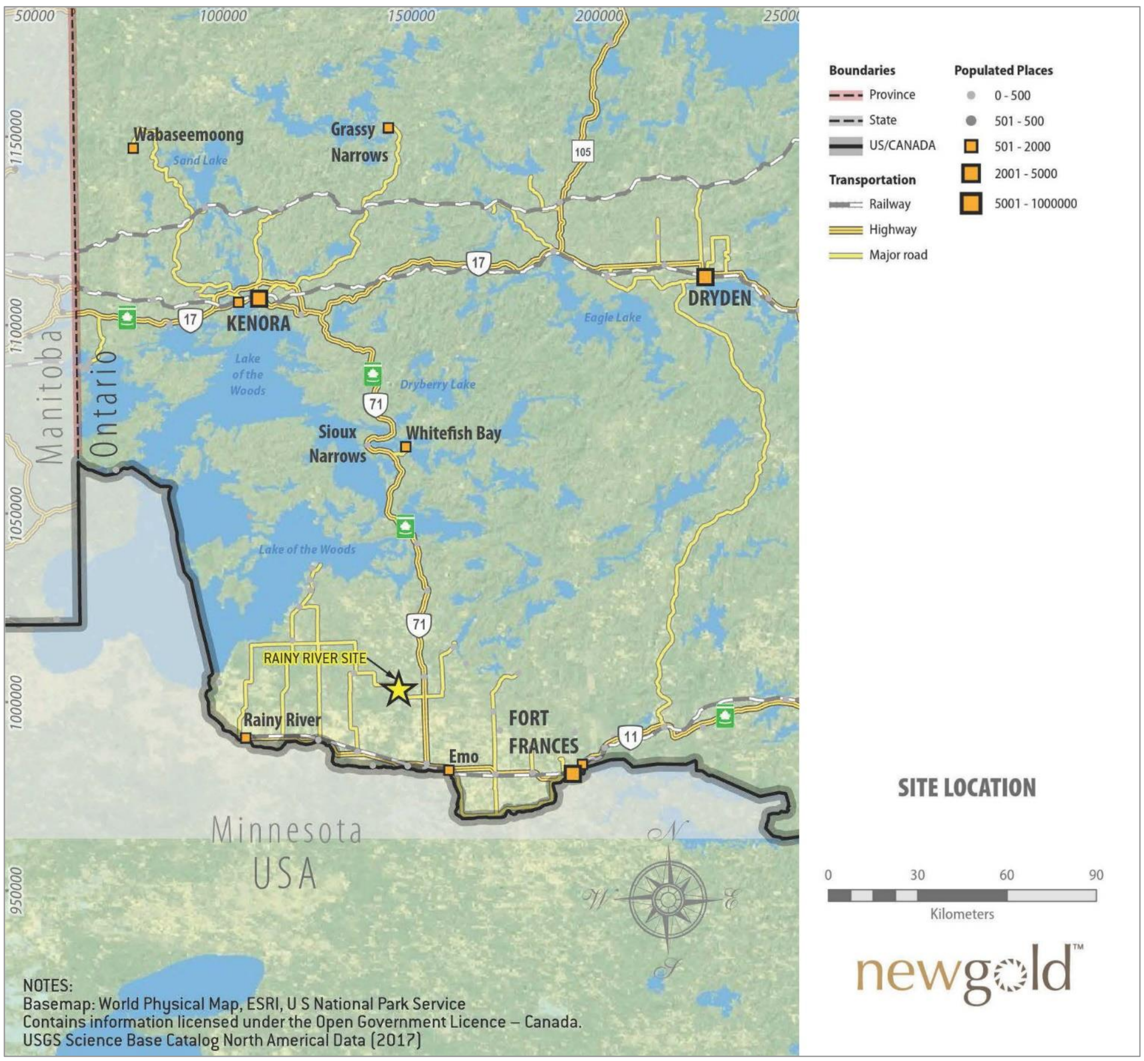
The QP is not aware of any environmental liabilities on the Property and New Gold has obtained all required permits to conduct the proposed work on the Property. The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed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

This item is more fully covered in Item 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.



Source: New Gold 2019.

Figure 5.1– Location and access to the Rainy River Mine site

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 Introduction 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 Item 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 Item 4.

6 HISTORY

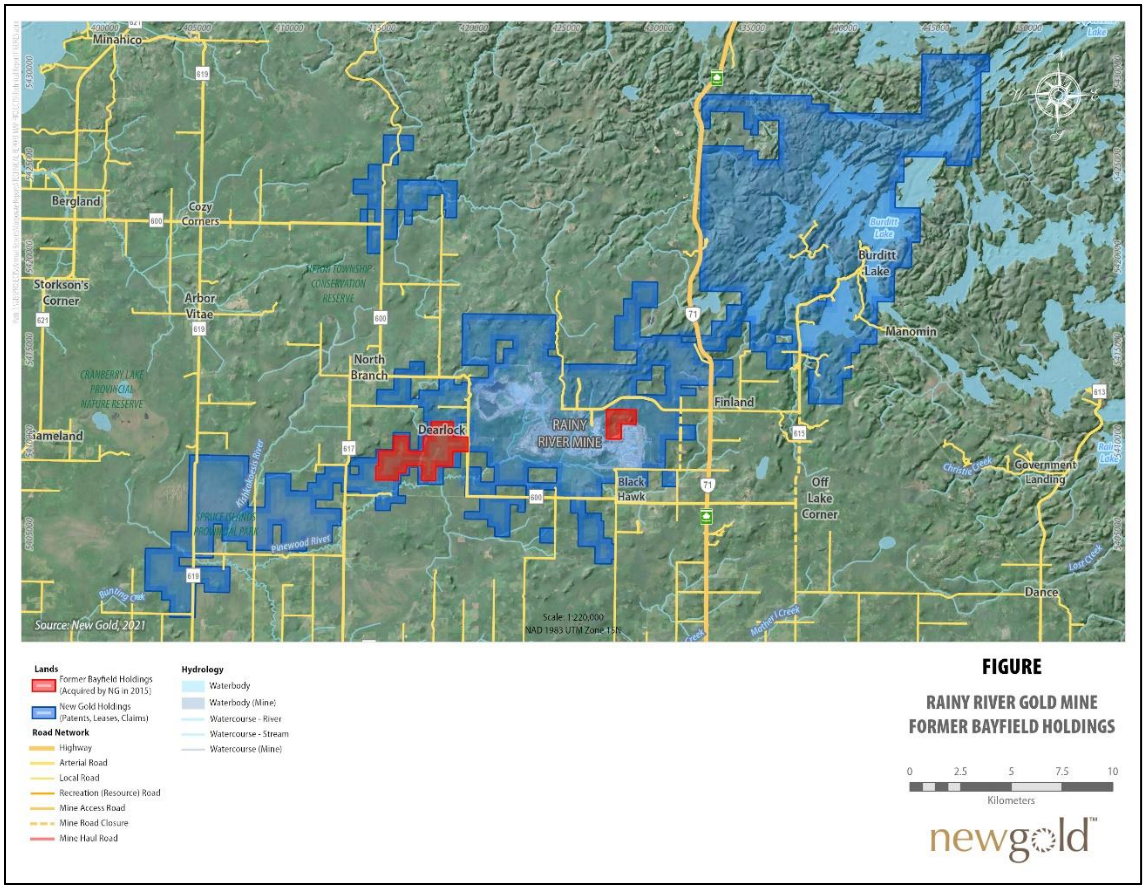
The following Item has been modified from the 2014 BBA Technical Report which in turn references documentation of exploration in north-western ON that is archived in the Ministry of Northern Development and Mines (MNDM) offices at Kenora.

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 Items, Figure 6.1 shows the land acquired during the Bayfield acquisition. Bayfield explored this ground from 2010 to 2014.



Source: New Gold January 2022.

Figure 6.1 –Claim map showing location of acquired Bayfield ground

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 Prior owners. 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

Year	Activity	Company
	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 Item 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
2011	88 km high-sensitivity potassium magnetometer ground survey	RDF Consulting
	Environmental baseline gap analysis	AMEC Earth and Environmental

Year	Activity	Company
	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 Item 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.

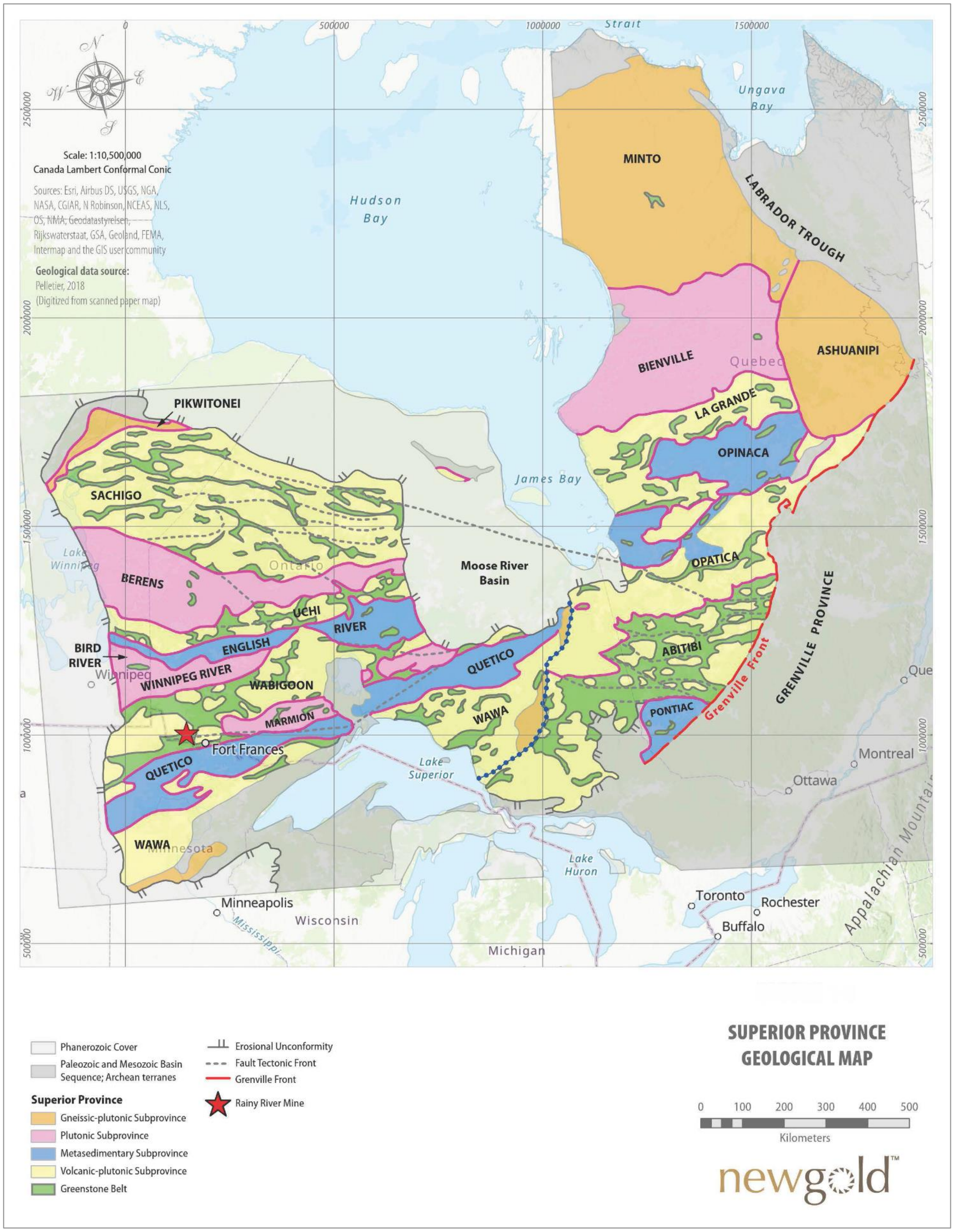
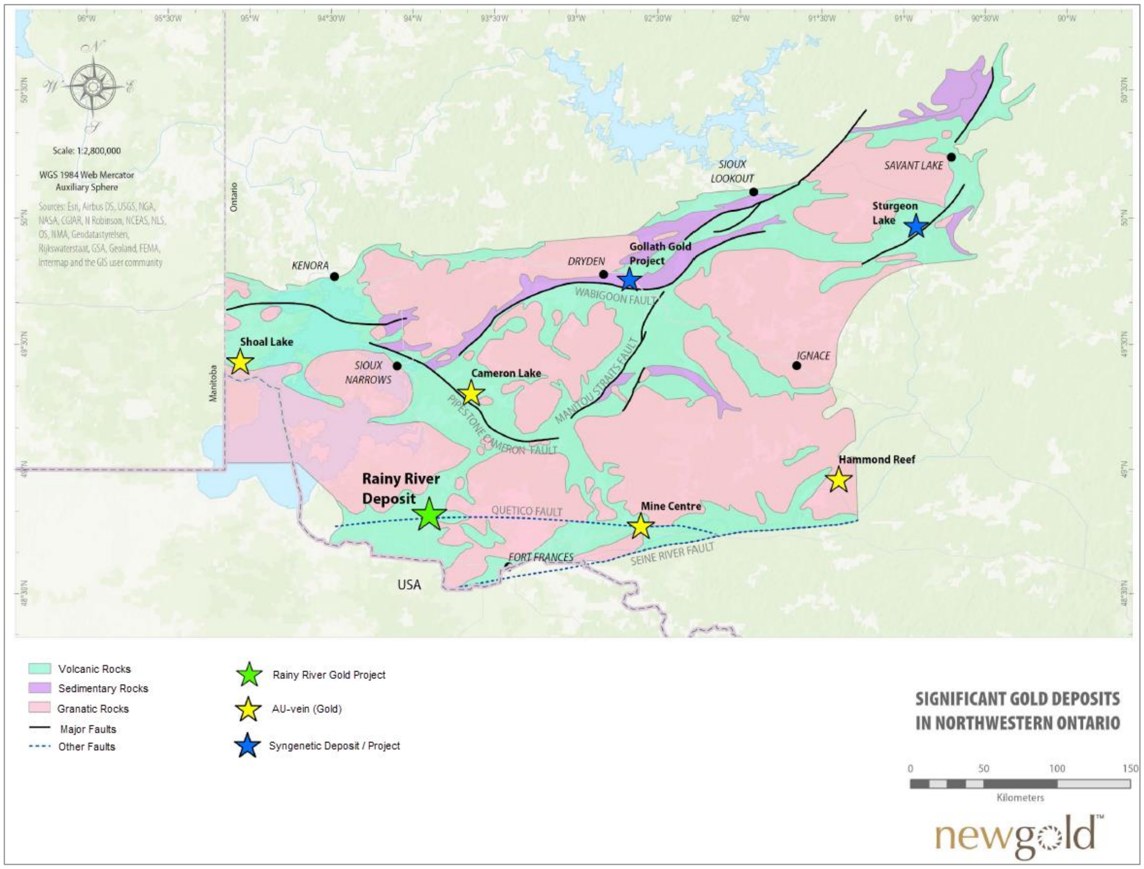


Figure 7.1 – Superior Province geological map



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

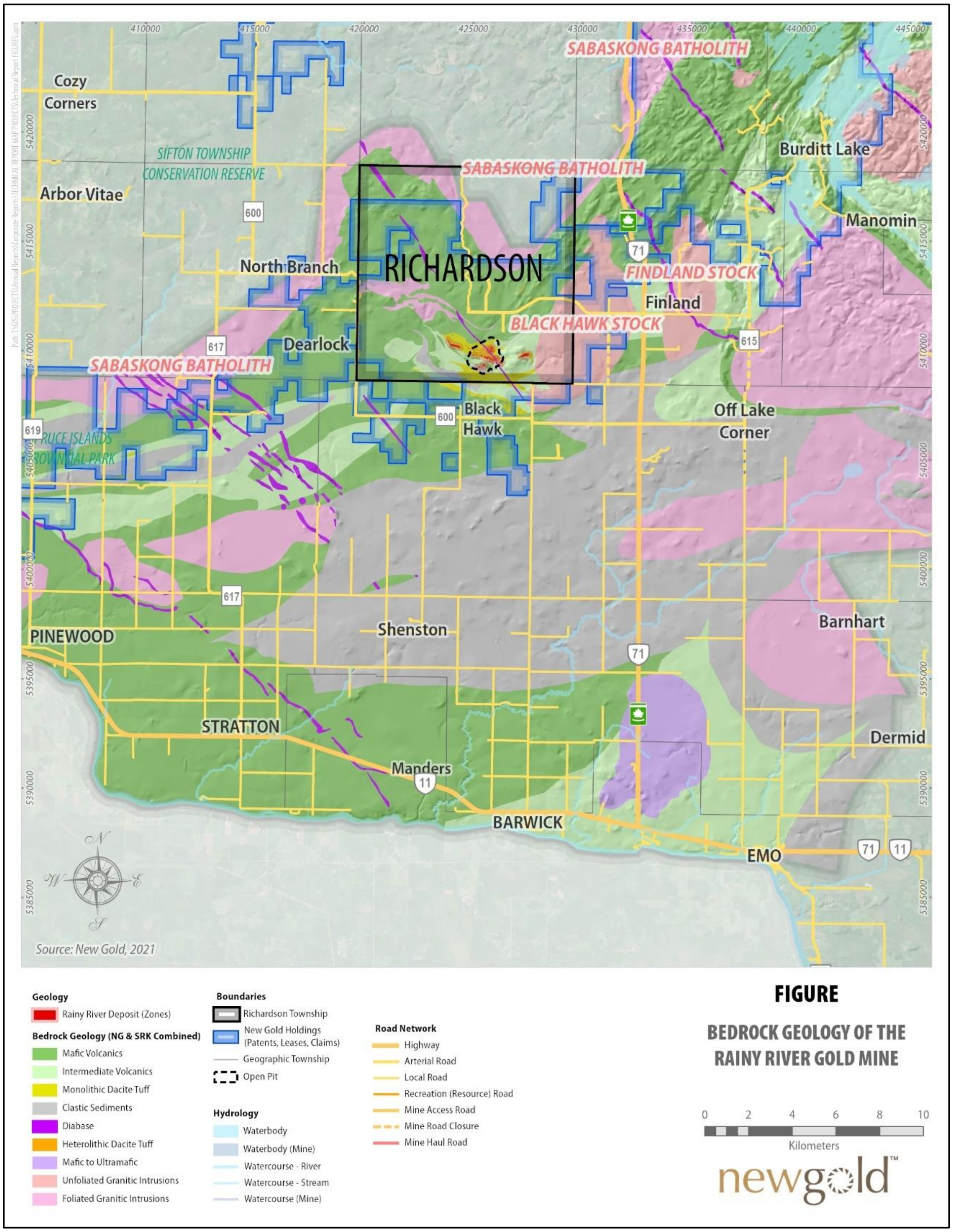
Figure 7.2 – Significant gold deposits in north-western ON

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.



Source: New Gold January 2021.

Figure 7.3 – Bedrock geology of the Rainy River Mine

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 Item 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 Item 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 meters 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 meters. 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 Item 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.

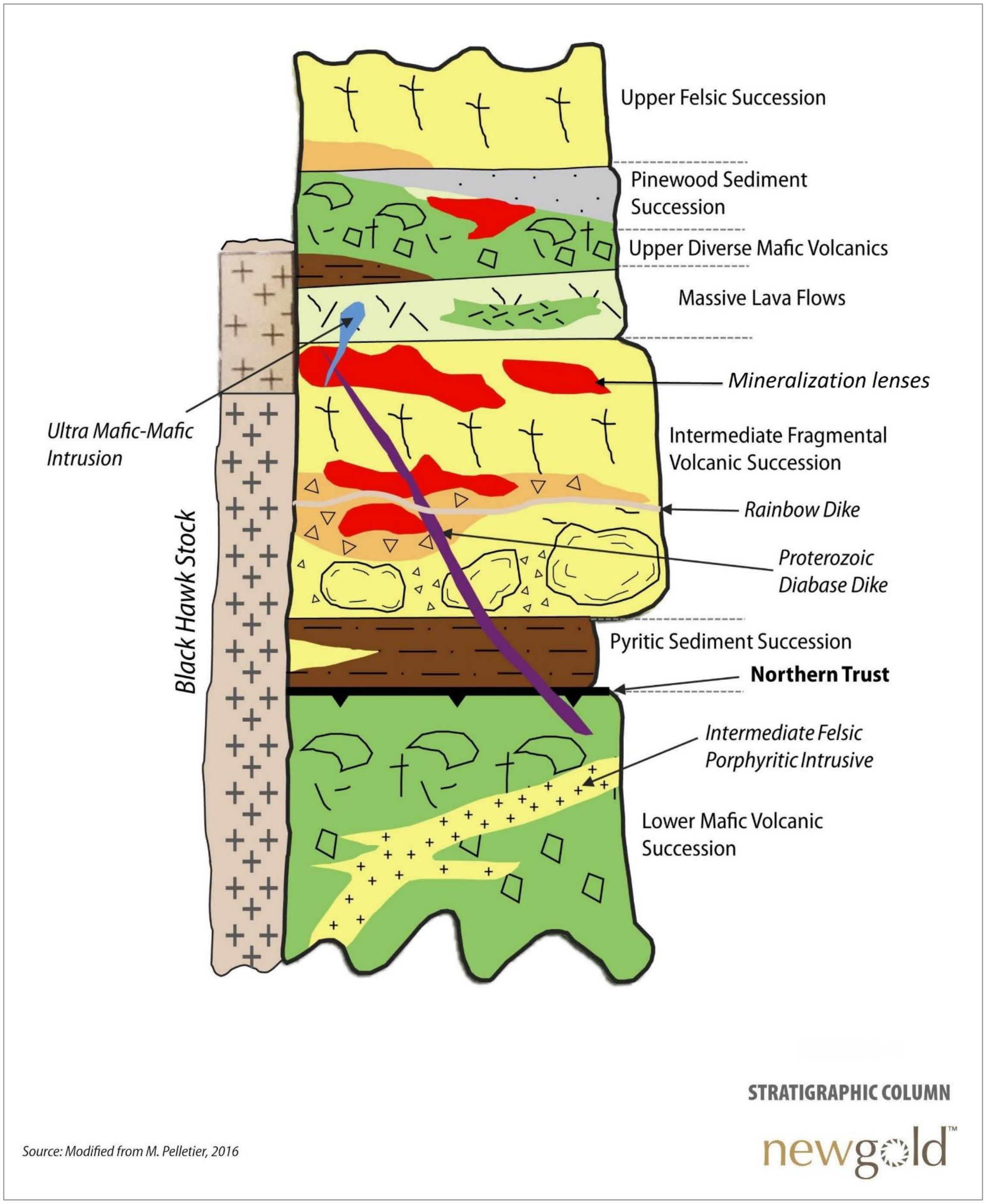


Figure 7.4 – Stratigraphic column

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.

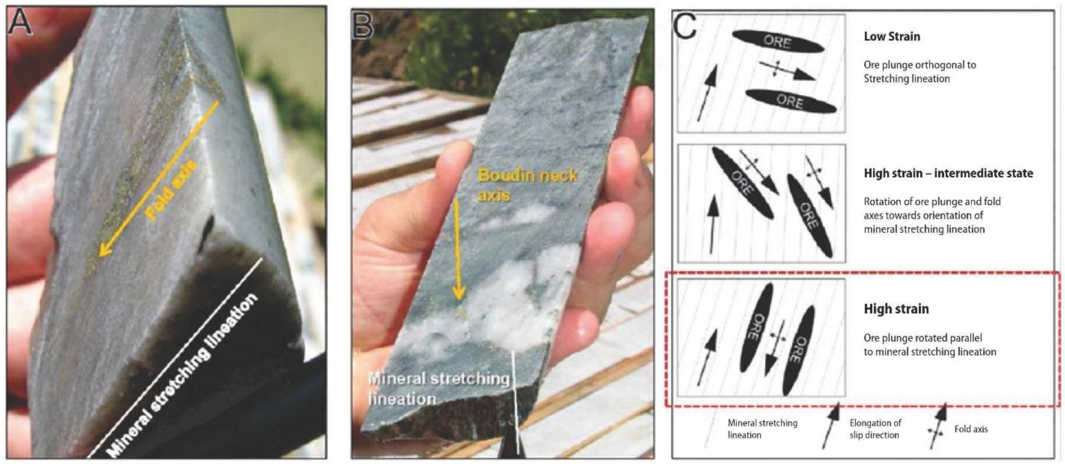
Key observations in core and outcrop include:

- 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.

Source: SRK 2011.

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



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

Source: Rainy River, 2013

STRUCTURAL CONTROL OVER THE PLUNGE OF GOLD MINERALIZATION AT THE RAINY RIVER GOLD MINE

newgold™

Source: New Gold from SRK 2011.

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

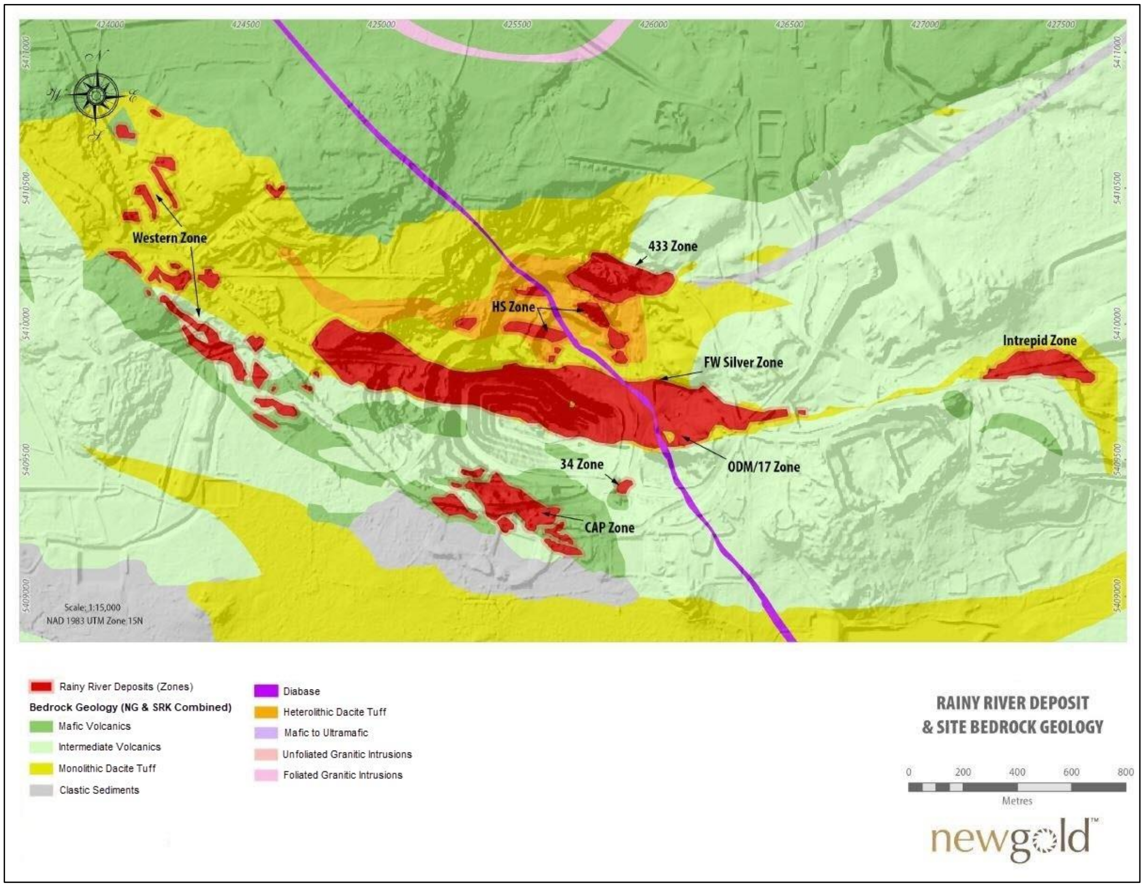
7.5 Deposit geology and mineralization

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 Item 14.



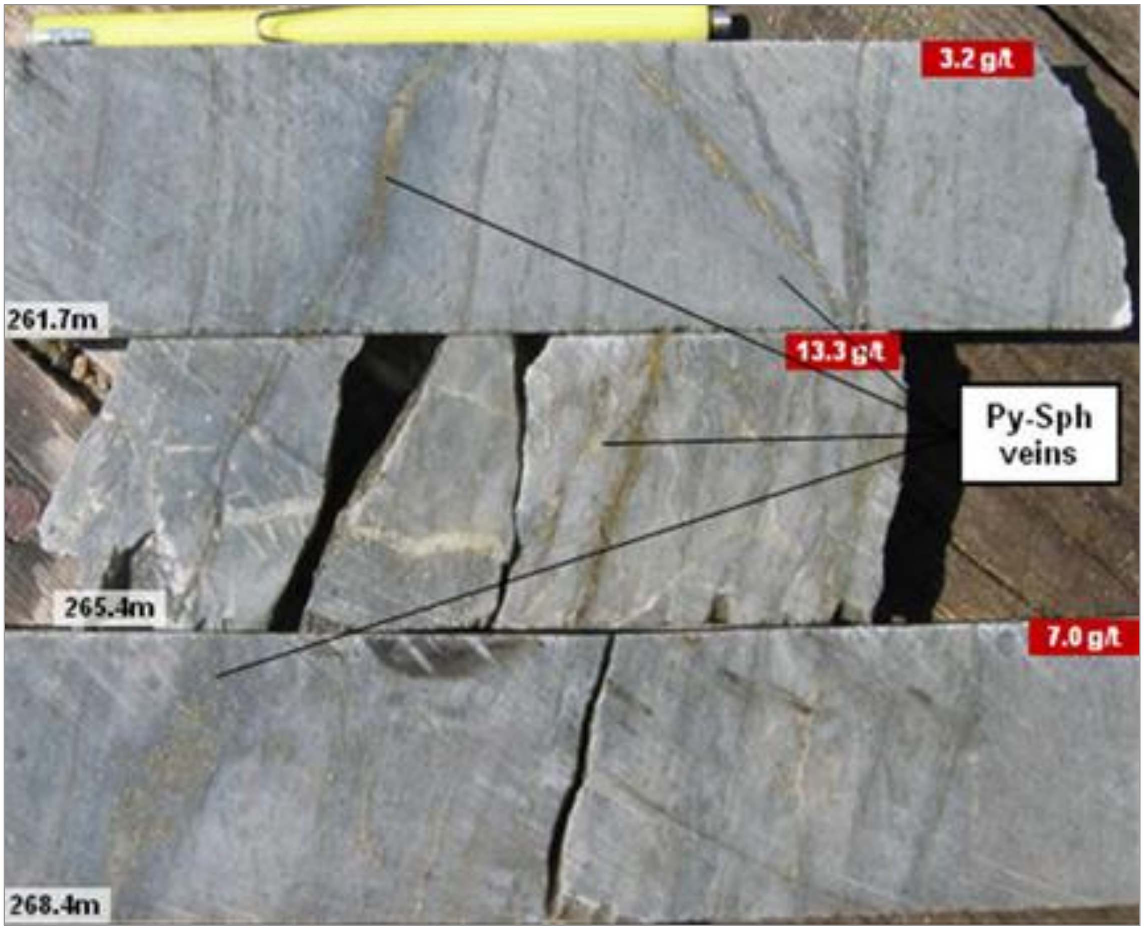
Source: New Gold 2019.

Figure 7.7 – Rainy River – mineralized zones

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.



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 7.8 – ODM/17 Zone 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.

Figure 7.9 – ODM/17 high grade gold mineralization

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.

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.

Figure 7.10 – 433 Zone high-grade gold mineralization

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.

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

Figure 7.11 – Higher-grade gold mineralization within the CAP Zone

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 Item 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.

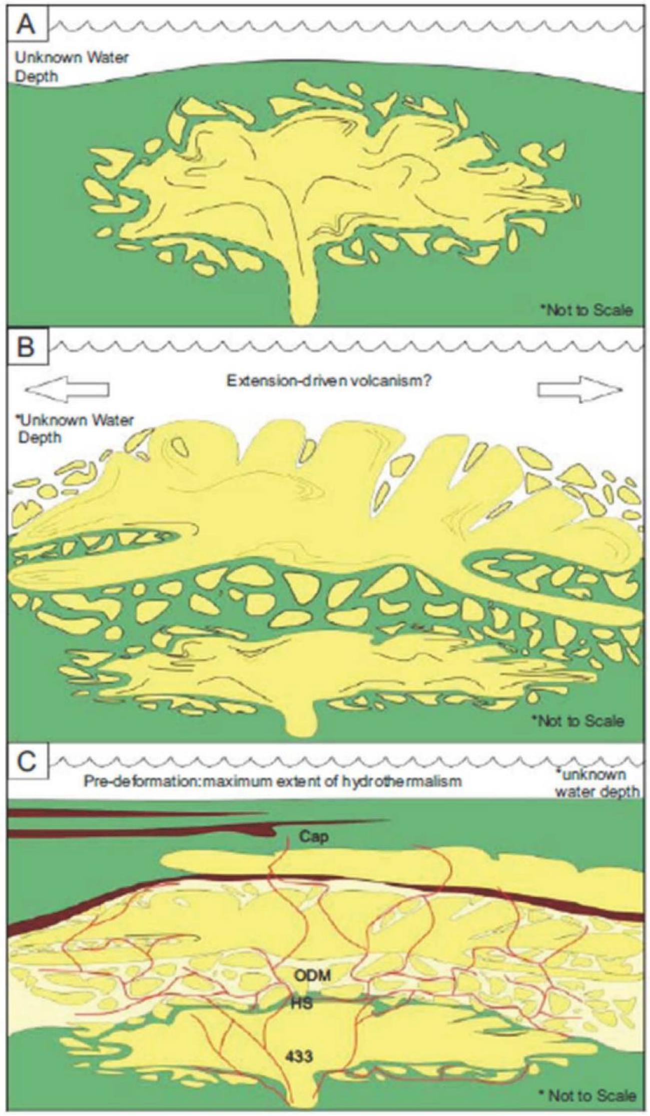
An early, pre-D2 origin for the alteration and sulphide zones is further supported by the strong control of the combined S2 and L2 fabrics on the shape of the mineralized zones and lithological contacts.

In VMS deposits, the main source of metals is 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).



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

newgold™

Figure 8.1 – Potential formation of the Rainy River deposit

9 EXPLORATION

New Gold has completed several 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
Jun 2020 – Nov 2021	231 rock chip samples, 1,303 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 Items 7 and 8.
Source: New Gold 2021.

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.

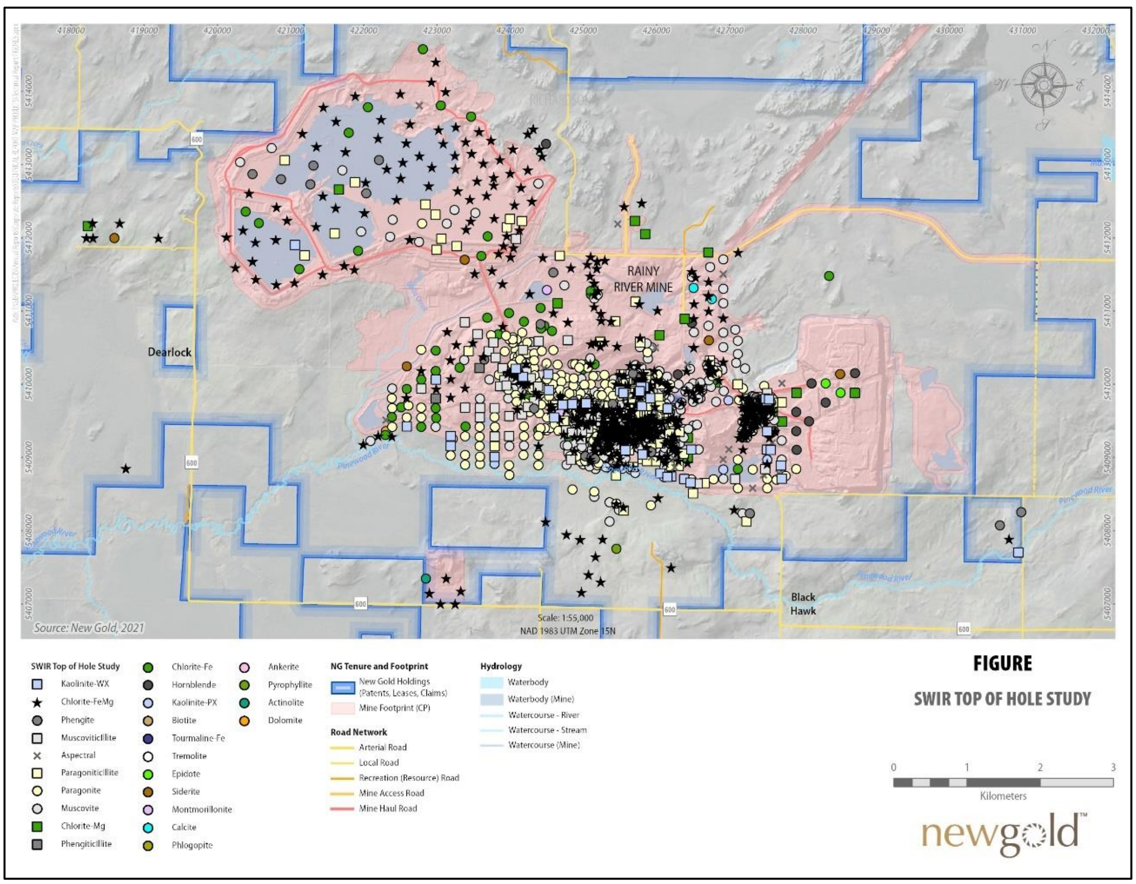
9.2 Relogging programs

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. New Gold has completed several 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.



Source: New Gold 2021.

Figure 9.1 – SWIR top of hole survey sample locations

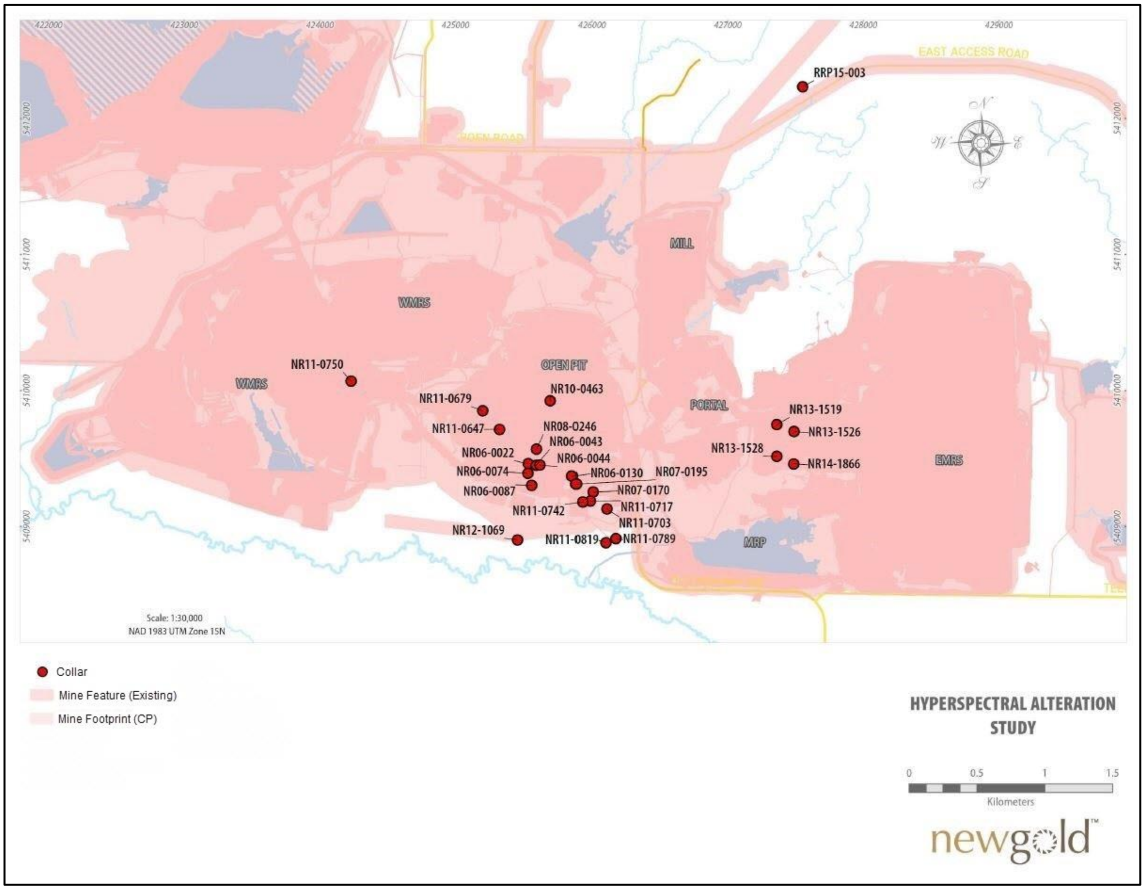
9.4 Hyperspectral alteration study

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.



Source: New Gold 2021.

Figure 9.2 – Corescan hyperspectral alteration study drillhole locations

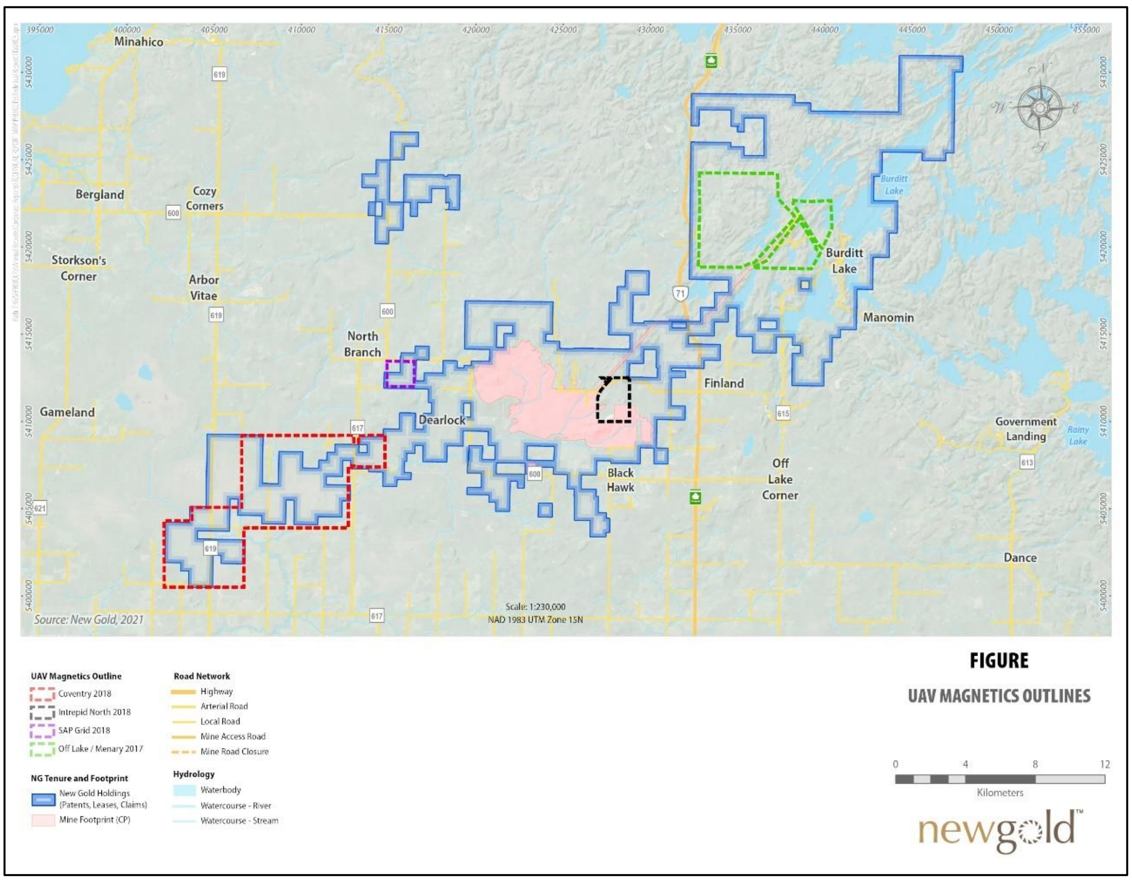
9.5 MSc research

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.



Source: New Gold 2021.

Figure 9.3 – 2017-2018 UAV magnetic survey areas

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 Item 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, the Vice President, Exploration and the Director, 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.

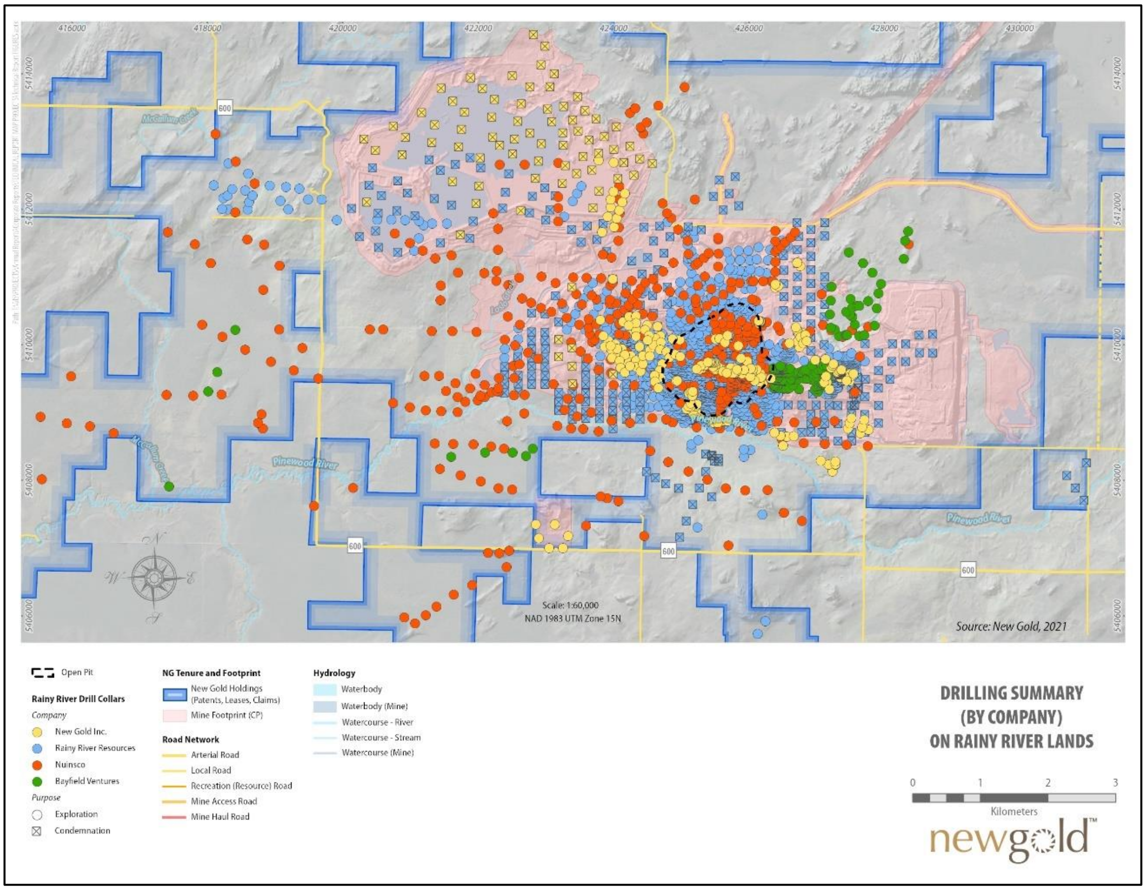
A complete summary of diamond core drilling completed at the Rainy River Mine is included in Table 10.1 and includes all diamond core drillholes drilled on the Property. RC, geotechnical, and abandoned holes are excluded. Drillholes used in the Mineral Resource estimate are a subset of this drilling 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 2020 2021	9 4 13	3,358 1,298 4,079		
	New Gold total	284	89,501	115	23,390
All	Overall total	2,211	930,423	305	66,018

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

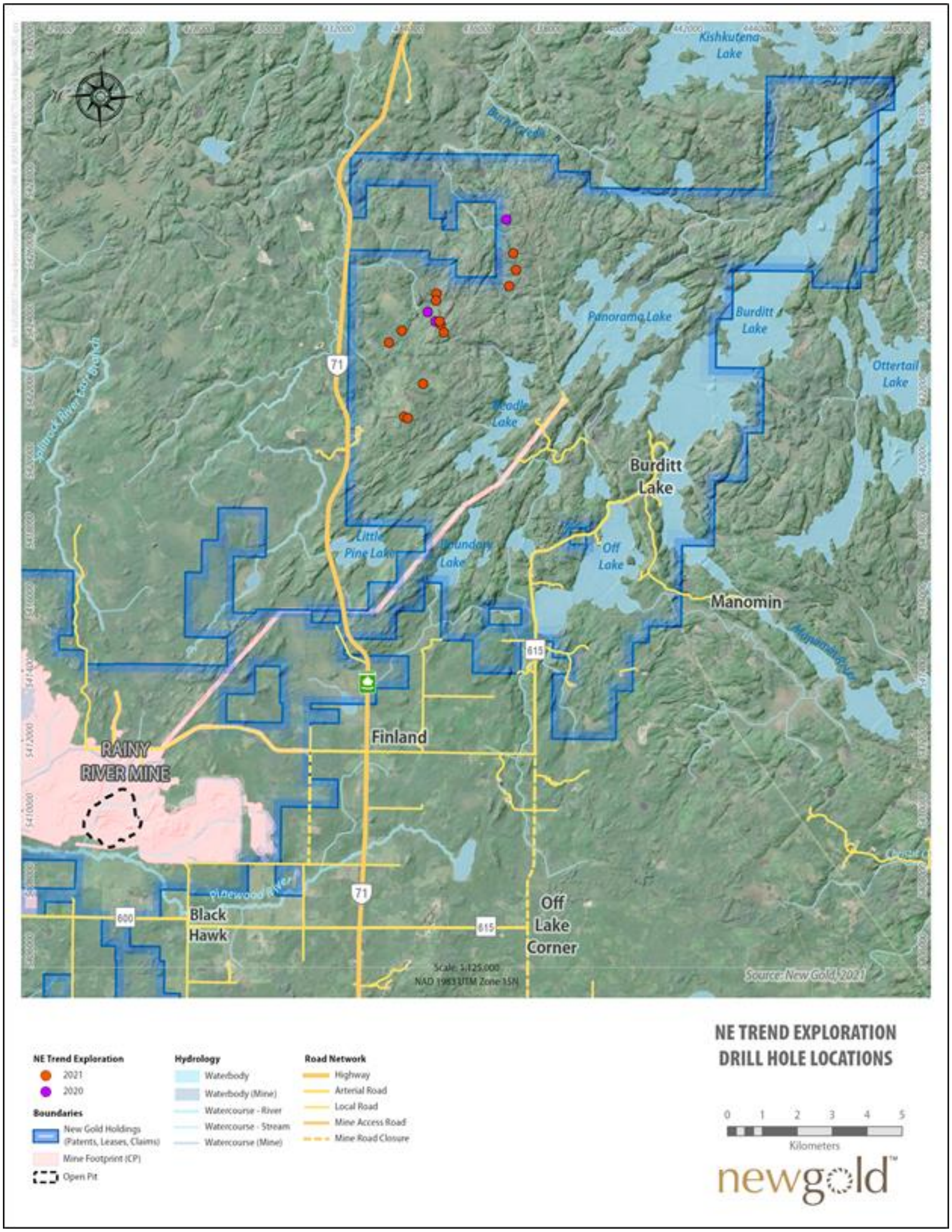
Drillholes were designed to provide sufficient information to delineate Mineral Resources. Representative cross sections of drilling completed at the Rainy River deposit are presented in Figure 10.2 to Figure 10.4.



Source: New Gold 2021.

Figure 10.1 – Rainy River Deposit Drillhole location map

In December 2020 New Gold started a reconnaissance drilling program on the north portion of the Campany's landholding in an area defined as NE Trend to explore for shear hosted gold mineralization within an interpreted ~15 km long north-northeast oriented structural corridor. As of December 2021, a total of 5,377 metres in 18 diamond drill holes were completed and drill hole collar's location are presented in Figure 10.2.



Source: New Gold 2021.

Figure 10.2 – NE Trend Drillhole location map

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 was adjusted where required to respect geological boundaries. Under New Gold from 2013 to 2015, sampling was performed at regular 1.5 m intervals. During the period 2016-2017 the average sampling length was changed to 1 m intervals for delineation drill holes completed within the open pit and adjusted to 1.5 m intervals from 2019 to present. 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

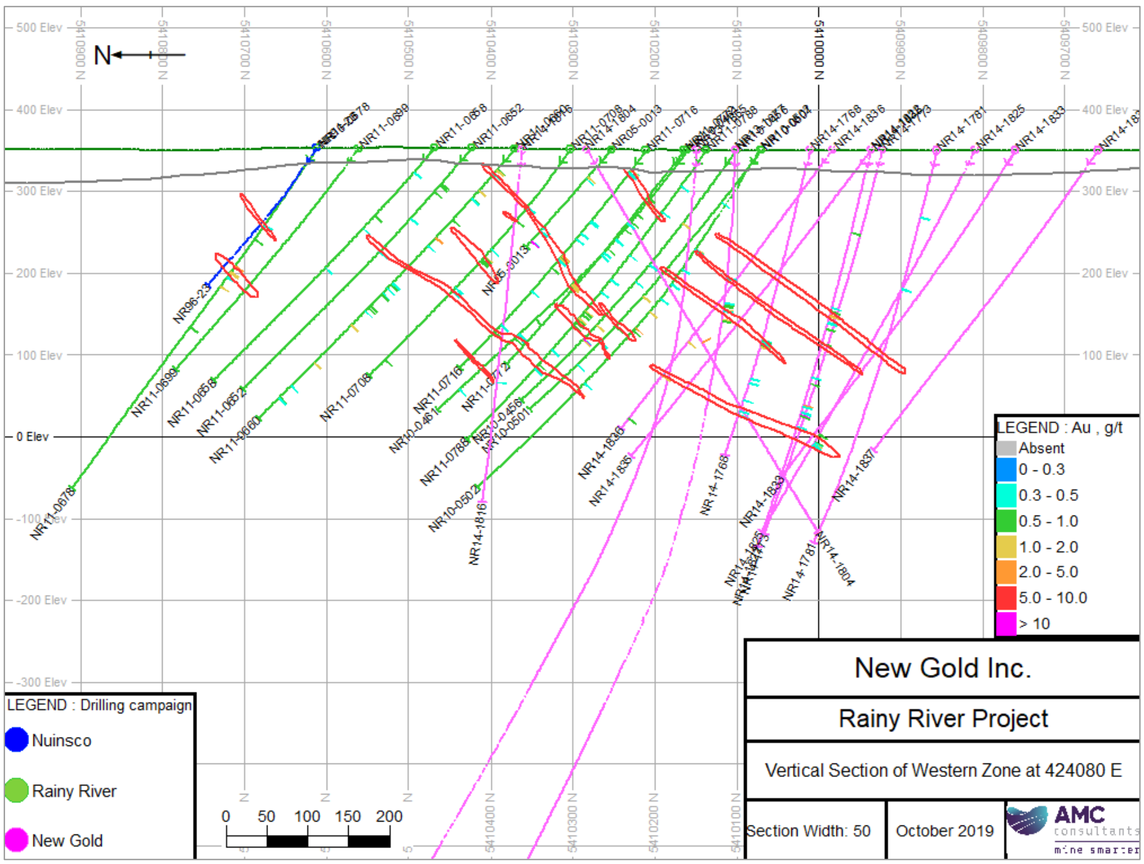
Diamond core sample recovery data has been collected since New Gold acquired the Property in 2013. Core recoveries from New Gold drill programs vary between 2.33% and 100% averaging 99.9%. A total of 219 of the 16746 intervals in the database have recoveries less than 90%.

10.6 Representative sections

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

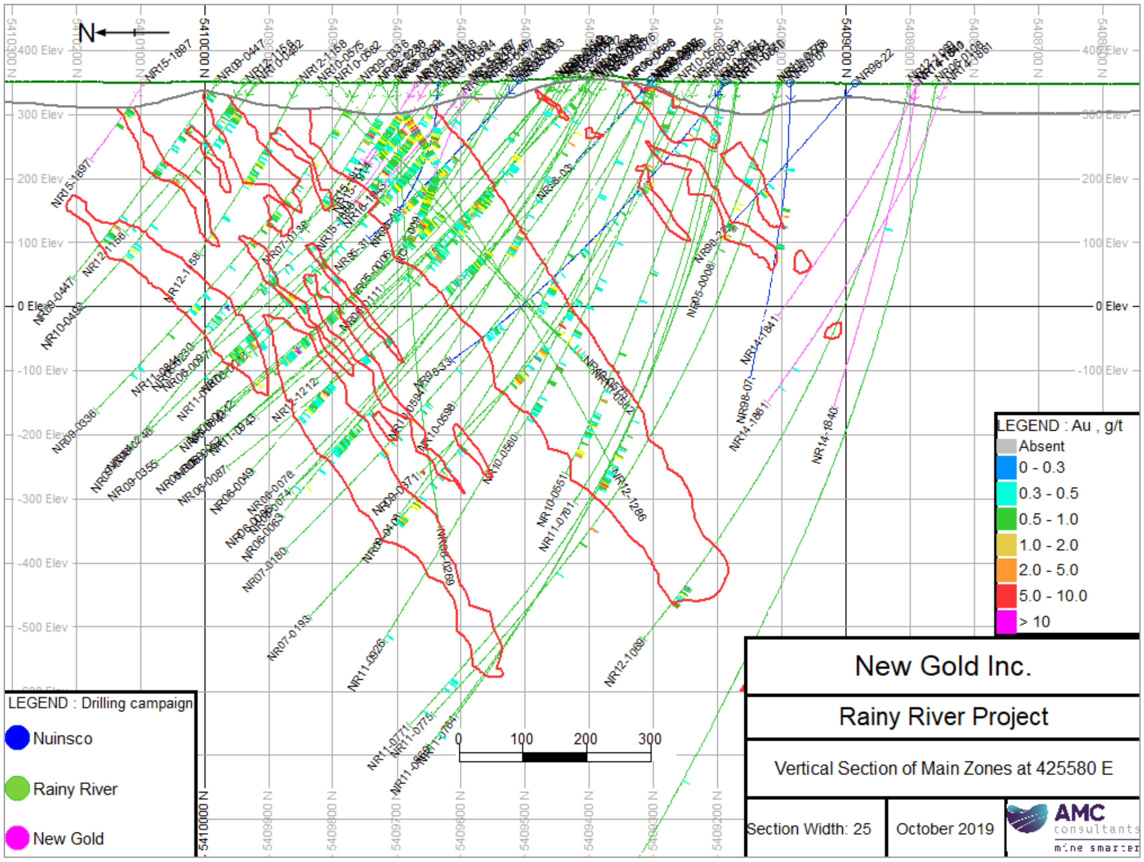
- Figure 10.2 is a section through the Western Zone.
- Figure 10.3 is a section through the main zones (including the ODM/17 Zone).
- Figure 10.4 is a cross section through the Intrepid Zone.

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



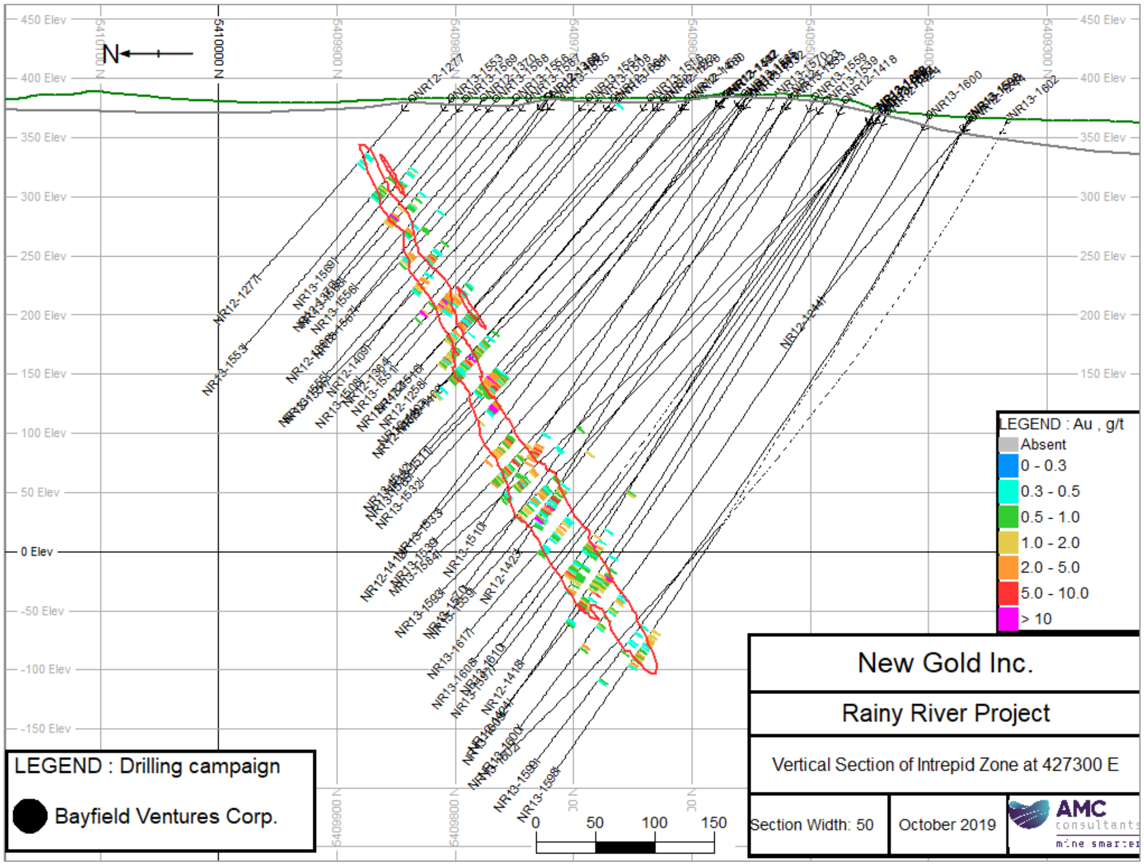
Source: AMC 2019.

Figure 10.3 – Vertical section through the Western Zone



Source: AMC 2019.

Figure 10.4 – Vertical section through the main zones (including ODM/17 Zone)



Source: AMC 2019.

Figure 10.5 – Vertical section through the Intrepid Zone

10.7 Conclusion

In QP'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 Item 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 Items 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 11.4Table 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 were 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)	NA	NA	NA	NA	NA

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 Item 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. The drilling in this period forms the basis of the Mineral Resource estimate.

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 add 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 derived from the data provided by New Gold.

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

Company			Rainy River Resources										New Gold			
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
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

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]
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	1.81	0.06							37	68	18	10				133

Company			Rainy River Resources										New Gold			
16a																
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				Total [1]
Laboratory			Actlabs		ALS							
CRM	Expected value (Ag ppm)	Stdv	2010	2011	2011	2012	2013	2014	2015	2016	2017	
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 Discussion on CRMs

CRMs are inserted to check the analytical accuracy of the lab. The QP 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 manufac-turer	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

CRM	Au value (ppm)	No. CRMS	Years	CRM manufac-turer	Analytical lab	Notes	Results
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.

The QP 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 11.3 – Gold CRM CDN-GS-1H

The QP considers a <5% failure rate acceptable for an individual CRM. While several CRMs have not met this criterion, the QP 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	Analyti-cal 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%

CRM	Expected Au value (ppm)	Stdv	Years used	Analyti-cal lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
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%
GBMS9 11-1	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	1.52	0.07	2010 –	Actlabs,	15	0	1	7%

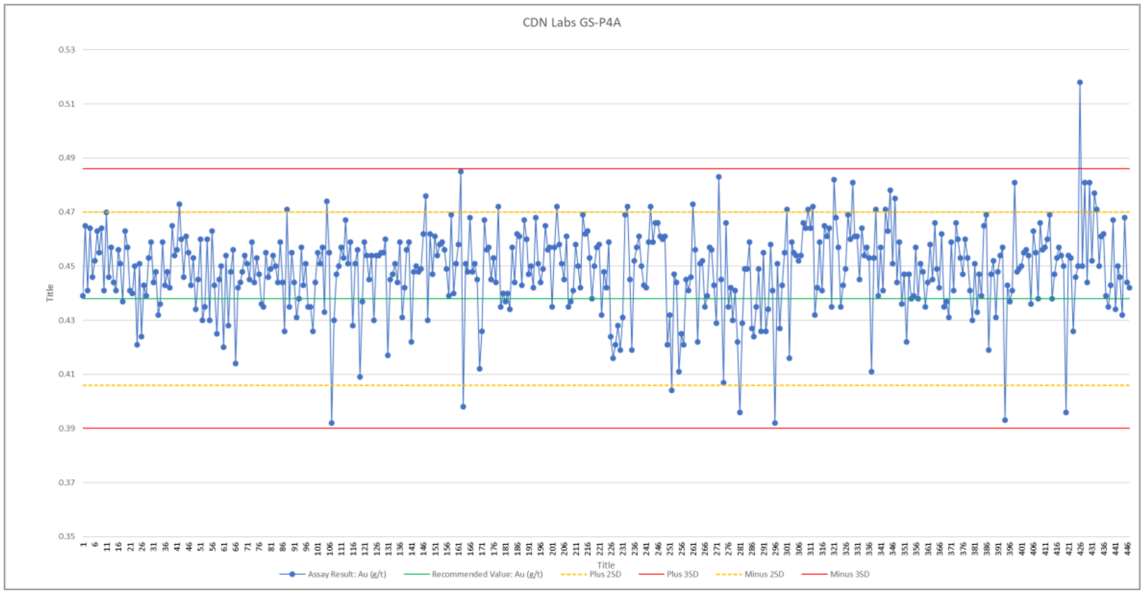
CRM	Expected Au value (ppm)	Stdv	Years used	Analytical lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
6Pc			2011	TSL				
OREAS 15d	1.559	0.042	2010 – 2014	Actlabs	200	33	24	12%
G311-8	1.57	0.08	2014 – 2017	ALS	196	1	0	0%
Si42	1.761	0.054	2009 – 2010	Accurassay, Actlabs	666	93	94	14%
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%
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 11.3.
Source: AMC, using data provided by New Gold.

Table 11.13 – Rainy River silver CRM results

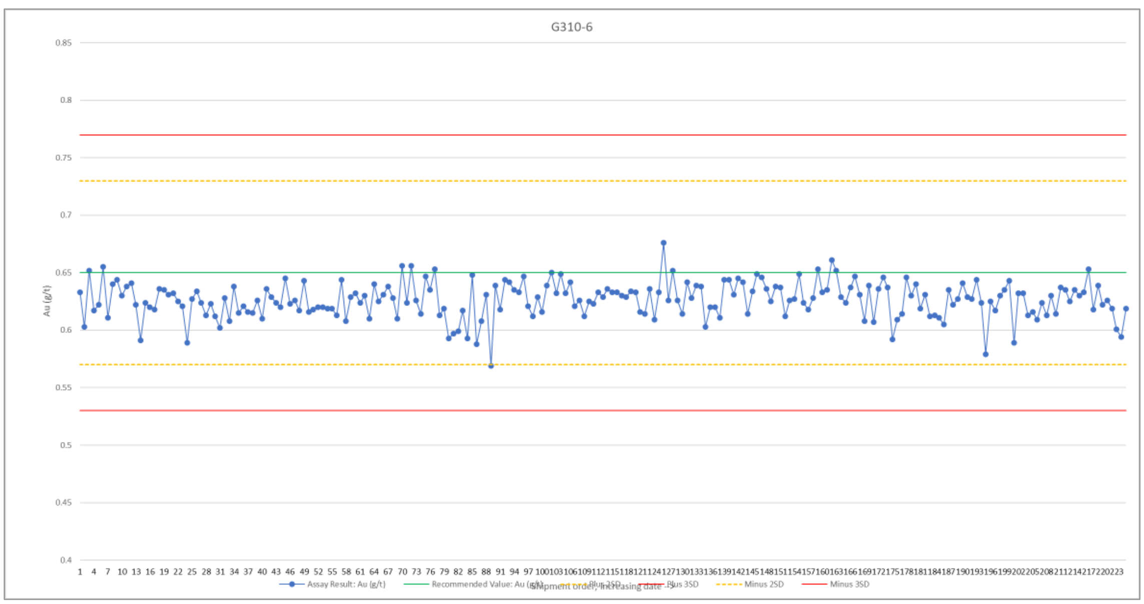
CRM	Expected Ag value (ppm)	Stdv	Years used	Analytical lab	Number of assays	Warning (>2 SD)	Fail (>3 SD)	Fail % (>3SD)
CDN-CM-26	2.5		2013 – 2014	ALS	134	0	0	0
GBM310-9	3.1	0.2	2014 – 2017	ALS	86	0	0	0%
OREAS H3	4.95	0.3	2010 – 2011	Actlabs	127	17	7	6%
OREAS 60b	4.96	0.31	2010 – 2011	Actlabs	77	7	8	10%
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.



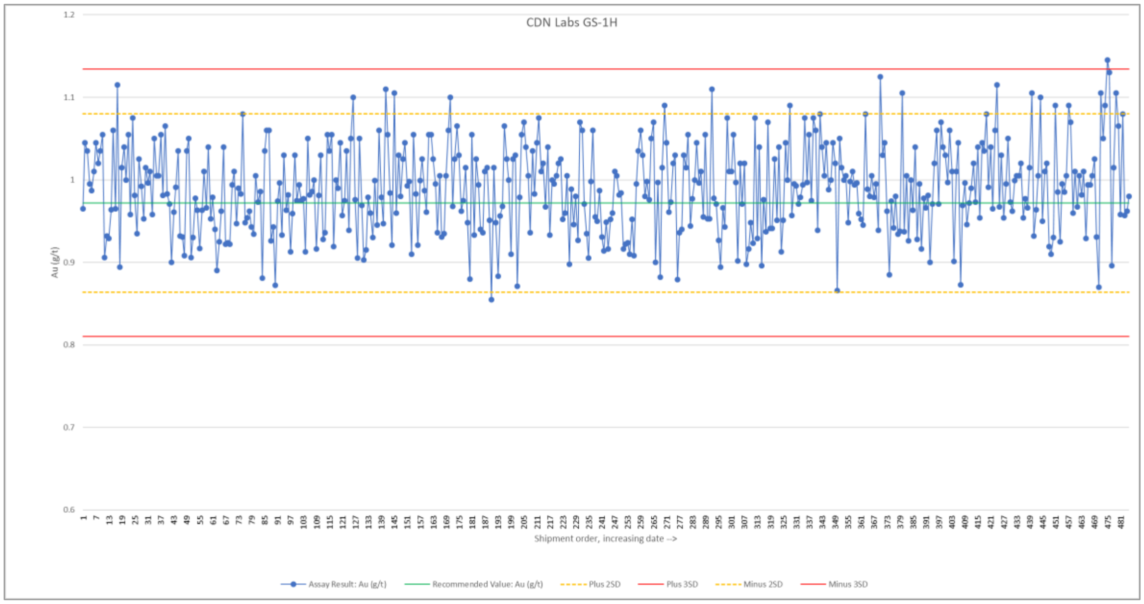
Note: All CRMs analyzed by fire assay with ICP-AAS at ALS Labs. AMC notes a positive bias with this CRM.
Source: AMC, using data provided by New Gold.

Figure 11.1 – Gold CRM CDN-GS-P4A



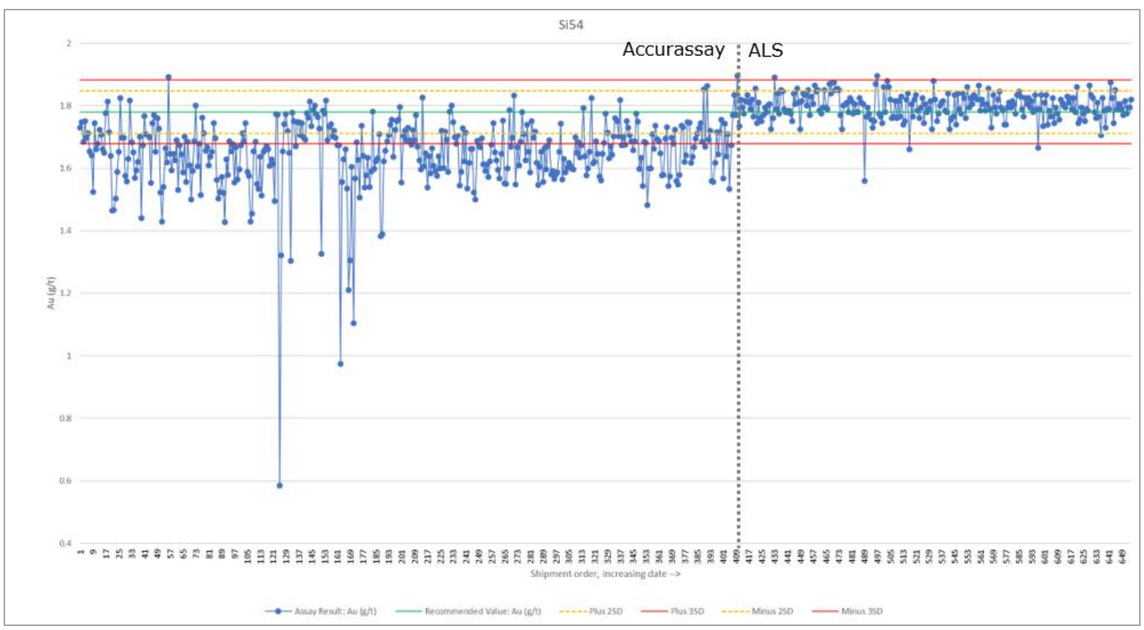
Note: All CRMs analyzed by fire assay with ICP-AAS at ALS Labs. AMC notes a negative bias with this CRM.
Source: AMC, using data provided by New Gold.

Figure 11.2 – Gold CRM G310-6



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

Figure 11.4 – Gold CRM Si54

The QP 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. Usually, 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 the QP 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.

It is noted that only 1% of the gold analyses are greater than 10 g/t gold.

11.7.1.3 Recommendations for CRMs

The QP recommends the following actions 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.

It is also recommended 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. This recommendation is 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 Discussion on blanks

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

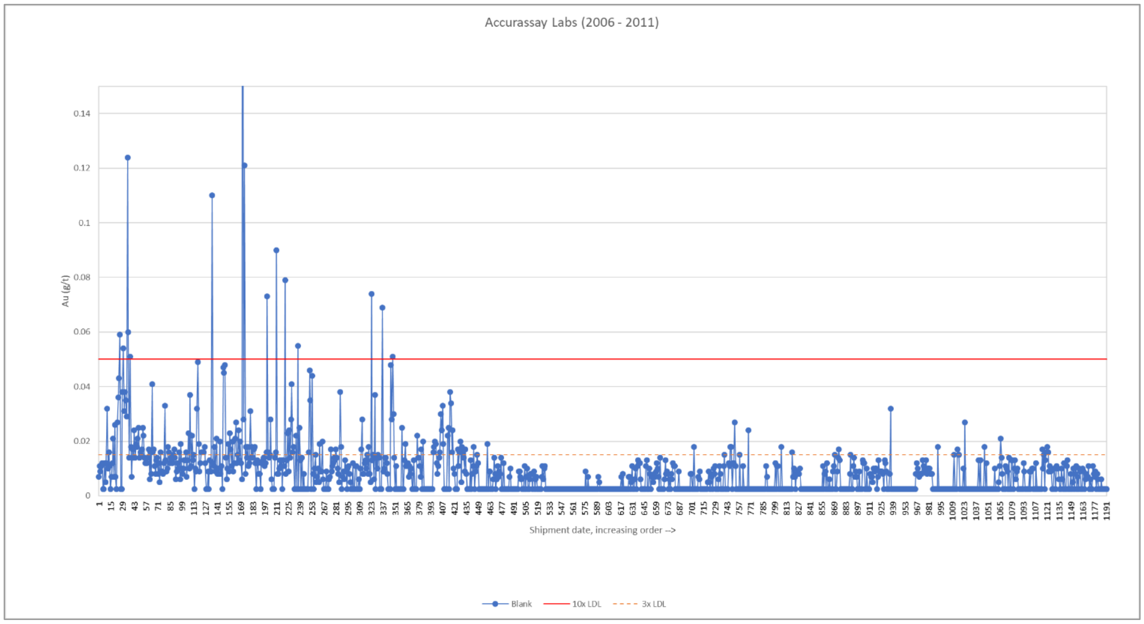
Table 11.14 shows the assay results from blank material for drilling completed between 2005 and 2017, and the results of both New Golds and the more stringent pass / fail parameters. Results from Accurassay and ALS are presented separately due to the differing performance of these labs during the period 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>

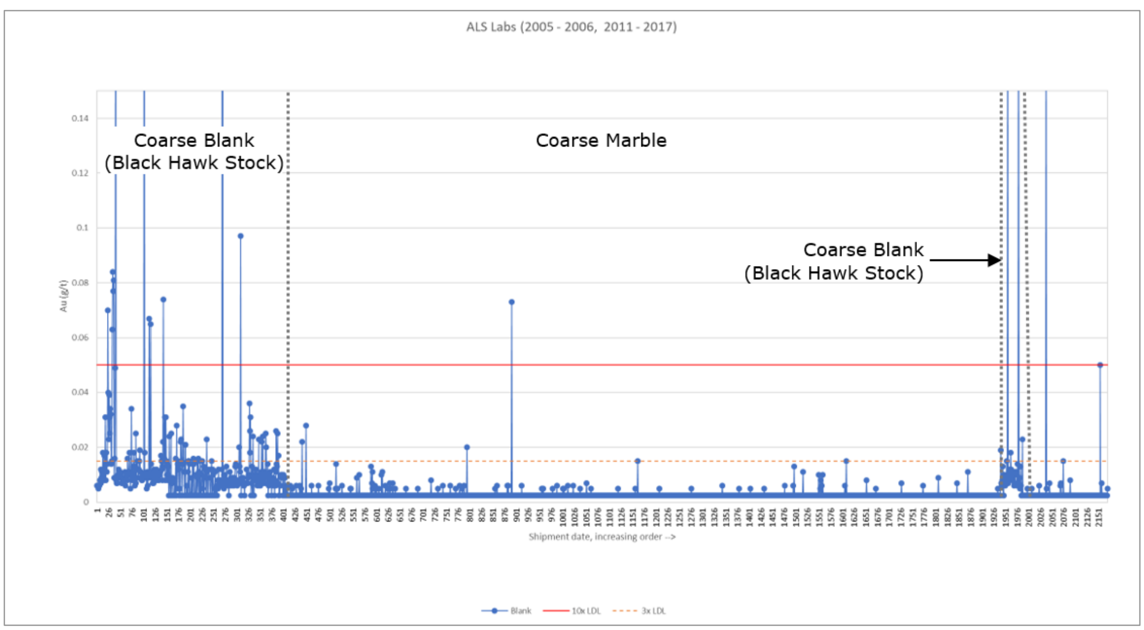
Notes: Year refers to year drilled. Blank samples run by Bayfield not included. Lower detection limit (LDL) is 0.005 ppm Au for fire assay analysis for all listed labs.
Source: AMC, using data provided by New Gold.

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.



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

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



Source: AMC, using data provided by New Gold.

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

11.7.2.3 Recommendations for blanks

The QP recommends the following actions 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 Discussion on duplicates

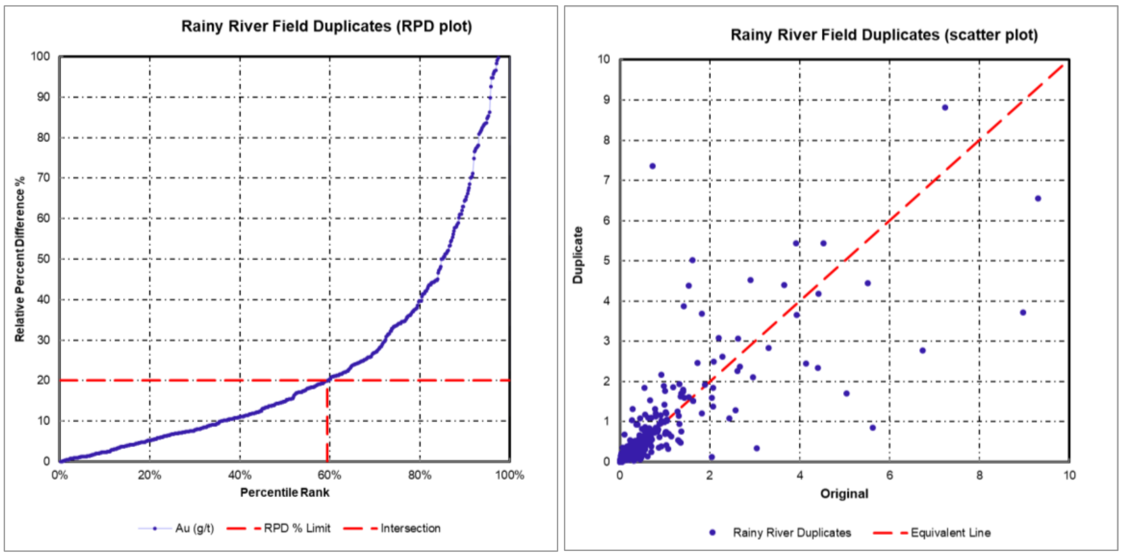
Field duplicates monitor sampling variance, sample preparation and analytical variance, and geological variance. Coarse duplicates monitor sample preparation, analytical variance and geological variance and pulp duplicates monitor analytical precision including homogenization and pulverization quality.

It is recommended that duplicate samples be selected over the entire range of grades seen at the Rainy River 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. The QP would typically assess 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 the QP's opinion that 80% pairs should be within 10% RPD (Stoker 2006). In these analyses, pairs with a mean of less than 15 times the lower limit of analytical detection have been excluded (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). The QP 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.

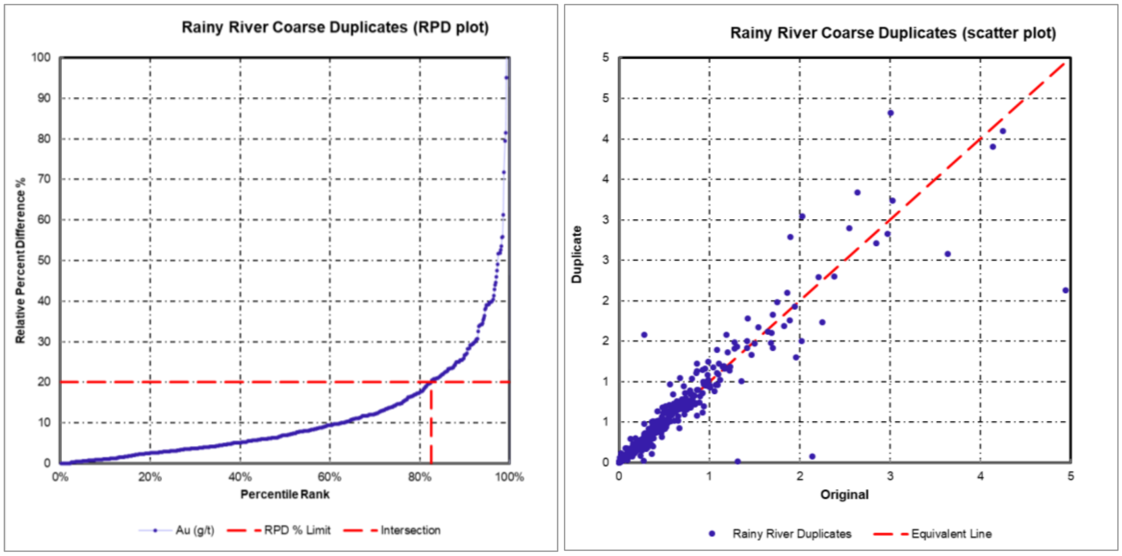
The proportion of duplicate samples with assay values within 20% RPD is less than desirable. This is most likely due to the combination of the heterogeneous nature of mineralization, as well as sampling variance.



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

Figure 11.7 – Rainy River field duplicate RPD and scatter plot

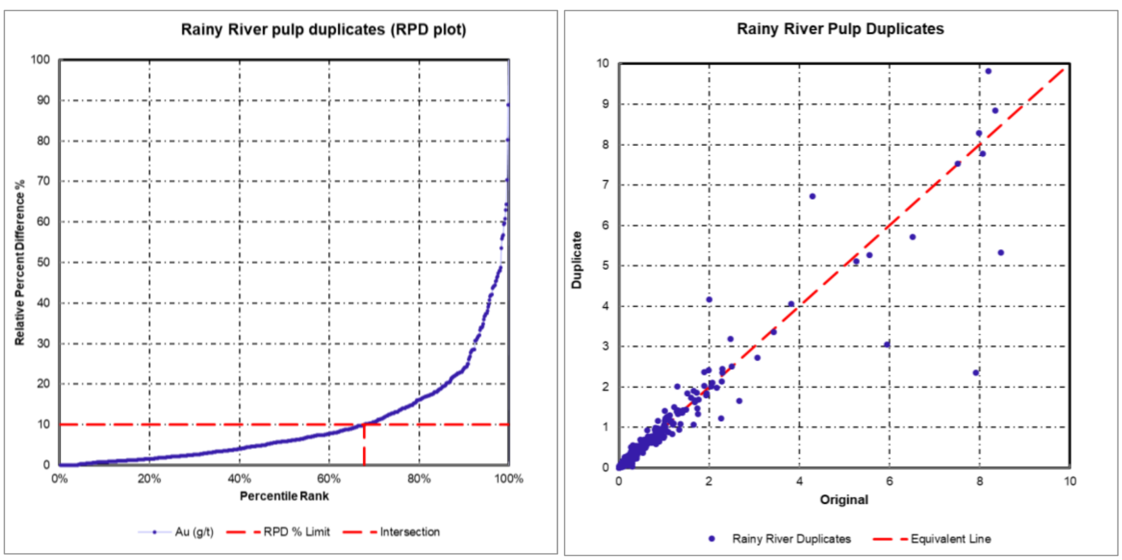
RPD and scatter plots for coarse duplicates are presented in Figure 11.8.. These plots show that ~82% of samples are within 20% RPD, with a negative bias towards the duplicate of ~12%. This higher bias is strongly skewed by two duplicate pairs that have an original high-grade analysis (> 50 ppm Au) paired 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.



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

Figure 11.8 – Rainy River coarse duplicate RPD and scatter plot

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.



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

Figure 11.9 – Rainy River pulp duplicate RPD and scatter plot

11.7.3.3 Recommendations for duplicates

The QP recommends the following actions for any future programs:

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

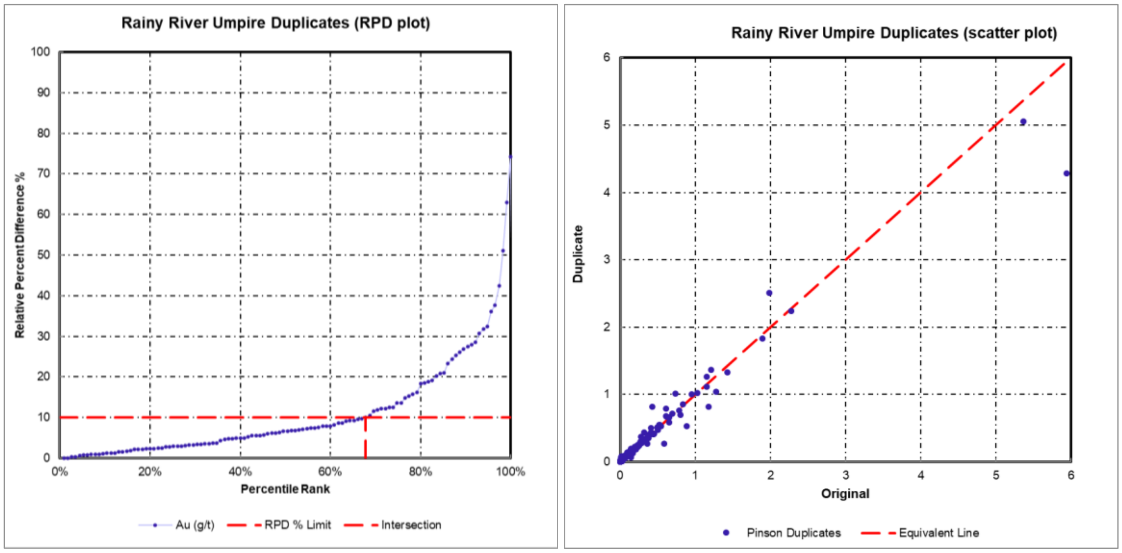
11.7.4.1 Description

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 Discussion on umpires

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 the QP'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 yields 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.



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

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

11.7.4.3 Recommendations for umpires

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. The QP has compiled and reviewed the available QA/QC data for this period.

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.

The QP 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), resource model, and DOM for gold, for the full year of 2021. In Table 12.2 the comparison is between DOM and the GC model; in Table 12.3 the comparison is between DOM and the resource model and in Table 12.4 the comparison is between GC model and the resource model.

Table 12.2 – Reconciliation for GC model to DOM

	Tonnes	Au g/t	Au ounces
DOM	14,471,532	0.70	325,341
GC model	15,500,389	0.74	368,933
DOM / GC	93%	95%	88%

In Table 12.2 the GC model tonnes are within 7% of the DOM tonnes and the DOM grade is 5% lower than the GC model, resulting in 12% fewer ounces of gold in DOM.

Table 12.3 – Reconciliation for resource model to DOM

	Tonnes	Au g/t	Au ounces
DOM	14,471,532	0.70	325,341
Resource model	15,686,155	0.80	403,185
DOM / resource model	92%	88%	81%

In Table 12.3 the resource model underestimates tonnes by 8% compared to DOM with the grades being lower by 12%, resulting in 19% fewer ounces of gold in DOM.

Table 12.4 – Reconciliation for GC model and resource model

	Tonnes	Au g/t	Au ounces
GC model	15,500,389	0.74	368,933
Resource model	15,686,155	0.80	403,185
GC / resource model	99%	93%	92%

The comparison between the GC model and the resource block model is shown in Table 12.4. This demonstrates that the two models estimate similar tonnes but with the grades being lower by 7%, there are 8% fewer ounces of gold in the GC model.

Note that the resource model used in the above comparisons is a regularized 10 m x 10 m x 10 m block model based on the underlying 2017 resource block model and thus includes some dilution.

All comparisons are at a 0.3 g/t AuEq cut-off as this is what site uses for grade control.

Reconciliation carried out by New Gold is detailed and thorough. It is carried out monthly and year to date figures are presented as tables. Table 12.4 indicates that the resource block model is functioning with acceptable reconciliation limits, albeit with a deficit of ounces of gold. It is recognized that the East Lobe of the orebody is driving the negative gold grade reconciliation and associated decrease in predicted contained ounces. New Gold has executed a reverse circulation drilling program in this zone during Q4-2021 to better predict the future production of this zone, with drill results expected in Q1-2022 which are to be included in an updated Mineral Resource block model during 2022.

In addition, Table 12.2 and Table 12.3 would tend to indicate that there are further losses during the mining process. New Gold is reviewing mining procedures to minimize dilution and ore loss during mining to improve these reconciliations. Positive reconciliation on the stockpile ore sent to the mill would seem to indicate possible ore loss to other mine polygons reporting to stockpile or possibly directly to mill. This is under investigation.

The QP 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 stockpile movements on a monthly basis in the mine.

In Item 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 drill core 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 Canada Inc. (SGS) 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 drill core and drill core 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	20.0
Z-433	12.0	
HS	1.0	
NZ	5.0	
ODM-1	35.1	80.0
ODM-2	3.5	
ODM-3	31.4	
ODM-4	9.9	
Master		100.0

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

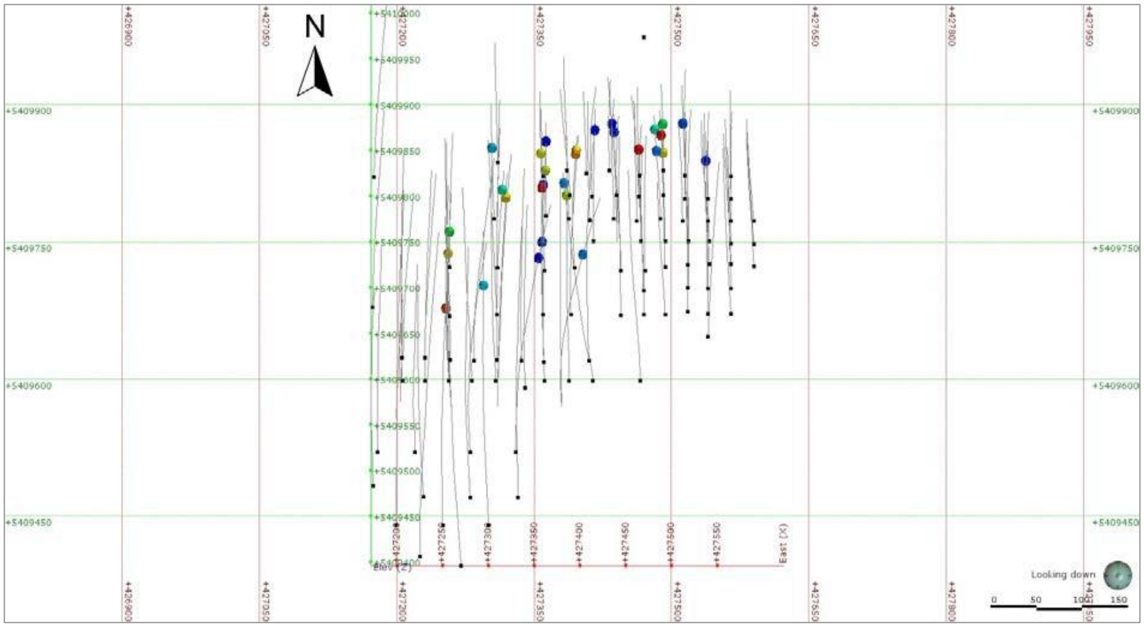
Note: Totals may not compute exactly due to rounding.

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

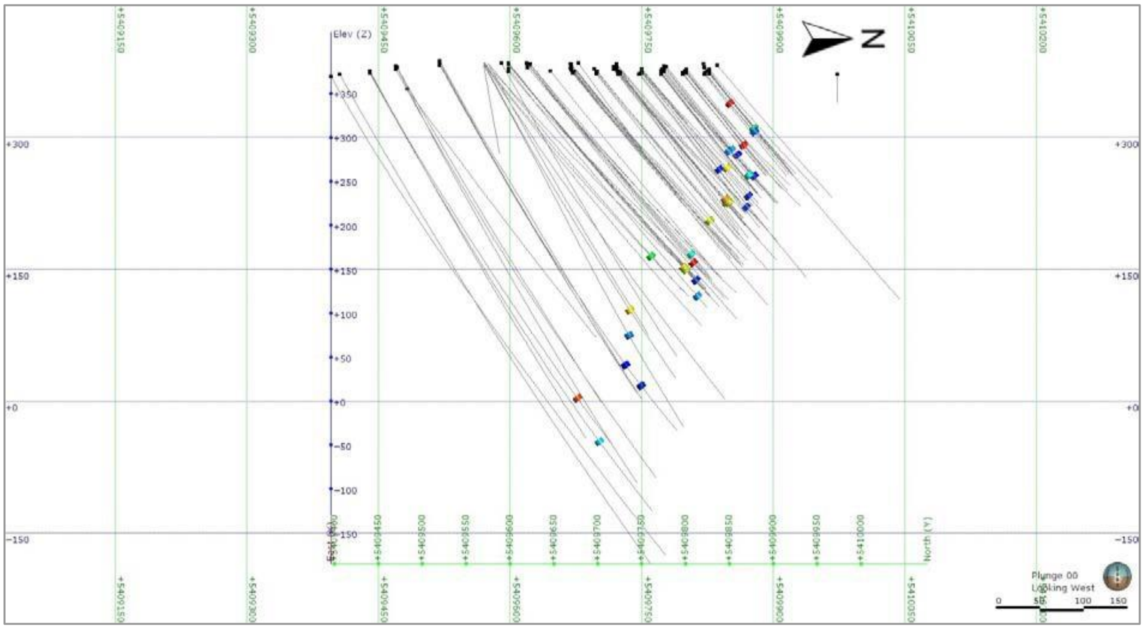
Sample variability testwork was performed, following flowsheet selection and development of the base test criteria. The variability testwork program included 162 comminution samples and 208 cyanide leaching samples from the Main Pit, and another 30 comminution and leaching samples from the Intrepid Zone. 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 Main Pit are presented in Figure 13.1 and Figure 13.2 in plan view and cross-section respectively.



Source: New Gold 2018.

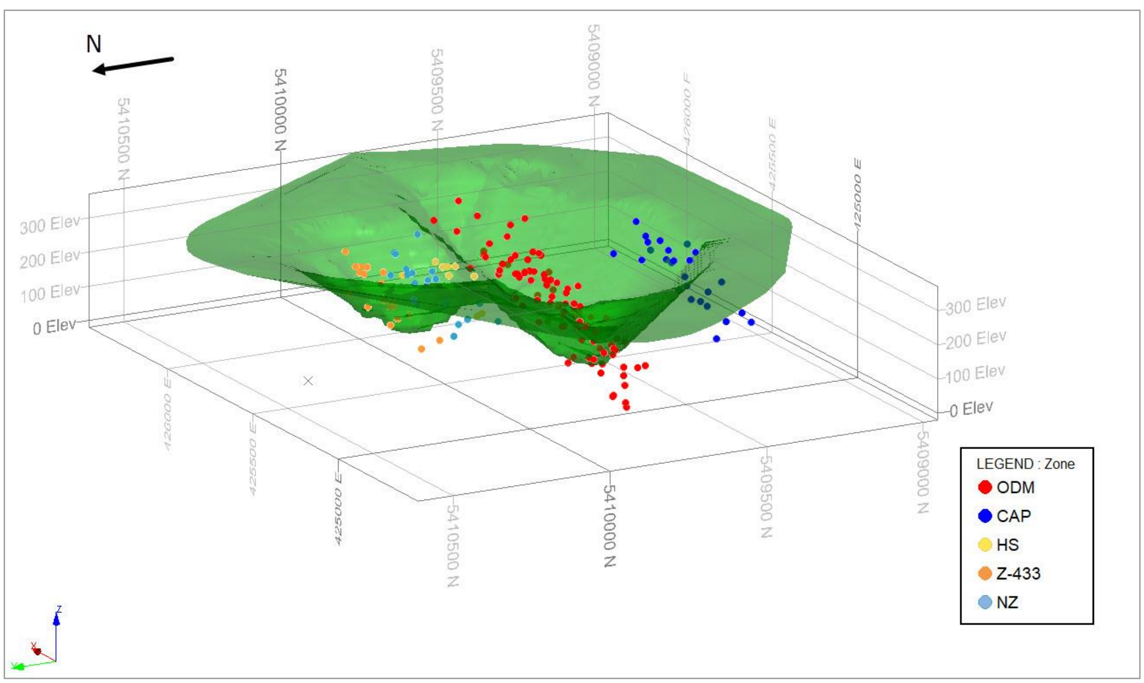
Figure 13.1 – Plan view of drillhole and sample locations in the Intrepid Zone



Source: New Gold 2018.

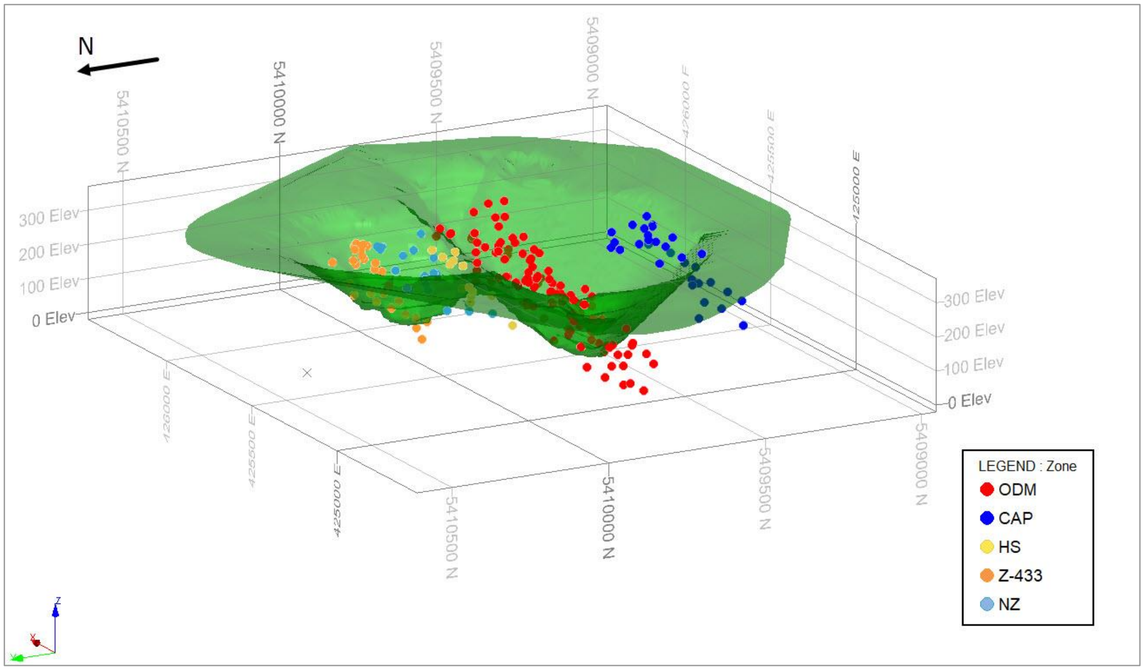
Figure 13.2 – Location of Intrepid Zone samples (cross-section looking west)

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



Source: New Gold 2020.

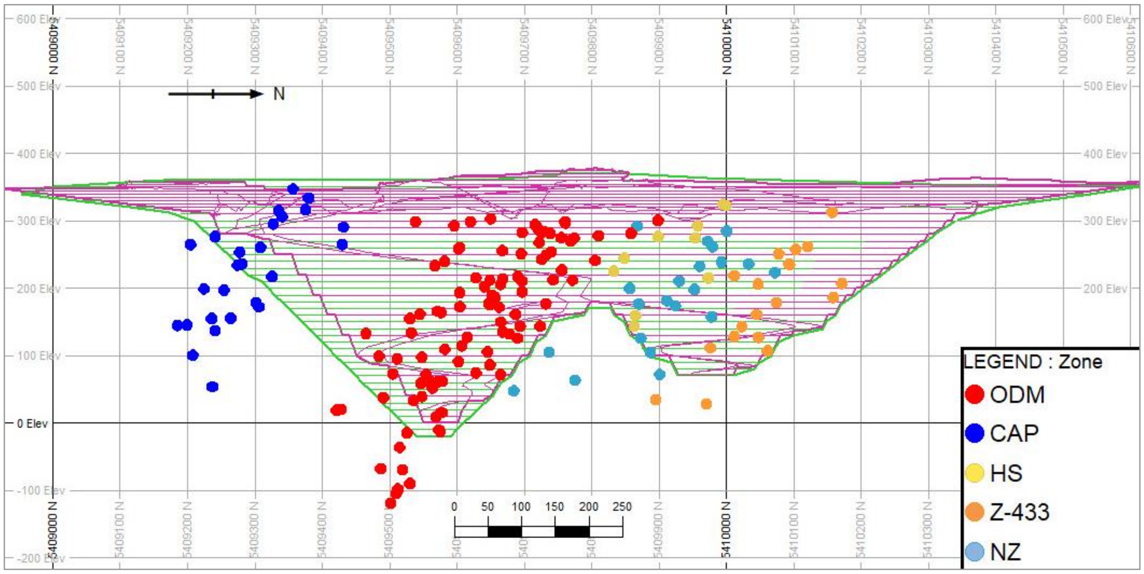
Figure 13.3 – Sample locations for cyanide leaching variability testwork



Source: New Gold 2020.

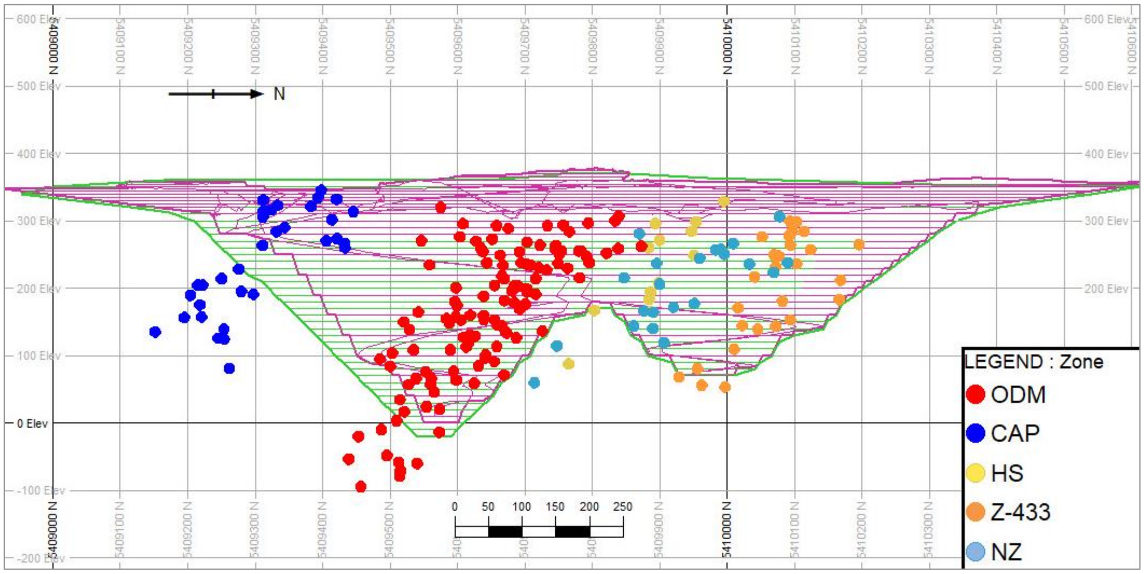
Figure 13.4 – Sample locations for comminution variability testwork

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.



Source: New Gold 2020.

Figure 13.5 – Sample locations for variability comminution 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.

Figure 13.6 – Sample locations for variability leaching testwork

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 sulfur and iron than the other zones. The Intrepid Zone has much higher silver levels than the other zones, however, the copper, iron, sulfur, 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.

13.1.6 Mineralogy

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

- Moderately to strongly deformed, auriferous sulfide and quartz-sulfide 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 sulfide 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 sulfide and quartz-sulfide 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, auricupride, 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 sulfides 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 sulfur 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 crushing work index 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 (CWi) 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 the majority 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 74 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 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 10^{th} percentile and 90^{th} 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
80^{th} 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 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®

Zone	A	b	JK DW tests A x b	ta	P (g/cm^3)	A	SMC 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 effect 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 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 P80 (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

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 Z433 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.

The gold and silver recoveries as a function of head grade are presented in Figure 13.7 and Figure 13.8.

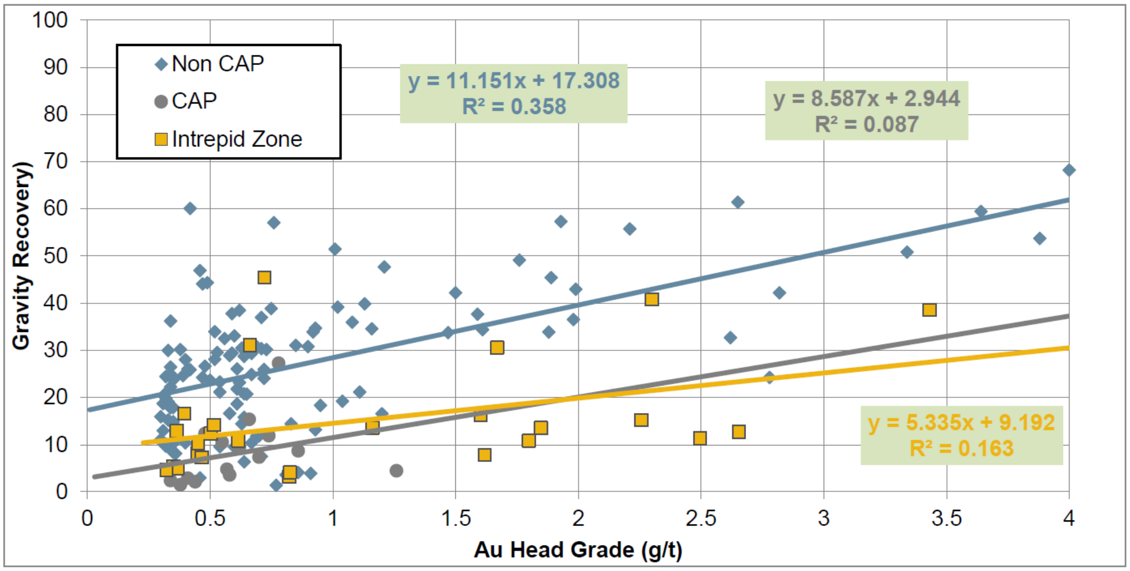


Figure 13.7 – Gold Gravity Recovery vs. Head Grade

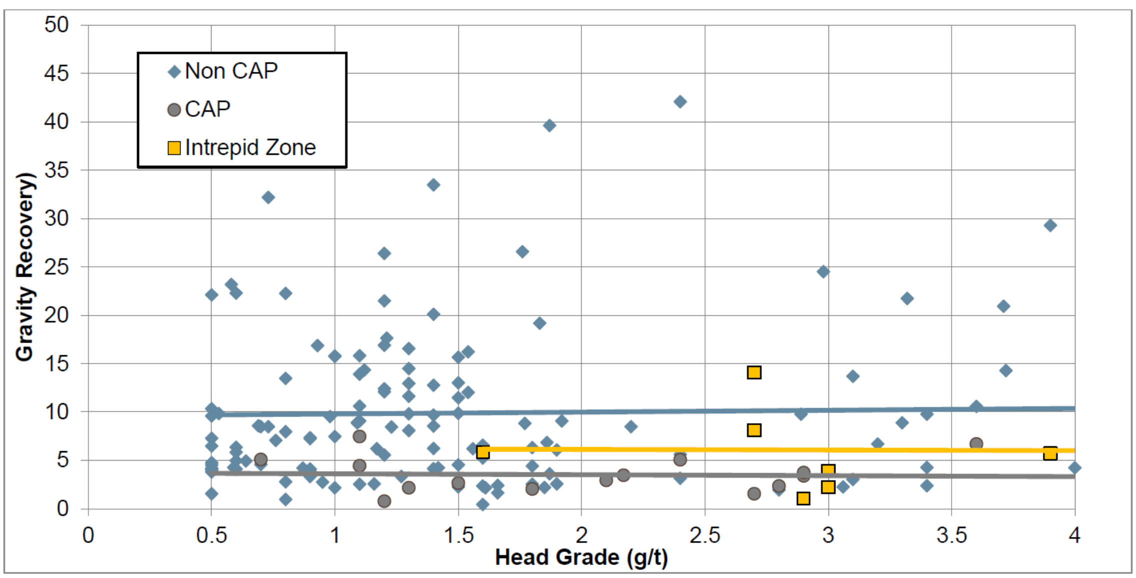


Figure 13.8 – Solver Gravity Recovery vs. Head Grade

It can be seen that the gravity recovery of gold is sensitive to the head grade, with higher grades giving higher recoveries. The same trend was noted for the Non-CAP, CAP and Intrepid Zones; however, the CAP and Intrepid Zone gold recovery were lower than the Non-CAP Zones. No trend was noted for the silver and it was assumed that silver gravity recovery is independent from the head grade. The silver gravity recoveries for the CAP and Intrepid Zones were lower than the non-CAP Zones, analogous to the gold gravity recovery. The average silver gravity recovery for the CAP Zone was roughly 3%; the

Intrepid Zone averaged 5% while the non-CAP Zones silver gravity recovery was approximately 10%.

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 liter (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 13.9 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.

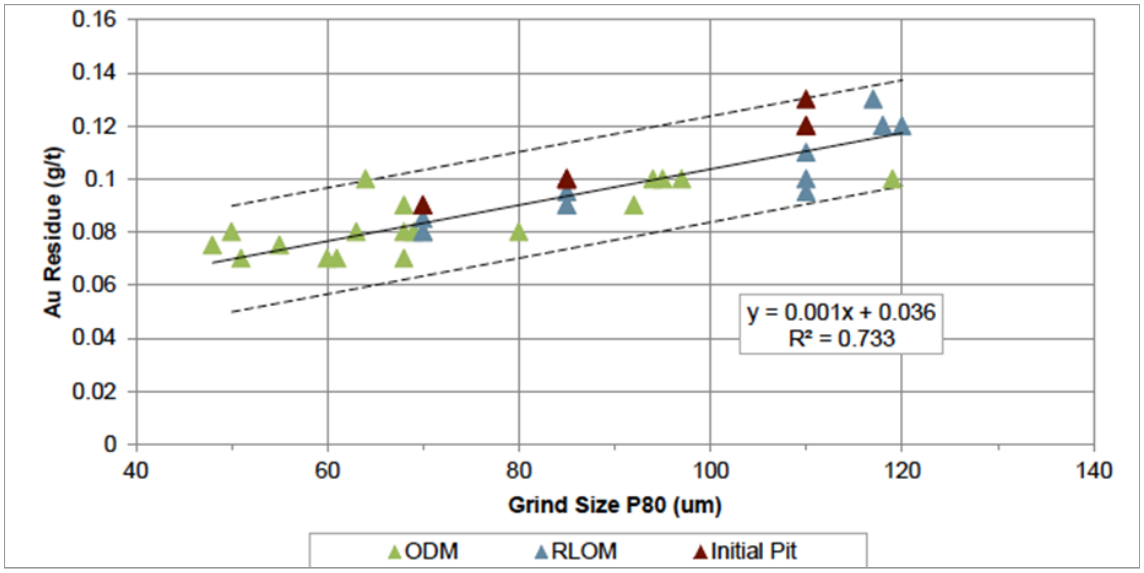


Figure 13.9 – Gravity tailings leach residue gold grade versus grind size

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 13.9. 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 13.10.

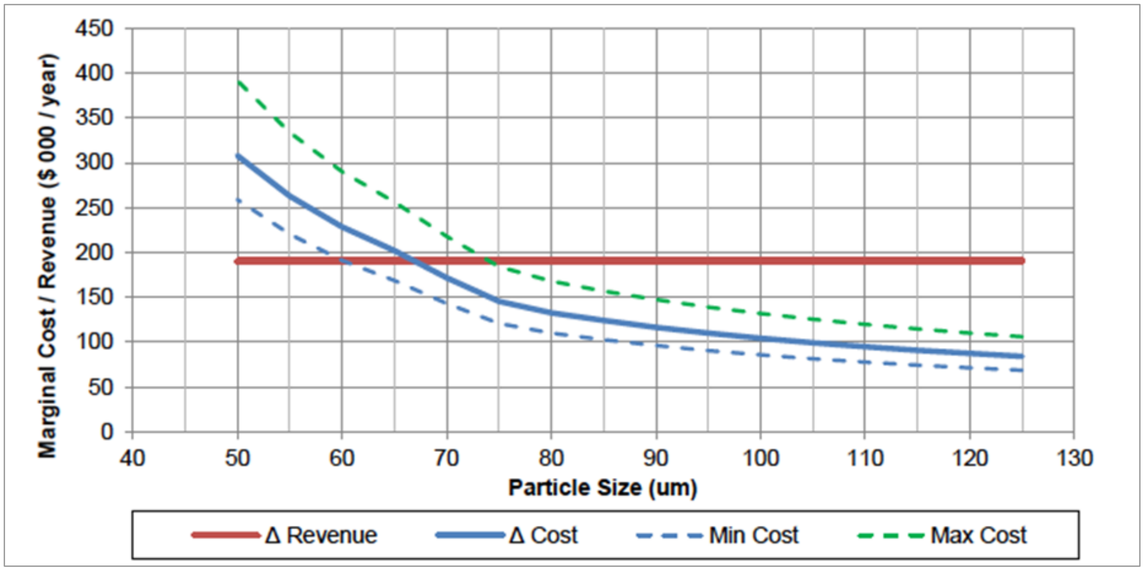


Figure 13.10 – Cost and revenue analysis by grind size

The results show that for the average costs of the listed parameters, grinding to 65 µm is still economic, however, when using the higher costs, it is only economic 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 13.11 shows cyanide leach gold recovery vs time for these tests.

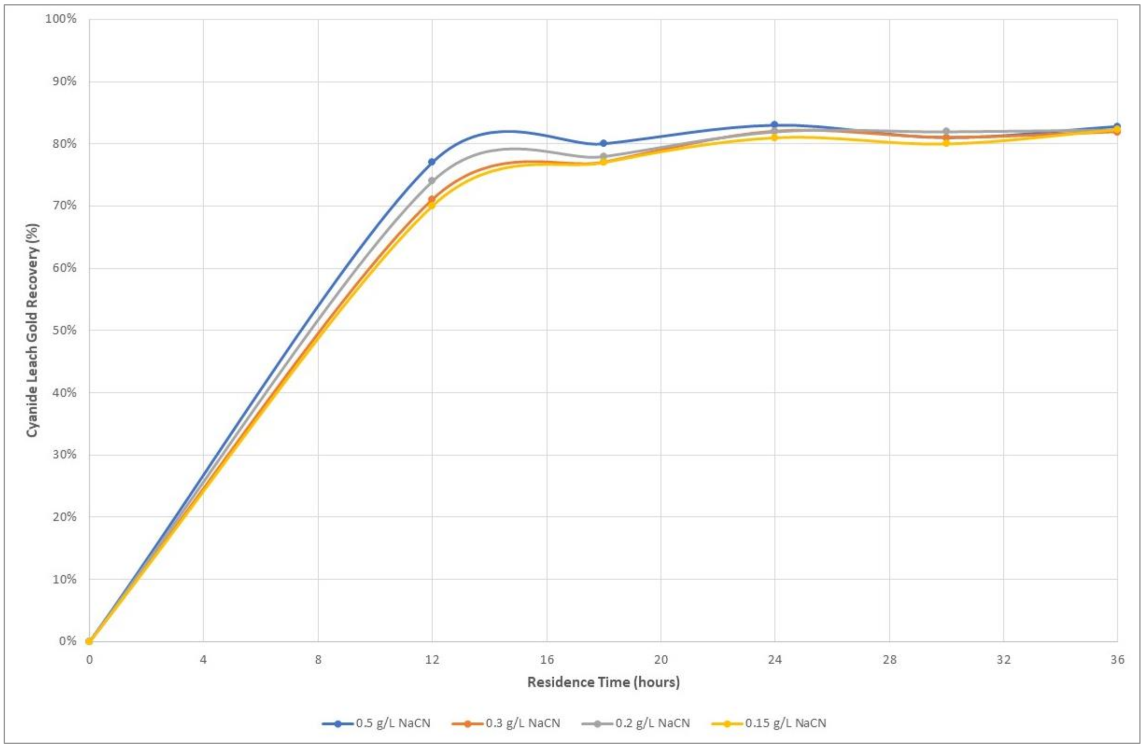


Figure 13.11 – Impact of gold recovery by NaCN concentration

Figure 13.11 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]				Gravity [2]	Gravity + cyanide leach [2]	Residue grade	Head grade
						12h	18h	30h	36h				
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 13.12 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.

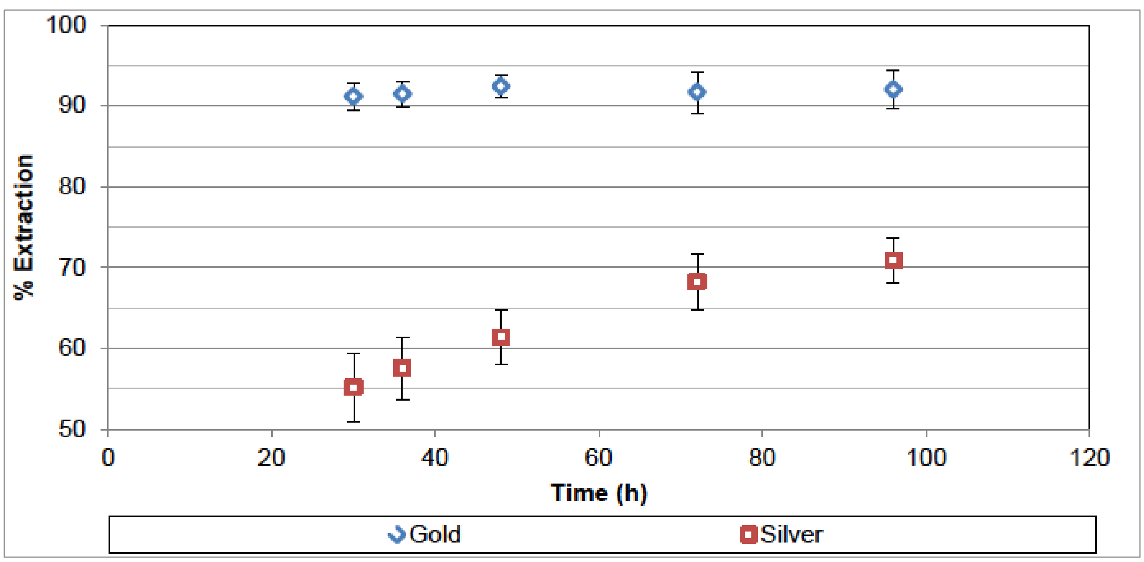


Figure 13.12 – Boxplot of Intrepid Zone gold and silver cyanide leaching kinetics

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

Due to the lower gold recovery observed in the CAP Zone samples, and a small percentage of the non-CAP Zones samples, 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 (HCl) leach followed by intensive cyanidation leach: Extraction of gold that is associated with pyrrhotite, calcite, ferrites, etc. This is done by leaching the tailings using hydrochloric acid to dissolve the pyrrhotite and other minerals, then performing the intensive cyanide leach to extract the liberated gold.
- Aqua regia (AR) leach: Extraction of gold associated with or encapsulated in sulfide minerals such as pyrite and arsenopyrite.
- The final residue from these tests is considered to be locked in silicates or associated with fine sulfides that are locked in silicates.

The gold deportments from these tests are shown in Figure 13.13.

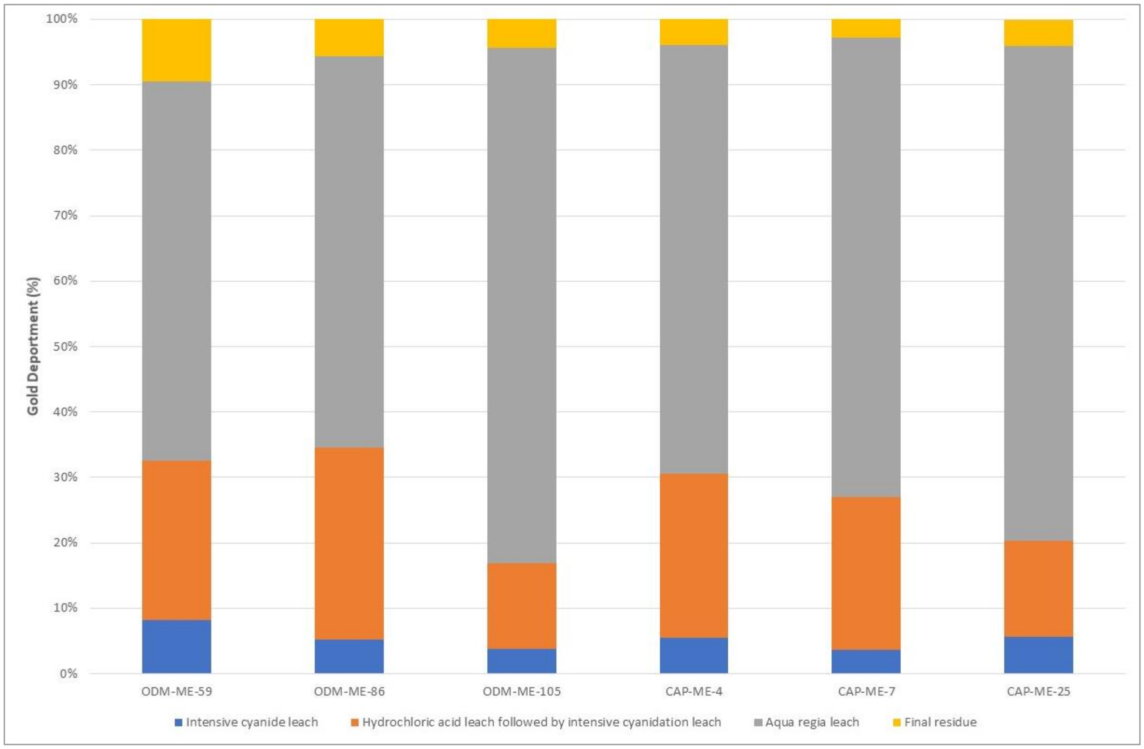


Figure 13.13 – Diagnostic leach test gold deportments on cyanide leach tails samples

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

- The amount of the residual gold recovered by the AR 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.

- AR leach: Extraction of gold associated with or encapsulated in sulfide 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 sulfides that are locked in silicates.

The gold deportments from these tests are shown in Figure 13.14.

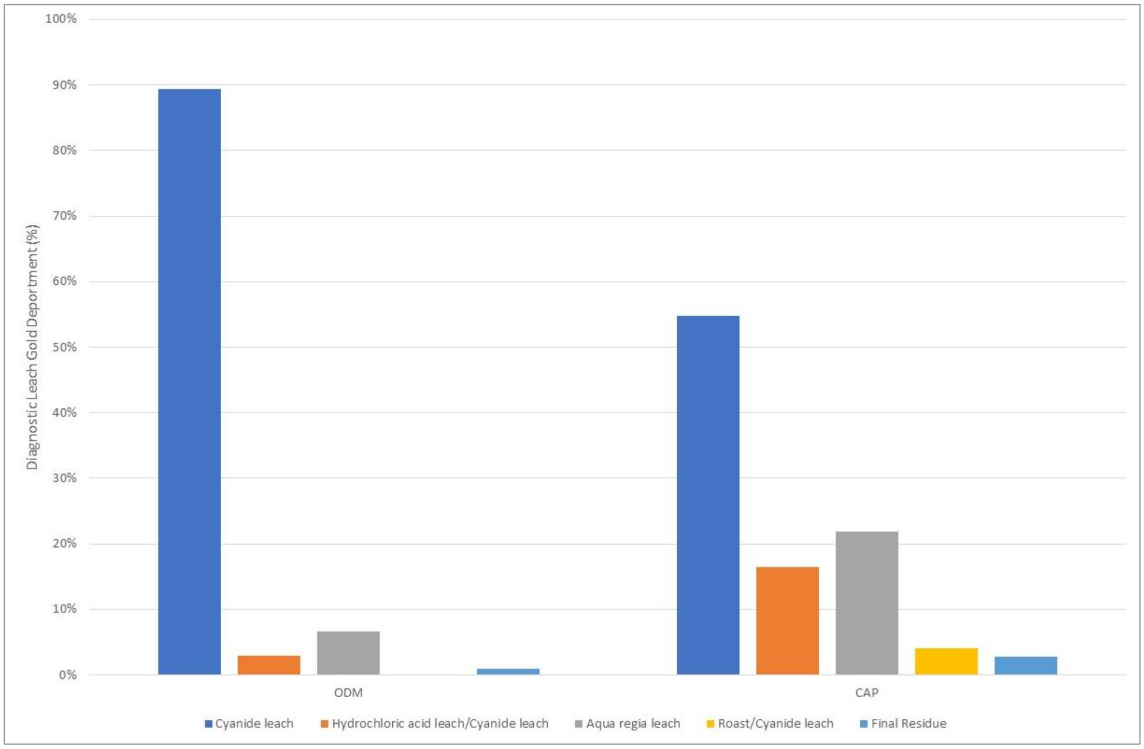


Figure 13.14 – Diagnostic leach test gold deportments on ore samples

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 sulfides (AR 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 sulfides (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_T (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 (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 13.15.

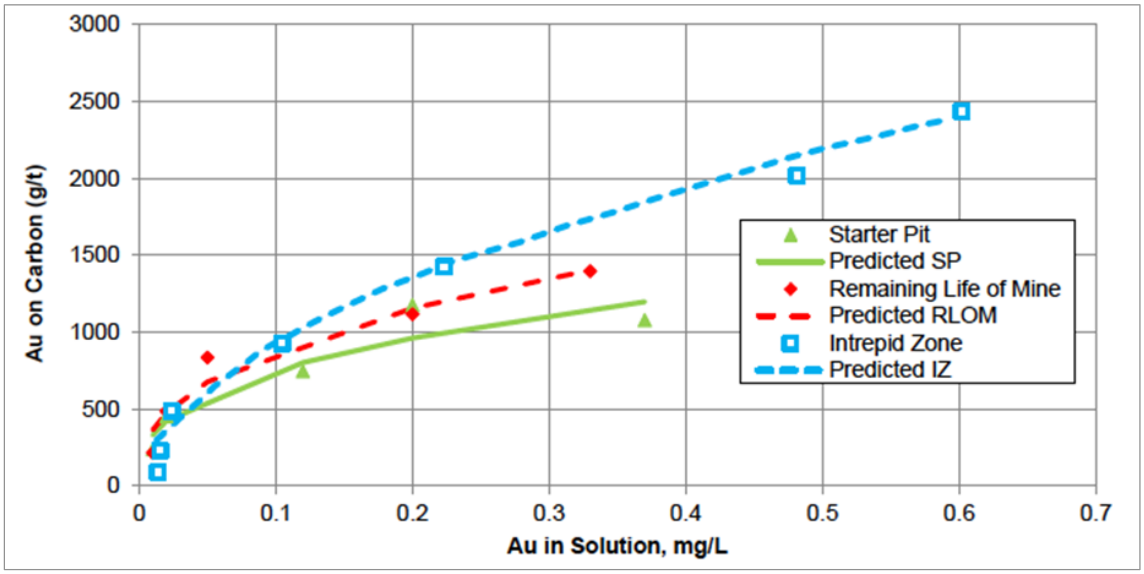


Figure 13.15 – CIP isotherms used for modelling

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 liter (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		meters	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.

Orway Mineral Consultants (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 13.16 shows carbon activity vs time for these tests.

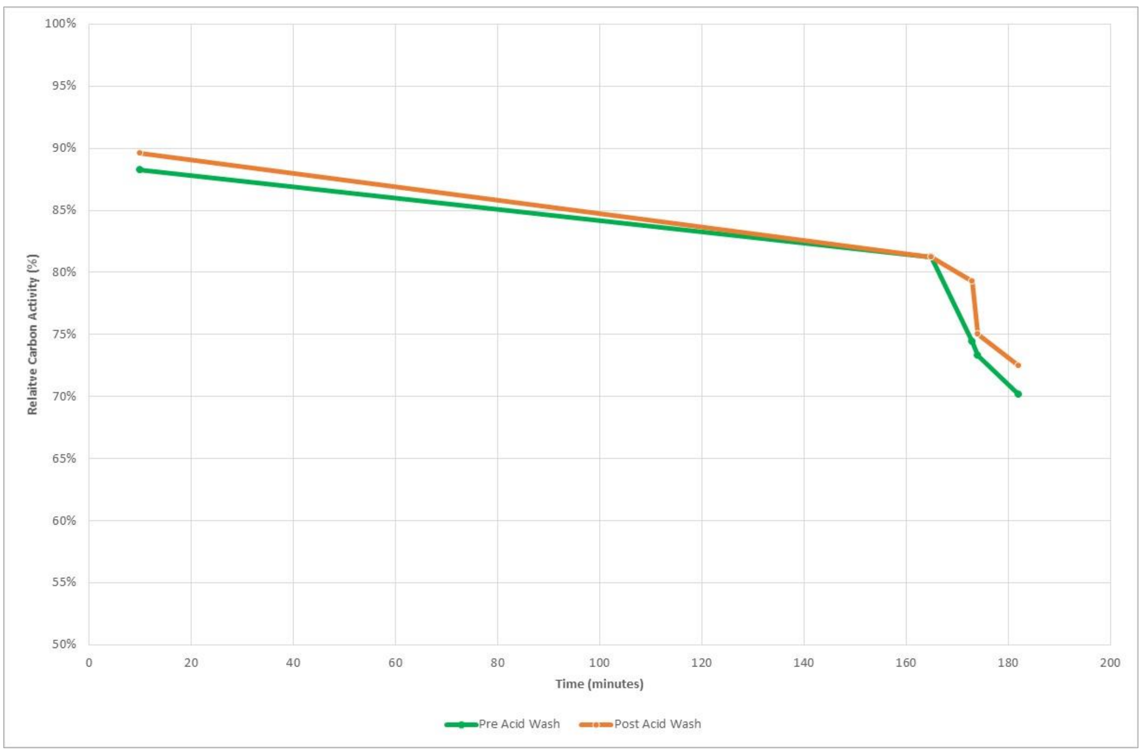


Figure 13.16 – Carbon activity vs time for acid wash tests

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 Magnafoc 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 meters 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.

Test work was conducted on the flocculant mixing system in 2019 concluded that the mixing system that was in place was not adequately mixing the dry polymer resulting in polymer waste and over consumption. To rectify the problem a new Polymer Slicing Unit (PSU) was installed in Q4 2020 and commissioned in Jan 2021.

A 0.017 kg/t and 0.013 kg/t flocculant consumption were observed with the previous polymer mixing unit and new PSU, respectively. The difference (0.004 kg/t) equates to a 21% reduction in flocculant consumption. The overall flocculant consumption has reduced from 0.050 kg/t in 2019 to 0.014 kg/t in 2021 (73%) with the combination of flocculant optimization, advanced process control (APC) systems, proper polymer mixing and effective use of the polymer.

13.2.4 Leach optimization testwork

SGS completed a leach optimization testwork in Q1 2021. The objective of the leach optimization testwork was to determine the response of the four new ore zones and evaluate how they would respond to the operating leach parameters back in Q4 2020 to Q1 2021. The five samples new ore zones included 433 high grade ore (433 HGO), low grade ore (LGO), HS medium grade ore (MGO HS), and ODM high grade ore (HGO ODM).

Gold and silver assays for the five samples are summarized in Table 13.21.

Table 13.21 – Results of supplier sedimentation testwork

Element	Unit	Jan 2021 - Variability Samples*				Plant Sample (Leach Feed)
		433 MGO	LGO	MGO HS	HGO ODM	Sep-20 ^
Au Cut A	g/t	0.46	0.27	0.66	1.03	0.93
Au Cut B	g/t	0.41	0.29	0.87	1.91	0.91
Au Avg.	g/t	0.44	0.28	0.77	1.47	0.92
Au Calc.	g/t	0.61	0.31	1.02	1.13	---
Ag	g/t	<0.5	0.9	0.6	8.6	4.3

* Samples used for whole ore cyanidation tests
^ Sample used for CIP modelling testwork
Au Calc = average calculated head from cyanidation tests

In total, 32 leach optimization tests were completed using the four variability samples. As noted above, the objective of the tests was to determine the response of the samples when applying the standard Rainy River leach parameters, and to also evaluate select conditions, which included:

- Grind Size
- Cyanide concentration
- Lead nitrate addition

All of the optimization tests were completed using standard bottle rolls. The baseline test conditions, which were based on the operation back in Q4 2020 to Q1 2021, were as follows:

- Grind size – varied
- Pulp Density – 59% solids (w/w)
- Cyanide Concentration – varied (maintained for 4 hours and then decayed until the end of test)
- Leach Retention Time – 24 hours (with subsamples as indicated)
- Pulp pH – 10.5-10.7 target (maintained with lime)
- Dissolved oxygen concentration – 5-8 mg/L (air sparged into bottles)
- Lead nitrate – 150 g/t (4 tests only, Set #3)

The results from the leach optimization study illustrated that the 433 MGO and MGO HS samples responded the best when applying the Rainy River leach conditions back in Q4 2020 to Q1 2022. Gold extractions were ~88-90%. The gold extractions for the LGO and HGO ODM samples were ~80%. The gold extractions did not improve when grinding finer from ~105 µm to ~90 µm or ~75 µm. The gold leach kinetic results at the various grind sizes are presented in Figure 13.17 to Figure 13.20 and indicated that the full 24 hours of leaching was needed to maximize gold recovery for each sample.

Increasing the cyanide addition by 50-100% also did not improve gold extraction for the samples. The addition of lead nitrate slightly improved the leach kinetics, but ultimately the final gold extractions were the same.

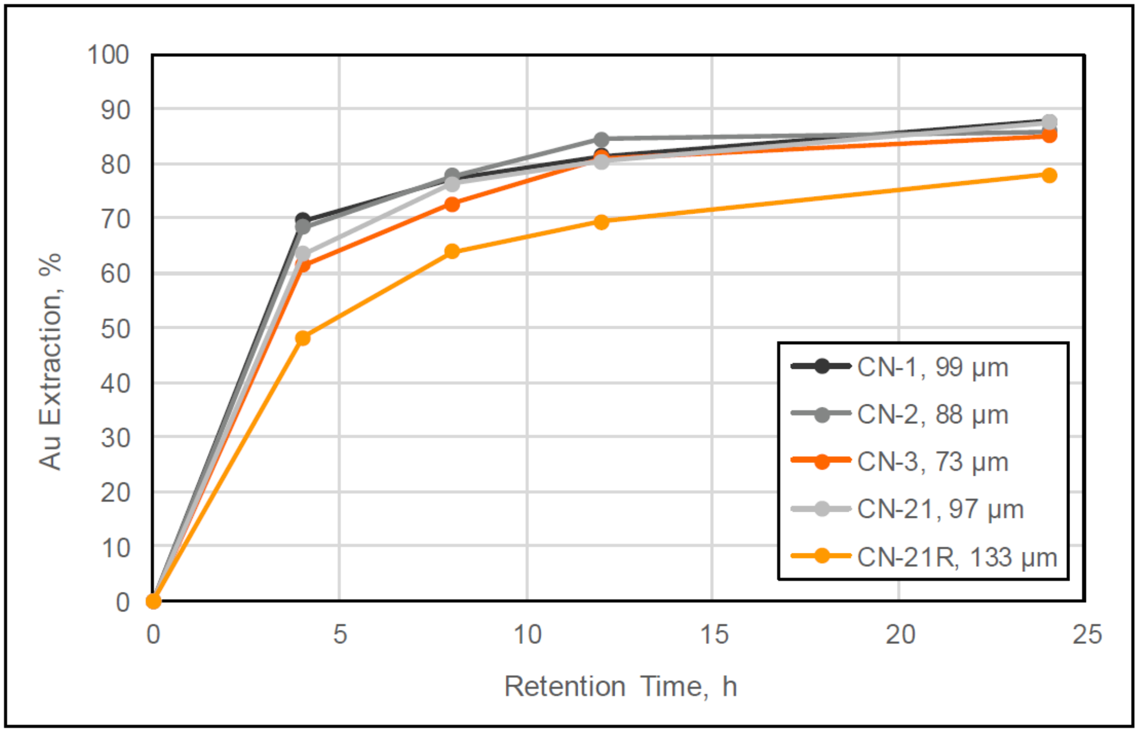


Figure 13.17 – Gold Leach Kinetic (Effect of Grind Tests) – 433 MGO

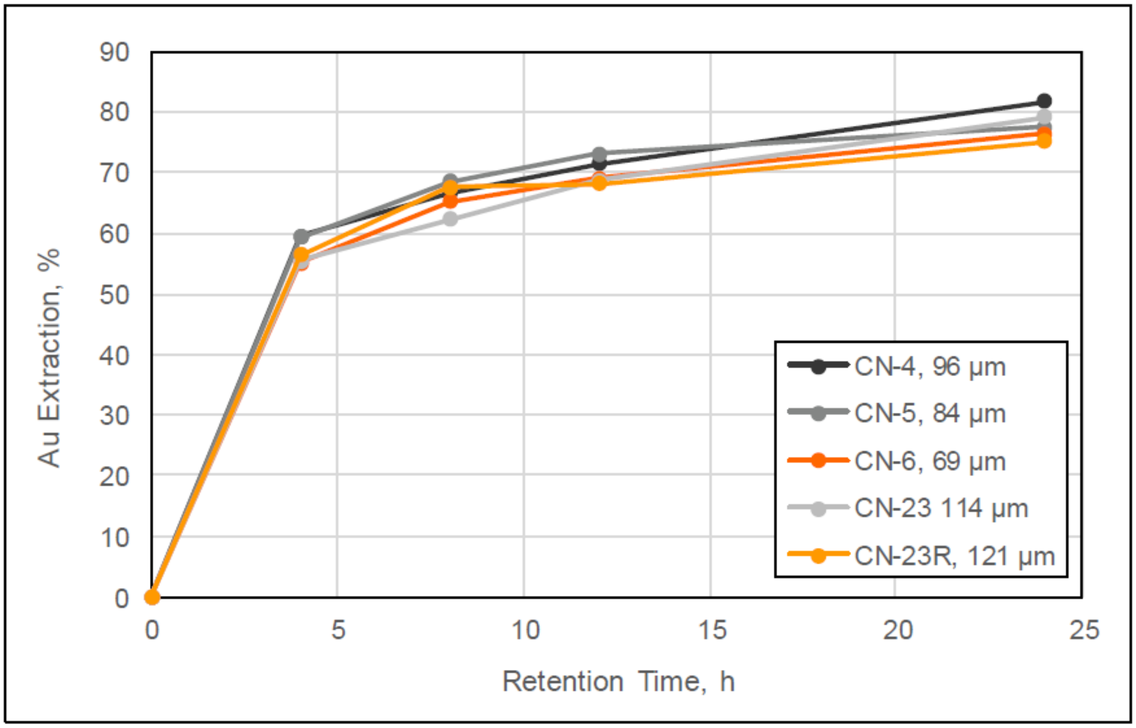


Figure 13.18 – Gold Leach Kinetic (Effect of Grind Tests) – LGO

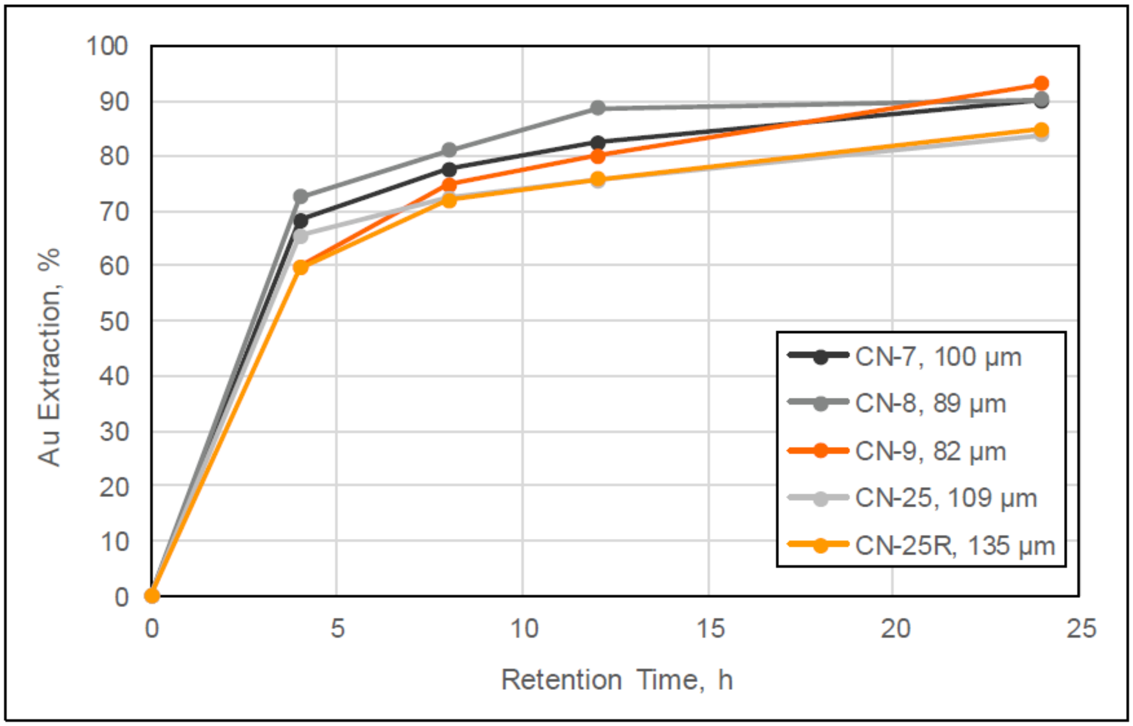


Figure 13.19 – Gold Leach Kinetic (Effect of Grind Tests) – MGO HS

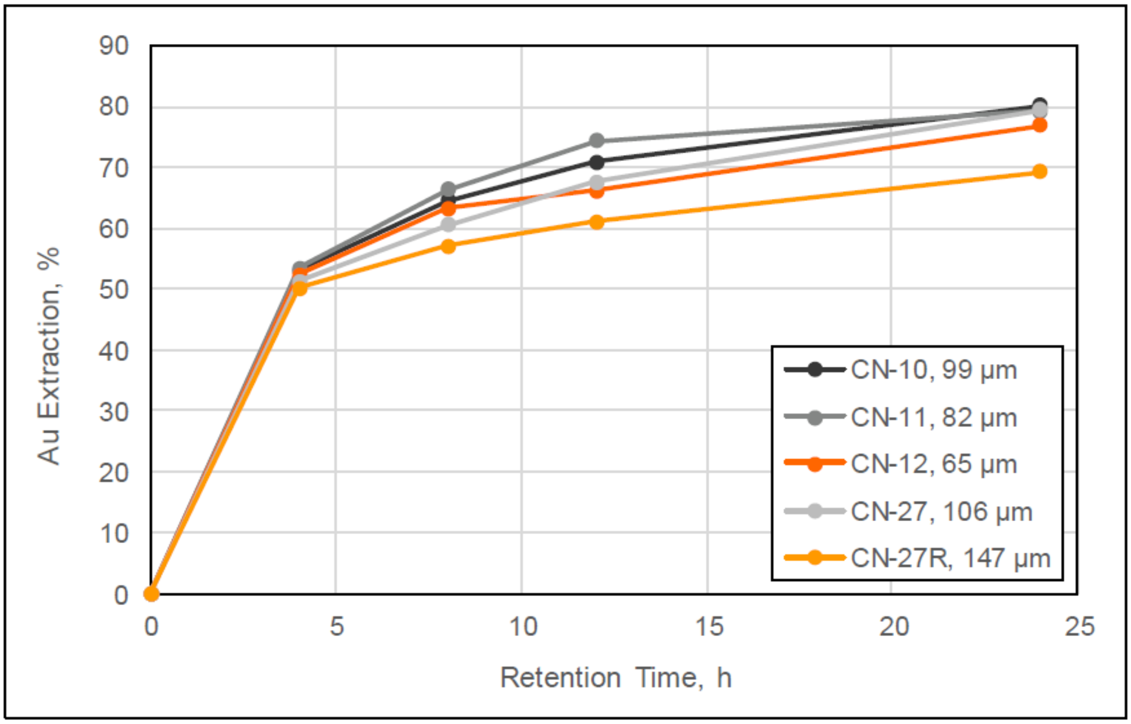


Figure 13.20 – Gold Leach Kinetic (Effect of Grind Tests) – HGO ODM

13.2.5 CIP modeling testwork

The rate of adsorption of gold cyanide on activated carbon is very slow by normal industrial standards, and the design criteria and operating philosophy for all CIP plants are such that the amount of gold loaded on to the carbon in each stage is always far less than equilibrium loading. Thus, the gold extraction efficiency in CIP plants is always based on the kinetics of adsorption and is not limited by equilibrium constraints.

Improving the kinetics of adsorption means more gold will load on the carbon in a given time, and this in turn translates to lower rates of carbon transfer to elution and regeneration, and lower operating and capital costs. However, the loading of gold cyanide on carbon is a reversible process, and the rate of loading decreases as the concentration of gold on the carbon increases. This means less gold is extracted from solution and soluble gold losses from the last adsorption stage increase with increasing gold on the carbon. But this trend can be countered by increasing the carbon concentration in each stage (because the carbon is not loaded to equilibrium) or by increasing the total number of adsorption stages. Arriving at the optimum design criteria for a particular plant is therefore an iterative process, which is best handled by models that describe the leaching of gold from a particular feed and the rate at which that gold loads onto activated carbon.

SGS completed a CIP modeling study in Q1 2021. The objective of the CIP modelling study was to assess the then operating parameters and to establish a model that could be used to optimize and evaluate the Rainy River CIP circuit. The model was generated using a plant sample and plant carbon and was calibrated to the plant data collected.

The CIP modelling results were very good, and the leach feed sample that was tested yielded results comparable to the plant data collected. Low barren solution losses (<0.01 mg/L Au) were achieved in basically all the scenarios tested. Modelling of different CIP operating strategies showed that increasing the carbon concentration or increasing the carbon advance rate (compared to plant practice back in Q1 2021) could lower soluble gold losses by up to 0.005 mg/L. The present plant design of 24 hours leaching prior to CIP is likely optimum. Overall, the Rainy River CIP circuit is operating well, and every effort should be made to continue to achieve low gold concentrations on the eluted carbon.

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. 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.

A new gold recovery formula for Non-CAP Zone was developed in October 2020. A multi-linear regression has been utilized to better represent gold recovery.

CAP Zone:

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

Non-CAP Zone:

- AuTG = 0.36349 + (AuHG) * 0.06667 + P_{80} * 0.00025 + (%ODM) * -0.34414 + (%433) * -0.38227 + (%HS) * -0.35209
- Au Rec = [(AuHG – AuTG)/ 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:

- AuTG is the process plant gold tailings grade in g/t
- 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 P80 will be coarser

- %ODM, %433 and %HS are all fraction of ore by tonnage

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.

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 Zone was carried out 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 Definition Standards for Mineral Resources and Mineral Reserves dated 10 May 2014 (CIM Definition Standards (2014)).

A summary of the timing, authorship, and responsibility of the current Mineral Resource estimates contained in this report is shown in Table 14.1. The data used for both the 2017 and 2015 block model estimates include the results of all drilling and updated geologic interpretation carried out on the Property to 31 December 2017, given that no drilling was carried out on the Intrepid Zone after 2015. The Main Zone and Intrepid model have been depleted to reflect remaining Mineral Resources as of 31 December 2021.

Table 14.1 – Mineral Resource estimates at Rainy River

Area	Year of estimate	Author	Responsibility	Statement date
Intrepid	2015	El-Rassi	Nussipakynova	31 December 2021
Main Zone	2021*	Bassotti	Nussipakynova	31 December 2021

Note: *Based on the 2017 block model

The Mineral Resource estimate of the Main Zone is based principally 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. The Mineral Resources are based on a combined model of Intrepid and Main Zone with a minor update of domain 114 of the ODM/17 Zone based on additional drilling completed within the area.

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. A breakdown of open pit and underground Mineral Resources is shown in Table 14.2.

Open pit Mineral Resources are reported at cut-offs (COGs) of 0.30 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 Item 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 1.7 g/t AuEq.

Measured and Indicated Mineral Resources are estimated to total 19.2 million tonnes (Mt) at grades of 2.50 g/t Au and 6.3 g/t Ag, containing 1,543 koz of gold and 3,894 koz of silver. Inferred Mineral Resources are estimated to total 2.5 Mt at grades of 2.37 g/t Au and 2.5 g/t Ag, containing 189 koz of gold and 196 koz of silver.

Table 14.2 – Mineral Resources as of 31 December 2021

Category	Tonnes & grade			Contained metal	
	Tonnes (000's)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Total Mineral Resources					
Measured	762	1.29	2.7	32	67
Indicated	18,413	2.55	6.5	1,511	3,827
Total M + I Mineral Resources	19,175	2.50	6.3	1,543	3,894
Total Inferred Mineral Resources	2,478	2.37	2.5	189	196

Notes:
1. CIM Definition Standards (2014).
2. The Mineral Resources are stated exclusive of Mineral Reserves.
3. Mineral Resources were estimated using a long-term gold price of US$1,500 per troy oz and a long-term silver price of US$21 per troy oz. The exchange rate used was C$1.25:US$1 (C$1:US$0.80).
4. Direct processing open pit Mineral Resources are reported at a gold equivalent (AuEq) cut-off grade 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 are reported at a gold equivalent cut-off of 0.30 g/t.
5. Gold equivalency was calculated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 90)].
6. Open pit assumptions include:
7. Average gold and silver recoveries of 90% and 60%, respectively.
8. Open pit Mineral Resources were constrained by a conceptual pit shell and exclude underground Mineral Reserves within the pit shell.
9. Inferred open pit Mineral Resources include Inferred material from within the Mineral Reserve open pit.
10. Direct processing underground Mineral Resources are reported at a gold equivalent cut-off grade of 1.70 g/t.
11. Gold equivalency was calculated as AuEq = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 95)].
12. Underground assumptions include:
13. Average gold and silver recoveries of 95% and 60%, respectively.
14. Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
15. Effective date of Mineral Resources is 31 December 2021.
16. Underground Mineral Resources were restricted by a vetting process that excluded clusters of blocks distal to the MSO Mineral Reserve shapes.
17. The Qualified Person for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC.
18. Totals may not compute exactly due to rounding.
19. Tonnes and grades are in metric units.

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.

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 the QP 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. The hardness values were estimated 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.

Mineral Resources were reported from a block model which was combined from the open pit and underground block models of Main Zone (2017) and Intrepid Zone (2015). The combined model in 2021 also included an update of domain 114, which is the top part on the west of the ODM/17 Zone. This update is based on 10 new drillholes completed in 2019 and assayed 2020.

The Rainy River Mineral Resource database has been exported by New Gold and provided to AMC as a series of Microsoft Excel files and includes drillhole collar locations, downhole survey, assay, and lithology data from 2,125 core boreholes (912,557 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	2,225
Total length (m)	912,557
Downhole survey entries	36,093
Lithology entries	499,792
Assay entries	492,619
Assay length (m)	712,507
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.

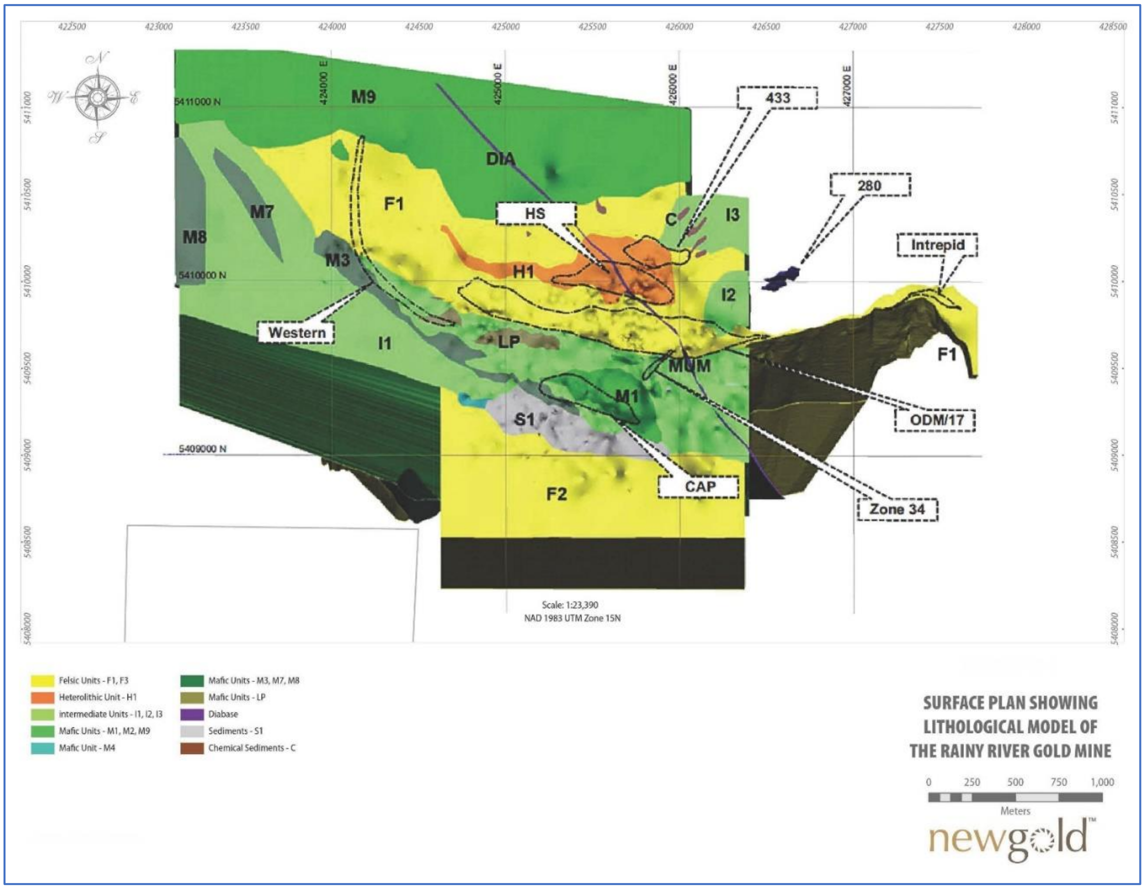
Upon receipt of the digital drilling data, the QP undertook the following validation step:

- 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).

14.2.1 Geological interpretation and 3D solids

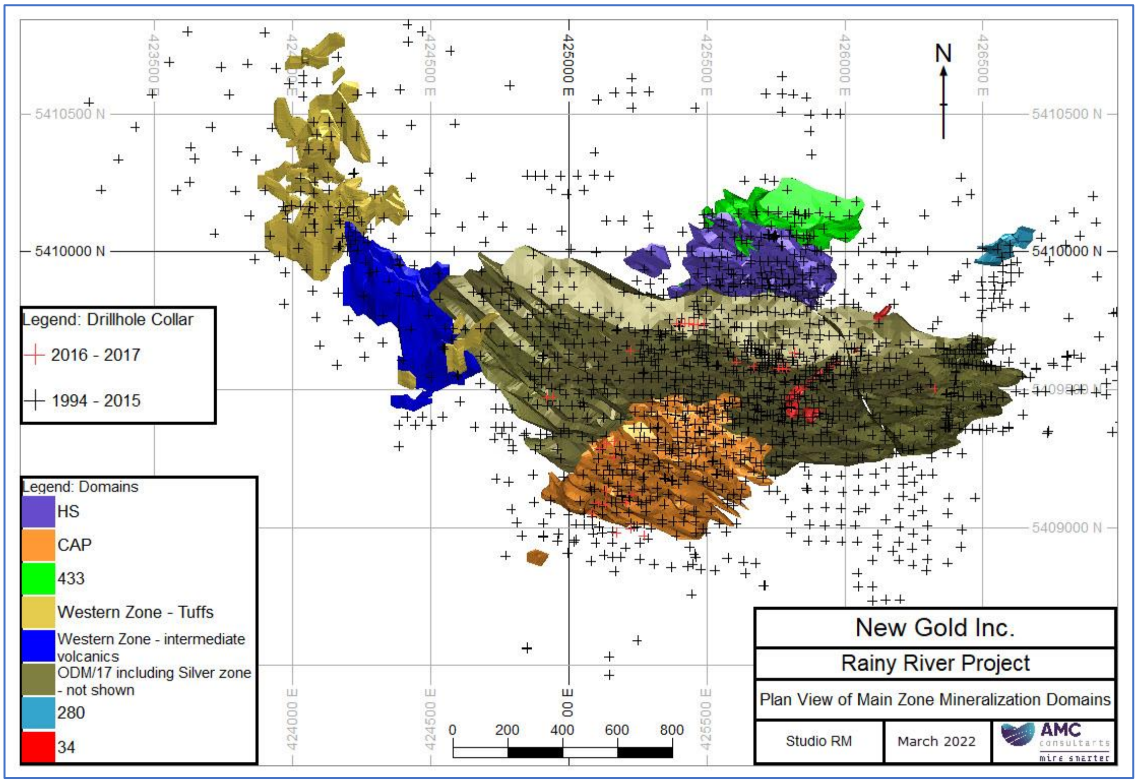
As it is currently defined by exploration drilling, the Rainy River deposit comprises a cluster of eight distinct zones of gold-silver mineralization, collectively referred to as the Main Zone. Intrepid Zone represents a satellite deposit located 1 km to the east of the Main Zone. A top-of-bedrock plan 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.



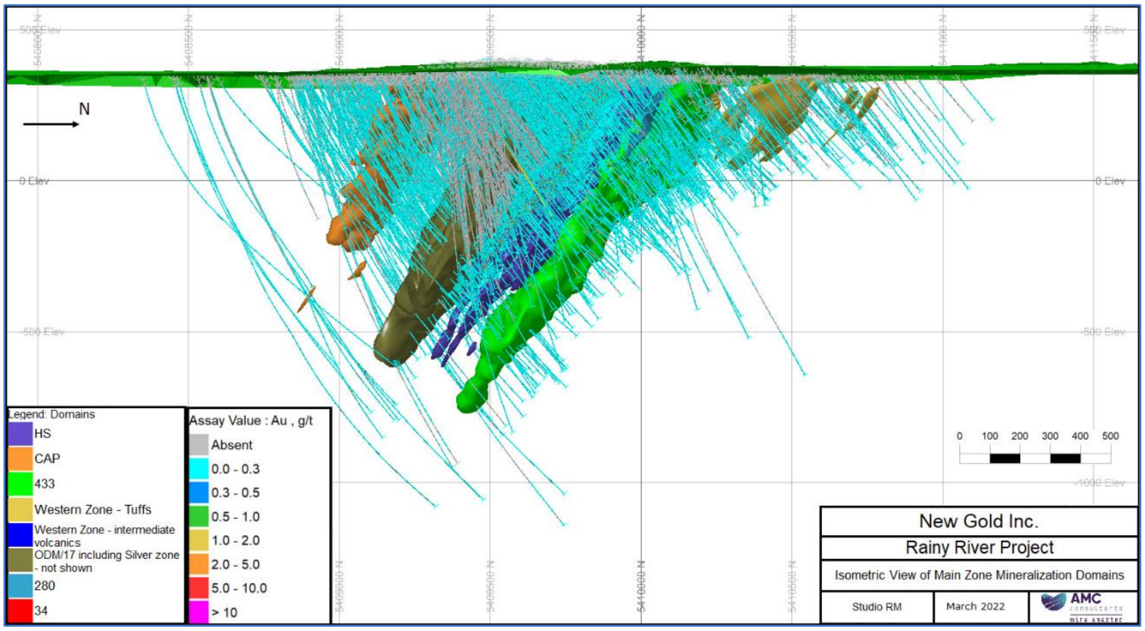
Source: New Gold 2022.

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



Source: AMC 2022.

Figure 14.2 – Plan view of Main Zone mineralization domains



Source: AMC 2022.

Figure 14.3 – Isometric view of Main Zone mineralization domains

14.2.1.1 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.

In 2020 the medium grade domain 114 of the ODM/17 Zone, close to the west wall of the pit was updated. 10 new drillholes were incorporated with the 2017 drillholes to update the wireframes. This update was only in the top part of the domain above 200 m RL, and slightly reduced the constraining open pit resource shell.

14.2.1.2 433 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:

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

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.

14.2.1.3 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.

14.2.1.4 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 level of 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.

14.2.1.5 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.

14.2.1.6 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:

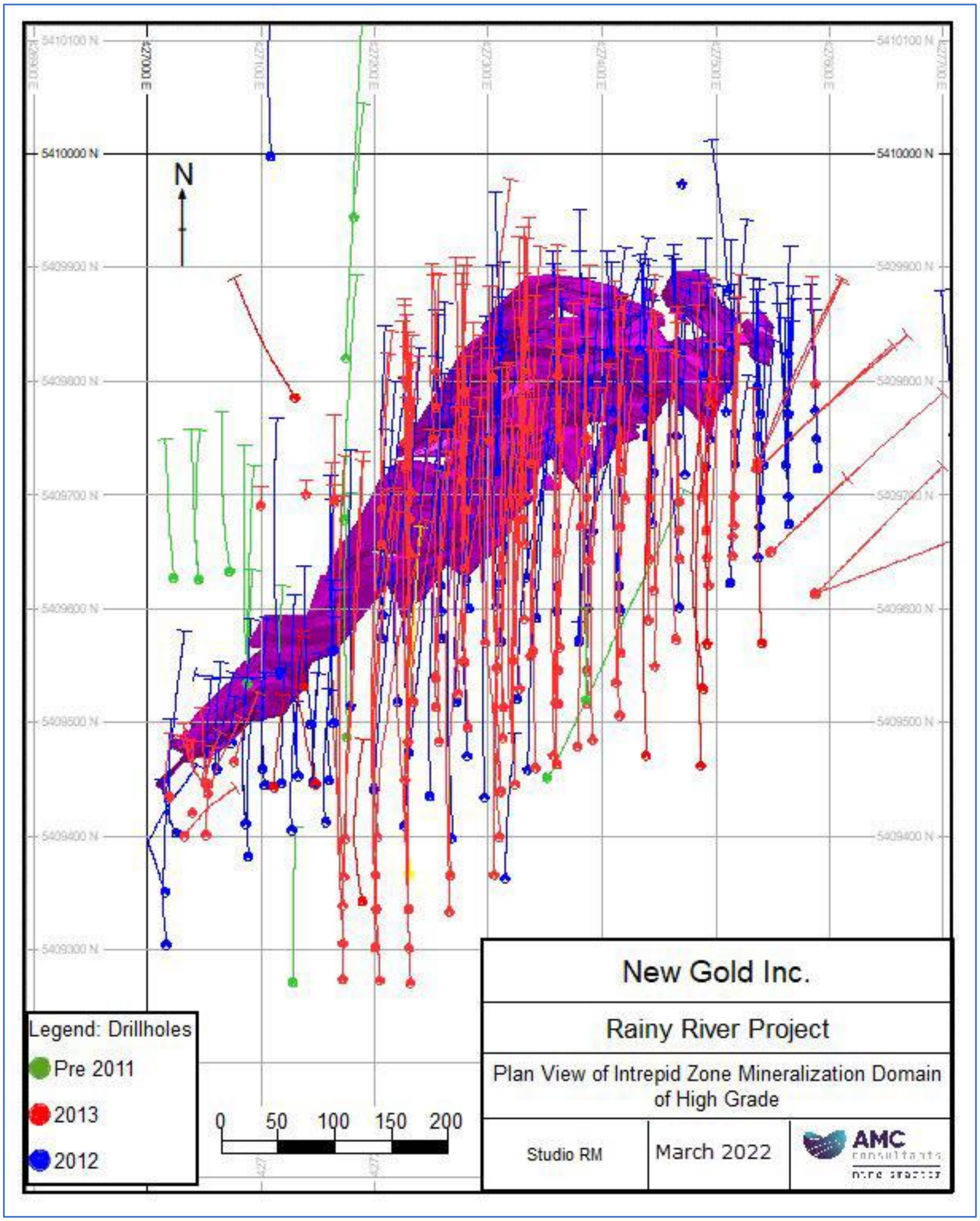
High grade:	Above 2.0 g/t Au.
Medium grade:	From 0.8 g/t to 2.0 g/t Au.
Low grade:	From 0.3 g/t to 0.8 g/t Au.

The Intrepid Zone has remained unchanged in the 2022 Mineral Resource estimation, with no new drilling data available. The general shape of the Intrepid Zone is shown in Figure 14.4.

14.2.1.7 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 crosscuts 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.

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



Source: AMC 2022.

Figure 14.4 – Plan view of Intrepid Zone high-grade domain

Table 14.4 – Mineralization and lithology domain codes

Mineralization/lithology	Zone/Grade domain name	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

Domain name	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	812	0.005	1,062.45	1.00	4.25	4.25
	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

Domain name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
	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
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 name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
ODM/17 Zone							
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	812	0.11	184.00	5.15	9.28	1.80
	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

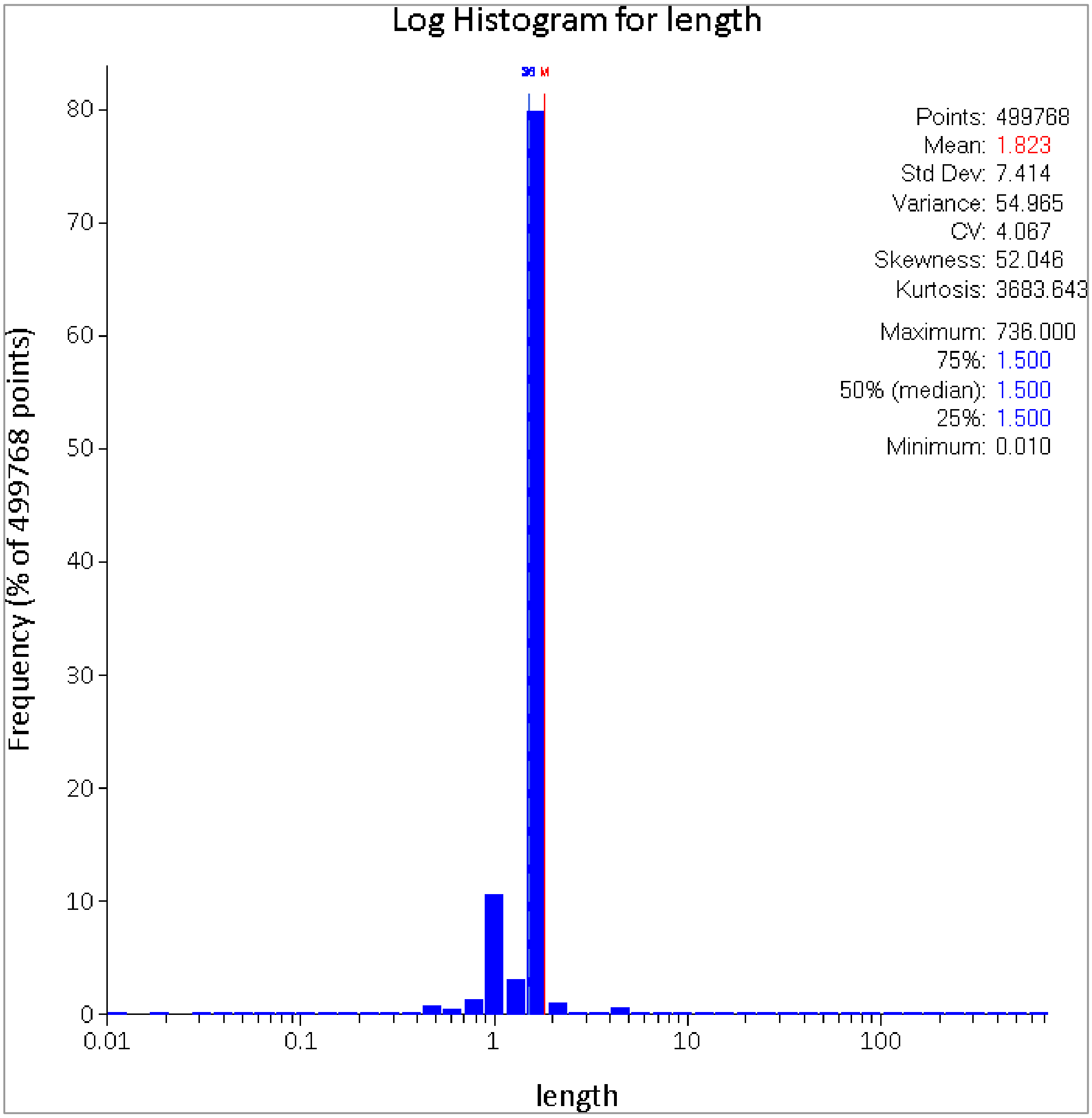
Domain name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
	124	2	0.80	20.80	10.80	12.25	1.13
	125	104	0.50	100.00	9.77	16.10	1.65
	126	316	0.60	2,580.00	77.20	202.79	2.63
34 Zone							
	200	230	0.10	59.00	2.45	5.60	2.29
280 Zone							
	280	269	0.10	11.40	0.95	1.34	1.41
433 Zone							
Low	300	11,419	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
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

Domain name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
	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.



Source: New Gold, 2018.

Figure 14.5 – Histogram of sample lengths at Rainy River

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, which were also estimated, see Item 1.1.1.

Table 14.7 – Summary of gold and silver capping thresholds

Zone	Domain code	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	10.00	78.98%	4	40.00	93.00%	8
	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

Zone	Domain code	Gold cap (g/t)	Gold percentile	No. capped	Silver cap (g/t)	Silver percentile	No. capped
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 name	Domain code	Count	Minimum	Maximum	Mean	Cut mean	CV	Cut CV
ODM/17	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	779	0.01	922.36	0.78	0.78	4.25	1.15
		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
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

Zone	Domain name	Domain code	Count	Minimum	Maximum	Mean	Cut mean	CV	Cut CV
	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.

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Table 14.9 - Statistical summary of silver composites

Zone	Domain name	Domain Code	Count	Minimum	Maximum	Mean	Cut mean	CV	Cut CV
ODM/17	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	779	0.11	184.00	5.15	4.79	1.80	1.20
		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

Zone	Domain name	Domain Code	Count	Minimum	Maximum	Mean	Cut mean	CV	Cut CV
		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 name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
ODM/17	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

Zone	Domain name	Domain code	Count	Minimum	Maximum	Mean	Stdv	CV
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	0.5	427,075	427,675
	North	5	0.5	5,409,500	5,409,950
	Vertical	5	0.5	-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, the QP imported all block models into Datamine software and integrated them into a single unified block model. The combined 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 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 combined block model.

Table 14.12 lists the block model parameters of the combined block model.

Table 14.12 – Block model parameters for the combined model

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 2020 AMC 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 code	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	-	-	-	-	-

Domain code	Nugget	Sill	Type	X1	X2	X3	Sill	Type	X2	Y2	Z2	Sill	Type	X3	Y3	Z3
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			

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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
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 un-estimated 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.

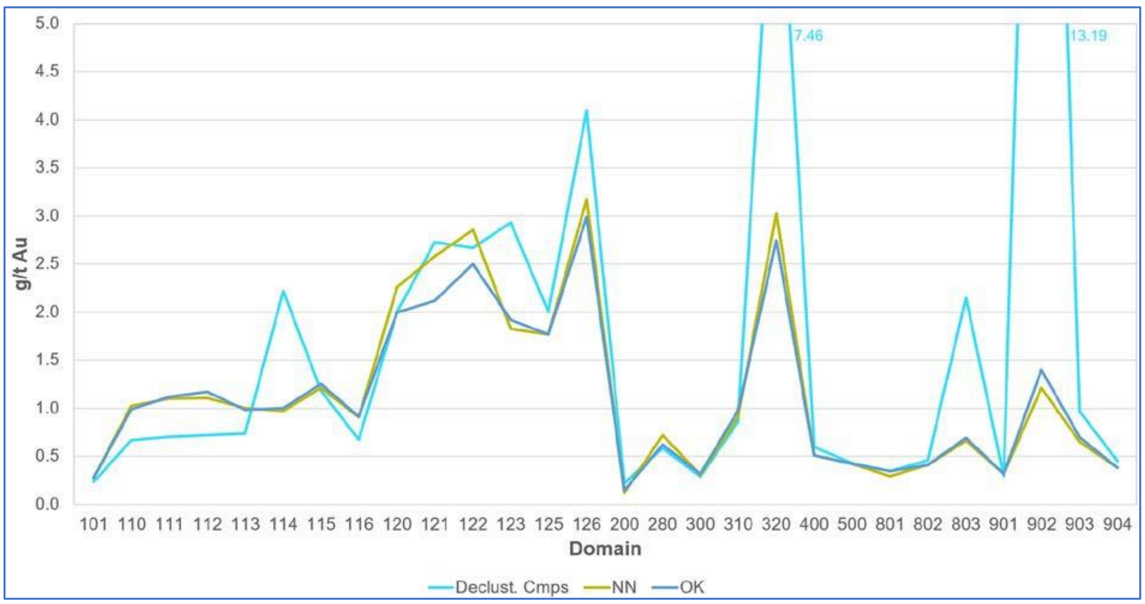
14.8 New Gold block model validation

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.

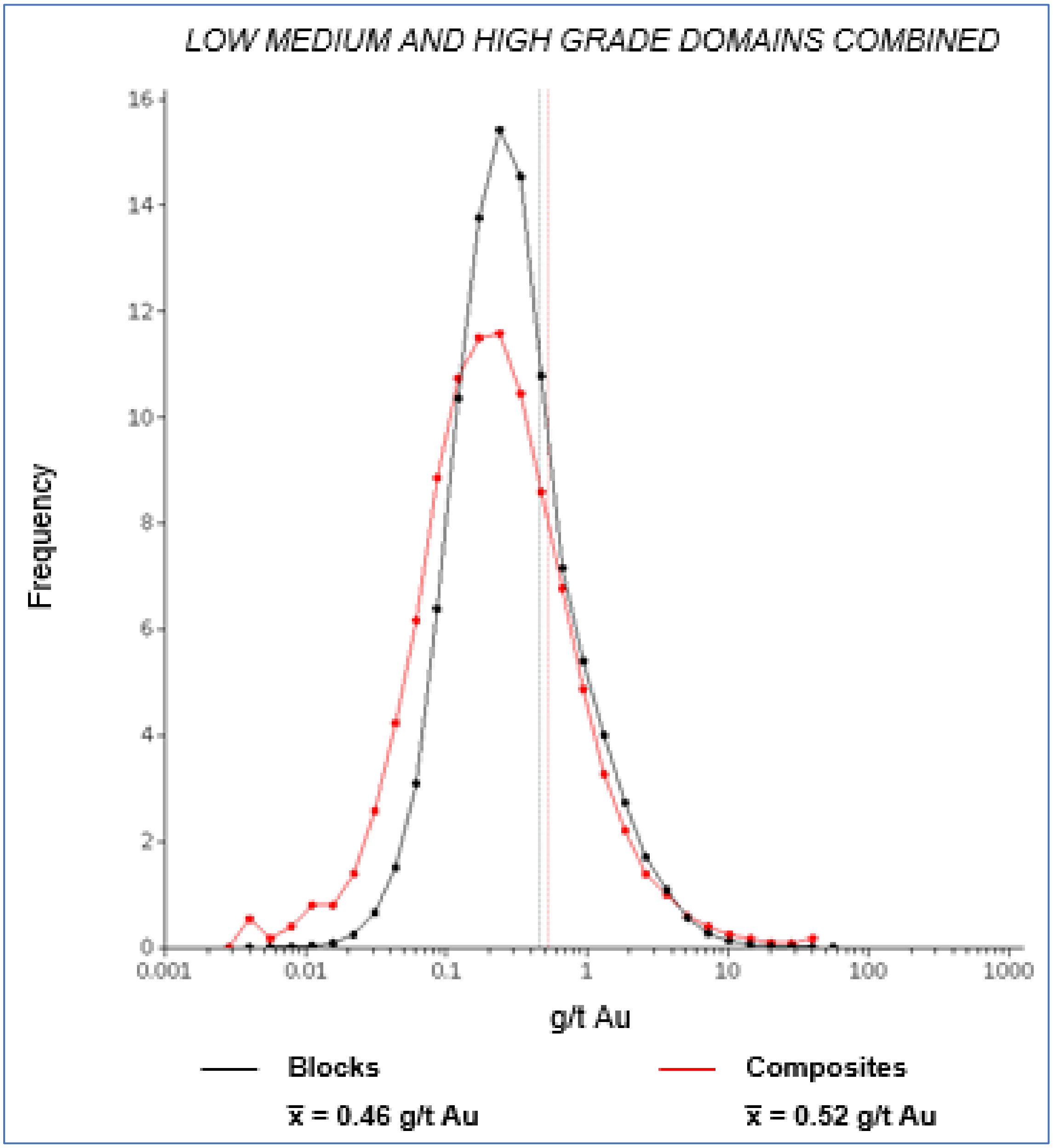
Selected comparative statistics are shown for gold in Figure 14.6.



Source: New Gold.

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

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.



Source: New Gold.

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

14.9 AMC block model validation

In addition to reviewing the validation undertaken by New Gold, the QP 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 checked for possible overlaps during the model combination process. This could lead to inadvertent double accounting of volumes during reporting. No overlaps were found.

14.9.1 Drillholes

Drillhole database files were provided in Excel format (collars, surveys, assays, and lithology) and are effective as of 31 December 2020.

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.

The QP 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.

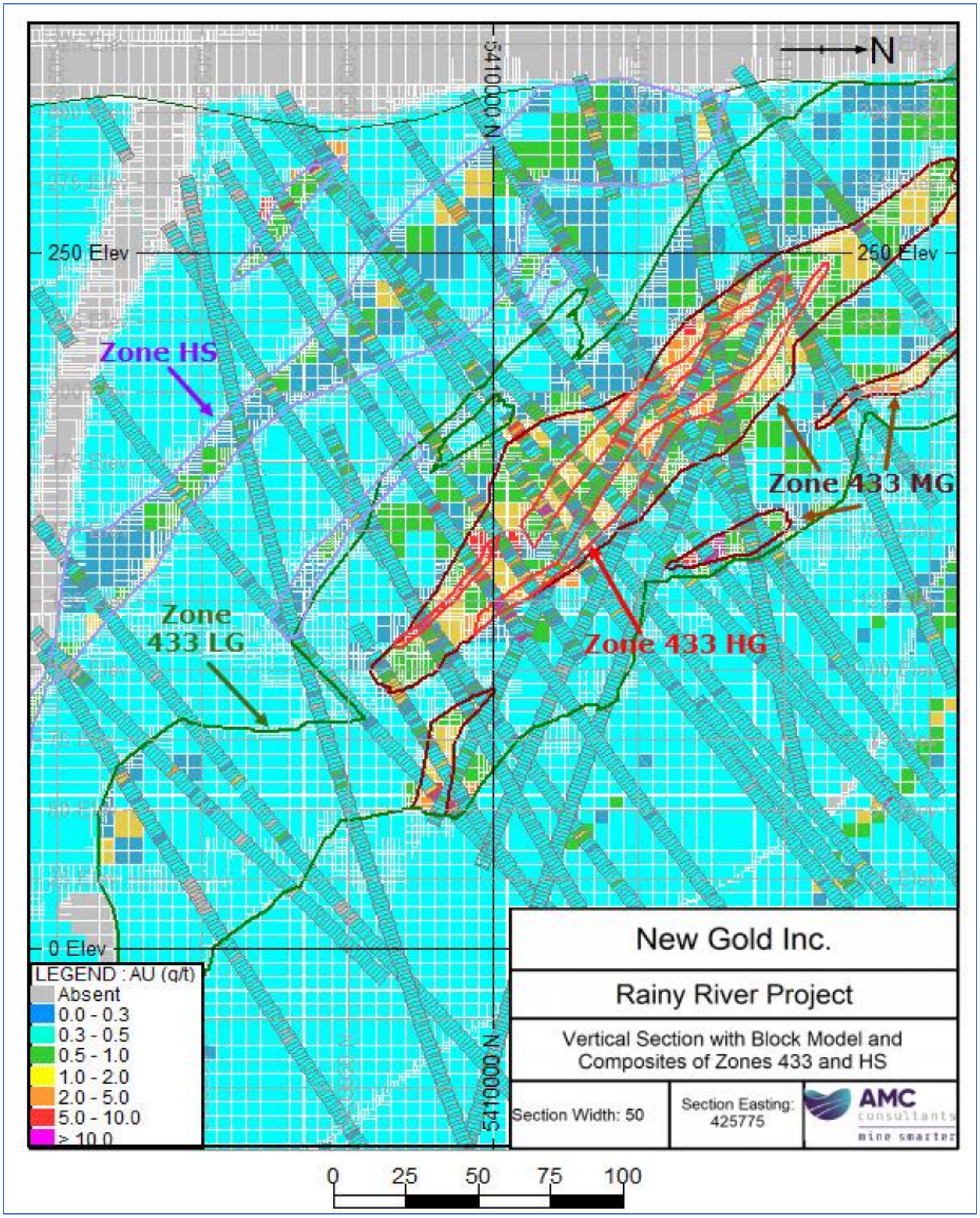
New Gold has provided 15 wireframe solids of mineralized domains. It was found that the file of ODM Zone Domain 126 is duplicating Domain 124. As Domain 124 was not estimated, there is no impact to the resource estimation. the QP is of the opinion that there are no domain flagging errors in the block 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. Five small lithology domains in unmineralized areas were identified which were missing and had not been assigned to the block model. The QP 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

A visual comparison of composite and block gold grades over the Main Zone was conducted. 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.



Source: AMC 2022.

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

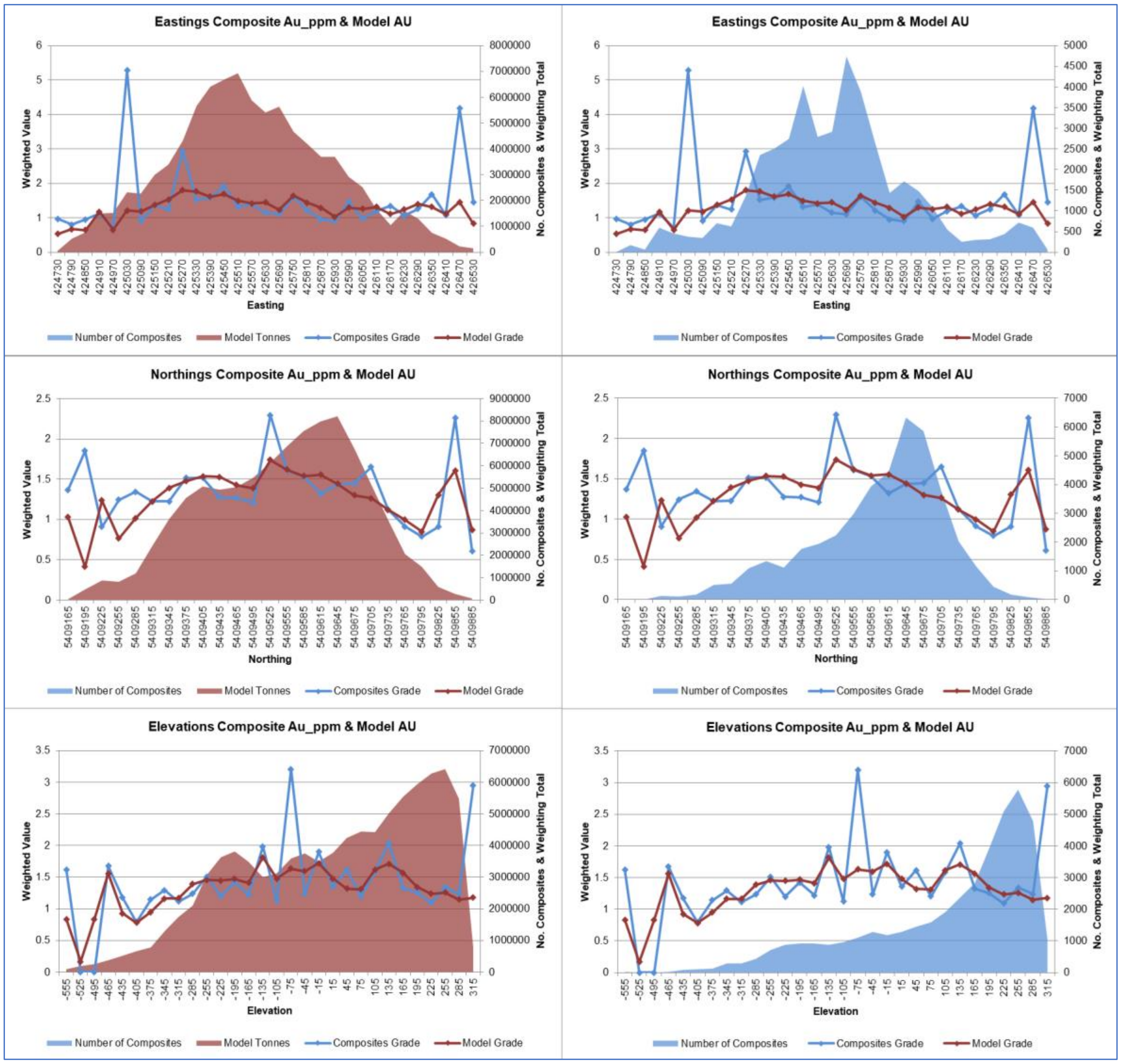
The QP compared the average composite and block gold and silver grades by domain and found them to show good agreement as shown in Table 14.18.

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

Domain code	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	0.79	0.88	4.78	5.14
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
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	0.45	0.38	19.12	17.04

Gold and silver grades of capped composites and blocks were compared 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 the QP for the gold grade distribution in the high and medium-grade domains of the ODM/17 Zone.

The QP is of the opinion that the methods used to produce the Mineral Resource gold and silver grades of capped composites and blocks were compared using swath estimate at the Main Zone and are in line with accepted industry practices.



Source: AMC 2022.

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

14.9.5 Intrepid model validation

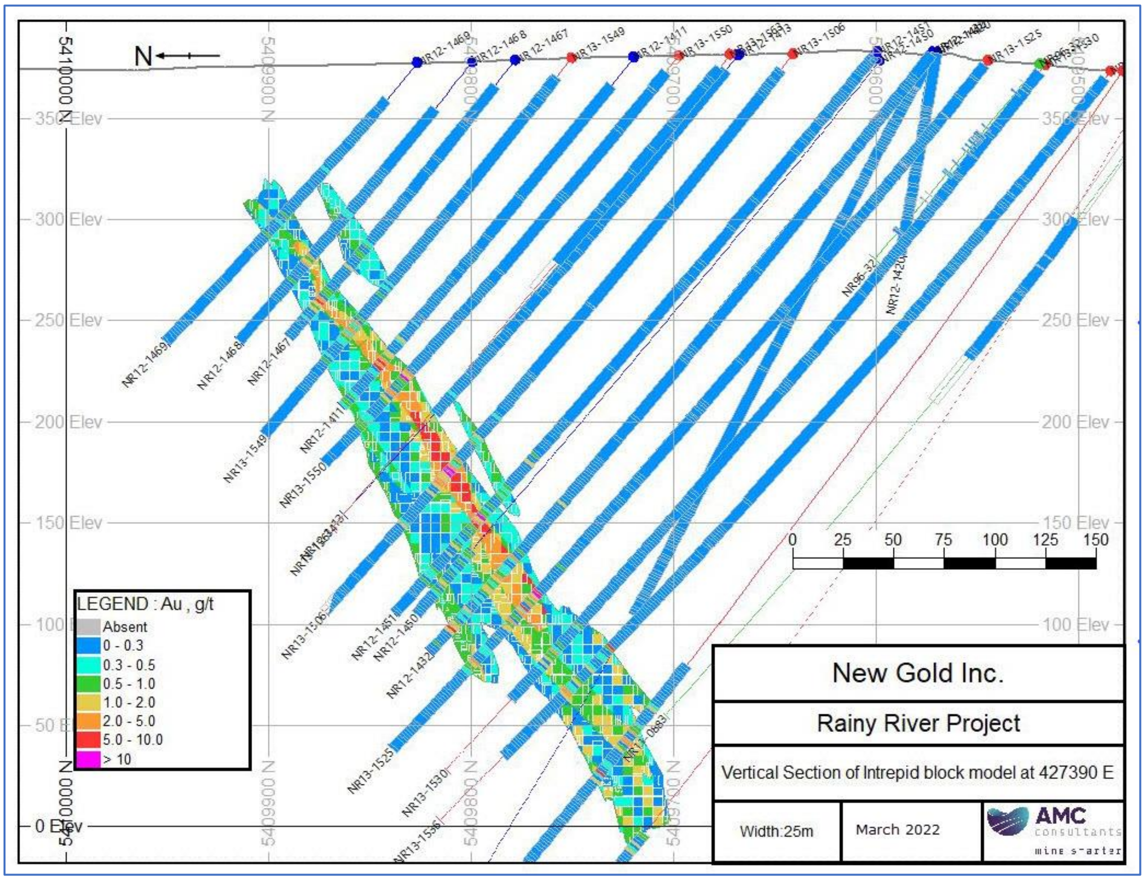
The review of the Intrepid Zone block model included the following:

- 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.

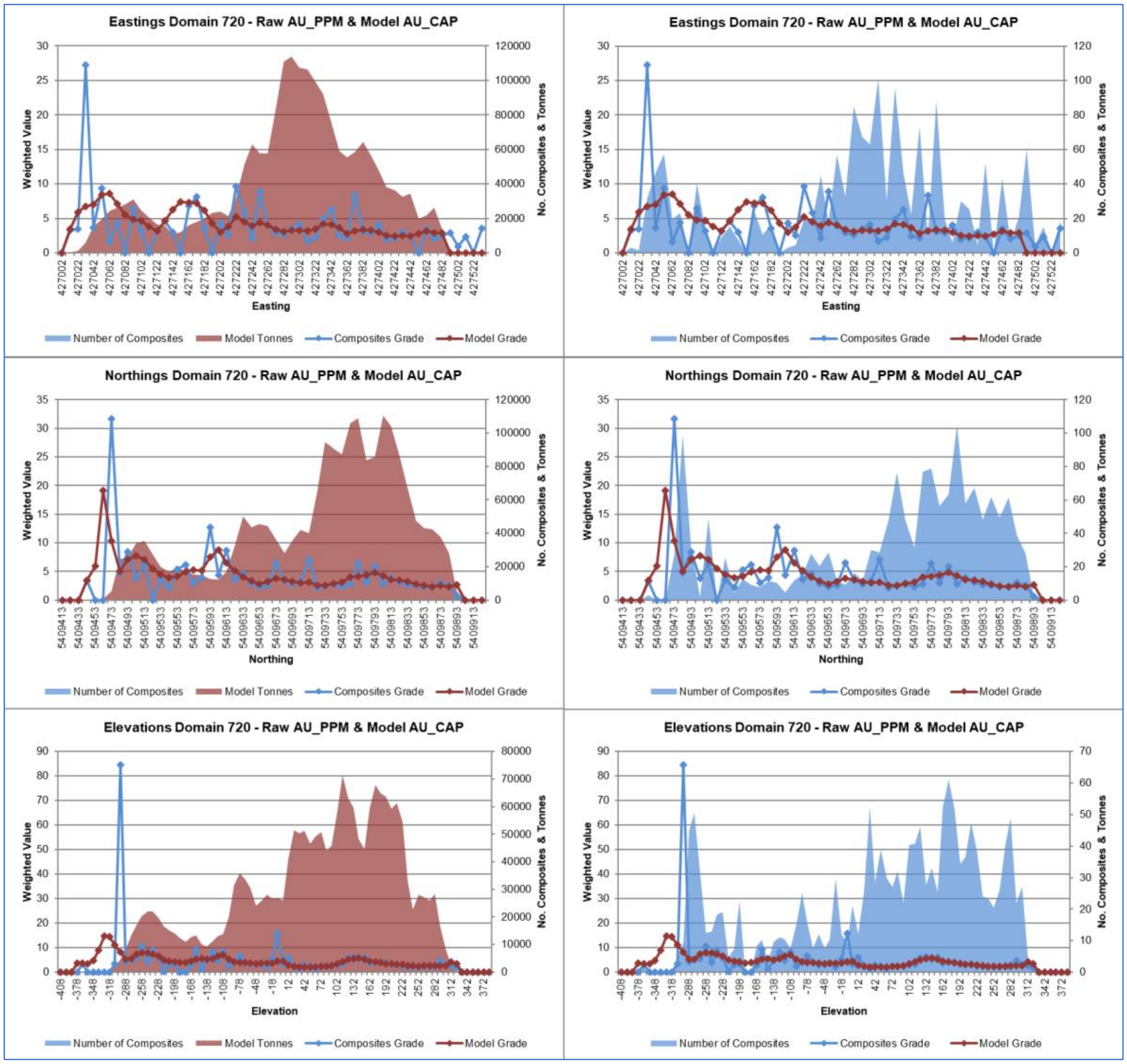
In general, AMC found there to be good visual agreement between the block model and drillhole grades as shown in Figure 14.10.



Source: AMC 2022.

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

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.



Source: AMC.

Figure 14.11 – Swath plots of gold grades for Intrepid Zone

No significant errors that would have an adverse material impact on Mineral Resources were found and the QP 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.

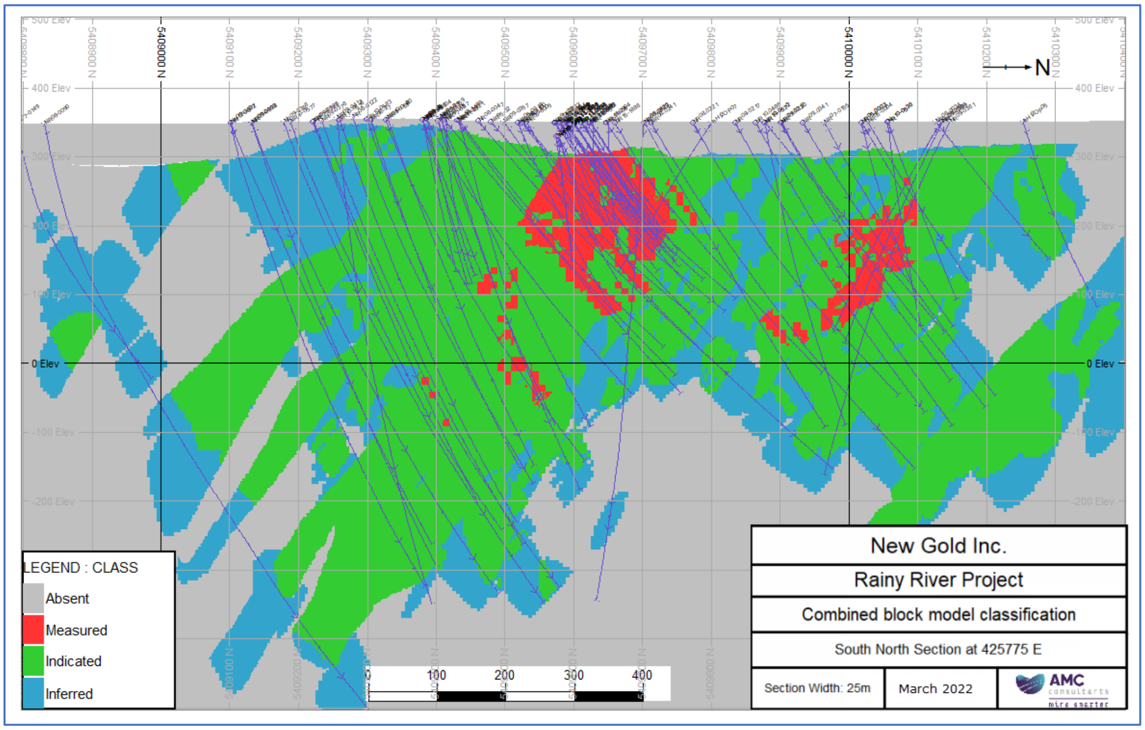
New Gold has assigned Measured classification where both drillhole density and bulk density measurements provide a high level of confidence in the geologic interpretation, grade continuity, and local grade and bulk density estimates. Currently, the ODM/17 and 433 Zones are the only areas with sufficient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

The QP is of the opinion that the classification criteria used to categorize blocks at Rainy River is reasonable.



Source: AMC 2022.

Figure 14.12 – Vertical section showing block model classification

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 (g/t) = Au (g/t) + [(Ag (g/t) *21*60)/1500*90)]
- Underground AuEq (g/t) = Au (g/t) + [(Ag (g/t) *21*60)/1500*95)]

Where:

- Gold price = $1,500 per ounce
- Gold recovery = 90% open pit and 95% underground
- Silver price = $21 per ounce
- Silver recovery = 60% for open pit and underground

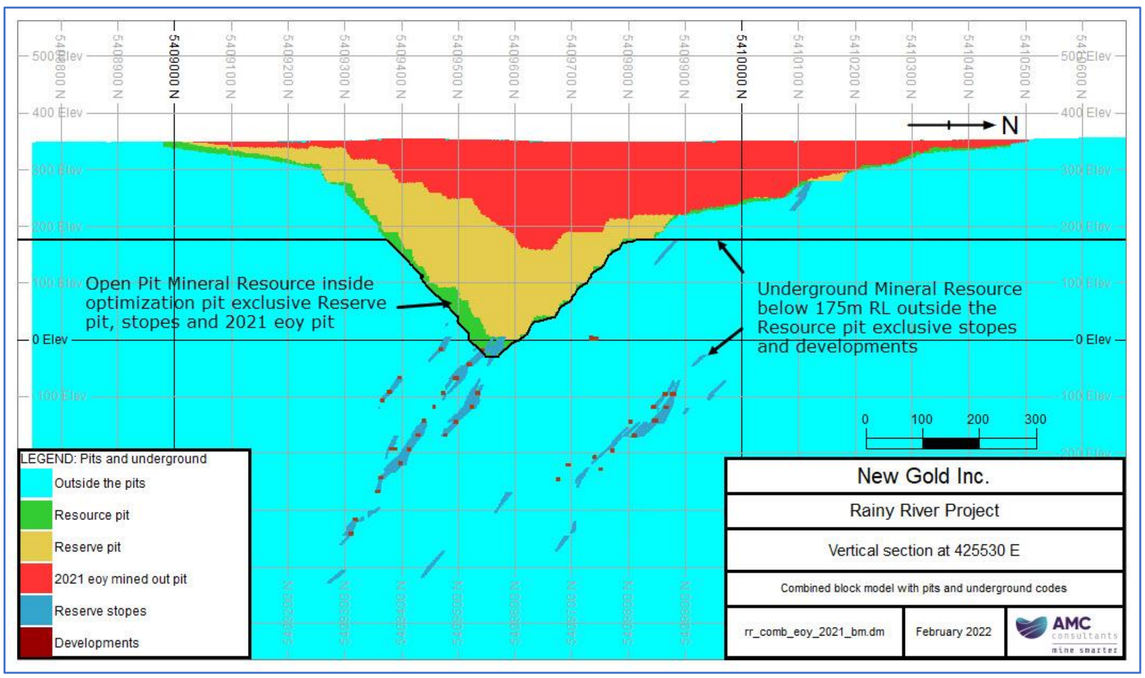
The assumptions for gold and silver prices and recoveries are discussed in more detail in Items 15 and 13.

14.11 Mineral Resource reporting

Mineral Resources for the Rainy River Mine have been updated to 31 December 2021. 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 2020 New Golds year end published Mineral Resource and Mineral Reserve (MRMR) statement. 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 $1,500 per ounce for gold and $21 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 175m (except Intrepid) reference elevation and peripheral to and below the conceptual resource pit shell. Underground Mineral Resources report continuous blocks above a 1.7 g/t AuEq COG within mineralized domains. Manual adjustments were applied removing the material with lower level of confidence or absence of continuity, and with a low probability of “reasonable prospects of eventual economic extraction” (RPEEE).

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.



Source: AMC 2022.

Figure 14.13 – Mineral Resource reporting criteria

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Table 14.20 – Mineral Resources as of 31 December 2021

Category	Tonnes & grade			Contained metal	
	Tonnes	**Gold**	**Silver**	**Gold**	**Silver**
	(t x '000)	**(g/t)**	**(g/t)**	**(K oz)**	**(K oz)**
Direct processing Mineral Resources					
Open pit					
Measured	570	1.61	3.0	30	55
Indicated	3,131	1.48	3.2	149	325
Sub-total open pit M + I	**3,701**	**1.50**	**3.2**	**179**	**380**
Inferred	481	0.98	2.5	15	38
Underground					
Measured	–	-	-	-	-
Indicated	14,014	2.99	7.6	1,348	3,422
Sub-total underground M + I	**14,014**	**2.99**	**7.6**	**1,348**	**3,422**
Inferred	1,593	3.30	2.7	169	141
Low grade Mineral Resources					
Open pit					
Measured	192	0.34	2.0	2	12
Indicated	1,268	0.34	1.9	14	80
Sub-total open pit M + I	**1,460**	**0.34**	**2.0**	**16**	**92**
Inferred	404	0.35	1.3	5	17
Total Mineral Resources					
Measured	762	1.29	2.7	32	67
Indicated	18,413	2.55	6.5	1,511	3,827
Total M + I Mineral Resources	**19,175**	**2.50**	**6.3**	**1,543**	**3,894**
Total Inferred Mineral Resources	2,478	2.37	2.5	189	196

Notes:

1. CIM Definition Standards (2014).
2. The Mineral Resources are stated exclusive of Mineral Reserves.
3. Mineral Resources were estimated using a long-term gold price of US$1,500 per troy oz and a long-term silver price of US$21 per troy oz. The exchange rate used was C$1.25: US$1 (C$1 = US$0.80).
4. Direct processing open pit Mineral Resources are reported at a gold equivalent (AuEq) cut-off grade 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 are reported at a gold equivalent cut-off of 0.30 g/t.
5. Gold equivalency was calculated as AuEq (g/t) = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 90)].
6. Open pit assumptions include:
7. Average gold and silver recoveries of 90% and 60%, respectively.
8. Open pit Mineral Resources were constrained by a conceptual pit shell and exclude underground Mineral Reserves within the pit shell.
9. Inferred open pit Mineral Resources include Inferred material from within the Mineral Reserve open pit.
10. Direct processing underground Mineral Resources are reported at a gold equivalent cut-off grade of 1.70 g/t.
11. Gold equivalency was calculated as AuEq = Au (g/t) + [(Ag (g/t) * 21 * 60)/ (1,500 * 95)].
12. Underground assumptions include:
13. Average gold and silver recoveries of 95% and 60%, respectively.
14. Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
15. Effective date of Mineral Resources is 31 December 2021.
16. Underground Mineral Resources were restricted by a vetting process that excluded clusters of blocks distal to the MSO Mineral Reserve shapes.
17. The Qualified Person for the Mineral Resource estimate is Ms D. Nussipakynova, P.Geo., of AMC.

18. Totals may not add exactly due to rounding.
19. Tonnes and grades are in metric units.

14.12 Comparison to previous Mineral Resource estimate

A comparison between the current Mineral Resource estimate, which is effective 31 December 2021, and the Mineral Resource statement dated 31 December 2020 is presented in Table 14.21. Principal changes since the 31 December 2020 estimate are:

- Ongoing depletion of Mineral Resources due to open pit mining.
- The updated underground Mineral Resources decreased due to a conversion of previous underground Mineral Resources to Mineral Reserves. See Item 15 for details.
- Updated costs reflecting the current cost of operation at the mine (mine, process, G&A, and relevant sustaining capital requirements). Overall, open pit mining costs have increased accompanied by decreases in processing and G&A costs.
- Updated geotechnical model resulting in slightly different overall pit slope angles.

The net result of the proceeding points is a reduction in Mineral Resources from previous estimate.

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

Table 14.21 – Comparison of 2021 and 2020 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 2021	Measured	762	1.29	2.7	32	67
	Indicated	18,413	2.55	6.5	1,511	3,827
	Measured & Indicated	19,175	2.50	6.3	1,543	3,894
	Inferred	2,478	2.37	2.5	189	196
31 December 2020	Measured	825	1.20	2.3	32	61
	Indicated	24,244	2.53	6.5	1,973	5,064
	Measured & Indicated	25,072	2.49	6.4	2,005	5,125
	Inferred	3,077	2.05	2.6	203	258
Difference %	Measured	-8	8	17	0	-10
	Indicated	-24	1	0	-23	-24
	Measured & Indicated	-24	0	-2	-23	-24
	Inferred	-19	16	-4	-7	-24

Notes for the 31 December 2021 estimate is shown in the footnotes under Table 14.20.
Notes for the 31 December 2020 estimate follow:

- CIM Definition Standards (2014).

- The Mineral Resources are stated exclusive of Mineral Reserves.
- Mineral Resources are estimated using a long-term gold price of US$1,500 per troy oz and a long-term silver price of US$20 per troy oz. The exchange rate used was 1:1.30 US$/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) * 20 * 60)/ (1,500 * 90)].
- Open pit assumptions include:
  a) Average gold and silver recoveries of 90% and 60%, respectively.
  b) Open pit Mineral Resources were constrained by a conceptual pit shell.
  c) 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 1.70 g/t.
- Gold equivalency was estimated as AuEq = Au (g/t) + [(Ag (g/t) * 20 * 60)/ (1,500 * 95)].
- Underground assumptions include:
  a) Average gold and silver recoveries of 95% and 60%, respectively.
  b) Underground Mineral Resources are excluded above 175 m RL except for the Intrepid Zone.
- Effective date of Mineral Resources is 31 December 2020.
- Underground Mineral Resources were restricted by a vetting process that included not reporting peripherals to the main zones.
- Totals may not add exactly due to rounding.
- Tonnes and grades are in metric units.

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 5,897 kt (24%), while gold grade has essentially stayed the same and the silver grade have decreased by 2% resulting in contained gold metal decreasing by 462 koz (23%) and silver metal decreasing by 1,237 koz (24%).
- Total Inferred Mineral Resource tonnes have decreased by about 600 kt (19%), gold grade increased by 16% and silver grade decreased by 4%. The metal content of gold and silver decreased by 14 koz (7%) and 24 koz (24%) respectively.

15 MINERAL RESERVE ESTIMATES

15.1 Introduction

The Mineral Reserve estimates presented herein conform to CIM Definition Standards (2014) and include Measured and Indicated Mineral Resources but do not include Inferred Mineral Resources. The Mineral Reserves represent the estimated tonnage and grade of ore considered economically viable for extraction.

New Gold has prepared open pit Mineral Reserves 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.

InnovExplo has prepared Underground (UG) Mineral Reserves under the guidance of Mr. Éric Lecomte, P.Eng., a mining engineer employed by InnovExplo. Mr. Lecomte is independent of New Gold and takes QP responsibility as defined in NI 43-101 for the underground Mineral Reserve estimate.

The Mineral Reserve estimates for the Rainy River deposits are summarized in Table 15.1.

Table 15.1 – Summary of Rainy River Mineral Reserves – effective December 31, 2021

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Open pit (including stockpile)					
Proven	26,276	0.72	2.2	605	1,837
Probable	31,288	0.95	2.1	953	2,101
Sub-total open pit	**57,563**	**0.84**	**2.1**	**1,558**	**3,938**
Underground					
Proven	-	-	-	-	-
Probable	12,657	3.05	7.6	1,241	3,084
Sub-total underground	**12,657**	**3.05**	**7.6**	**1,241**	**3,084**
Total					
Proven	26,276	0.72	2.2	605	1,837
Probable	43,944	1.55	3.7	2,194	5,185
Total Mineral Reserves	**70,220**	**1.24**	**3.1**	**2,799**	**7,022**

Notes:

- CIM Definition Standards (2014) were used for reporting these Mineral Reserves.
- Refer to the footnotes of Dilution Factor Calculation for prices, cut-offs, recoveries, etc.
- Totals may not add exactly due to rounding.

The Mineral Reserves herein supersede the Mineral Reserves reported previously at year-end 2020 by New Gold for the Rainy River Mine.

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

15.2 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 conventional truck-shovel open pit mining operation to extract the Mineral Reserve.

15.2.1 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). Additionally, material is identified as being associated with the CAP Zone or labelled as Non-CAP Zone. The CAP Zone is described in Item 7 and has been identified as having a different metallurgical recovery response than other mineralized zones within the deposit in Item 13. Non-Cap Zone is a reference to all other mineralized zones not identified as CAP Zone within the open pit deposit. Open pit Mineral Reserve estimates presents a breakdown of the material classification and their respective applied COG.

Table 15.2 – Material classification

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

DPO is material identified as higher-grade and is to be directly processed as excavated from the open pit mine and / or is preferentially re-handled first from stockpiles as part of an elevated COG operating policy.

LGO material is lower grade material that is preferentially stockpiled to not displace better quality material from being processed early in the mine plan, but as required is used as mill feed to supplement excess process capacity.

See Item 15.2.4 for a description on how the COG's have been calculated and are applied.

15.2.2 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 Item 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 2021 New Gold utilized the afore mentioned model and undertook a dilution / ore loss study 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 potential 3.3 m dilution skin was applied to all blocks.
- A potential 0.2 m ore loss skin was applied to all blocks.
- Amount of dilution and ore loss will depend upon the grade of the adjacent blocks. Each block is able to pull dilution and give ore loss material from any adjacent block with a lower grade and to give dilution and pull ore loss material to any adjacent block with higher grade.

In addition, because of a significant negative reconciliation in the East Lobe during 2021, New Gold (New Gold, 2022) applied a modifying factor against the gold grade in the East Lobe of the mine planning model of 89% to better reflect current results.

Overall net impact of the modifying factors within the Mineral Reserve pit design applied against the regularized model at a COG of 0.3 g/t AuEq is an increase of 16% in ore tonnes and a 13% decrease in grade. This diluted model is designated as the mine planning resource model.

2021 Reconciliation of Measured and Indicated Resource within the mine planning resource model to the GC model and DOM at Rainy River, is as follows:

Table 15.3 – Reconciliation January – December 2021

	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Model and reconcilation values					
Mine planning resource model (MP)	15,850	0.74	3.0	378	1537
GC model	15,500	0.74	3.9	369	1,920
DOM	14,472	0.70	3.2	325	1,500
Reconciliation mine planning resource model					
GC vs MP	98%	100%	128%	98%	125%
DOM vs MP	91%	94%	107%	86%	98%
DOM vs GC	93%	95%	84%	88%	78%

The mine planning resource model reconciliation provides a better overall prediction of tonnes, grade, and contained metal when compared to the regularized resource model (see Item 12). The reconciliation of the grade control model is good compared to the mine planning resource model. However, mining and / or metal accounting practices during reconciliation appear to be impacting the DOM recorded metal mined from the deposit. New Gold has undertaken additional drilling in 2021, particularly in the East Lobe to improve model predictability and is focusing attention on mining practices for the open pit and metal accounting practices during reconciliation to improve results. Results of the drilling program are expected in Q1-2022 and according to New Gold will be included within an updated Mineral Resource model during 2022.

Modifying factors should be reviewed as new mining areas are exposed and additional reconciliation information is gathered to continue validating the model performance.

15.2.3 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 Item 13.3.

The predictive gold recovery formulae are as follows:

CAP Zone Predictive metal recovery curves

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

Non-CAP Zone Predictive metal recovery curves

- AuTG = 0.36349 + (AuHG) * 0.06667 + P80 * 0.00025 + (%ODM) * -0.34414 + (%433) * -0.38227 + (%HS) * -0.35209
- Au Rec = min(95,[(AuHG – AuTG)/ AuHG] * 100)

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

Where:

- AuTG is the process plant gold tailings grade in g/t
- 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. For recovery estimation within the block model, this is set to the expected 100 um average.
- %ODM, %433 and %HS are all fraction of ore by tonnage

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

CAP Zone predictive formulas for silver recovery

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

Non-CAP Zone predictive formulas for silver recovery

- Ag Rec = [([AgHG – (0.4409 * AgHG$^{0.9285}$)] / AgHG) * 100] * 0.966
- Where:
- Ag Rec is the silver recovery in %.
- AgHG is the silver head grade in g/t.

15.2.4 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 2022 Budget models. Open pit metallurgical recoveries summarizes the open pit COG assumptions. A COG of 0.49 g/t AuEq for CAP Zone and 0.46 g/t AuEq for Non-CAP zone material was used for the estimation of Direct Processing Mineral. Low grade Mineral Reserves are set at a COG of 0.30 g/t AuEq for all rock types.

Table 15.4 – Open pit COG calculation parameters

Parameter field	Unit	Open pit parameter value
Metal prices		
Gold	$/oz	1,400.00
Mining cost		
Ore	$/t mined	2.70
Waste	$/t mined	2.90
Incremental per bench (below 340 m elevation)	$/t mined	0.04
Re-handle	$/t ore	2.69
Sustaining capital	$/t ore	0.48
	$/t waste	0.55
Process cost		
Process base cost	$/t milled	6.70
Process variable cost	$/t milled	3.08
Sustaining capital	$/t milled	0.03
Tailings management	$/t milled	1.80
Treatment & refining		
Gold	$/oz recoverable	1.89
Silver	$/oz recoverable	0.85
Royalties		
Gold	$/oz recoverable	9.93
Silver	$/oz recoverable	0.11
G&A		
G&A	$/t processed	3.05
Gold recovery at cut-off (utilized)		
CAP zone		
Direct processing ore	%	73.9
Low grade-ore	%	73.1
Non-CAP zone		
Direct processing ore		
ODM	%	83.7
433	%	92.0

Parameter field	Unit	Open pit parameter value	
HS	%	85.4	
Low grade-ore			
ODM	%	78.5	
433	%	91.3	
HS	%	81.2	
COG	**Unit**	**Calculated**	**Utilized**
CAP zone			
Direct processing ore	g/t AuEq	0.43	0.49
Low grade-ore	g/t AuEq	0.23	0.30
Non-CAP zone			
Direct processing ore			
ODM	g/t AuEq	0.38	0.46
433	g/t AuEq	0.34	0.46
HS	g/t AuEq	0.37	0.46
Low grade-ore			
ODM	g/t AuEq	0.21	0.30
433	g/t AuEq	0.18	0.30
HS	g/t AuEq	0.22	0.30

The COG is expressed as an AuEq grade which is estimated as follows:

$$AuEq = Au\ (g/t) + [(Ag\ (g/t) * 19 * 60) / (1{,}400 * 90)]$$

Where, the factors in the equivalence calculation are:

- Gold price $1,400/oz
- Silver price $19/oz
- Gold recovery 90% (estimated preliminary overall average)
- Silver recovery 60% (estimated preliminary overall average)

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". This is calculated as 0.43 g/t AuEq and 0.34 – 0.38 g/t AuEq for CAP and Non-CAP material respectively, but as mentioned earlier, an elevated COG is applied to this material classification as part of an elevated COG policy giving preference to processing better quality material first. Material between the calculated and utilized COG's for DPO is subsequently classified as LGO.

The remaining LGO Mineral Resources below the respective DPO calculated COG's do not cover all fixed process costs or site G&A costs. As such, they are not included within the open pit optimization process but are rather reported 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, and thus can be considered incremental.

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 considers the open pit COG calculation to be appropriate for the deposit based upon the assumptions used.

15.2.5 Open pit optimization

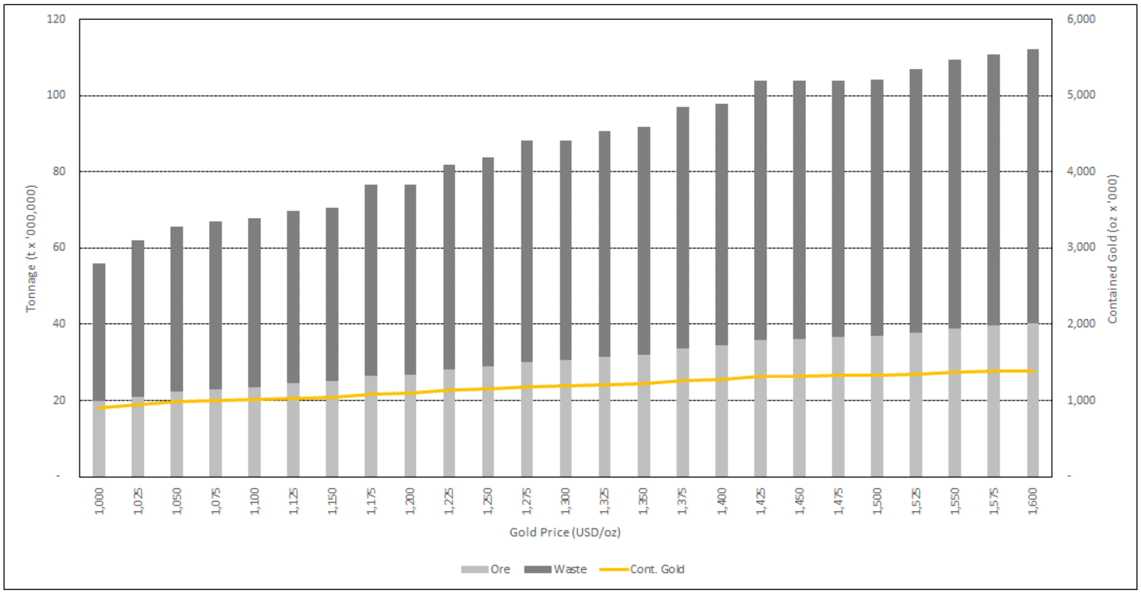
The pit optimization was conducted by AMC on the mine planning resource model described in Item 15.2.1 using metal prices of $1,400/oz gold and $19/oz silver. The parameters used for open pit optimization are provided in Table 15.4. Only Measured and Indicated Mineral Resources were included in the pit optimization process. The portion of the low-grade Mineral Resources which cannot cover all costs, as described in the previous Item, are excluded from the pit optimization process. GEOVIA Whittle™ was the software used for the open pit optimization.

The open pit operation is planned to transition to an underground mining operation in the future. As part of the pit optimization, a preliminary cut-off cost of $80.09/t (inclusive of all mining, process, treatment and refining, royalties, applicable sustaining capital, and G&A costs) was applied to limit the pit optimization to only extract Mineral Resource more economically mined by an open pit operation than by an underground operation.

A subsequent optimization was done without consideration of the underground operation, maximizing the open pit size. In general, the pit expanded by 35 m to 70 m in the southwest portion of the pit as well as approximately 20 to 30 m in depth. The two optimizations were subsequently utilized to guide the design of the final open pit limit in this expansion area taking into account previous underground designs.

Open pit slope estimates were included in the open pit optimization process utilizing overburden slope recommendations from Golder Associates (Golder), and hard rock slope recommendations from SRK Consulting (SRK) as presented in Item 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 40° to 54°.

Open pit optimization results at incremental gold metal price are provided in Figure 15.1 for the pit optimization that considered an underground operation as mentioned previously. The revenue factor 01 pit at $1,400/oz gold was used as the principal guide to undertake the pit design.



Note: Ore tonnages and contained metal reflect DPO material and the portion of LGO material that cover full operating costs only. Impact of incremental LGO and existing stockpiles is not represented in the open pit optimization chart.
Source: AMC.

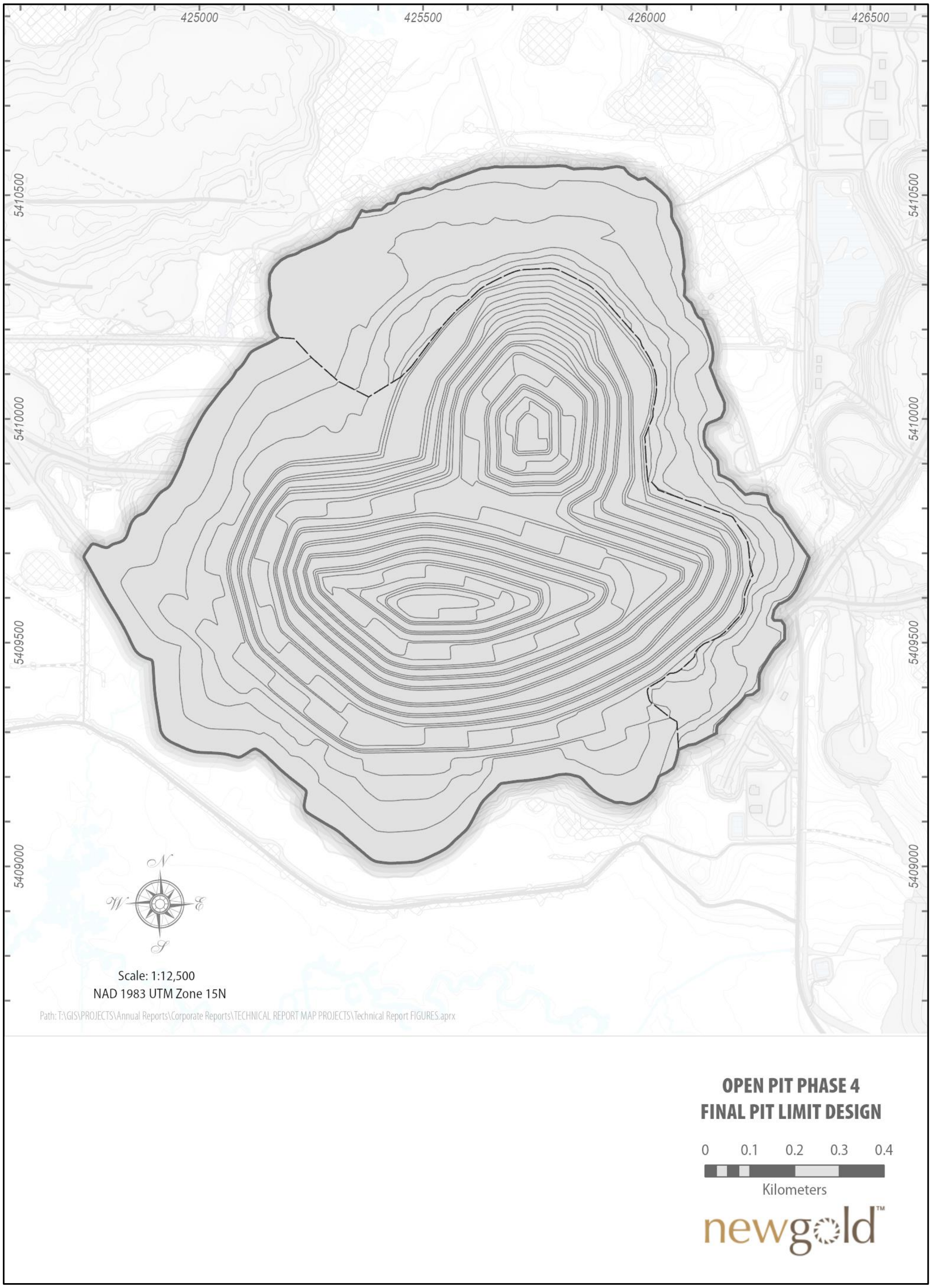
Figure 15.1 – Open pit optimization results at incremental gold metal price

15.2.6 Reserve pit design

The optimized pit solution resulting from the criteria presented in the preceding Items 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 1,550 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-2021, including waste, is 145 Mt.



Source: New Gold 2022.

Figure 15.2 – Open pit final limit design

15.3 Underground Mineral Reserve Estimates

The underground Mineral Reserves were estimated through the application of mine development and stoping plans to convert Indicated Resources to Probable Reserves. The Mineral Reserves for the Rainy River deposits incorporate dilution and mining recovery factors based on the selected mining methods and design. An economic analysis was completed to validate the economic viability of all areas of the Mineral Reserves.

15.3.1 Estimation procedure

The most up-to-date mineral resource block models (as of December 31, 2017) were used to estimate the mineable tonnage in the mine plan for the Rainy River and Intrepid deposits (see Item 14 for more details on the block models).

For the Rainy River deposit, two COGs were calculated for the UG Main Zones (all underground reserves below the pit). The first, applicable during Phase 1, considers the feed from the LG Stockpile and the mill at full capacity (until 2028). The second, for Phase 2, reflects underground production only and the associated downsizing of the mill after depletion of the stockpile. A third and separate COG was calculated for the satellite Intrepid deposit during the 2021 optimization process, which converted the lower part of the zone to reserves. The main reason for having a separate COG is the higher mining cost for Intrepid ore.

Stope shapes were optimized based on their respective COGs. Optimizations are a function of various parameters, such as geometry and dilution. The final Mineral Reserve estimate was obtained after completing the stope and underground mine designs and the economic validation, while considering additional factors such as mining recovery.

15.3.2 Underground COG

A cut-off grade is a fundamental component in mineral reserve estimations, mine designs and mine production schedules. The COG calculations for the Rainy River and Intrepid deposits use parameters that were provided by the issuer or obtained from previous InnovExplo estimates.

As described in 15.3, two COGs were used for stope optimization in the UG Main Zones to reflect the pre- and post-stockpile depletion phases. A different COG was calculated for Intrepid using a separate set of parameters.

The COG for the Main Zones considers a combined production from the open pit LG stockpile at surface and the orebody underground. The second COG applies to underground production only. The COG for the underground operation before and after the depletion of the LG stockpile is listed below. After open pit mining ceases in Q2 2026, the mill feed will comprise the LG stockpile and underground ore from the Main and Intrepid zones. When the stockpile is depleted by Q2 2028 and production is entirely from underground ore, the COG will change from 1.74 g/t AuEq to 2.25 g/t AuEq.

The COG for the Intrepid Zone is 1.93 g/t Au. It will be constant over the life of the orebody until production ceases in 2028.

The table of COG assumptions for the UG Main Zones (Estimation procedure) show that the main differences between Phase 1 and Phase 2 lie Item 15.3 in the processing costs. The differences for the Intrepid Zone were inevitable given its distance to the mill and differences in the mining methods, resulting in a higher ore overall mining cost. Other assumptions include a gold price of US$1,400/oz, a silver price of US$19/oz (not shown in the summary), a metallurgical recovery of 94% and an estimated mining dilution of 14%.

Table 15.5 – Cut-off grade parameters for the underground zones

Parameters	Units	Intrepid	UG Main Zones	
			Phase 1 : With LG Stockpile	Phase 2 : No LG Stockpile
Gold price	$US/oz	1,400	1,400	1,400
Exchange rate	$US/$US	1.25	1.25	1.25
Royalty	%	0.81%	0.81%	0.81%
Royalty	$US/oz	11.30	11.29	11.29
Refining cost	$US/oz	5.34	5.34	5.34
Cost of selling	$US/oz	16.63	16.63	16.63
Total processing cost	$US/t processed	10.00	10.00	18.15
Metallurgical recovery	%	94.0%	94.0%	94.0%
Mining recovery	%	95.0%	95.0%	95.0%
Mining dilution	%	14.00%	14.00%	14.00%
Ore mining cost	$US/t processed	55.57	48.73	48.73
Administration & General	$US/t processed	1.83	1.83	11.56
Total Cost	$US/t processed	67.39	60.55	78.44
Cut-Off Grade AuEq		1.93	1.74	2.25

The COG calculation is appropriate for the deposits based upon the assumptions used and the current company strategy relative to metal prices and combined open pit and underground operations.

15.3.3 Dilution Factor Calculation

Internal dilution involves optimizing stope shapes and converting them into planned mineable shapes. External dilution is also considered for the production stopes using average ELOS values (equivalent linear overbreak slough) of HW = 0.6 m and FW = 0.3 m. These estimated values were used based on experience in similar conditions or on similar mining projects; the gap in geotechnical knowledge associated with the underground portion of the project did not allow each stope to be characterized based on location and rock mechanic properties.

Backfill dilution was added afterward, based on the location of each stope and the mining sequence for each mining horizon.

The following parameters were used to estimate stope dilution:

- Dilution is expected to come from the hanging wall (0.6 m) and footwall (0.3 m) for most stopes in the Main Zones.
- A backfill dilution of 0.5 m is estimated for each exposed face of a backfilled stope.
- A lower recovery is assumed for sill pillar stopes caused by expected geomechanical challenges.
- Secondary ground support (cables) is planned to reduce dilution for specific stopes (poor conditions and / or width > 12.0 m).
- No dilution is assumed for the stope floors.
- For internal, external, and backfill dilution, the calculations assumed a dilution grade of 0 g/t.

In summary, an estimated external waste dilution of 14% was used in the COG calculations for both the Main Zones and Intrepid.

15.3.4 Mining Losses

Mining loss (or mining recovery) is based on the material in the model that is left behind, for example, to provide structural support when facing blasting or operational challenges and rock mechanics issues.

The estimated mining recoveries for all underground deposits is estimated at 95%. Recovery varies mainly according to blasting method and the associated challenges, as well as rock mechanic conditions such as sill pillar recovery. Recovery values are based on experience and estimates for typical stopes. The factors are considered acceptable given the selected mining method and known ground conditions.

The geological block model was the primary input in Deswik Shape Optimizer (DSO) version 2021.2. Deswik software is used to optimize individual stope shapes from the block model using Stope Shape Optimizer algorithms from Alford Mining System. The parameters used for the optimization are listed in Item 16.

15.3.5 Cut-off Grade (COG)

A COG of 1.74 g/t AuEq was used to estimate the Mineral Reserves for Phase 1 (UG plus LG Stockpile) based on the cost estimates, metal prices and exchange rate summarized in Estimation procedure.

A COG of 2.25 g/t AuEq was used to estimate the Mineral Reserves for Phase 2 (after depletion of the LG Stockpile) based on the cost estimates, metal prices and exchange rate summarized in Estimation procedure.

A COG of 0.83 g/t AuEq was used for development ore, recognizing this as incremental material that must be mined during the stope development process.

A COG of 1.93 g/t AuEq was used to estimate the Mineral Reserves for Intrepid based on the cost estimates, metal prices and exchange rate summarized in Estimation procedure.

15.4 Open pit and underground Mineral Reserve estimates

Open pit and underground Mineral Reserves at Rainy River are summarized in Table 15.6.

Table 15.6 – Mineral Reserve Estimates – Effective December 31, 2021

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Direct processing Mineral Reserves					
Open pit					
Proven	9,486	1.26	2.2	385	657
Probable	21,861	1.20	2.2	845	1,513
Sub-total open pit	31,347	1.22	2.2	1,230	2,170
Stockpile					
Proven	1,247	0.65	2.5	26	98
Probable	-	-	-	-	-
Sub-total stockpile	1,247	0.65	2.5	26	98
Underground					
Proven	-	-	-	-	-
Probable	12,657	3.05	7.6	1,241	3,084
Sub-total underground	12,657	3.05	7.6	1,241	3,084
Total direct processing Mineral Reserves	45,251	1.72	3.7	2,498	5,353
Low grade Mineral Reserves					
Open pit					
Proven	2,982	0.36	1.7	34	164
Probable	9,426	0.36	1.9	108	588
Sub-total open pit	12,408	0.36	1.9	142	752
Stockpile					
Proven	12,561	0.39	2.3	159	918
Probable	-	-	-	-	-
Sub-total stockpile	12,561	0.39	2.3	159	9218
Total low grade Mineral Reserves	24,969	0.38	2.1	301	1,670
Total Mineral Reserves					
Open pit (including stockpile)					
Proven	26,276	0.72	2.2	605	1,837
Probable	31,288	0.95	2.1	953	2,101
Sub-total open pit	57,563	0.84	2.1	1,558	3,938
Underground					
Proven	-	-	-	-	-
Probable	12,657	3.05	7.6	1,241	3,084
Sub-total underground	12,657	3.05	7.6	1,241	3,084

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Total					
Proven	26,276	0.72	2.2	605	1,837
Probable	43,944	1.55	3.7	2,194	5,185
Total Mineral Reserves	70,220	1.24	3.1	2,799	7,022

Notes:

1. CIM Definition Standards for Mineral Resources and Mineral Reserves (2014) were used for reporting of Mineral Reserves.
2. Mineral Reserves are estimated using a long-term gold price of US$1,400 per troy oz and a long-term silver price of US$19 per troy oz. The exchange rate used was 1:1.25 US$:C$.
3. 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) * 19 * 60)/ (1,400 * 90)].
4. Open pit assumptions include:
    - COGs applied to a regularized 10 m x 10 m x 10 m mine planning block model, which was generated from re blocking the original resource model. Modifying factors representing a potential dilution of 3.3 m and ore loss of 0.2 m were applied, including a factor of 0.89 applied against the gold grade in the East Lobe.
    - Metal recoveries are variable dependent on metal head grade. At Mineral Reserve COG, the gold recoveries are as follows:
        a. DPO
           CAP zone gold = 73.9%
           Non-CAP zone gold: ODM=83.7%, 433=92.0%, HS=85.4%
        b. LGO
           CAP zone gold = 73.1%
           Non-CAP zone gold: ODM=78.5%, 433=91.3%, HS=81.2%
        c. Average gold and silver recoveries of 90% and 60%, respectively, have been used for the gold equivalency calculation.
5. Underground Mineral Reserves for UG Main are estimated at an AuEq COG of 1.74 g/t for Phase 1, AuEq COG of 2.25 g/t for Phase 2, and 0.83 g/t for development. Underground Mineral Reserves for Intrepid are estimated at an AuEq COG of 1.93 g/t.
6. Underground assumptions include:
    - In UGMain Zones and Intrepid, the hanging wall (HW) and footwall (FW) dilution of 0.6 m and 0.3 m, respectively, with total unplanned dilution of 14% approximately.
        a. Average mining recovery estimated as 95% for UG Main Zones and Intrepid.
        b. Average gold and silver mill recovery of 95% and 60%, respectively, for UG Main Zones and Intrepid
    - Cut-off value of CDN$84.24/t, CDN$75.69/t & CDN$98.05/t (Intrepid, UG Main Phase 1 & Phase 2, respectively), inclusive of costs for mining, processing, General and Administrative (G&A), refining & transport and royalties
7. The qualified persons responsible for this item of the technical report are not aware of any mining, metallurgical, infrastructure, permitting or other relevant factors that could materially affect the mineral reserve estimates.
8. Effective date of Mineral Reserves is 31 December 2021.
9. Totals may not add exactly due to rounding.

The Mineral Reserves reported herein supersede the Mineral Reserves reported previously at year-end 2020 by New Gold for the Rainy River Mine.

15.5 Comparison with previous Mineral Reserve estimates

The most recent Mineral Reserve estimate published by New Gold was in a press release titled "New Gold Provides 2021 Operational Estimates for the Rainy River Mine and Updated Mineral Reserves and Mineral Resources", dated February 10, 2021. The Mineral Reserve statement was effective December 31, 2020.

The current Mineral Reserve estimate described in this NI 43-101 Technical Report is effective December 31, 2021.

Table 15.7 and Table 15.8Comparison with previous Mineral Reserve estimates provide a comparison of the end-2020 and end-2021 Mineral Reserve estimates for the open pit and underground, respectively.

Table 15.7 – 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 2020					
Proven	28,320	0.82	2.2	746	2,009
Probable	40,198	0.91	2.6	1,180	3,348
Total open pit	**68,517**	**0.87**	**2.4**	**1,926**	**5,357**
Effective 31 December 2021					
Proven	26,276	0.72	2.2	605	1,837
Probable	31,288	0.95	2.1	953	2,101
Total open pit	**57,563**	**0.84**	**2.1**	**1,558**	**3,938**
Difference over 2020					
Proven	-7%	-12%	-	-19%	-9%
Probable	-22%	4%	-19%	-19%	-37%
Total open pit	**-16%**	**-3%**	**-13%**	**-19%**	**-26%**

Note: Totals may not add exactly due to rounding.

Changes to the open pit Mineral Reserve estimate from end-2020 to end-2021 are due predominantly to:

- 2021 Mineral Reserve depletion from mining activities of 9.2 Mt @ 0.88 g/t gold and 3.4 g/t silver, totalling 262 koz of contained gold and 1,010 koz of contained silver.
- Updated costs reflecting the current cost of operation (mine, process, G&A, and relevant sustaining capital requirements). Overall, mining costs have increased by approximately 10%accompanied by a reduction in processing and G&A costs by 3% and 5%, respectively.
- Updated geotechnical model having variable impact across various pit sectors.

- Updated mine planning resource model with modifying factors impacting dilution and ore loss.

The net result of these items has been a decrease in the Mineral Reserves, primarily a result of 2021 Mineral Reserve depletion.

Table 15.8 – Comparison with previous Mineral Reserve estimates - Underground

Category	Tonnes & grade			Contained metal	
	Tonnes (000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)
Underground					
Effective 31 December 2020					
Proven	-	-	-	-	-
Probable	5,399	3.87	10.3	672	1,795
Total underground	5,399	3.87	10.3	672	1,795
Effective 31 December 2021					
Proven	-	-	-	-	-
Probable	12,657	3.05	7.6	1,241	3,084
Total underground	12,657	3.05	7.6	1,241	3,084
Difference over 2018					
Proven	-	-	-	-	-
Probable	234%	-21%	-26%	185%	172%
Total underground	**234%**	**-21%**	**-26%**	**185%**	**172%**

Note: Totals may not add exactly due to rounding.

The changes to the underground Mineral Reserve estimates between year-end 2020 (previous estimate) year-end of 2021 (current estimate) are primarily a reflection of the changes in the COGs and the re-design of the underground components of the Rainy River mine. The underground reserves are mined alongside the open pit and stockpile reserves until the stockpile is completely depleted.

In 2021, development started at Intrepid and a new feasibility study was completed for the UG Main Zones. Under the current mine schedule at a rate of 4,500 tpd, the LOM of the underground mine extends until 2031, three years after the stockpile is completely depleted.

InnovExplo recognizes that there is a significant quantity of what is currently considered marginal material in underground Mineral Resources and recommends that regular reassessment of that material be undertaken relative to the metal price environment and company strategy.

15.6 Conversion of Mineral Resources to Mineral Reserves

Table 15.9 shows the proportions of total Measured and Indicated Mineral Resources converted to Mineral Reserves in terms of contained gold ounces.

Table 15.9 – 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)	
Open pit				
Measured	612	Proven	605	99%
Indicated	1,159	Probable	953	82%
OP M&I	1,770	OP P&P	1,558	88%
Underground				
Measured	-	Proven	-	-
Indicated	2,306	Probable	1,241	54%
UG M&I	2,306	UG P&P	1,241	54%
Total M&I	**4,077**	**Total P&P**	**2,799**	**69%**

Note: Totals may not add exactly due to rounding.

15.7 Factors that may affect the Mineral Reserves

Areas of uncertainty that may materially impact the Mineral Reserve estimate include the following:

- Commodity prices, market conditions and foreign exchange rate assumptions
- Cut-off grade estimates
- Capital and operating cost assumptions
- Geological complexity and mineral resource block modelling
- Slope and stope stability, dilution and mining recovery factors
- Metallurgical recoveries and contaminants
- Rock mechanics (geotechnical) constraints and the ability to maintain constant underground access to all working areas

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

16 MINING METHODS

16.1 Introduction

Mining at Rainy River currently uses open pit methods. It will transition into a combined open pit and underground (OP+UG) operation. Underground development commenced at Intrepid in 2021. The average processing rate will be approximately 27,000 tpd while the open pit ore is processed. When all the open pit ore, including the stockpiled ore, has been exhausted, the only mining operations will be underground. During this phase, UG ore will be processed at a rate of approximately 4,500 tpd.

Over the LOM, the open pit (including stockpile rehandling) and underground operations are scheduled to provide 82% and 18% of the processed ore tonnage, with 56% and 44%, respectively, of the processed contained gold ounces.

The open pit mine is a conventional truck and shovel mining operation, with a fleet of 220t payload haul trucks combined with diesel-powered hydraulic excavators and large front-end loaders (FELs) as primary loading units. The open pit operates at an annualized peak mining rate of 153,000 tpd of ore, and the waste has an overall strip ratio of 2.32:1 (waste:ore).

The plan for the Main Zones underground operation is to access the various ore zones from two portals. The IntrepidDeposit will be accessed from one portal, with a decline started in 2021 (still under development as part of an orebody investigation project).

The underground operations will be accessed via declines and follow a mechanized longitudinal long-hole open stoping technique to mine the underground Mineral Reserves. Underground ore production rates will be variable but will reach the planned maximum of approximately 5,500 tpd in 2026.

The combined OP+UG operations have a remaining mine life through to Q4-2031.

16.2 Open pit mining

16.2.1 Production to end 2021

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-2021 are provided in Table 16.1 and Table 16.2, respectively.

Table 16.1 – Open pit mine production to end-2021

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
2020	11,777	0.73	2.5	276	958	39,354	51,131	3.34
2021	14,496	0.71	3.3	331	1,543	39,206	53,7023	2.7
Total	47,207	0.92	2.5	1,402	3,850	147,227	194,434	3.12

Note: Totals may not add exactly due to rounding.

Table 16.2 – Mill production to end-2021

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
2020	8,819	0.91	2.5	229	362
2021	9,250	0.88	3.4	234	611
Total	33,615	1.01	2.5	973	1,547

Note: Totals may not add exactly due to rounding.

16.2.2 Hydrologic considerations

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, overburden dewatering wells, 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.

Localized loss of surface water control above the crest of the ODM shear zone area has occurred where the rock mass is more altered and weaker, leading to operational disruption at times in active lower benches and the pit bottom. This zone along the West wall should be reviewed to identify any alteration / weathering influence or faults which could exacerbate the loss of surface water control. Where possible, the lower benches of the mine should not be advanced during freshnet periods and preferabley used as in-pit sumps for water collection and pumping.

16.2.3 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 six design sectors based on the bedrock geometry and interpreted clay thickness. The design sectors are as follows:

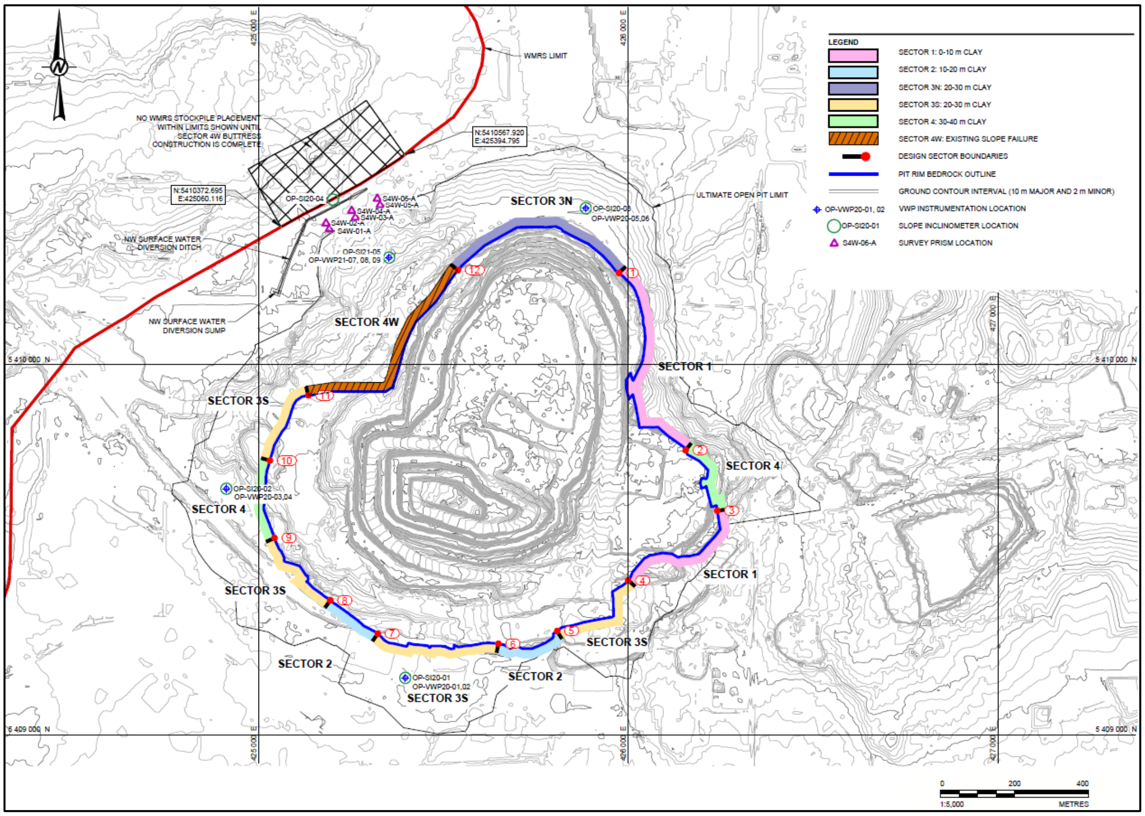
- Sector 1: 0 m – 10 m clay thickness
- Sector 2: 10 m – 20 m clay thickness
- Sector 3S and 3N: 20 m – 30 m clay thickness
- Sector 4 and 4W: 30 m – 40 m clay thickness

A rockfill toe berm has been designed for all the sectors. Design Sectors 3 and 4 include an inset upper bench. The advantage of the rockfill toe berm and upper bench 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 berm and upper bench is required for longer term stability. Excavation of the final overburden slopes should be top down to prevent excessive strain. Placement of the rockfill toe berm and upper bench 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 excavation slopes that vary between 4H:1V to 8H:1V and the bottom sand till deposit (WST) has a constant design excavation slope of 3H:1V for all the sectors. The summarized overburden slope design 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. A 10 m height benched excavation of the Sectors 3S and southern Sector 4 overburden slopes is proposed to provide a more cost-efficient overburden excavation technique, while satisfying the overall design slope.

Design Sector 4W was developed in response to a retrogressive slumping of the overburden clay slope observed in the northwest extent of the open pit. The design modification includes an increased waste rock buttress keyed down to bedrock to support the as-built slope geometry.

The open pit overburden slopes utilize an extensive geotechnical instrumentation monitoring and surveillance system to continually assess the performance of the slopes. Instrumentation includes slope inclinometers, monitoring prisms, and vibrating wire piezometers. Trigger action response plans are in place for the instrumentation to inform mine operations should any abnormal measurements be detected so that appropriate actions could be taken to ensure safe construction of the facilities.



Source: Golder 2020b.

Figure 16.1 – Overburden slope design sector layout plan

Table 16.3 – Summary of design geometries

<table>
<tr><th colspan="2" rowspan="2">Design geometries</th><th colspan="6">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><th>Sector 4W</th></tr>
<tr><td rowspan="2">Overburden cut slope grade</td><td>Through WST unit</td><td colspan="5">3H:1V</td><td>-</td></tr>
<tr><td>Through Clay units (BRE, WML, and WYL)</td><td>4H:1V</td><td>4H:1V</td><td>7H:1V</td><td>7H:1V</td><td>8H:1V</td><td>In-Place</td></tr>
<tr><td rowspan="5">Rockfill berm</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><td>45 m</td></tr>
<tr><td>Slope grade</td><td colspan="6">1.3H:1V</td></tr>
<tr><td>Upper buttress distance below the crest</td><td>2.5 m</td><td>6 m</td><td>8 m</td><td>8 m</td><td>8 m</td><td>0 m</td></tr>
<tr><td>Upper bench height</td><td>Varies</td><td>Varies</td><td>7 m</td><td>7 m</td><td>7 m</td><td>6 m</td></tr>
<tr><td>Upper bench crest width</td><td>Varies</td><td>Varies</td><td>80 m</td><td>80 m</td><td>80 m</td><td>50 m</td></tr>
</table>

Notes: WST = Whiteshell Till; BRE = Brenna Formation; WML = Whitemouth Lake Formation; and WYL = Wylie Formation

16.2.4 Open pit geotechnical considerations – hard rock

SRK carried out an open pit slope stability assessment and design update in 2019. The work 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:

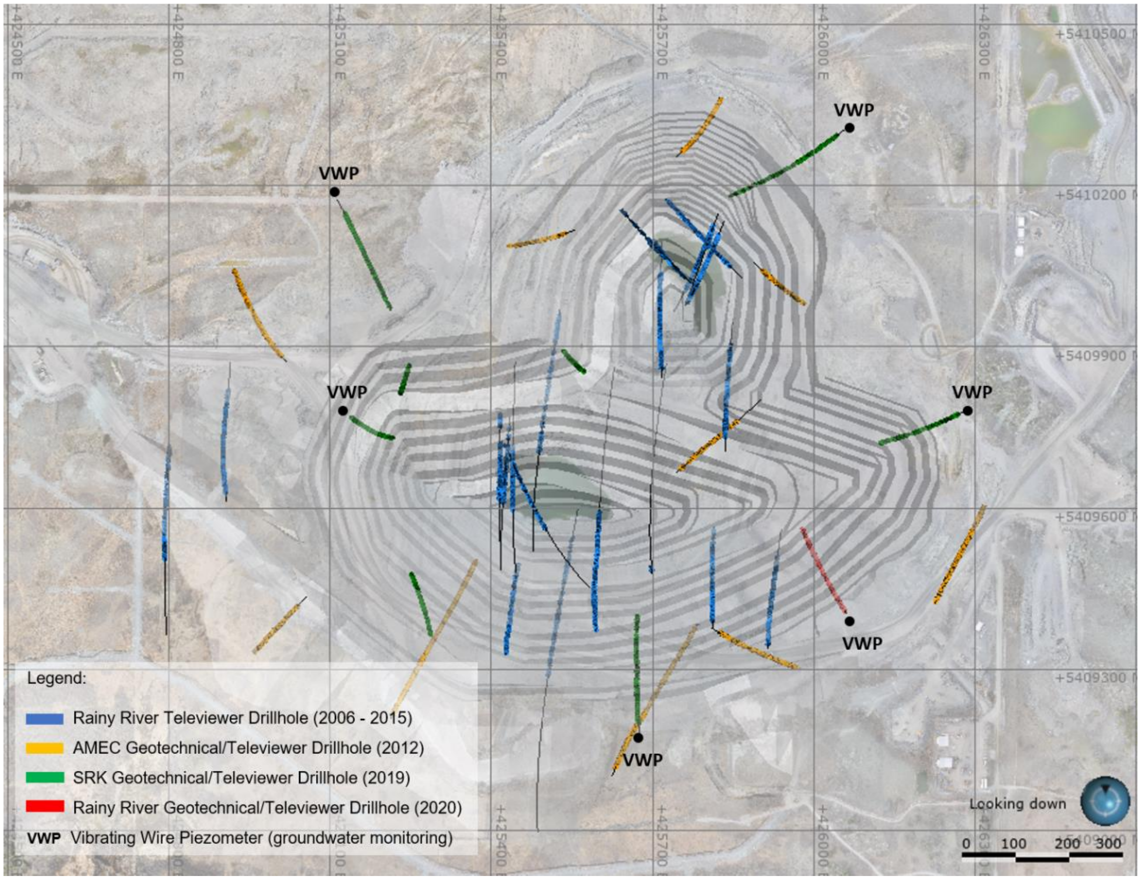
- The orientation of the regional south-southwest dipping foliation structures (north walls).
- The kinematic stability related to the major joint sets (all pit walls).

Ongoing stability and design work is completed with respect to the implementation of the current slope design. Currently, there are recommendations to perform blast trials to evaluate potential back-break and bench-scale rock hazards through the IMV prior to excavation in the southwest design sectors. Based on these trials there maybe requirements to modify the design recommendations to improve performance and safety around the planned Phase 4 southwest ramps.

The design work carried out to date does not account for the in-pit portal locations, and there may be instances where adjustments are required to achieve the long-term stability and access objectives for the underground.

16.2.4.1 Field and laboratory 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. Prior to this, targeted geotechnical drilling and televiewer surveys were conducted during the feasibility project. Since 2019, investigations comprised eight diamond drill holes (DDHs), geotechnical logging and field testing, hydrogeological testing, and five nested vibrating wire piezometer (VWP) installations. The locations of the previous geotechnical drillholes and televiewer surveys are shown in Figure 16.2.



Note: Orientation data annotated in legend colour, with the drillhole traces in black.
Source: SRK 2022.

Figure 16.2 – Geotechnical drillholes and televiewer surveys completed between 2006 and 2020

16.2.4.2 Stability assessment

A rock slope stability assessment was carried out using multiple approaches with a combination of software packages, as summarized in Table 16.4.

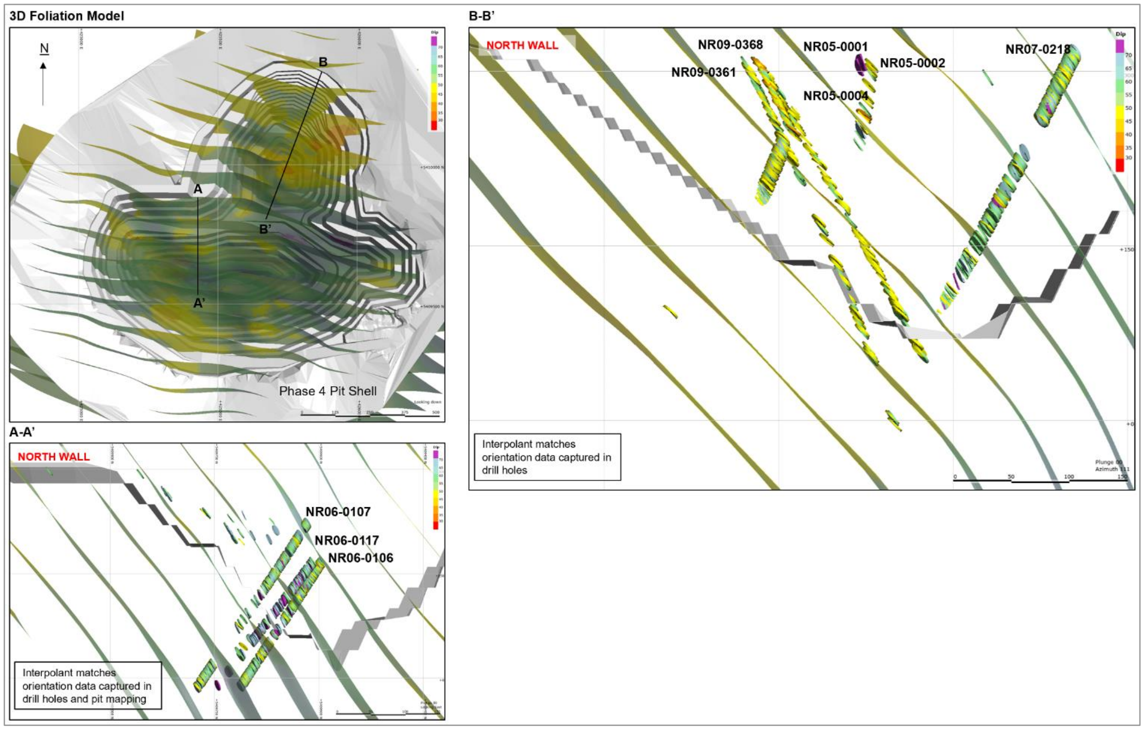
Table 16.4 – Overview of stability assessment approach and software

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™

16.2.4.3 Foliation and structural geology 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 processed in Leapfrog™ to generate a 3D model, as shown in Figure 16.3.

In 2021, SRK has updated the 3D brittle-structural model using drillhole and pit mapping data. The structural model forms the basis of the ongoing kinematic stability work.



Source: SRK 2020.

Figure 16.3 – Pit scale 3D foliation model

16.2.4.4 Kinematic and overall stability assessments

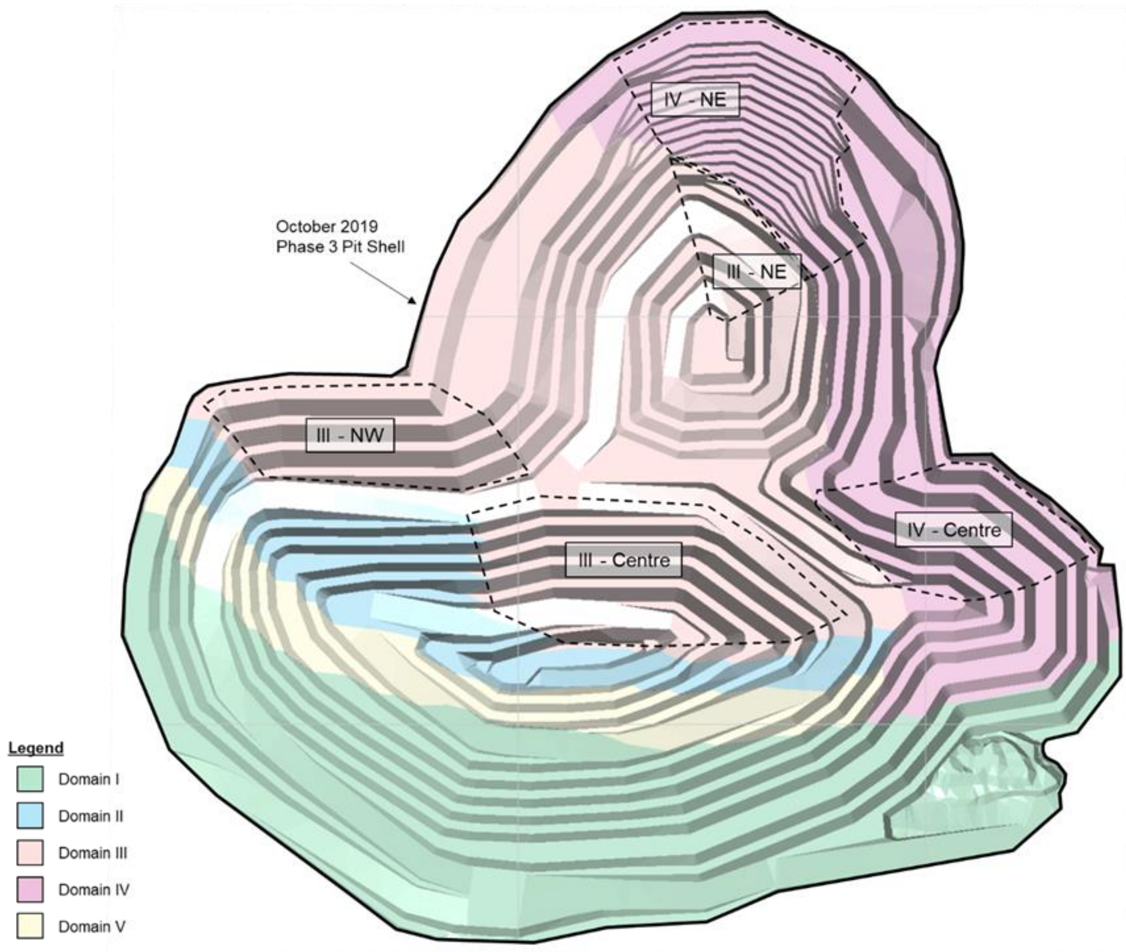
Kinematic analyses were carried out using Dips™ and SBlock™ for defined litho-structural domains and all applicable slope face directions. The analyses were carried out for 30° segments to identify the potential kinematic failure modes that could limit the design. Both wedge intersection and planar sliding mechanisms were assessed to be high risk at the bench scale for most of the analyzed slope aspects. These higher risks indicate that strict management practices are requirement to implement the design configuration and reduce residual rock fall risks. The ongoing kinematic stability work includes the updated structural geology model (SRK, 2021).

In 2020, two-dimensional slope stability analyses were used to evaluate the expected design rock slope stability conditions. The analyses were conducted using Slide2D™ and RS2™ (SRK, 2020). The stability analyses considered the potential for overall non-circular failure through the anisotropic rock mass. These higher risks indicate that strict management practices are requirement to implement the design configuration and reduce residual rock fall risks. The ongoing kinematic stability work includes the updated structural geology model (SRK, 2021).

16.2.4.5 Rock slope design criteria

Pit slope design recommendations are presented in Table 16.12 and are based on the litho-structural domains shown in Figure 16.4.

Note that the 62° BFA recommendation presented in Table 16.12 was modified by New Gold to 65° in their open pit designs to reflect drill fleet capabilities. Inter-ramp angles were maintained.



Source: SRK 2020.

Figure 16.4 – Litho-structural design domains

Table 16.5 – 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	62	30	14.0	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	62	30	12.0	47			Bench scale wedge intersection on JS3 / JS5.
	160	230	62	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	62	30	13.0	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

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)	
									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)	30	10.5 (NW)	44 (NW)			Orientation of south-dipping Foliation (FOL) structures. Design configuration to limit planar sliding mechanisms. Benched along foliation structures.
			62 (Centre)	30	12.0 (Centre)	47 (Centre)			
			50 (NE)	10	5.0 (NE)	37 (NE)			
	230	250	62	30	15.0	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) 62 (Centre)	10 (NE) 30 (Centre)	75.0 (NE) 12.0 (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.
Source: SRK 2020

In addition, the following design guidelines were provided:

- 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 and 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.

16.2.5 Open pit mine design

The mine design was developed on the optimized pit shell detailed in Item 15 by rationalizing the shape into a feasible mining geometry and incorporating haulage ramps and detailed slope design criteria as presented in Item 16.2.3 and Item 16.2.4. 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.

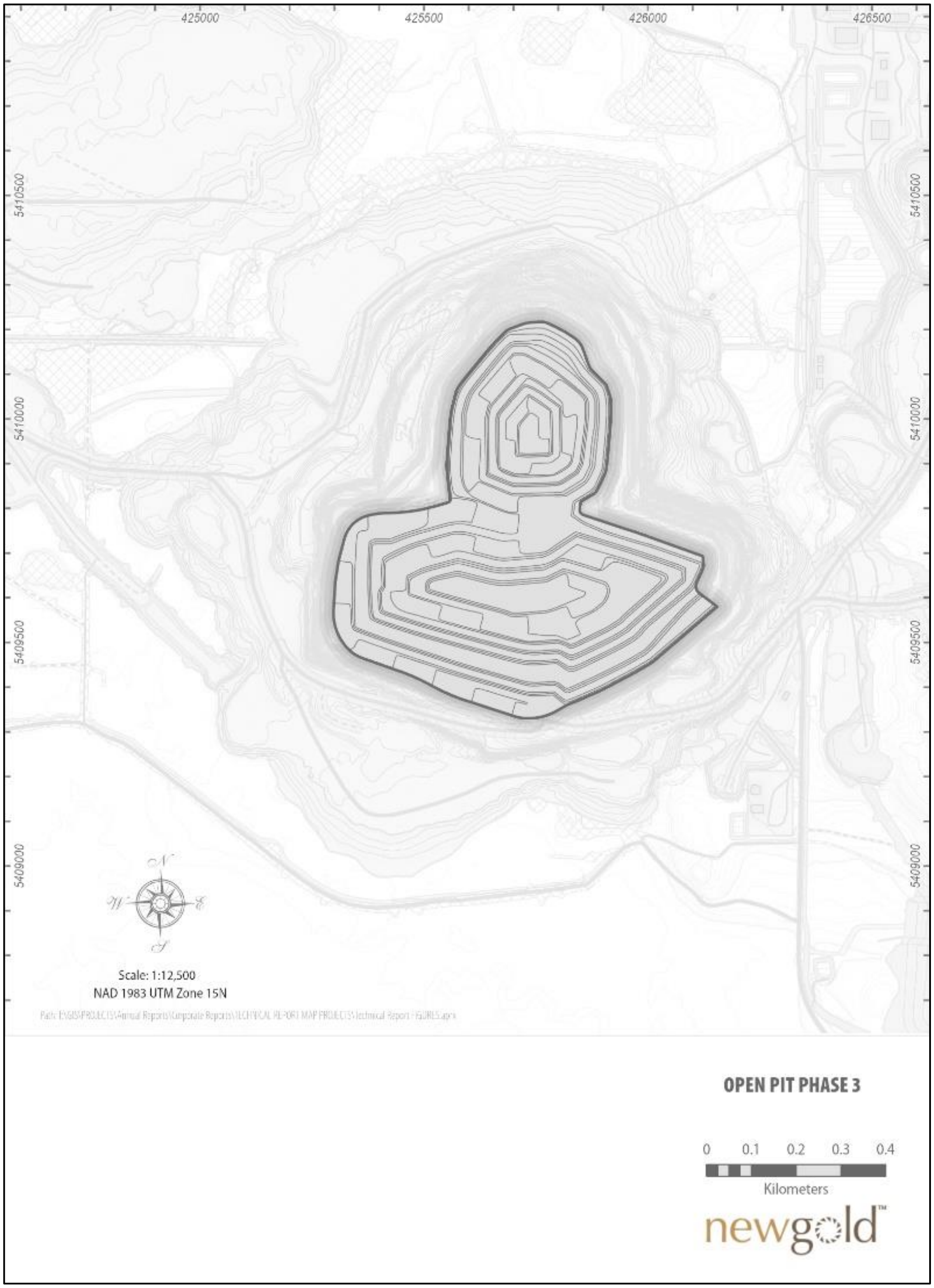
Following the design of the ultimate pit, the pit was subdivided into a series of mining phases. Phases are mining shapes, which except for the final pit limit, never exist exactly as depicted during the LOM. The phases determine the conceptual development of the open pit, with the objective of outlining 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.

Two additional mining phases were developed by New Gold beyond the current Phase 1 and Phase 2 which have already been excavated. These are designated Phase 3, and Phase 4, with Phase 4 also representing the final pit limit design. Figure 16.5 through Figure 16.6 illustrate the phase designs developed for Rainy River.

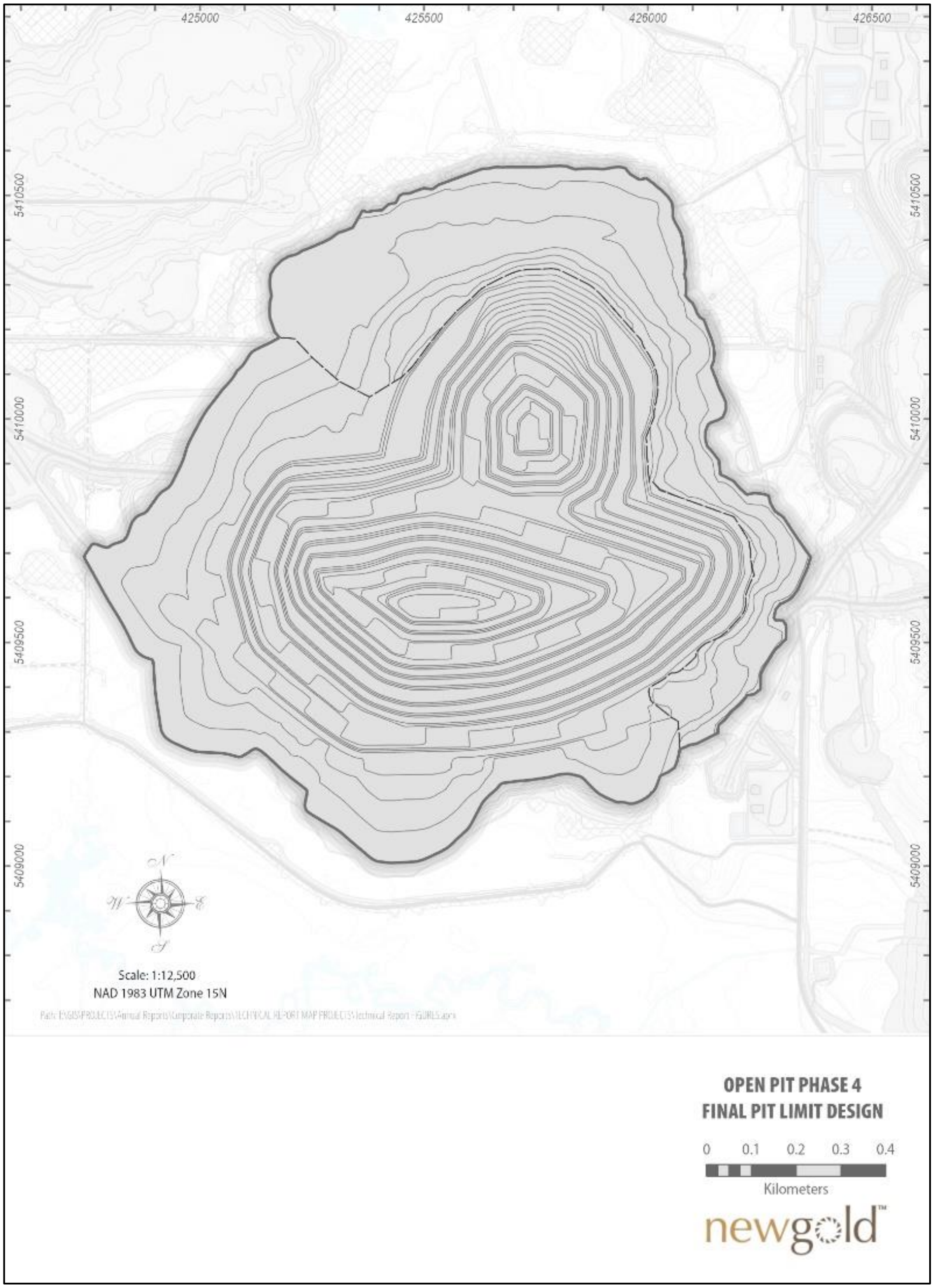
AMC has reviewed these phases and they appear reasonable. It is noted that there is only a single-ramp access to the pit bottom which may impact future operations in the case of a geotechnical event on the Phase 4 south and / or west wall. A second-ramp access should be considered, pending the results of a risk assessment.

It is also recommended that further optimization should be undertaken to confirm the best pit bottom elevation upon which to transition from open pit mining to underground. This will permit additional pit limit optimization related particularly to the south-west area of the final pit limit design.



Source: New Gold 2022.

Figure 16.5 – Open pit Phase 3



Source: New Gold 2022.

Figure 16.6 – Open pit Phase 4

16.2.6 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 153,000 tpd of ore and waste and has an overall strip ratio of 2.32:1 (waste:ore).

16.2.6.1 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, three Sandvik DI650i, and one DR580 down-the-hole drills which drill 171 mm diameter drillholes. Blasthole drills are configured to drill the 10 m height of the bench plus 0.5 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 DI650is and the DR580 drills with a 140 mm diameter drillhole spaced every 1.8 m linearly.

Where more maneuverability is required for pioneering on overburden / bedrock, the Sandvik DI650i and DR580 drills may be utilized.

Drill productivities are estimated at a rate of 21 m/operating hour.

16.2.6.2 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, blasting pattern, and blasting domain.

16.2.6.3 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) to assist primarily with rehandle. Preferentially, the PC8000 is attempted to be scheduled in waste and / or large continuous ore blocks, with the PC5500s scheduled primarily in ore where improved selectivity is required and also in waste as necesary. The FELs, due to their mobility, are assigned to ore or waste as required and are utilized for stockpile rehandle.

An additional Komatsu PC3000 (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.

16.2.6.4 Hauling

Hauling is performed by a fleet of Komatsu 830E / 830E-AC electric drive rear-dump haul trucks in the 220 tonnes 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 TMA for construction purposes.

16.2.7 Mine planning

The mine plan is executed to take advantage of the installed mine fleet productive capacity, allowing an elevated COG policy to be employed, 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. As it is not always possible to separate the DPO from the LGO in the field resulting in a blending of the material types, the current mine plan includes an increased proportion of LGO stockpiles being rehandled and blended with DPO on an annual basis, to better reflect operational experience.This results in an open pit mine life extending to Q1 - 2025 with stockpile rehandling occurring in parallel to the underground operations through to Q4 - 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. Ore sent to stockpile is included at approximately 47% of ore mined on average during peak mining years. The amount of material delivered to stockpile is under continuous review pending operational realities of the mine and evolving economic conditions.

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. Additional waste rock rehandle is included in the mine plan for in-pit and ex-pit road armouring, safety berm construction, etc.

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 and is stored at three locations, the East Mine rock stockpile (EMRS), the West Mine rock stockpile (WMRS) and the In-Pit Mine rock stockpile (IPRS). The IPRS will be commissioned in the north lobe of the open pit upon completion of mining in this area at end-2022.

The EMRS is designed to accommodate a combination of overburden and either PAG or NAG waste mine rock, while the WMRS is designed to accommodate a combination of overburden waste and NAG waste mine rock. The IPRS is designed to accommodate principally PAG but can accommodate NAG if required. See Item 18 for details. 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.

NAG requirements for TMA construction are currently fulfilled from in-pit mine production. NAG has been found to be 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 past. There is sufficient NAG to fulfill the LOM TMA construction schedule requirements after taking into account NAG recovery factors. However, NAG quantities being extracted from the mine after 2023 will be significantly reduced and it is recommended that New Gold review mitigating strategies to ensure sufficient quantities are available when required, 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. No further mining of the EOC for NAG construction rock is included in the current mining schedule.

Table 16.6 presents the open pit mine production schedule.

Table 16.6 – Open pit mine production schedule

Year	Ore tonnes	Grade		Contained metal		Waste tonnes	Total tonnes	Strip ratio	Re-handle tonnes[1]	Total tonnes moved (000s)
	(000s)	Gold (g/t)	Silver (g/t)	Gold (koz)	Silver (koz)	(000s)	(000s)	(w:o)	(000s)	(000s)
2022	12,994	0.90	2.1	376	871	42,972	55,966	3.3	6,507	62,473
2023	12,400	0.84	2.3	335	928	41,280	53,680	3.3	4,388	58,067
2024	15,688	1.13	1.9	571	963	16,868	32,556	1.1	2,975	35,531
2025	2,674	1.05	1.9	90	160	581	3,255	0.2	7,473	10,728
2026	-	-	-	-	-	-	-	-	8,578	8,578
2027	-	-	-	-	-	-	-	-	8,504	8,504
2028	-	-	-	-	-	-	-	-	5,713	5,713
Total	**43,755**	**0.98**	**2.1**	**1,373**	**2,922**	**101,702**	**145,457**	**2.32**	**43,738**	**189,195**

Notes: Totals may not add exactly due to rounding.
[1] Rehandle tonnes include in pit and ex-pit rehandle. Excludes underground ore rehandle from portal stockpiles.

16.2.8 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. 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 16.7.

Table 16.7 – Peak principal open pit mining equipment requirements

Description	Manufacturer	Model	Units
Production drill	Sandvik	DI650i	3
Production drill	Sandvik	DR580	1
Production drill	Sandvik	D75KS	4
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	24
Dozer	Komatsu	D475	2
Dozer	CAT	D10T	3
Dozer	CAT	D9T	3
Dozer	CAT	D8T	1
Dozer	CAT	18M	1
Grader	CAT	16M	3
Excavator	CAT	390F	1
Tire handler	Komatsu	WA600	1

The peak open pit mining equipment requirements correspond to the 2022 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 maintenance equipment, FEL's, trucks, crew buses, lighting plants, compactors, etc.

No further additional nor replacement open pit mine principal equipment fleet is considered for purchase during the remaining LOM plan.

16.3 Underground Mining

The Rainy River underground mine has been designed to create a safe working environment. The planned methods and technologies will provide security for New Gold workers and the company's assets. Mining at Rainy River is currently conducted using open pit mining methods. It will transition into a combined OP+UG operation over seven years, with underground production commencing in 2022 from the Intrepid deposit. An average processing rate of approximately 27,000 tpd is scheduled over the LOM until the low-grade (LG) stockpile is depleted. The underground mine will continue for another 4 years at an average rate of 4,500 tpd for the UG Main Zones (below the pit) and 850 tpd for Intrepid.

The Rainy River mine will thus entail a continuation of the existing OP operation, plus two UG operations using mining methods optimized to the deposit's geometry and employing long-hole (longitudinal and transverse) stoping methods. The initial main ramp will be excavated from the middle of the in-production pit to access the UG Main Zones below the pit (180 RL). Starting at the 17 East portal, the primary purpose of the ramp will be to reach the ventilation infrastructures and start pre-production in the ODM zones as soon as possible. Secondary ramps will provide access to the ODM East, Zone 433 and ODM Main Zones from the main ramp. Other secondary ramps will connect ODM Main to ODM West and ODM East to 17 East Lower. From the portal to the ODM Main ventilation raise, the total decline development length is estimated at 1,893m. The numerous ventilation and development phases are highlighted in Item 16.3.9 - *Mine* .

Once open pit production finishes, a second main ramp will be developed at the bottom of the pit to reach ODM Main. Both ramps will sustain material handling, personnel and equipment, but most of the production will be hauled through the ODM Main portal when available. The total projected underground development was optimized to benefit from leading-edge methods and technologies. The total project will entail around 65 km of development (horizontal and vertical) to access all six UG Main Zones: 17 East Lower, 17 East Upper, ODM East, ODM Main, ODM West and Zone 433. Zone sizes vary, but the average dips for most are between 50° and 85°.

The Intrepid Zone is the only zone with an existing portal. It is independent of the UG Main Zones. Approximately 2,644 m of total horizontal development and 168 m of Capex vertical development have already been completed at Intrepid. The portal is at 365 RL, and planned stopes are located between 300 RL and -325 RL. The zone extends approximately 300 m to 120 m horizontally and dips between 50° and 70°.

Combining the production from the Rainy River (OP and UG Main Zones) and Intrepid deposits will allow the project to sustain the 27,000 tpd production rate through to the end of 2028. At this point, the LG stockpile will have been depleted and downsizing the mill will allow it to sustain a feed of 4,500 tpd, exclusively extracted from the UG Main Zones.

Mining of the UG Main Zones will take place in two phases. Phase 1 will ensure maximum productivity by using a lower cut-off grade during pit production and milling of the LG stockpile. After the stockpile is depleted, the underground operation will aim to maximize the grade and value of the mined material to recover the remaining reserves at a higher cut-off grade. Using multiple ore sources and access points will provide more flexibility and maximize productivity in Phase 1 and will sustain an economically viable operation during Phase 2. The cut-off grades and phases are also discussed in Item 15.

Mining voids will be filled using a combination of cemented rock fill (CRF) and non-cemented rockfill to increase mining recovery, provide stable rock conditions, and minimize the impact of open stopes on general ground conditions.

At Intrepid, underground mining will commence in April 2022. Production from the UG Main Zones will start with the 17 East Upper Zone in Q1 2024, followed by Zone 433 in Q2 2024 and the ODM Main Zone in Q3 2024. The underground Mineral Reserve will have been mined in its entirety by Q4 2031.

Intrepid development started in June 2021. It included 2,388 m in Capex horizontal development, 257 m in Opex horizontal development and 168 m in Capex vertical development, for a total of 2,812 m in 2021.

Ore grade gold/silver mineralization occurs in subvertical horizons ranging from 3 m to 20 m thick, with a weighted average thickness of 6.5 m for UG Main and 5.7 m for the

Intrepid zone. A minimum width of 3.4 m was used to define Mineral Reserves. The ore zones generally dip at 60° or steeper but can flatten locally to 50°. The planned mining methods rely on gravity for ore flow along the footwall.

Stopes will mostly be mined using a long-hole retreat method, except where the width of the deposit is greater than 20 m and allows for a long-hole transverse method (only in the ODM Main Zone). Most stopes will be drilled as downhole unless the overcut is unnecessary or at the top of the production centres (i.e., the sill pillars). Irregularities in the lenses and geology require that waste rock gaps be used as natural pillars as much as possible.

Ore from the UG Main Zones will be mined at approximately 4,500 tpd from Q3 2026 onwards and blended with open pit stockpiles to maintain the total (UG+OP) mill feed rate at approximately 27,000 tpd until the LG stockpile is depleted. Production ramps up from Q1 2024 to Q3 2026 in the UG Main Zones to initially achieve 3,500 tpd. As soon as the second portal at the bottom of the pit is available, the production is fully ramped up to the targeted 4,500 tpd. Development production is planned with a mining contractor for the project's duration to minimize the risk of manpower shortage. Total development requirements for UG Main Zones amount to 65 km, whereas 56% is capital development.

Mine development will employ numerous production fronts to maximize productivity and flexibility to reach the 4,500 tpd target. Two main long-hole mining methods will be employed: longitudinal and transverse. Transverse stoping is concentrated in part of the ODM Main Zone, the widest zone in the mine. Mining areas have individual production centres based on the main mining method of each sector. Mining of each production centre will ascend from the lowest to the highest level. For the UG Main Zones, an average of 9.4 stopes will be blasted each month, while an average of 7.0 stopes will be backfilled monthly. Excluding development, an average of 3.4 simultaneous stopes must be active to achieve the production targets.

16.3.1 Geotechnical Consideration

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 Map3DTM linear elastic modelling exercise was undertaken to review the proposed mining design and sequence. The induced stresses around the planned development and stopes were evaluated based on this modelling and the ground support requirements were estimated.

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.

16.3.1.1 Rock mass characterization

The overall rock mass quality 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 16.8 summarizes Rock Mass Classification data (Q’, RMR76, and Geological Strength Index (GSI)) by mining zone (AMEC 2013).

Table 16.8 – 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 16.14 (AMEC 2013).

Table 16.9 – 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 16.15, indicating strong to very strong rocks. Field assessments by NRMS confirmed these ranges as accurate and representative.

Table 16.10 – Summary of UCS test results from the stoping domains

Zone	Rock type	Test #	Avg (MPa)	Stdv	CV	Min (MPa)	Max (MPa)
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

Notes: 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. Figure 16.11 summarizes the hydraulic radius (HR) design limits and associated maximum strike length for both unsupported and supported cases for anticipated rock mass conditions.

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Table 16.11 – 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 elastic 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 most 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.

16.3.1.2 Recovery and dilution

Dilution is discussed in detail in Item 15.

In summary, average ELOS values of 0.3 m footwall (FW) and 0.6 m hanging wall (HW) are used throughout the UG Main Zones. The gap in geotechnical knowledge associated with the underground portion of the project did not allow each stope to be characterized according to location and rock mechanic properties.

Intrepid dilution was calculated using the empirical estimation of wall slough from past reports. The average ELOS values of 0.3 m FW and 0.6 m HW were estimated relative to considerations of rock mass quality, stope dimensions, structure, dip and depth. This corroborates the assumptions used for the UG Main Zones.

For the UG Main Zones, a mining recovery of 95% has been applied to the production portion of the project, and a recovery of 80% and 90% has been applied to sill pillars and transverse secondary stopes, respectively.

For Intrepid, a mining recovery of 95% has been applied to the estimates. Meanwhile, the mining recovery of sill pillars, where needed, is estimated to be 60%.

16.3.2 Ground control

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 ground control can be done through various geomechanics instruments. For stopes, a CMS (Cavity Monitoring System) is standard and should be done on a regular sequence to review the stability of the Hanging Wall and Footwall of a stope. Ground support can be modified to minimize dilution. Rock bolt pull testing will be performed on two percent of installed bolts and tested to 75% of yield. 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 16.12.

Table 16.12 – Ground Support Requirements for Lateral Development

Support class (SC)	Ground support requirements	
	Less than 5 m span	**5 m span or above**
Q 10 – 40 GSI 75 - 100	Primary support: Minimum 9 gau. wire mesh and 1.2 m long 33-39 mm split sets at the face. Minimum 9 gau. 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 1.2 x 1.35 m spacing in back. 1.8 m long 33-39 mm split sets or #6 fully encapsulate rebar bolt in walls as required.	Primary support: Minimum 9 gau. wire mesh and 1.2 m long 33-39 mm split sets at the face. Minimum 9 gau. gal. 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 1.2 x 1.35 m spacing in back. 1.8 m long 33-39 mm split sets or #6 fully encapsulated rebar bolt in walls as required. Secondary support for span 8 m to 15 m: Minimum 5 m long 0.6''-0.7" single-strand bulbed cablebolt on 2.5 m square pattern or an approved equivalent.
Q 4 – 10 GSI 55 - 75	Primary support: Minimum 9 gau wire mesh and 1.2 m long 33-39 mm split sets at the face. Minimum 9 gau. 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 1.2 x 1.35 m spacing in back, with 0.6 m 'Dice-5' offset row. 1.8 m long 33-39 mm split sets or #6 fully encapsulate rebar bolt in walls as required.	Primary support: Minimum 9 gau. wire mesh and 1.2 m long 33-39 mm split sets at the face. Minimum 9 gau. gal. 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 1.2 x 1.35 m spacing in back, with 0.6 m 'Dice-5' offset row. 1.8 m long 33-39 mm split sets or#6 fully encapsulate rebar bolt in walls as required. Secondary support for span 8 m to 15 m: Minimum 5 m long 0.6''-0.7" single-strand bulbed cablebolt on 2.5 m square pattern or an approved equivalent.

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 cm to 10 cm of plain or fibre reinforced shotcrete applied to the back, walls, or other specified target areas.
- 3 m to 10 m long bulged twin-strand cable bolts 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.
- These may be required to support larger spans, potential wedges & blocks formed by persistent structure, fault zones, etc. These may also be regularly required at depths greater than >500 m, where stress effects due to mining are indicated to become more apparent and impactful.

16.3.3 Hydrogeology

The groundwater surface tends to be at or near ground surface in the northern area of the resource area, and within 3 m of ground surface towards the south of the project. Artesian pressures, with pressure head up to 2.3 m above ground surface were noted in the Whiteshell Formation on borehole logs (AMEC 2013).

Packer-test hydraulic conductivity values determined during SRK's 2019 pit investigation studies are provided in Table 16.13.

Table 16.13 – Packer-Test Hydraulic Conductivity Values

Hole ID	Drillhole Depth (m)		Hydraulic Conductivité (m/s)
	From	To	
SRKOP19-01	36	106	7.4E-10
	108	178	3.0E-10
	180	244	3.0E-10
	246	319	1.0E-10
SRKOP19-03	54	60	6.0E-07
	60	146	5.5E-09
	147	242	4.5E-10
	243	344	1.0E-10
SRKOP19-04	20	76	1.7E-09
	77	130	Test not successful
	77	130	8.1E-10
	131	184	5.8E-10
	187	271	1.0E-10
	275	364	1.0E-10
SRKOP19-06	260	299	1.0E-10
	230	260	1.0E-10
	17	230	7.9E-09

There is a general downward trend in deep VWPs around the north pit limits that follows the pit advancement with depth. This indicates the pore water pressure is dissipating behind the northern pit wall which are parallel to foliation orientation. In the south, limited overall drawdown has occurred to date with some recent downward trends observed where the active mining face is within approximately 50 m of the VWP. Blasting in the vicinity of VWPs can result in observed nearly instantaneous slope depressurization, indicating a relatively tight rock mass with low conductivity and limited flow conditions prior to the dilation of new open fracturing. Similarly, porewater pressure increases corresponding to large precipitation events or periods are also regularly observed. Limited observed inflows into underground workings corroborate these tight conditions.

16.3.4 Mine Design

UG Main Zones

The UG Main Zones are designed as a mine with mechanized ramp access that will use long-hole open-stope techniques to exploit the underground Mineral Reserves. The location, size, shape, orientation (dip), and physical properties of a mineral deposit generally determine the selection of the appropriate mining method.

Level spacing is set at 25 m. This has been evaluated as the best alternative between 20 m and 30 m level spacing to maximize profitability while minimizing drill & blast challenges (mainly excessive deviation and dilution). Including planned dilution, the minimal stope width and average stope width are 3.4 m and 6.5 m, respectively. The main method used is long-hole longitudinal retreat, which includes 87% of all ore production. The remaining production stopes, found in zones thicker than 20.0 m, are mined with a long-hole transverse mining method (only in the ODM Main zone). Drilling will be conducted with a combination of 4-inch production holes and a V-30 cut (30.0" hole). Hanging walls and footwalls have dips ranging from 47º to 80º.

The main ramp in the UG Main Zones will provide access to all the zones through connecting ramps and will house the major infrastructure components, like the service bay and main services. Fresh air will be supplied to the mine through two ventilation raises, with high-efficiency fans installed on surface to provide a combined total of 1.3 Mcfm when at full production.

As a satellite deposit the Intrepid Zone will have its own ramp and underground infrastructure. The ore body is more continuous than its counterpart in the Rainy River deposit, thus, the ramp, access and ventilation design adhere more closely to the FW ore outline all along the zone.

Figure 16.7 presents an overview of the project at completion.

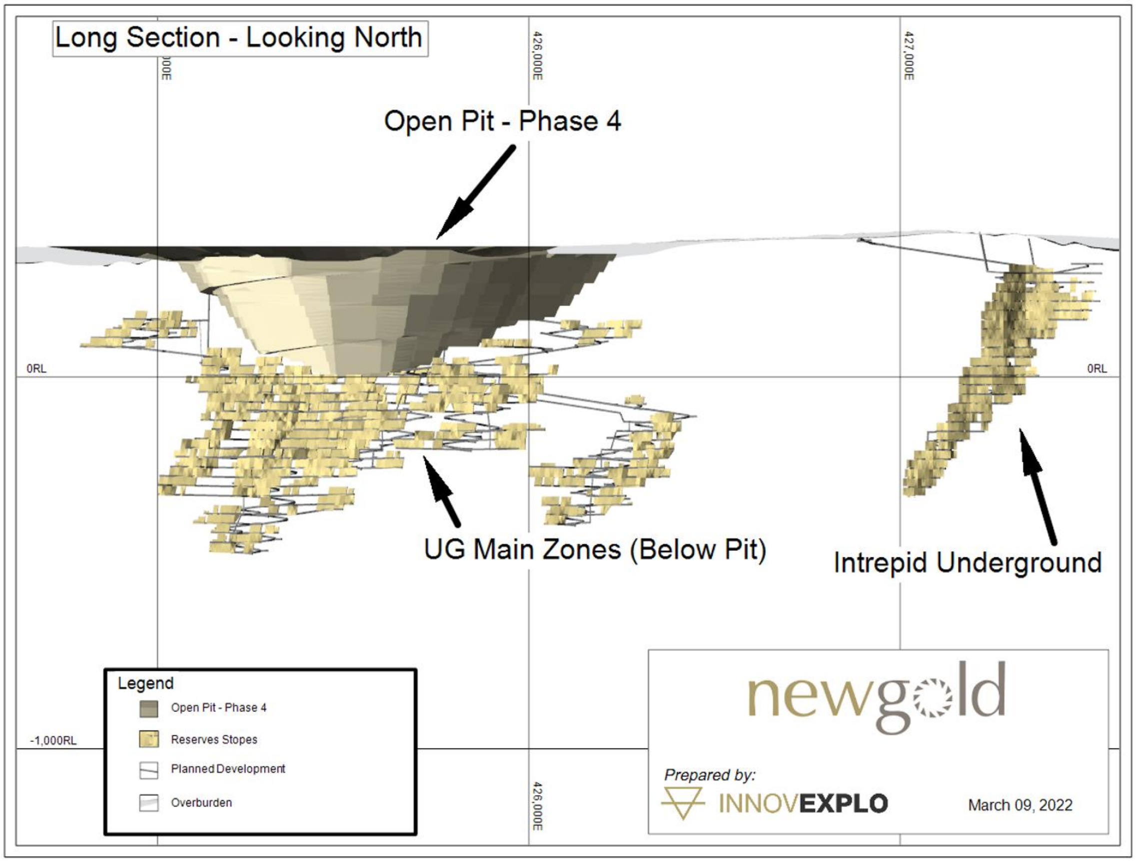

Figure 16.7 – Overview of the Rainy River Project

UG Main Zones

The underground mine designs for the UG Main Zones (primarily 17 East Upper, ODM East, ODM Main, ODM West and Zone 433) are all below the open pit or along the open pit walls. Underground stopes will not daylight into the open pit to avoid stability issues. A horizontal separation distance of 50.0 m and a vertical separation (crown pillar) distance of 40.0 m between the pit and the planned excavation have been applied 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.

Slot blasting will occur once raise boring (V-30), drilling of slot holes and drilling of production rings are completed. All ore will be mucked from the stope's undercut for both opening blasts and mass blasts, using a combination of manual mucking when the brow is filled with ore, and remote LHD operation when the brow is open. Confirmation of the stope completion by operations will initiate the engineering process, which includes CMS (Cavity Monitoring System) and reserves extraction validation. When the engineering department confirms the stope has been completed, the ongoing mining cycle is allowed to progress.

Mining voids are filled using a combination of cemented rock fill (CRF) and non-cemented rockfill to increase mining recovery, provide stable rock conditions, and minimize the impact of open stopes on general ground conditions. Every final stope in a sequence or secondary transverse stope are filled with rockfill only to minimize CRF costs. Upper stopes are not backfilled; instead, rib pillars are left to stabilize the ground and minimize dilution.

Intrepid Zone

The plan for the Intrepid Zone uses a similar mining approach as the UG Main Zones. A combination of downhole and Uphole stopes will recover the main orebody in five distinct production centres or phases, each mined from bottom to top. No transverse mining is planned.

The drilling and blasting will follow the same methods and techniques as the UG Main Zones. Rockfill and CRF will also be used as backfill material in order to stabilize the excavations and maximize recovery. Because of the surface proximity, crown pillar and geotechnical considerations are considered in the upper part of the zone.

Since part of Intrepid will have already been mined by the time production starts in the UG Main Zones, the best practices and experience gained at Intrepid will be implemented in both deposits.

16.3.5 Stope design

UG Main Zones

The Deswik Stope Optimizer™ (DSO) module was used on the Mineral Resource block model to generate mineable shapes that were subsequently used to optimize the proposed design. Once the preliminary stopes were generated, a check was made to remove any outlying stopes that would not be economic if development and mining costs were considered. Parameters used in the DSO module are presented in Table 16.14. Additional key design parameters are presented in Table 16.15.

Table 16.14 – DSO Parameters for Underground Mining (UG Main Zones)

Parameters	Field	Default	Units
Density (waste)	Density	2.85	t/m^3
Optimization field	AU_EQ20	0	g/t
COG (stopes)	AU_EQ20	1.74 & 2.25	g/t
Slice interval		2.0	m
Default dip		55	Degrees
Strike azimuth		100	Degrees
Sub-blocking		Yes	
Optimization length		25	m
Minimum mining width		3.4	m
HW dilution		0.6	m
FW dilution		0.3	m
Maximum strike change		90	Degrees
Stope maximum side-length ratio		2	ratio

Notes:

- Cut-Off grade (COG): 1.74 g/t = Phase 1 (with LG stockpile), 2.25 g/t = Phase 2 (without LG stockpile)
- Au price US$1,500 per troy ounce, Ag price US$19 per troy ounce
- The exchange rate used was 1:1.25 US$/C$.
- AuEq is equal to Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,500 * 95)]

Table 16.15 – Key Design Parameters (UG Main Zones)

Parameters	Long hole Mining	Units
AuEq	1.74 & 2.25	g/t
Minimum mining width	3.4	m
Mining height	25	m
Mining length	25	m
Minimum HW angle	55	Degrees
Minimum FW angle	55	Degrees
Mining recovery	95	%

Notes:

- Cut-Off grade (COG): 1.74 g/t = Phase 1 (with LG stockpile), 2.25 g/t = Phase 2 (without LG stockpile)
- Au price US$1,500 per troy ounce, Ag price US$19 per troy ounce
- The exchange rate used was 1:1.25 US$/C$.
- AuEq is equal to Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,500 * 95)]

Intrepid Zone

Deswik Stope Optimizer™ (DSO) module was also used generate minable shapes for the Intrepid Zone. Optimization was completed prior to the current study. Parameters differ slightly from those used for the UG Main Zones and are presented in Table 16.16. Additional key design parameters are presented in Table 16.17Table 16.15.

Table 16.16 – DSO Parameters for Underground Mining (Intrepid Zone)

Parameters	Field	Default	Units
Density (waste)	Density	2.83	t/m³
Optimization field	AU_EQ	0	g/t
COG (stopes)	AU_EQ	1.93	g/t
Slice interval		10.0	m
Default dip		60	Degrees
Strike azimuth		90	Degrees
Sub-blocking		Yes	
Optimization length		30	m
Minimum mining width		2.9	m
HW dilution		0.58	m
FW dilution		0.29	m
Maximum strike change		90	Degrees
Stope maximum side-length ratio		2	ratio

Notes:

- Cut-off grade (COG) = 1.93 g/t
- Au price US$1,500 per troy ounce, Ag price US$19 per troy ounce
- Exchange rate = 1:1.25 US$/C$.
- AuEq = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,500 * 95)]

Table 16.17 – Key Design Parameters (Intrepid Zone)

Parameters	Long hole Mining	Units
AuEq	1.93	g/t
Minimum mining width	2.9	m
Mining height	25	m
Mining length	25	m
Minimum HW angle	55	Degrees
Minimum FW angle	55	Degrees
Mining recovery	95	%

Notes:

- Cut-off grade (COG) = 1.93 g/t
- Au price US$1,500 per troy ounce, Ag price US$19 per troy ounce
- Exchange rate = 1:1.25 US$/C$.
- AuEq = Au (g/t) + [(Ag (g/t) * 19 * 60)/ (1,500 * 95)]

The AuEq grade is estimated directly in the Mineral Resource block model, using the following equivalence:

$$AuEq = Au\ (g/t) + [(Ag\ (g/t) * 19 * 60) / (1{,}500 * 95)]$$

where the factors in the equivalence calculation are:

- Gold price $1,500/oz
- Silver price $19/oz
- Gold recovery 95%
- Silver recovery 60%

Following the optimization, all stopes were reviewed to ensure the economic viability of the project.

The final optimization, after economic analysis and design, is presented in Figure 16.8.

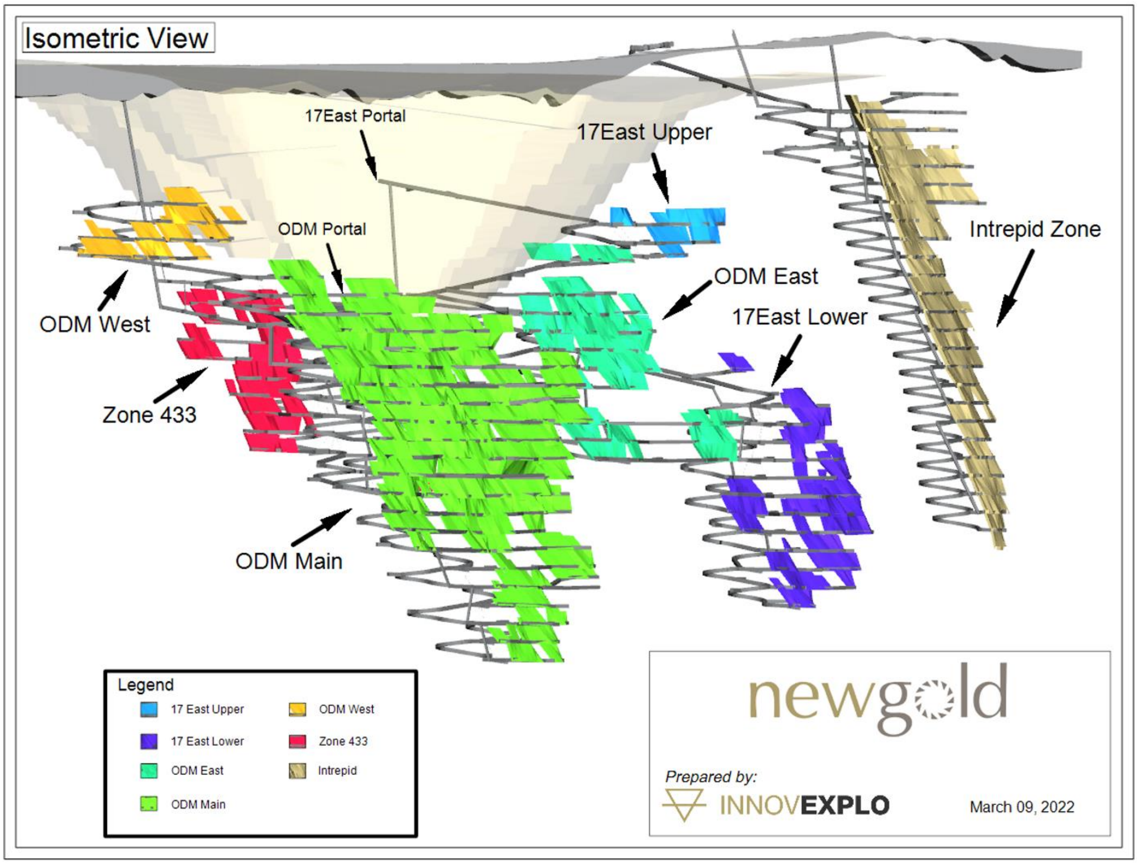

Figure 16.8 – Isometric View of the UG Main Zones (Below Pit) and Intrepid Zone

Existing infrastructure

Several pieces of critical infrastructure have already been installed for the underground mining operations:

- 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

In addition to the components listed above, all surface support infrastructures necessary for the underground operations are already present and in use (truck shop, mill plant, mine dry, mine offices). No major changes are projected. Infrastructure currently dedicated to the open pit operation will transition to the UG Main and Intrepid Zones as production shifts from surface to underground.

16.3.6 Main infrastructure

UG Main Zones

Most major infrastructure will be located underground and centralized on level UG 350 (Main Ramp). On surface, this will include the two primary ventilation fans. The main one is located west above the pit boundary; it has 1200HP and generates 800 kcfm at full production. The other, and the first to be installed, is inside the pit, between the North and Main pits; it outputs 800 HP and generates 500 kcfm at full power. Pads are needed around the ventilation installation for natural gas reservoirs to power the fan burners. Temporary ore and waste pads may be needed to facilitate material haulage in the pit. To properly use the waste rock pile in the North Pit, a mobile rock breaker system will be needed to prepare material for screening and backfilling.

Due to the challenges associated with having portals directly in the open pit (no access for the piping system during pit production), no compressed air system has been planned for the UG Main Zones. A support fleet of compressors will instead be used underground.

Intrepid Zone

The Intrepid Zone takes advantage of the existing surface infrastructure used for the pit. In addition, the main underground infrastructure already includes a ventilation network (with ventilation doors and walls), a dewatering network and sumps, a main pumping station, refuge stations and electrical substations. A mechanic shop & washbay and powder & cap magazine are also planned underground (one of each).

Service bay

The service bay in the UG Main Zones will have an access area, welding bay, garage, tire storage, washing bay, warehouse, greasing bay, fuel bay and parking. The service bay will be located on Level UG 350. The garage will be able to simultaneously accommodate up to two large and one small piece(s) of equipment, and the parking area will have room for at least 5 vehicles. The overall service bay area will have a total volume of 12,000 m^3 for a linear-equivalent total of 380 metres.

The overall maintenance strategy underground is to prioritize emergency reparations, small preventive maintenance, and work on slower critical equipment (production drills), while planned maintenance on larger equipment will take place surface. This will make maximum use of the existing open pit workshop.

Figure 16.9 presents an overview of the Main Ramp around the ODM East Zone and level UG 350, the main service hub for the project.

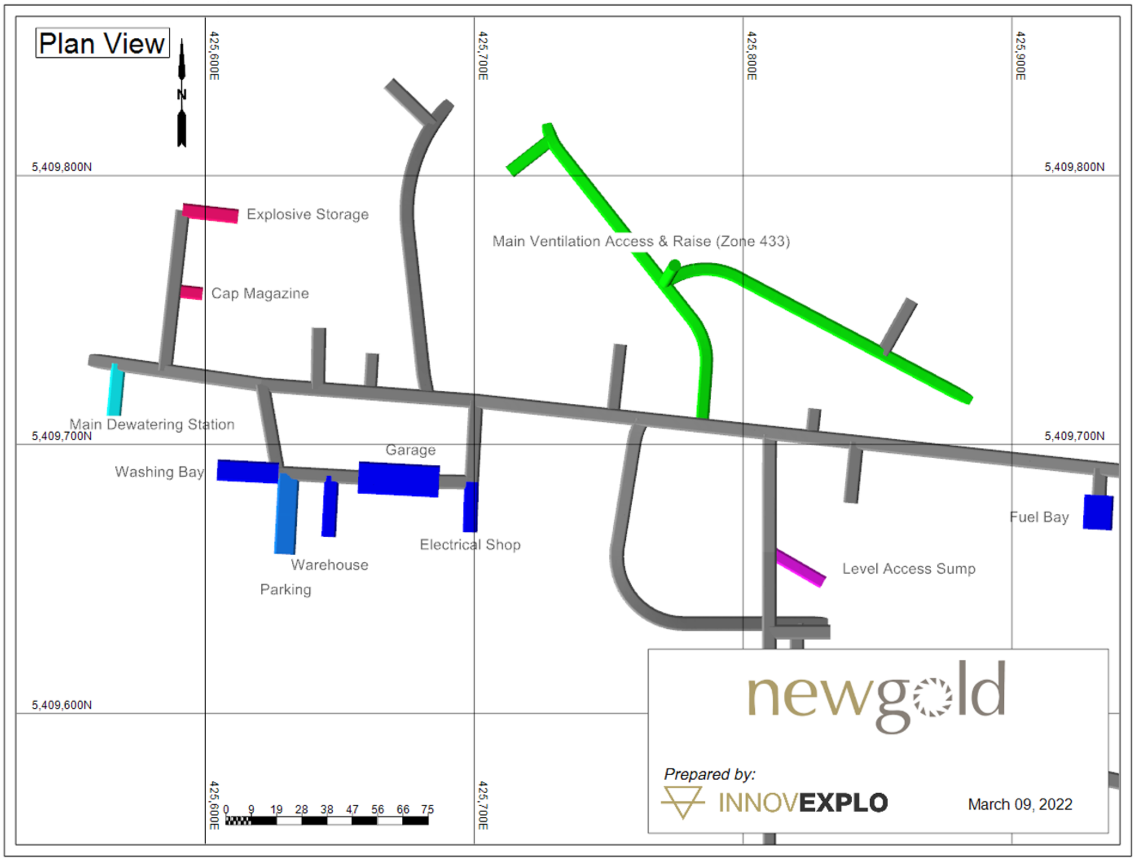

Figure 16.9 – Main Ramp & Service Bay (Level UG 350)

As mentioned in a previous Item, Intrepid will have its own service bay or mechanical shop, situated at on level UG 175.

16.3.6.1 Additional infrastructure

UG Main Zones

Additional infrastructure includes emergency underground refuge stations, powder & cap magazine, and internal ventilation raises. The plan is for a single powder & cap magazine, situated on level UG 375 (connected to the Main Ramp); which can easily accommodate the explosive requirements of the project.

Each underground refuge station is designed and located to accommodate the necessary number of workers at any given time. The refuge stations are located closer than the required 1,000 m to ensure no delays in the development sequence. The occupancy of the refuge stations, either 12 or 24 workers, depends on the volume of activity in the mining area. All three 24-worker refuge stations are located in major or isolated zones: ODM Main, ODM East and 17 East Lower.

Since compressed air is not available, all refuge stations are equipped with a Refuge One ON_2 Solutions system to ensure proper air support in case of an emergency.

The refuge stations will be equipped with tables, chairs and a communication system so they can serve as lunchrooms.

Intrepid Zone

The Intrepid Zone will have 24 electrical sub-stations and 22 sumps fitted with a pumping system (including the main pumping station).

Ventilation walls fitted with doors are needed for each ventilation access to complement Intrepid's ventilation network. A total of 14 ventilation walls are planned for the duration of the zone.

Unlike the UG Main Zones, Intrepid has access to an air compressor at surface, which will be connected to the main refuge station, on level UG 175. For the rest of the zone, temporary refuge stations will be used as a safety measure on active levels.

16.3.6.2 Mine design criteria

UG Main Zones

Permanent drifts (ramps and access drifts) are generally 5.5 m wide by 5.5 m high, whereas the ore and waste drifts not used by trucks are 5.0 m wide by 5.0 m high. The following subsections describe the different types of rockworks and headings (e.g., main ramps, typical level, loading bay, and emergency egress). Various development parameters are summarized in Table 16.18, whereas the general pillars set by rock mechanics are listed in Table 16.19.

Remucks are generally spaced every 150 m for development efficiency.

Table 16.18 – Mine Design Parameters (UG Main Zones)

Development Heading	Width (m)	Height (m)	Gradient
Ramp	5.50	5.50	13 @ 15%
Remuck (Ramp)	5.50	5.50	2.0%
Remuck (Access Level)	5.50	7.30	2.0%
Remuck (Haulage drift)	5.00	5.00	2.0%
Level Access	5.50	5.50	2.0%
Truck Loading/Unloading Bay	5.50	7.30	2.0%
Level Haulage & Waste Drift	5.00	5.00	2.0%
Ore Drift	6.00	5.00	2.0%

Table 16.19 – Mine Design Pillars

Pillar Type	Minimal Distance (m)
Ramp / Stope	30
Haulage Drift / Stope	15
Raise / Stope	15
Ramp / Level Drift	13
Drift / Drift	8
Raise with manway / Ramp	30
Ramp / Stope	30

Intrepid Zone

Intrepid permanent drifts (ramps and access drifts) are planned 5.5 m wide by 5.5 m high, whereas the main ore drifts are 6.0 m wide by 5.0 m high. Additional small excavations are also planned for this zone: Ventilation access (4.5 m wide by 4.5 m high) and exploration drift (4.0 m wide by 4.0 m high). Intrepid development parameters are summarized in Table 16.20.

Table 16.20 – Mine Design Parameters (Intrepid Zone)

Development Heading	Width (m)	Height (m)	Gradient
Ramp	5.50	5.50	13 @ 15%
Remuck	5.50	5.50	2.0%
Level Access	5.50	5.50	2.0%
Level Haulage & Waste Drift	5.00	5.00	2.0%
Ore Drift	6.00	5.00	2.0%
Ventilation Access	4.50	4.50	2.0%
Exploration Drift	4.00	4.00	2.0%

16.3.6.3 Main ramps

UG Main Zones

The Main Ramp starts development in Q2 2023 at elevation 125 RL in the pit. The portal starts on the middle bench between the North pit in the still-active Main pit. Extensive ground control analysis and mitigation are planned to ensure the proper long-term stability of the portal. Furthermore, the final pit design and bench position may be optimized to maximize security and flexibility around the portal (see Item 26 for more details).

The main ramp width and height is 5.5 m x 5.5 m at a maximum gradient of 15%. Secondary electrical stations are excavated every 300 m, which is the maximum conservative distance without loss of charge. Sumps are positioned every 500 to 600 m and will only be used during development.

As mentioned above, the first goal of pre-production development is to reach and excavate the first ventilation raise (Ventilation Raise Zone 433, which daylights in the pit). Level access and other infrastructures are partially developed to facilitate future development and waste haulage, but the development priority is always toward ODM Main and the two ventilation raises.

As soon as the Zone 433 ventilation raise is available, additional development teams are added to maximize pre-production development.

Figure 16.10 presents the evolution of the main ramp's development between Q2 2023 and Q3 2024.

Figure 16.10 – Main Ramp Evolution for the UG Main Zones (Q2 2023 to Q3 2024)

Intrepid Zone

Intrepid Zone has only one main ramp, which has already been developed down to level UG 175. About 5,100 m of ramp development is planned for the zone. The ramp follows the deposit on the FW side using an eight pattern, respecting both pillars and gradient limitations. Remucks are planned in the ramp between each level.

16.3.6.4 Typical level

UG Main Zones

A typical production level includes an access drift, a sump, a primary/secondary electrical station, a loading bay, a ventilation access (generally connected to the level), and haulage and ore drifts (see Figure 16.11). Depending on the location, the level can also include a refuge station, an exploration drift, a CRF/remuck bay and other infrastructure elements.

Sumps are excavated roughly at 60° and -15% from the access to facilitate the mucking of excess mud. Each level has at least one secondary electrical station; main electrical stations are positioned every three or four levels.

Sumps and electrical stations are vertically aligned (whenever possible) to facilitate the setup of the pumping and electrical networks.

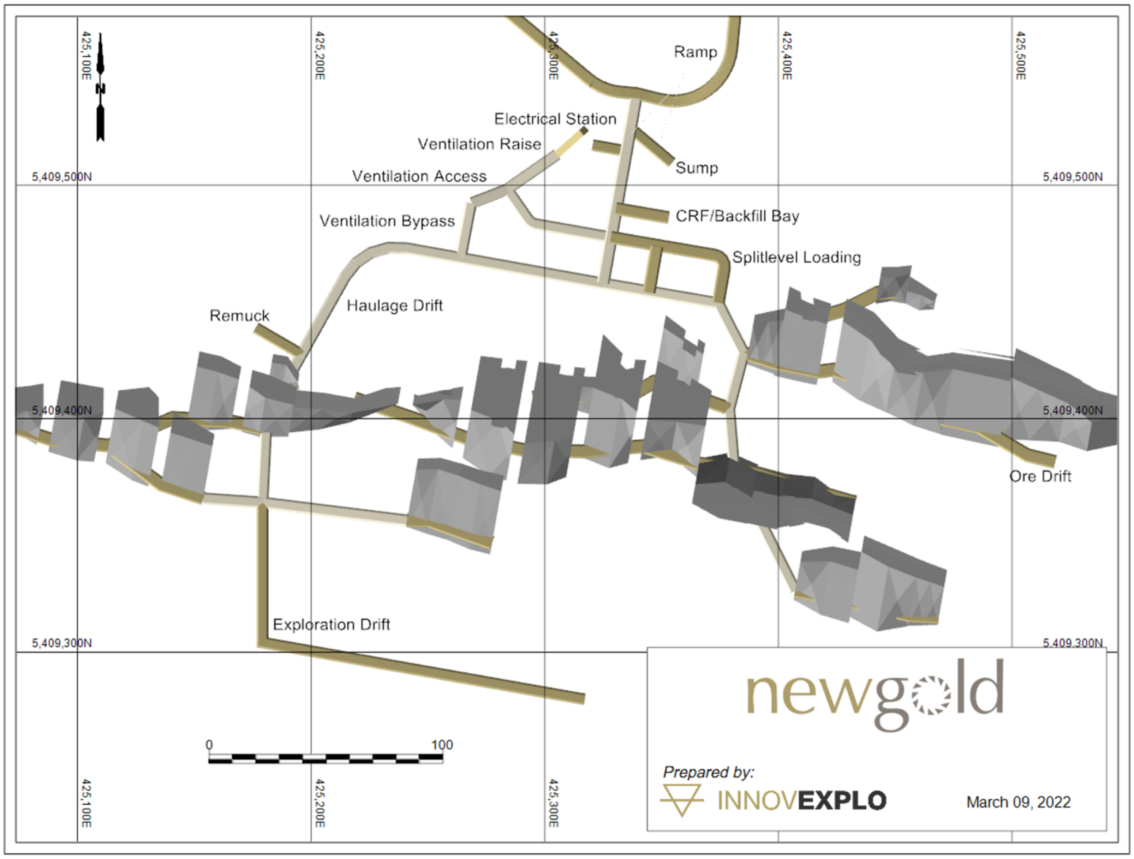

Figure 16.11 – Typical Level (ODM Main - Level UG 600)

The access drifts and loading bay (5.5 m x 5.5 m, 5.5 m x 7.3 m) are used by trucks and LHDs, whereas haulage drifts will only accommodate LHDs and production drills (5.3 m x 5.0 m). Waste haulage drifts are typically offset by at least 30 m from the ore body to respect the pillars established by rock mechanics analysis. Because multiple lenses are present on each level, a combination of waste drifts and ore drifts will serve as drawpoints, minimizing redundant development. The maximum haulage distance is around 300 m, with some exceptions reaching the 350 m mark.

By comparison, transverse mining requires a drawpoint for every stope panel, boosting development in exchange for increased productivity.

Developments are designed to respect the 2% minimal gradient to facilitate water runoff to the level's sump.

Where multiple access and an increased tonnage per level are found (ODM Main), a split-level loading concept has been used for the design. This entails excavating a higher drift for LHDs dumping, maximizing visibility and fill ratio. The double-access shape also ensures flexibility by having the option to dump backfill or ore in the second access. Supplier input and past experiences have proven that this type of layout is optimal to maximize productivity.

Intrepid Zone

The Intrepid Zone uses a similar approach as the UG Main Zones. Each level consists of an access, an electrical sub-station, a sump, a ventilation access, a crosscut for a refuge or a temporary refuge, a remuck and the ore/waste drift. Given the continuous geometry of the Intrepid orebody, the levels are mostly identical, following the FW and the center of gravity on each level.

16.3.7 Emergency egress

UG Main Zones

By the end of the UG Main Zones mine life, both in-pit portals will provide alternative exits from the main ramps and be considered as the two emergency egress routes out of the mine. An additional manway will be outfitted in the Zone 433 ventilation raise to allow production to start before the end of the pit and to increase safety.

Each zone's ventilation network will also serve as an egress between production levels, with internal ventilation drop raises outfitted with manways.

The ODM West Zone, being the only zone without a dedicated ventilation network, is planned with two raise-bored 1.5 m openings, designed with a Safescape™ system. As a low-cost and effective alternative to conventional manways, this technology may be used in other parts of the mine to replace costly manway construction, if needed.

Intrepid Zone

The Intrepid Zone uses the same logic: manways in ventilation raises as alternative egresses for each level. As mentioned above, additional temporary refuge stations are positioned on each active level to ensure proper worker safety.

16.3.8 Mining methods

UG Main Zones

Mine development in the UG Main Zones will employ numerous production fronts to maximize productivity and flexibility to reach the 4,500 tpd target. Two main long-hole mining methods will be employed: longitudinal retreat and transverse (see Figure 16.12). The transverse stoping is only present in the ODM Main Zone, the widest zone of the mine, where stope width exceeds 20.0 m. Mining zones have individual production centres, or mining horizons, based on the overlapping and interconnected lenses found in each sector. Production centres are also dependent on haulage access and waste gaps between different lenses or levels.

Figure 16.12 presents the mining method for the UG Main Zones.

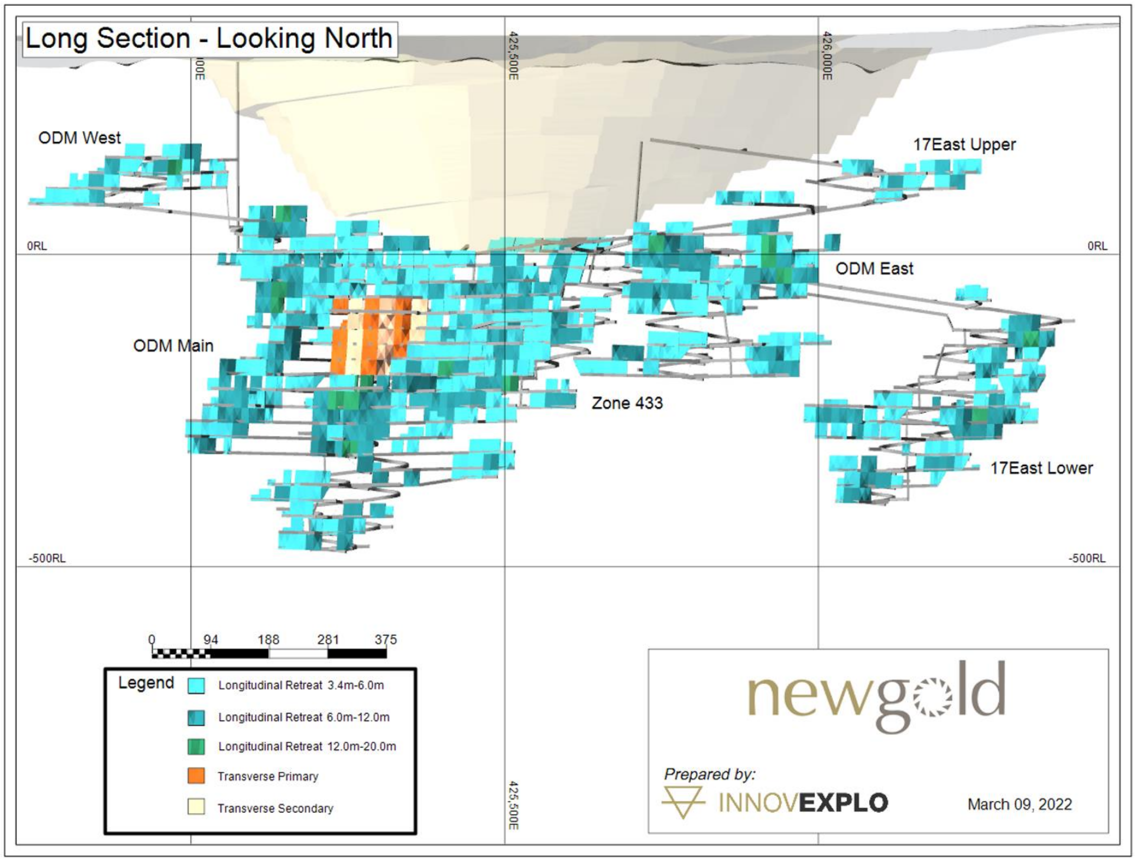

Figure 16.12 – Mining Methods (UG Main Zones)

The mining of each production centre will ascend from the lowest to the highest level. Horizontal sill pillars and vertical rib pillars are positioned strategically to minimize ore loss and to maximize the use of natural waste pillars.

The last level in a sequence, the sill pillar, will be recovered by uppers and will not be backfilled. To ensure stability, 5.0 m rib pillars are positioned efficiently every 25 m to 50 m to separate upper stopes. The lowest (first level) of a production centre will be backfilled with CRF with a high cement proportion (7.0%) to ensure safety and maximum sill pillar recovery. Some additional stopes will also be drilled using upper drilling (e.g., stopes at the apex of zones or stopes without a need for an overcut). An adjusted recovery (85.0%) has been applied to stopes recovered in the sill pillar.

Figure 16.13 presents the production centres for UG Main Zones.

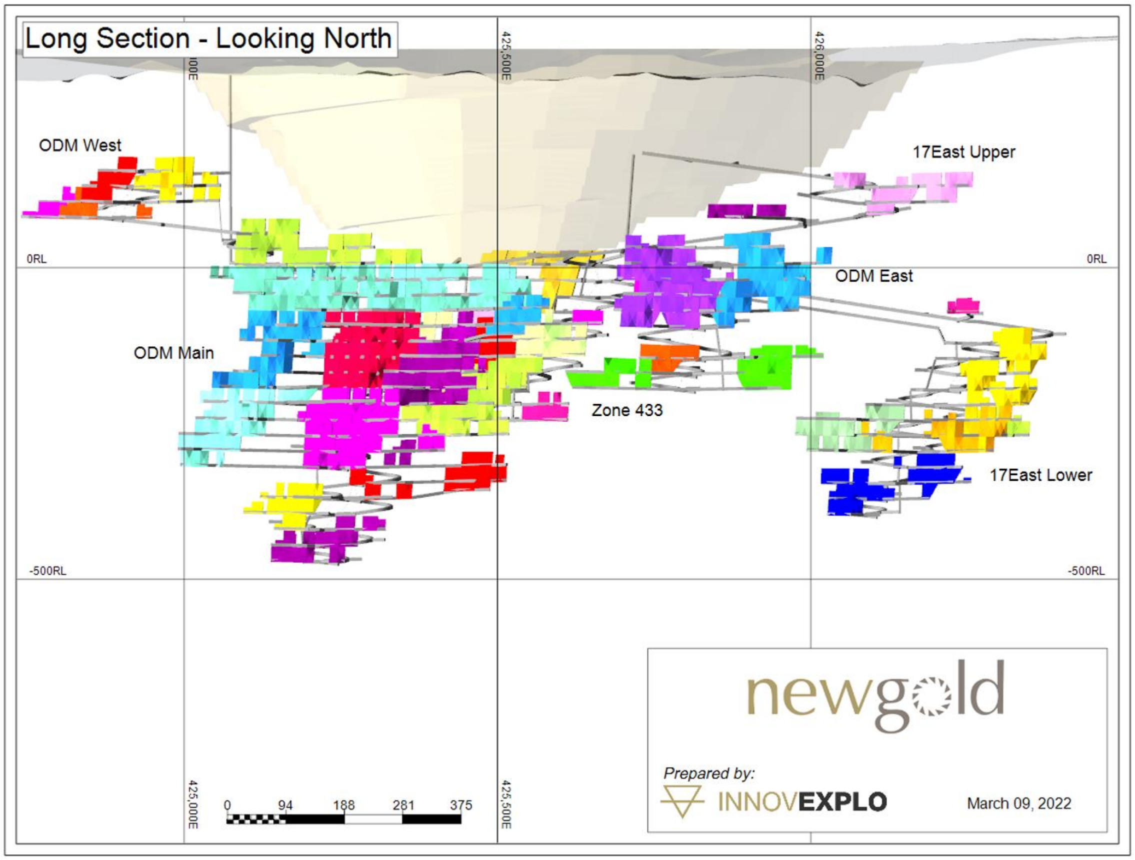

Figure 16.13 – Production Centres (UG Main Zones)

Intrepid Zone

Once again, the simpler design of the Intrepid Zone minimizes the number and variability of the production centres. Only five production centres or phases are needed for the Intrepid Zone, each with a sill pillar and a bottom-to-top production.

Figure 16.14 presents the production centres for the Intrepid Zone.

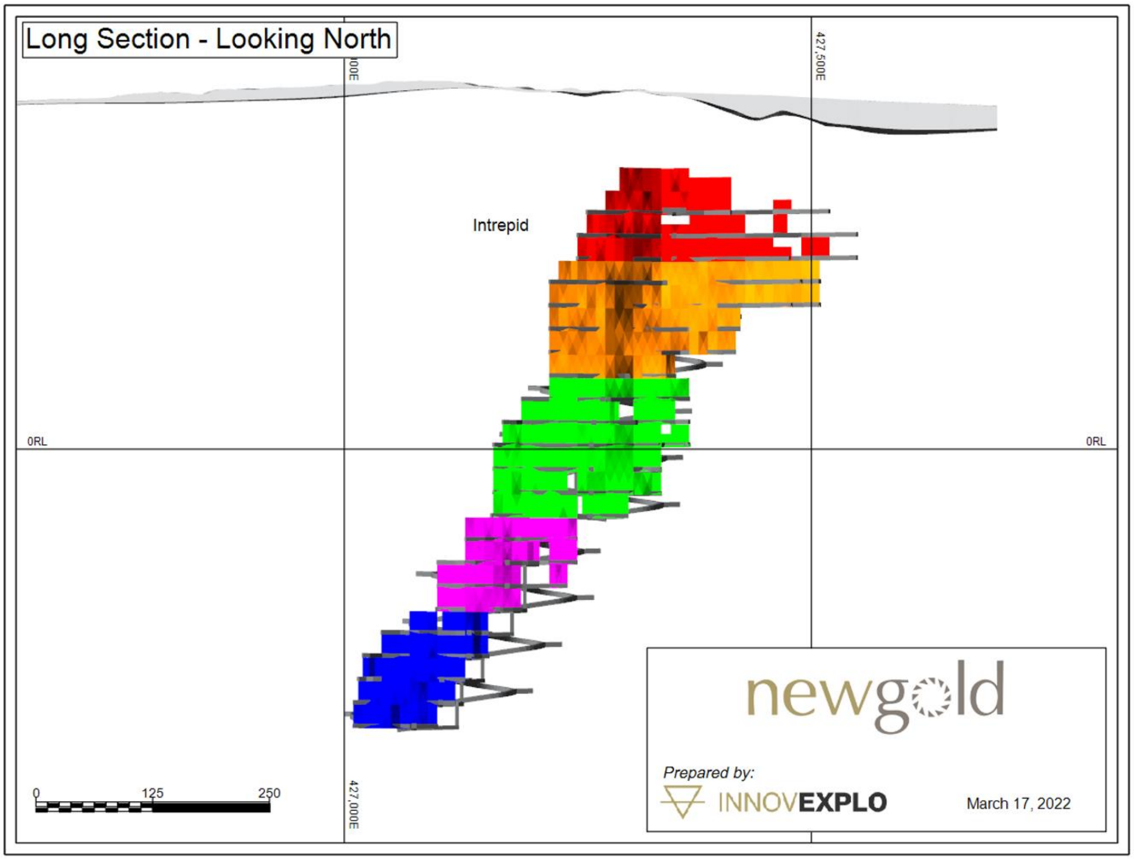

Figure 16.14 – Production Centres (Intrepid Zone)

16.3.8.1 Longitudinal long-hole retreat

Longitudinal long-hole methods will be used for stopes less than 20 m wide (see Figure 16.15 and Figure 16.16 for an example). These stopes are classified based on their average width and have corresponding parameters like drilling factor, number of holes per stope, powder factor and quantity of consumables. The resulting total tonnage mined by the longitudinal long-hole method in the UG Main Zones is 7.72 Mt (83% of total stope production). The Intrepid Zone consists entirely of longitudinal stopes (1.89 Mt in stope production).

Table 16.21 summarizes the resulting tonnage.

Table 16.21 – Mining Methods – Longitudinal Stoping Summary

Method - Longitudinal	Number Of Stopes	Tonnage
UG Main - Longitudinal 3.4m – 6.0m	454	3,620,137
UG Main - Longitudinal 6.0m – 12.0m	212	3,561,789
UG Main - Longitudinal 12.0m – 20.0m	18	542,066
Intrepid - Longitudinal	146	1,889,077

A typical mining cycle includes secondary ground support, where required. V-30 slot-drilling is done in advance of the production drill mobilization, followed by the complete production drilling of the stope. Longitudinal stopes are blasted in two phases: a primary blast for the void and a secondary blast after the first blast is mucked out. The second blast may be prepared for loading during mucking to maximize efficiency. Once the stope is blasted and mucked out, a barricade is built. The stope is backfilled with CRF. Rockfill is used when possible or to finalize and level the drift floor. The longitudinal retreat method is used to create pyramidal shapes as mining progresses in a production centre. This maximizes the stability of the mining area by diverting the induced stresses.

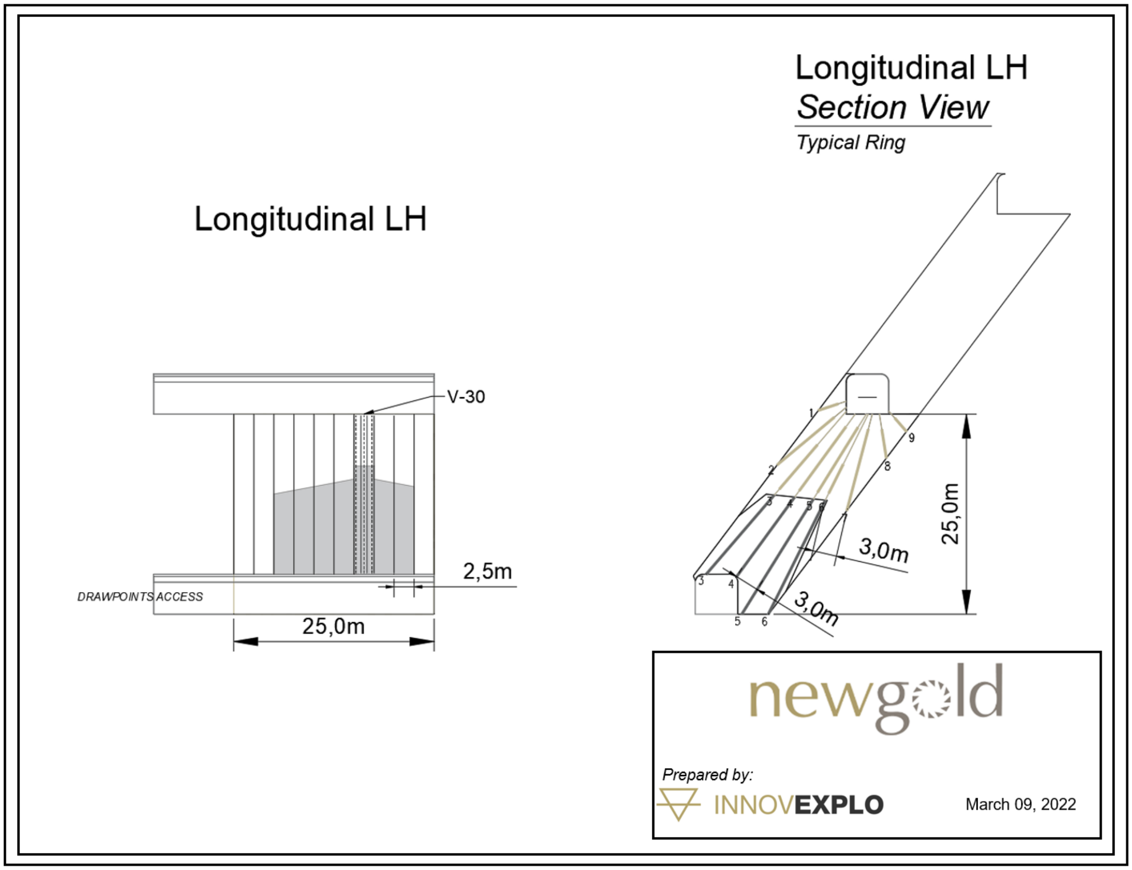

Figure 16.15 – Mining Method – Longitudinal Long-Hole Retreat

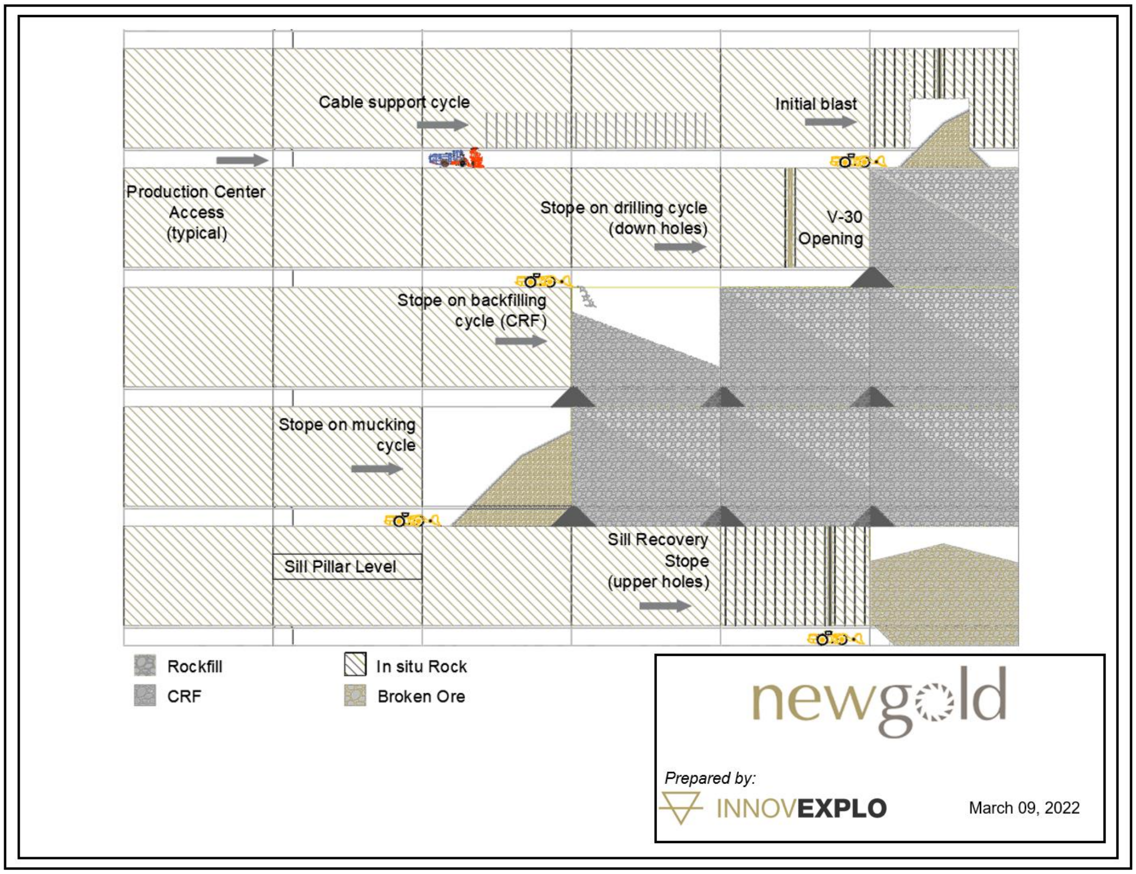

Figure 16.16 – Mining Cycle – Longitudinal Long-Hole Retreat

16.3.8.2 Transverse long-hole

A transverse long-hole method will be used with the remaining zones (i.e., width > 20 m). Only one production centre in the ODM Main zone uses this method and it entails having a drawpoint for each stope panel. These stopes are differentiated into primary and secondary categories depending on the sequence. Due to the complexity of the stope geometries and variabilities in this sector and to facilitate planning, design parameters have been evaluated for the average transverse stope, and are used for all stopes using the transverse method. The resulting total tonnage mined by transverse long-hole method is 1.13 Mt (17% of total stope production). Table 16.22 summarizes the resulting tonnage.

Table 16.22 – Mining Methods – Transverse Stoping Summary

Method - Transverse	Number Of Stopes	Tonnage
Transverse Primary	26	562,878
Transverse Secondary	28	565,170

Like longitudinal stoping, typical mining cycles include secondary ground support where required, V-30 slot-drilling, production drilling, mucking and backfilling. The mining sequence starts with the primary stopes from bottom to top, whereas the secondary stopes are blasted when both adjacent primaries on two levels are backfilled. For the same drawpoint, the farthest stope is mined first, and the sequence retreats towards the hauling drift. This sequence creates a pyramidal shape with the mining voids when the mining progress in a centre of production and is beneficial with respect to the rock mechanics and production aspects. Most transverse stopes need two blasts.

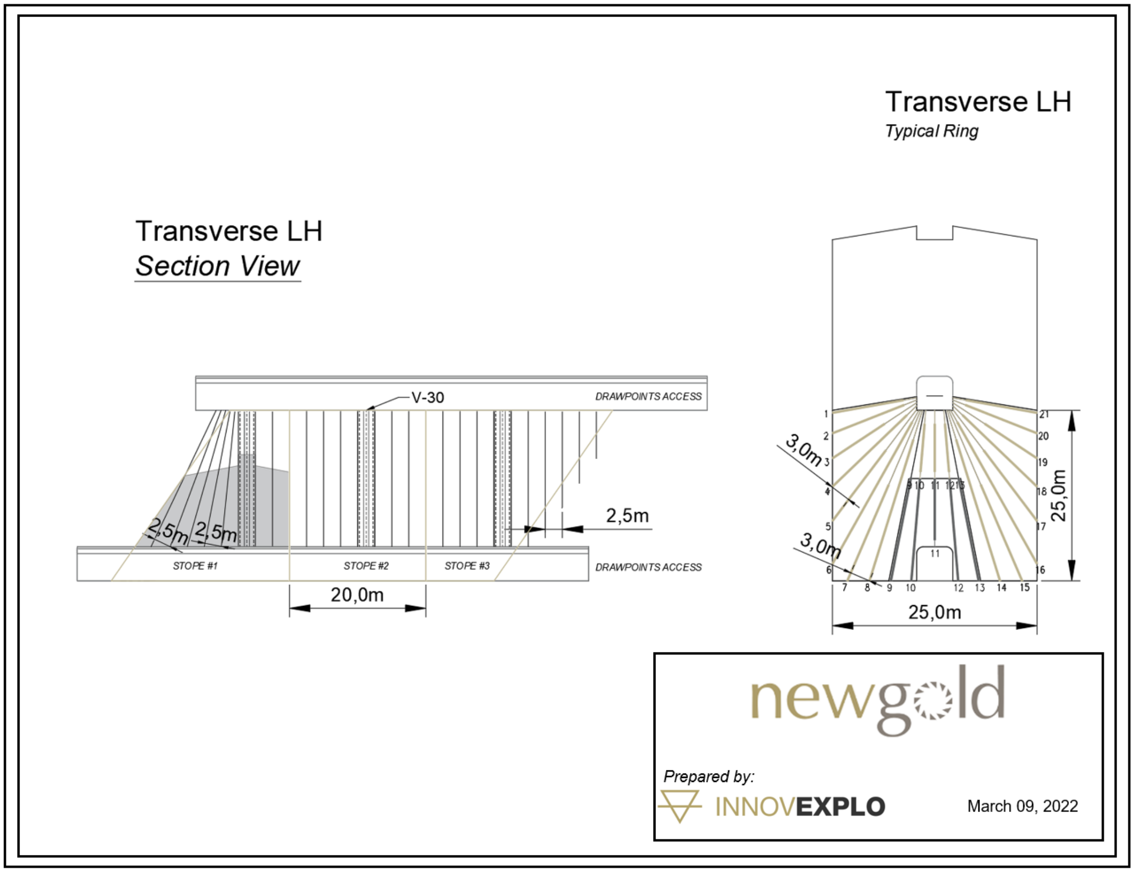

Figure 16.17 – Mining Method – Transverse Long-Hole

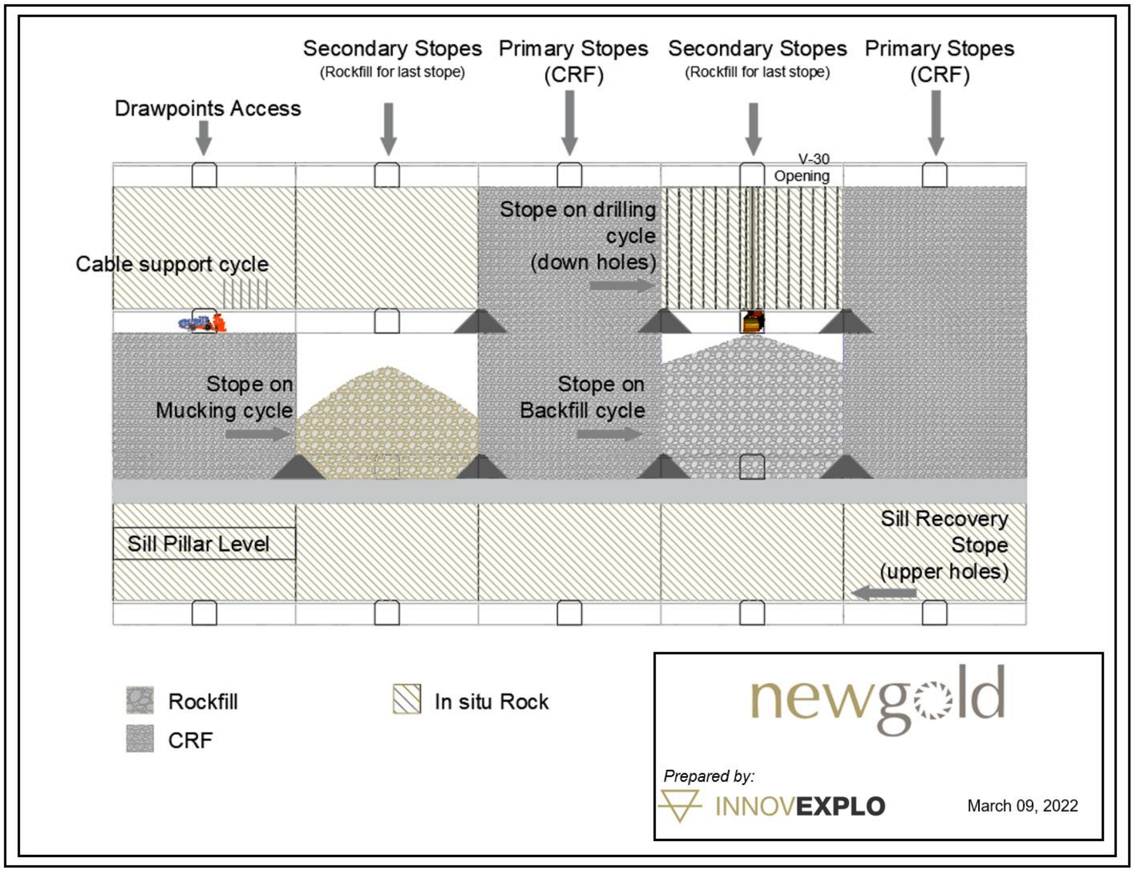

Figure 16.18 – Mining Cycle – Transverse Long-Hole

16.3.8.3 Drill and Blast Design

The long-hole methods chosen for the underground Rainy River project make use of fan drilling to maximize recoveries from single overcut or undercut drifts. Production drill holes are drilled with a 101.6 mm (4.0") diameter using a Sandvik DL432i. Most holes are 3.0 m to 20.0 m long (level spacing at 25.0 m), but some of the longer holes can reach 29.0 m to 30.0 m.

With the selected equipment, deviation can easily be controlled and avoided. The impact on dilution is minimized. To limit personnel exposure and production cycle times, the main method for cut opening will be a raise-bored hole. The most economical, flexible, and risk-free method is achieved using Machines Roger as a contractor to drill the necessary V-30 openings (30" holes). Other raise boring rigs have been evaluated; however, the dip of the ore body does not allow the use of most of the common rigs in the industry. For safety purposes, the excavated hole will be mechanically capped or not fully raise bored to the overcut, eliminating open hole exposure.

As stope geometry can vary greatly between each method, some adjustments are to be expected between each stope variant (smaller burden for narrow stopes, increased burden and spacing for wider stopes). Some alterations to these configurations are made for upper drilling to maximize recovery (over-drilling, dipping section). The burden for each method is averaging at 2.5 m and the spacing varies from 2.0 m to 3.0 m depending on the stope width and the drilling pattern. The method to assess the drilling and the resulting load on planning is to evaluate the drilling ratio (t/m) and re-drilling factor (%).

Table 16.23 – Mining Methods – Drilling Ratio and Re-Drill

Methods	Drilling Ratio (t/m)	Re-Drilling Factor
Longitudinal 3.4m – 6.0m	9.81	10%
Longitudinal 6.0m – 12.0m	9.28	10%
Longitudinal 12.0m – 20.0m	8.39	10%
Transverse Primary	7.31	10%
Transverse Secondary	7.31	10%

The targeted void for blasting is 20%. All methods can achieve this easily with two mass blasts. With the experience acquired during production, parameters like the length and height of the first blast can be optimized to improve safety, production rate and rock stability.

Generally, collars will be 2.0 m with 1.5 m of stemming. Electronic detonators will be used to optimize flexibility and fragmentation. Additional parameters include two boosters and detonators and the use of bulk emulsion to minimize risks and costs.

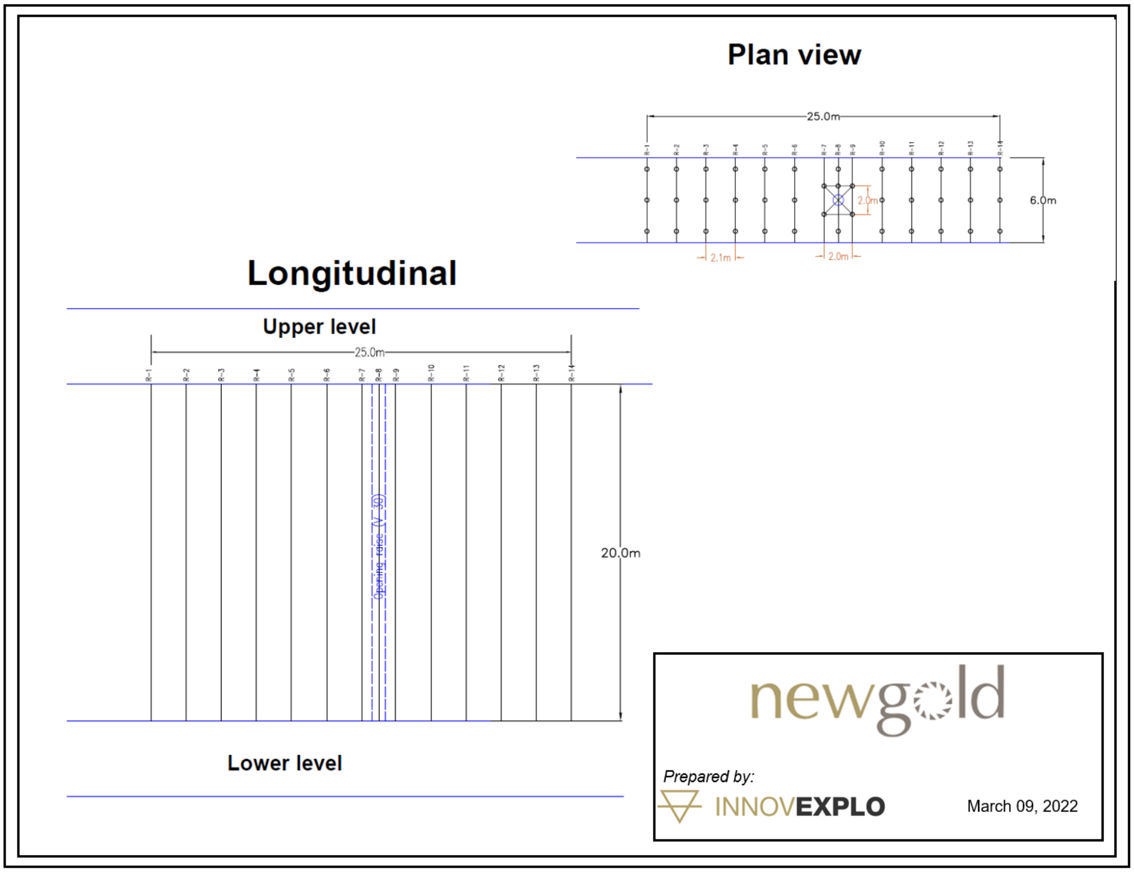

Figure 16.19 – Drill and Blast - Longitudinal and Plan View

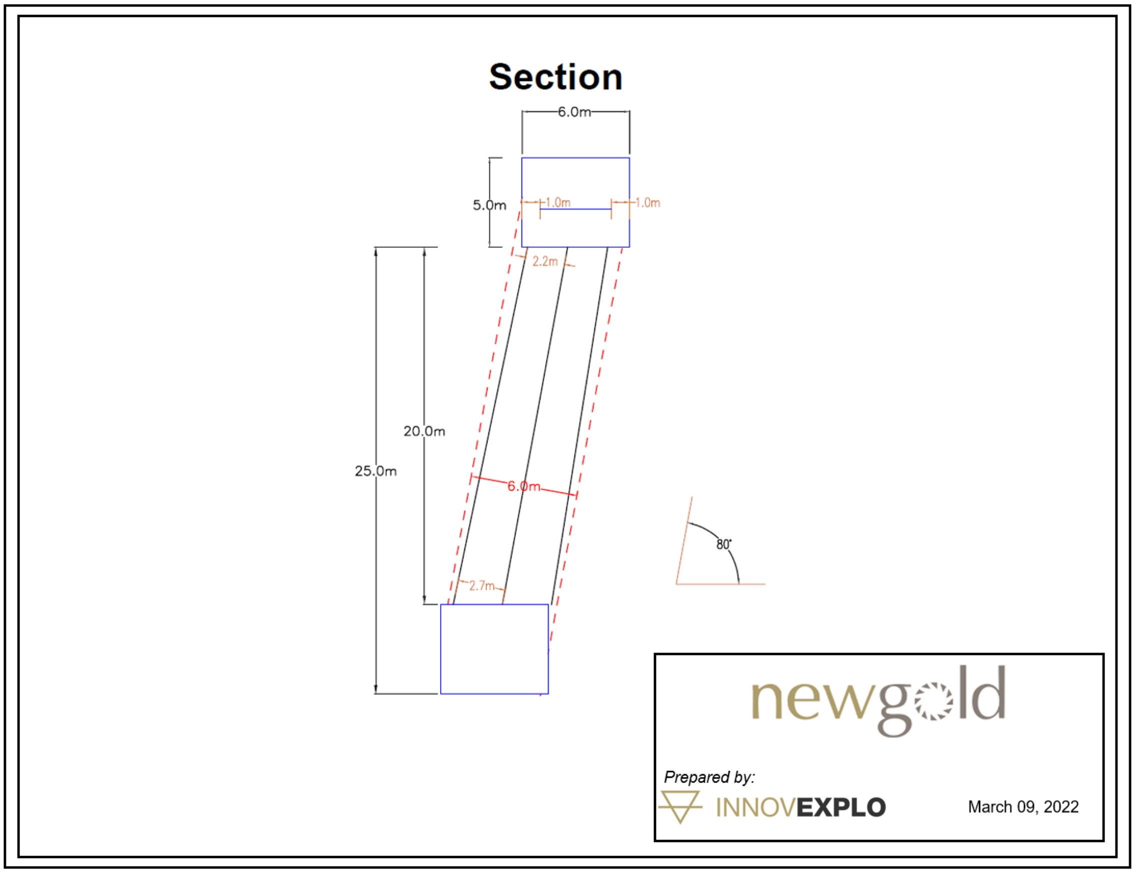

Figure 16.20 – Drill and Blast - Typical Section

16.3.8.4 Backfill

Two types of backfill are used at Rainy River. The primary backfill method is cemented rockfill (CRF) with a 4.0% cement binder, except above sill pillars where the cement binder is increased to 7.0%. This percentage may change depending on the results encountered underground. Simple rockfill will be used as much as possible, especially at the end of a longitudinal sequence, for secondary transverse stopes or for stopes with no direct effects to adjacent excavations.

For the purpose of this technical report, material movement and cost estimation are based on the use of CRF and RF to backfill the stopes. Further analysis to reduce cost by the use of other types of binders to replace cement and the use of a pulling a void in between the backfill stope and the new block to be blasted will be examined to reduce operating costs.

Development waste rock will be used as CRF or rockfill as a priority. Excess waste rock will be stocked in unused or depleted levels, whenever possible. Some remucks may also be used to stockpiled temporarily excess waste rock for future backfill. Excess waste rock will be hauled to the North pit near the 17 East Portal, although waste material hauled to surface is minimized as much as possible.

By the end of the mine life and the slowing of waste development, additional waste material will be required to be crushed and hauled underground from the North Pit waste reserve for backfill purposes. A portable crusher is planned to be installed in the North pit to optimize the granulometry of the backfill rock. A system of screens and sieves is also planned for backfill optimization and to provide material for the roadbed decline. Including maintenance of the pit ramp and underground development, 130k tonnes of crushed rock is required for the roadbed throughout the LOM.

The required backfill strength will depend on the size of the stope and the mining sequence. A typical stope would require a backfill strength of 780 kPa (lab testing) or of 260 kPa (field backfill strength). To maximize safety and flexibility, the curing time used is 21 days before exposing a backfilled face (other activities, like stope preparation and drilling, can start sooner).

Table 16.24 summarizes the backfill schedule (CRM and rockfill) for the UG Main and Intrepid Zones.

Table 16.24 – Backfill Schedule for UG Main and Intrepid Zones

			2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
UG MAIN	CRF	tonnes	-	-	24,276	142,943	350,326	372,454	519,068	508,424	583,711	436,669
	Rockfill	tonnes	-	-	16,000	135,307	299,048	527,707	392,447	341,903	422,025	184,256
INTREPID	CRF	tonnes	55,448	138,657	44,356	84,945	17,673	40,732	4,038	-	-	-
	Rockfill	tonnes	57,231	69,329	45,202	74,759	16,471	74,160	7,126	-	-	-

A SWATcrete contractor was approached for the whole backfilling process using their manpower and equipment to backfill the stopes. The service includes the equipment and operation/supervision but excludes all consumables (cement, maintenance, material haulage, etc.).

The Figure 16.21 presents a typical CRF backfilling operation.

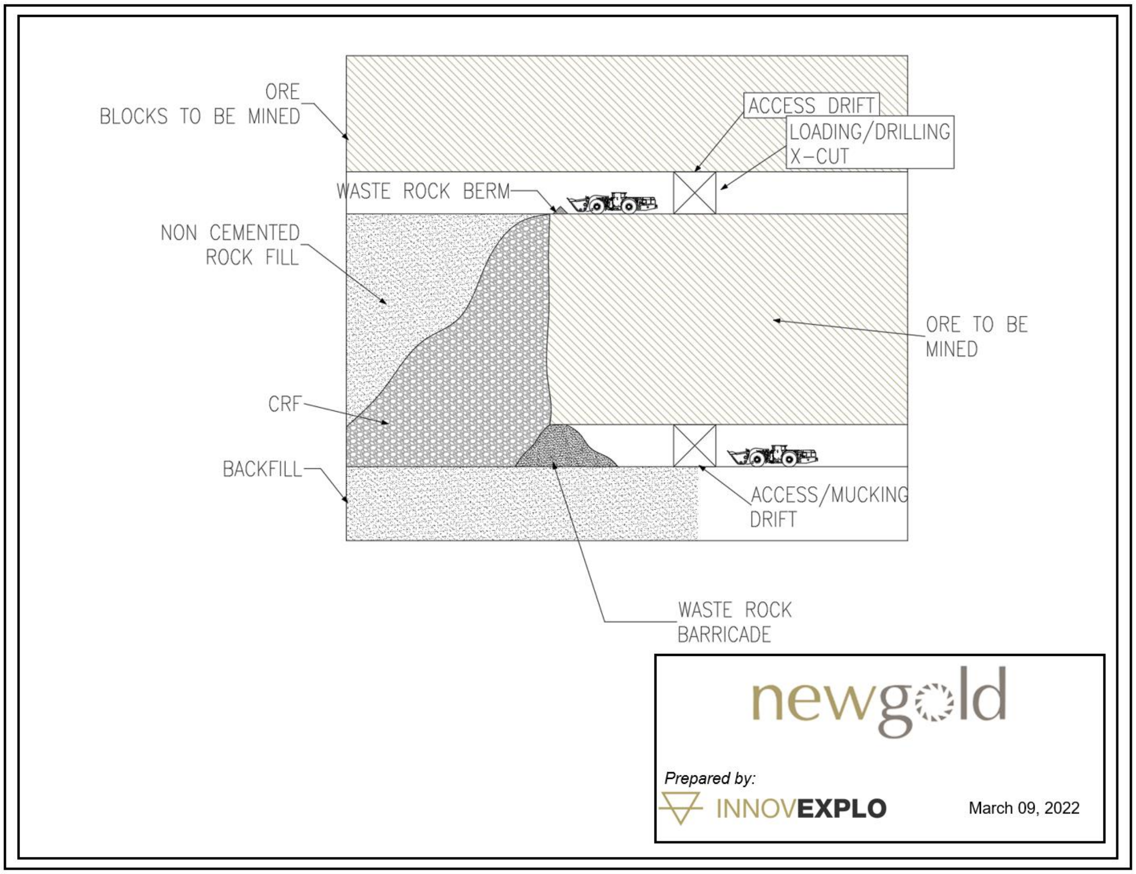

Figure 16.21 – Cemented Rockfill (CRF) Overview

16.3.8.5 Underground production plan

The plan is to access the UG Main Zones from one portal, 17 East, reaching the separate ore zones (in order): 17 East Upper, ODM East, ODM Main, Zone 433, 17 East Lower and ODM West. Another portal, ODM Main Portal, will be constructed in the bottom of the pit once the production is completed. This will provide more flexibility and increase production earlier. The Intrepid Zone is independent of the Main Zones, and the different levels will be accessed through its own portal. These are shown in Figure 16.8 (longitudinal view).

For the UG Main Zones, underground development will start at the end of Q1 2023. The first ore development is planned for Q1 2024, and the first stope is mined in Q2 2024. The commercial production period is scheduled to start Q2 2026, when the mine reaches 4,500 tpd for the first time after three years of pre-production. During the pre-production period, major infrastructures like the main ventilation raises and escapeways will be excavated. All associated equipment will be installed and commissioned.

The main haulage method is trucks hauling ore and waste to the surface.

The LOM plan shows a rapid ramp-up in production in the first year, with production rising to approximately 140,000 oz AuEq per year for the subsequent 5 years only for the main zone. Average gold production is expected to be 190,000 oz/year over the LOM of 8 years. The ounces and other material reported in Item 15 refer to diluted reserves that consider mining recovery and other underground mining factors but do not consider mill recovery. Based on the current Mineral Reserves, Rainy River has a mine life to Q4 2031, but the potential conversion of Mineral Resources and the exploration potential could extend the mine life. Contractors will be employed to develop the two access ramps, major infrastructures, and all development afterward. An average of 8,700 m of horizontal development are realized per year, with a maximum of 15,200 m in 2026.

During the life of the mine, a median of 16 levels in operation are required at the same time, including all main activities.

The Intrepid Zone is the only zone with an existing portal and is independent of the UG Main Zones. It has approximately 2,644 m of lateral development already completed, and development will resume with the current contractor (cementation).

A summary of the underground schedule, overall and by mining area, is provided in Table 16.25 and Table 16.26.

Table 16.25 – Underground Schedule Summary

Zone	Item	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
UG Main	Horizontal Development	m	-	1,685	7,238	11,939	12,586	11,417	9,498	5,580	2,328	-	**62,272**
	Vertical Development	m	-	97	433	380	288	292	377	156	45	-	**2,068**
	Total Development	m	-	1,782	7,671	12,319	12,875	11,709	9,876	5,736	2,373	-	**64,340**
	Ore Production	kt	-	-	165	711	1,133	1,367	1,361	1,378	1,529	1,208	**8,852**
	Ore Development	kt	-	1	114	383	364	273	286	264	97	-	**1,782**
	Total Ore	kt	-	1	280	1,094	1,497	1,639	1,647	1,643	1,625	1,208	**10,634**
	Ore per day (average)	t/d	-	3	764	2,997	4,101	4,492	4,500	4,501	4,453	3,310	**-**
	Gold (g/t)	g/t	-	0.94	2.58	2.87	2.85	2.94	3.03	3.10	3.45	3.07	**3.04**
	Silver (g/t)	g/t	-	2.1	6.5	4.2	3.6	4.2	6.8	6.4	4.3	3.8	**4.9**
	Gold (oz)	koz	-	0	23	101	137	155	160	164	180	119	**1,039**
	Silver (oz)	koz	-	0	59	148	173	221	361	336	224	149	**1,670**
	Waste Produced	kt	-	150	567	648	695	576	523	205	96	-	**3,459**
	Rockfill	kt	-	-	24	143	350	372	519	508	584	437	**2,938**
	Cemented Rock Fill	kt	-	-	16	135	299	528	392	342	422	184	**2,319**
Intrepid	Horizontal Development	m	2,962	2,967	2,869	2,863	2,339	-	-	-	-	-	**14,000**
	Vertical Development	m	84	112	68	74	97	-	-	-	-	-	**435**
	Total Develpment	m	3,046	3,079	2,937	2,937	2,436	-	-	-	-	-	**14,435**
	Ore Production	kt	109	275	382	359	366	321	137	-	-	-	**1,949**
	Ore Development	kt	39	35	-	-	-	-	-	-	-	-	**74**
	Total Ore	kt	147	310	382	359	366	321	137	-	-	-	**2,022**
	Ore per day (average)	t/d	404	849	1,046	984	1,004	880	375	-	-	-	**-**
	Gold (g/t)	g/t	2.46	2.35	3.36	3.35	3.16	3.68	2.46	-	-	-	**3.09**
	Silver (g/t)	g/t	24.2	22.6	24.2	20.7	18.6	22.8	19.1	-	-	-	**21.8**
	Gold (oz)	koz	12	23	41	39	37	38	11	-	-	-	**201**
	Silver (oz)	koz	114	225	297	239	219	235	84	-	-	-	**1,414**
	Waste Produced	kt	196	220	179	222	219	10	-	-	-	-	**1,047**
	Rockfill	kt	79	125	174	200	134	185	44	-	-	-	**940**
	Cemented Rock Fill	kt	34	83	61	91	64	50	16	-	-	-	**399**

Table 16.26 – Underground Summary per Zone

Zone	Item	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
UG Main	17 East Lower												
	Total Ore	kt	-	-	-	-	9	96	275	268	135	284	**1,066**
	Gold (oz)	koz	-	-	-	-	1	9	26	26	13	26	**100**
	Silver (oz)	koz	-	-	-	-	8	73	226	223	123	55	**708**
	Total Development	**m**	**-**	**-**	**-**	**382**	**1,699**	**2,364**	**2,417**	**1,914**	**927**	**-**	**9,703**
	17 East Upper												
	Total Ore	kt	-	-	112	41	-	-	-	-	-	-	**153**
	Gold (oz)	koz	-	-	9	4	-	-	-	-	-	-	**13**
	Silver (oz)	koz	-	-	51	16	-	-	-	-	-	-	**67**
	Total Development	**m**	**-**	**153**	**681**	**-**	**-**	**-**	**-**	**-**	**-**	**-**	**833**
	ODM East												
	Total Ore	kt	-	1	1	285	641	274	425	140	-	-	**1,767**
	Gold (oz)	koz	-	0	0	25	61	29	41	10	-	-	**167**
	Silver (oz)	koz	-	0	0	80	107	51	63	19	-	-	**321**
	Total Development	**m**	**-**	**519**	**1,073**	**3,978**	**3,988**	**948**	**-**	**-**	**-**	**-**	**10,506**
	ODM Main												
	Total Ore	kt	-	-	120	718	847	1,162	755	811	1,269	724	**6,407**
	Gold (oz)	koz	-	-	9	69	74	106	73	77	134	73	**615**
	Silver (oz)	koz	-	-	6	50	58	92	55	58	79	49	**448**
	Total Development	**m**	**-**	**1,013**	**4,177**	**7,272**	**6,541**	**5,343**	**4,330**	**2,905**	**1,446**	**-**	**33,028**
	ODM West												
	Total Ore	kt	-	-	-	-	-	0	26	128	137	125	**417**
	Gold (oz)	koz	-	-	-	-	-	0	2	14	21	13	**50**
	Silver (oz)	koz	-	-	-	-	-	0	9	23	18	43	**93**
	Total Development	**m**	**-**	**-**	**10**	**146**	**298**	**835**	**1,113**	**916**	**-**	**-**	**3,318**
	Zone 433												
	Total Ore	kt	-	-	46	49	-	107	167	296	84	75	**824**
	Gold (oz)	koz	-	-	5	3	-	11	18	37	13	7	**94**

Zone	Item	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	Total
	Silver (oz)	koz	-	-	2	2	-	4	8	12	4	2	**33**
	Total Development	**m**	**-**	**97**	**1,730**	**541**	**349**	**2,219**	**2,016**	**-**	**-**	**-**	**6,952**
Intrepid	Intrepid												
	Total Ore	kt	147	310	382	359	366	321	137	-	-	-	**2,022**
	Gold (oz)	koz	12	23	41	39	37	38	11	-	-	-	**201**
	Silver (oz)	koz	114	225	297	239	219	235	84	-	-	-	**1,414**
	Total Development	**m**	**3,046**	**3,079**	**2,937**	**2,937**	**2,436**	**-**	**-**	**-**	**-**	**-**	**14,435**

16.3.8.6 Truck estimation (UG Main Zones)

Truck operating hours were estimated using Deswik.LHS software. A cycle time was estimated for all hauling tasks in the mine planning, based on the truck parameters provided by Sandvik (i.e., cycle time calculated for truck loading, etc.) The truck cycle times considered design constraints, such as grade and optimal dropping points to surface.

For the waste hauling strategies, trucks will haul to the closest backfilling activities or otherwise to the waste dump (North pit waste reserve).

Figure 16.22 shows the average estimated number of operating trucks per year for the LOM. From this estimate, contingencies and spare equipment have been added to the final cost estimation.

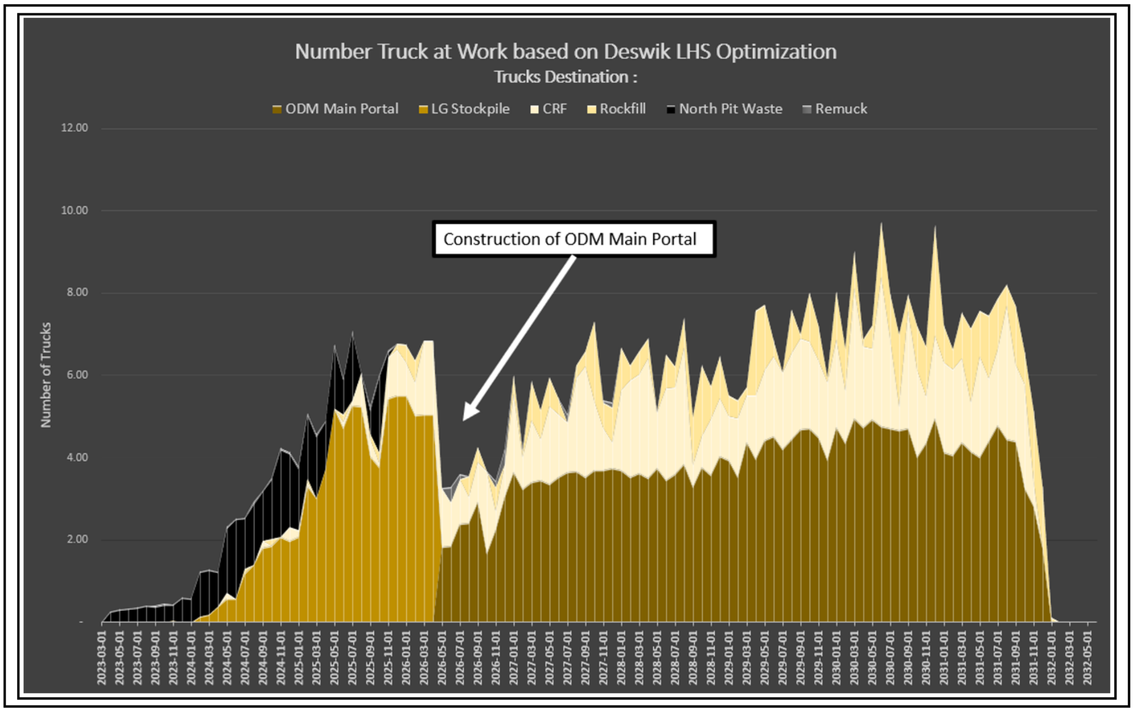

Figure 16.22 – Truck Simulation (Number of Trucks based on Destination)

16.3.9 Mine services

16.3.9.1 UG Main Zones

Electrical services

The electrical distribution was designed in association with ASDR Canada (specialized consultant) based on the requirements for the equipment, such as Jumbos, production drills, bolters, and fans for each level of every ore zone.

All levels will need an electrical substation (600 V) to operate the development and production equipment. All electrical substations are fed by one main electrical station (13,800/600 1.5 MVA). A main electrical station can power 3 to 4 substations.

There are 70 levels in the 6 main zones of the mine (one substation per level). In addition to the stations, there is a requirement for additional elements, such as cable extensions, PTO 3 plug, leaky feeder and optic fibre for communication. There will also be a 15 KV container on surface.

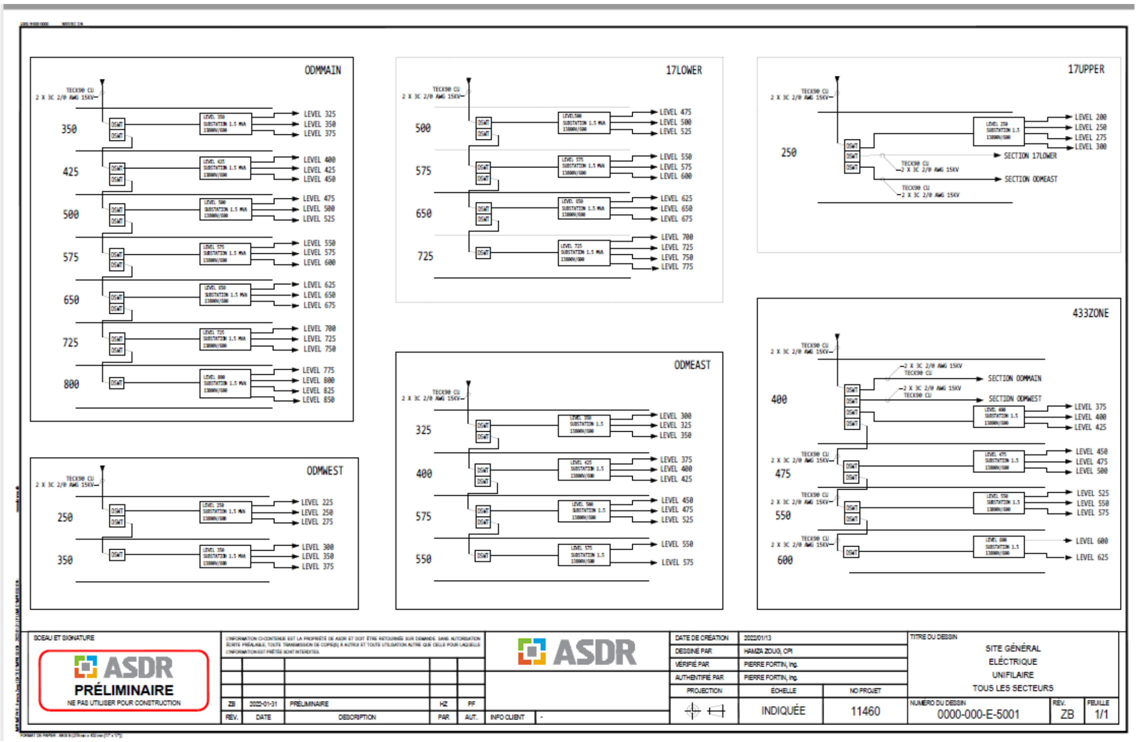

Figure 16.23 – Typical electrical Distribution per Zones

The construction of the electrical stations follows the development sequence as a priority excavation on each level. The cost of the electrical network is 25.8 M $US (CAN$32.2M) over the mine life, being the most expensive services capital cost to feed 70 levels over six ore zones.

16.3.9.2 Communication network

The underground communications network will consist of fibre-backbone between level and a leaky-feeder network for radio communication.and co-wireless equipment, with cyber security recommended based on the size of the mine. This equipment includes non-redundant EPC cores for up to 1,100 users with VoLTE and Push-To-Talk application server with a capacity to support 250 users. Underground personnel will be equipped with smartphones with radio and tracking capability.

The fibre-optic network will be installed between level throughout the ramps as it isthey are developed to facilitate communication with the mine production equipment connected to the underground electrical rooms and mine power centres.

16.3.9.3 Fuel Distribution network

Fuel supply will be stored at surface in two 90,000 L tanks. One underground fuel bay will be positioned strategically near the main underground services. A 2-inch steel pipe and automated pumping system will ensure constant flow to the fuel stations. The fuel line cannot be installed during pre-production until the fuel bay is commissioned. Until then, a service truck will deliver fuel to the equipment underground.

16.3.9.4 Permanent mine pumping network

The pumping flowsheets for each zone were designed in association with Technosub, a specialist in water management for underground operations.

Permanent dewatering is conducted with a system of drain holes and submersible pumps able to handle high solid content. Muddy water is transported by gravity whenever possible, and strategically placed secondary pumping stations allow the water to be transported to the main pumping station. These secondary pumping stations consist of a larger excavation sump to separate muddy water to clear water equip with one or two submersible pumps (Technoprocess 40HP and combination of Tsurumi 5HP on upper sub level), depending on the estimated flow. The estimated inflow per level per zone is 60 gpm and another 72 gpm from equipment when in operation. With a contingency of 10%, it makes 145 gpm per zone for a total of 870 gpm to surface.

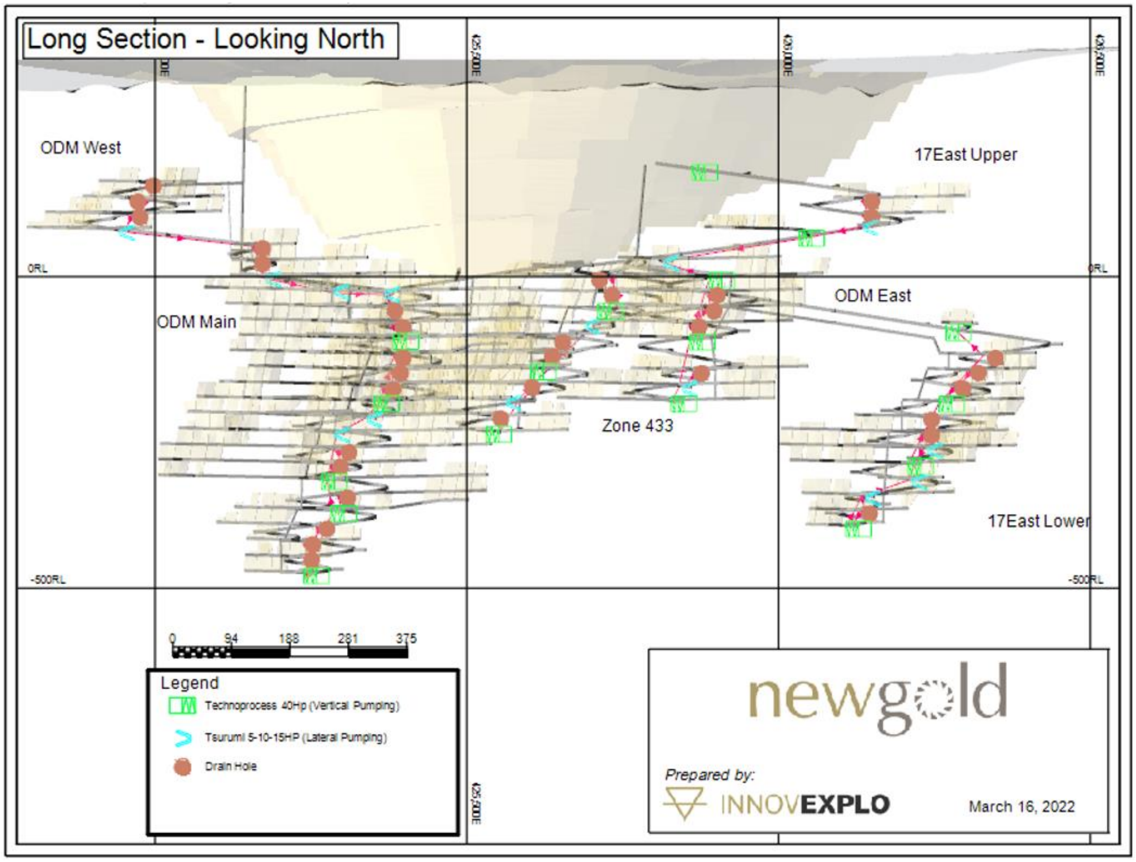

Figure 16.24 – Overview of Pumping Network

Intrepid Zone

Intrepid electrical distribution, communication network, fuel distribution and permanent pumping network will be similar to the ones planned in UG Main Zones. These systems will be installed first as we are developing Intrepid. Lessons learned from these installations in Intrepid will be optimized when it will be time to proceed in the UG Main Zones.

Intrepid is planned entirely with a contractor. All services (electrical, water, pumping, ventilation) are installed by said contractor and are calculated on a linear meter basis.

The standardization of equipment, consumables and methods will need to be reviewed between the UG Main Zones and Intrepid. Costs, logistics and purchasing can be easily improved by combining the two approaches.

16.3.10 Ventilation

16.3.10.1 UG Main Zones

To comply with Ontario regulations concerning underground operations, 1.3 Mcfm will be required at full production. Conservative utilization rates were applied to account for the time when machines may be mechanically unavailable or simply not in use: 75% for production equipment and 50% for most service equipment and machinery that operates primarily with electricity. Table 16.27 shows the ventilation rate for each piece of diesel equipment and the fresh air volumes needed to respect regulation and protect workers.

Table 16.27 – Fresh Air Requirement Ventilation Rate per Diesel-powered Equipment

Name	Power		Air requirement per unit		QTY	Utilization factor	Total Airflow	
(Equipment type)	(kW)	(HP)	(m^3/s)	(cfm)			(m^3/s)	(cfm)
JUMBO	110	147	6.6	13,985	4	50%	**13.2**	**27,958**
975 Bolter	110	147	6.6	13,985	4	50%	**13.2**	**27,958**
Scissor Lift	110	147	6.6	13,985	3	50%	**9.9**	**20,968**
Anfo Loader	88	118	5.3	11,188	2	50%	**5.3**	**11,183**
Production LHD	310	415	18.6	39,413	2	75%	**27.9**	**59,092**
Minetruck 50tm	515	690	30.9	65,477	2	75%	**46.4**	**98,169**
Production LHD	310	415	18.6	39,413	4	75%	**55.8**	**118,184**
Minetruck 50tm	515	690	30.9	65,477	10	75%	**231.8**	**490,847**
Production LHD	310	415	18.6	39,413	3	75%	**41.9**	**88,638**
Swatcrete equipment	170	228	10.2	21,614	2	75%	**15.3**	**32,405**
Production Drill	110	147	6.6	13,985	3	50%	**9.9**	**20,968**
Explosive truck (Emulsion)	155	208	9.3	19,707	1	75%	**7.0**	**14,773**
ITH Cubex Drill (slot raise)	105	141	6.3	13,350	1	50%	**3.2**	**6,672**
Cable Bolter	110	147	6.6	13,985	1	75%	**5.0**	**10,484**
Scissor Lift	110	147	6.6	13,985	1	50%	**3.3**	**6,989**
Shotcrete Sprayer	75	101	4.5	9,536	1	50%	**2.3**	**4,766**
Fuel - Lube truck	148	198	8.9	18,817	1	50%	**4.4**	**9,404**
Boom truck	110	147	6.6	13,985	1	50%	**3.3**	**6,989**
Deck Truck - CRF service vehicule	110	147	6.6	13,985	1	50%	**3.3**	**6,989**
Underground Grader	93	125	5.6	11,824	1	50%	**2.8**	**5,909**
Water Cannon	110	147	6.6	13,985	1	50%	**3.3**	**6,989**
Services LHD	256	343	15.4	32,548	1	50%	**7.7**	**16,266**

Name	Power		Air requirement per unit		QTY	Utilization factor	Total Airflow	
(Equipment type)	(kW)	(HP)	(m³/s)	(cfm)			(m³/s)	(cfm)
Personnel Carrier	130	174	7.8	16,528	2	50%	**7.8**	**16,520**
Light vehicule	95	127	5.7	12,078	12	50%	**34.2**	**72,436**
Mine rescue - Light vehicule	95	127	5.7	12,078	1	50%	**2.9**	**6,036**
						Total:	**560.7**	**1,187,594**
						Contingency 10%:	**56.1**	**118,759**
						Total with contingency :	**616.8**	**1,306,354**

Note: Occupational Health and Safety Act, R.R.O. 1990, REGULATION 854, MINES AND MINING PLANTS. Art. **183.1** (1)

16.3.10.2 Ventilation network

A step-by-step ventilation system will be implemented to allow for continuity in the production from the open pit. A first network will be put in place during the pre-production phase as a temporary system. Once the primary ventilation circuit will be completed, the primary ventilation system will be ready to be put into operation and in two phases: a first network during the operation of the open pit mine and a second when the operation of the pit is completed.

Ventilation Network during pre-production

Development will begin by excavating a portal located between the two open pits. The objective is to reach the ventilation raise to create a first internal air circuit.

The fresh air requirements for the first section of the ramp have been estimated for two trucks and a loader (170 kcfm). The ventilation strategy during the pre-production phase is to use two 48'' lines of rigid ducting which will each provide 85 kcfm. The fans used are 2 x 200hp in series per ducting.

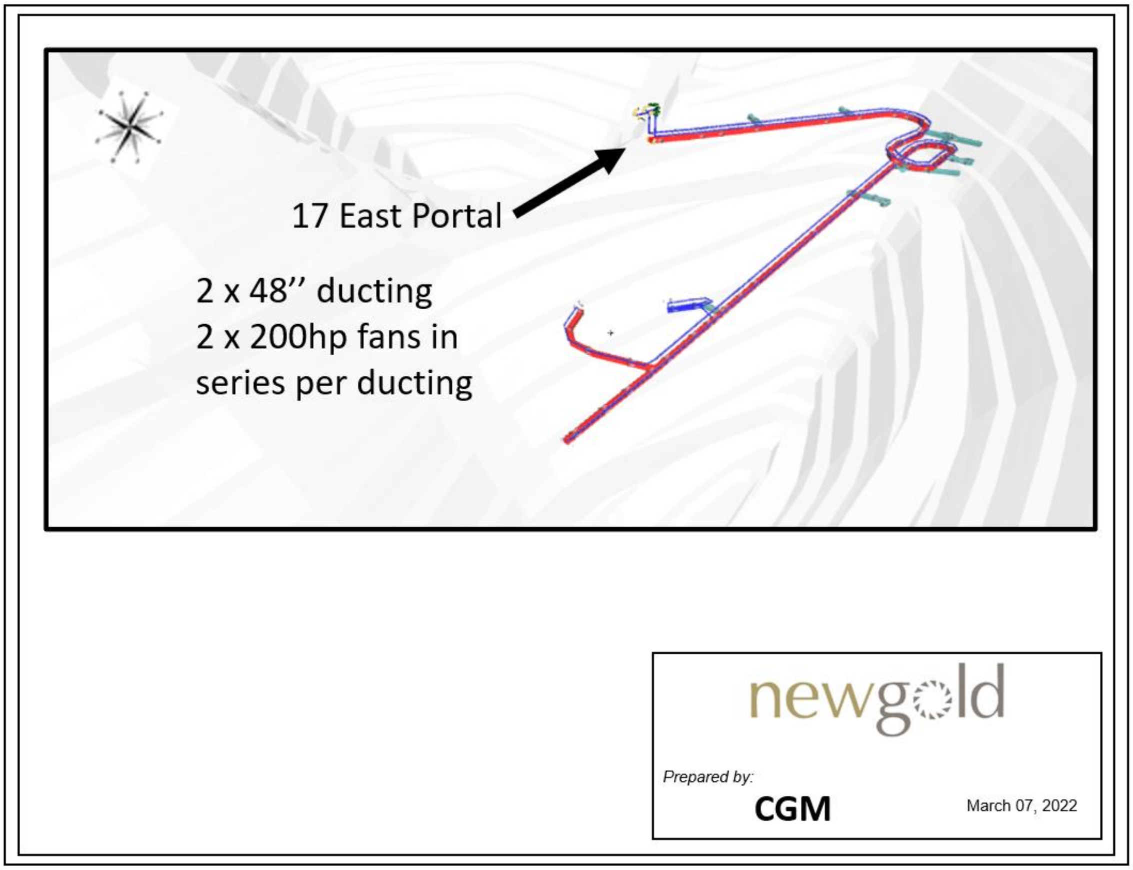

Figure 16.25 – Pre-production Ventilation Arrangement towards East Ventilation Raise

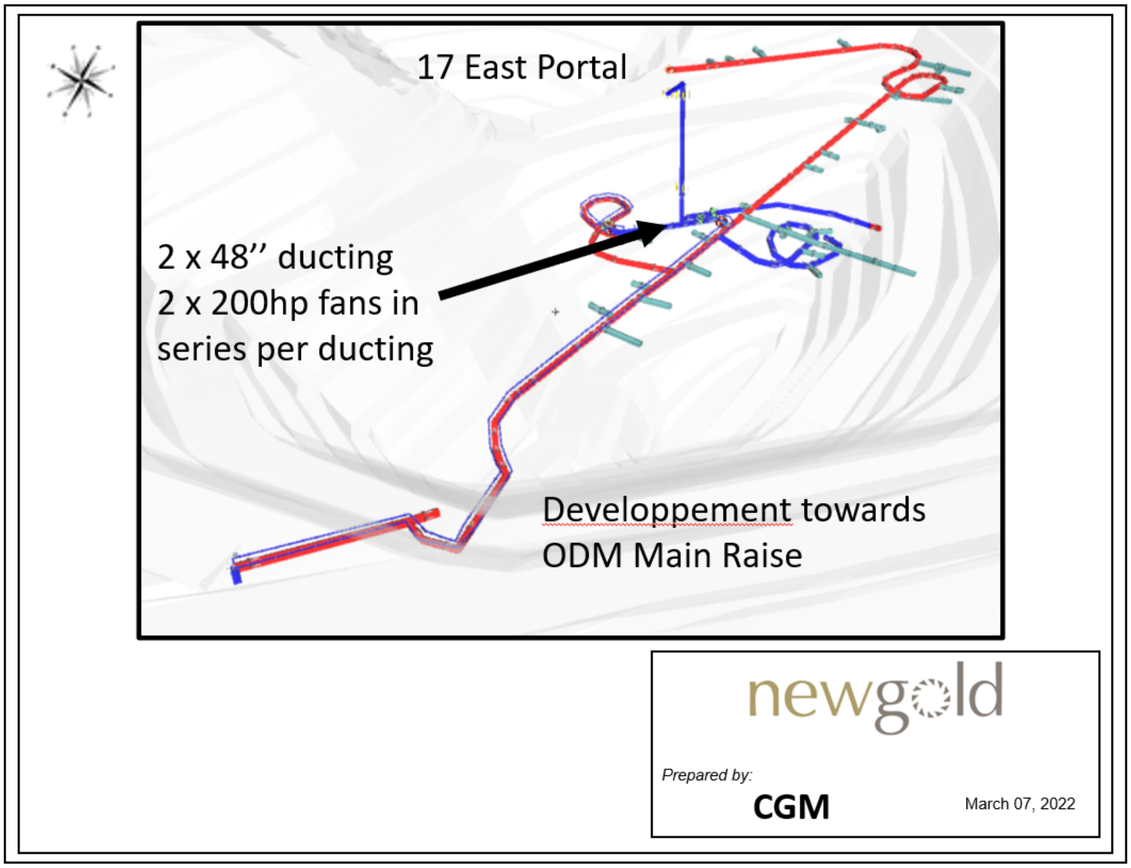

Figure 16.26 – Pre-production Ventilation Arrangement towards West Ventilation Raise

Ventilation Network during production – Phase 1

To be able to operate with all the equipment necessary and to achieve sustained full production of 4,500 tpd, some infrastructure components need to be installed prior to initial production.

The ventilation network for Phase 1 of the mine consists of generating a mechanical air thrust through the intake raise (FAR) located above the ODM Main Zone. Fans capable of delivering 800 kcfm (2 x 400 kcfm) in the network will be installed in parallel above this FAR. During this period, the East ventilation raise and the East Portal will be used as exhaust, as presented in Figure 16.27.

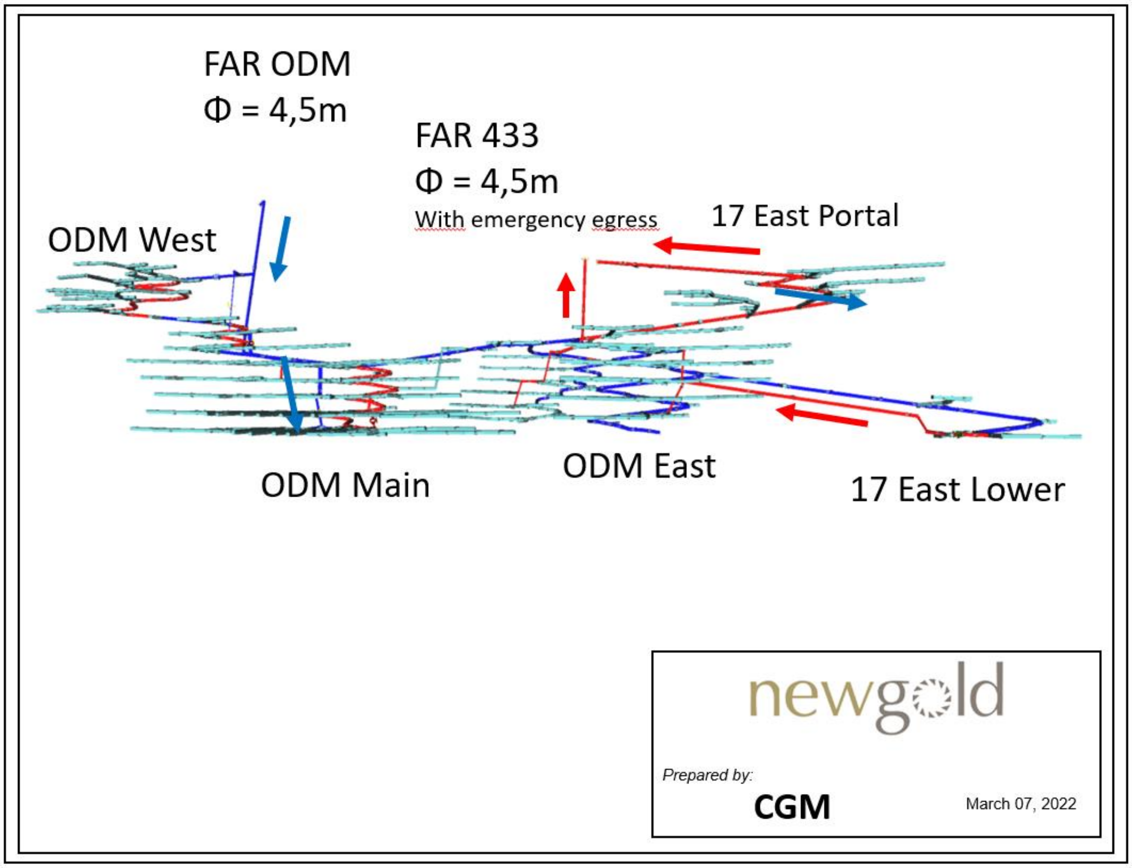

Figure 16.27 – Longitudinal View of the Ventilation System (Production Phase 1)

Ventilation Network during production – Phase 2

During the second phase of production (once the production of the pit is completed), the ventilation system will become a push-push system. The fan system already installed on the West Raise will remain in place, and a new fan system capable of bringing an additional 500 kcfm to the network will be installed on top of the East Raise (formerly an exhaust). An additional portal (West) will be connected to the bottom of the open pit, allowing for a second exhaust. Figure 16.28 presents the longitudinal view of the ventilation network.

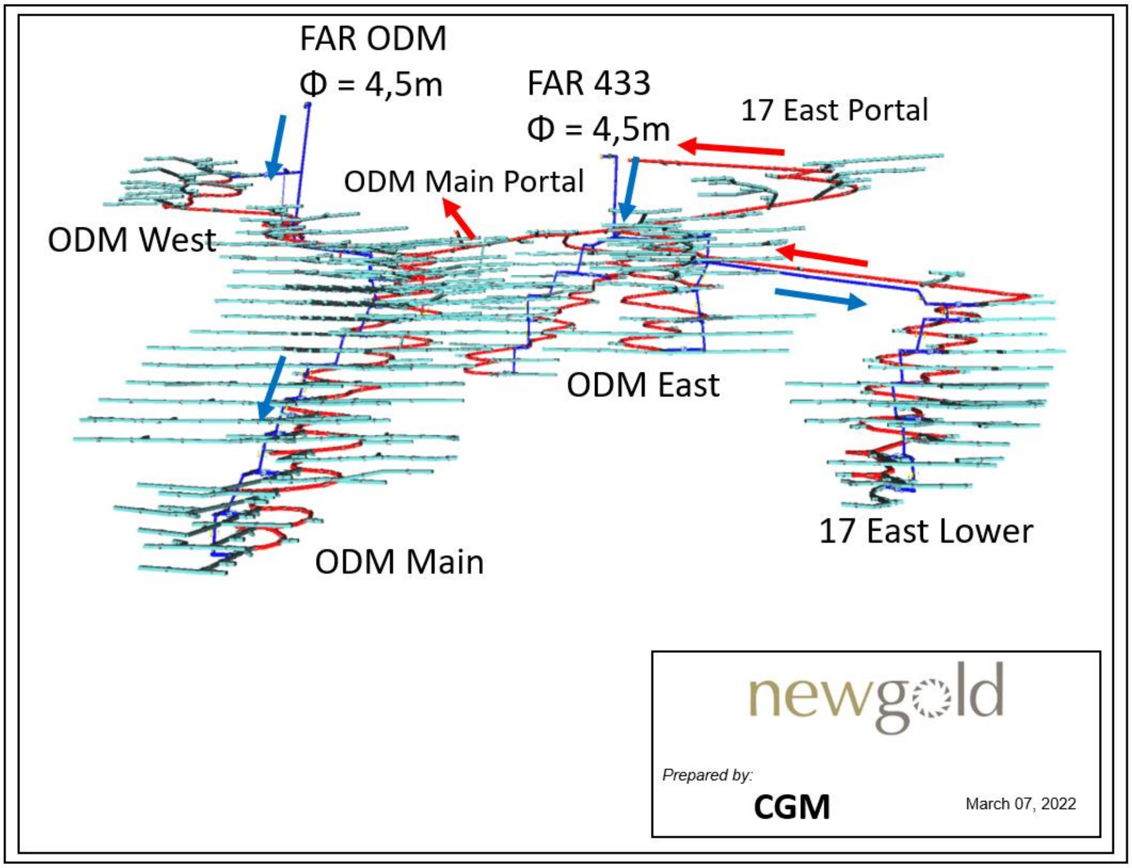

Figure 16.28 – Longitudinal View of the Ventilation System (Production Phase 2)

Typical level ventilation

Two situations were assessed to establish the ventilation requirement on the levels. The needs were established by considering one truck and one LHD per level. The truck will be loaded at the loading bay. Figure 16.29 shows the situation when the air is taken into the ramp, and the Figure 16.29 shows the situation when the internal raise breaks through the level. In general, two auxiliary fans of 100hp are necessary to ventilate a level. A 48" flexible duct has been considered for the design.

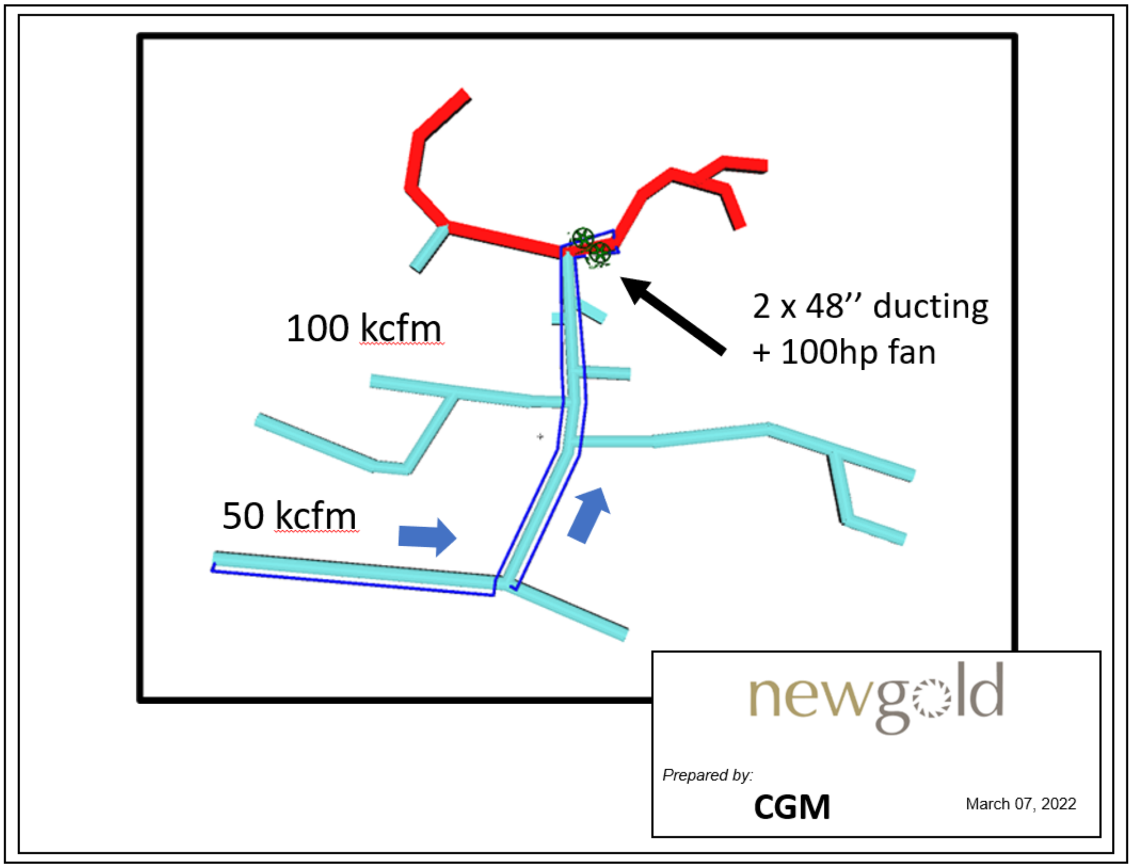

Figure 16.29 – Typical Auxiliary Ventilation Arrangement for a Level when the air is taken from the Ramp

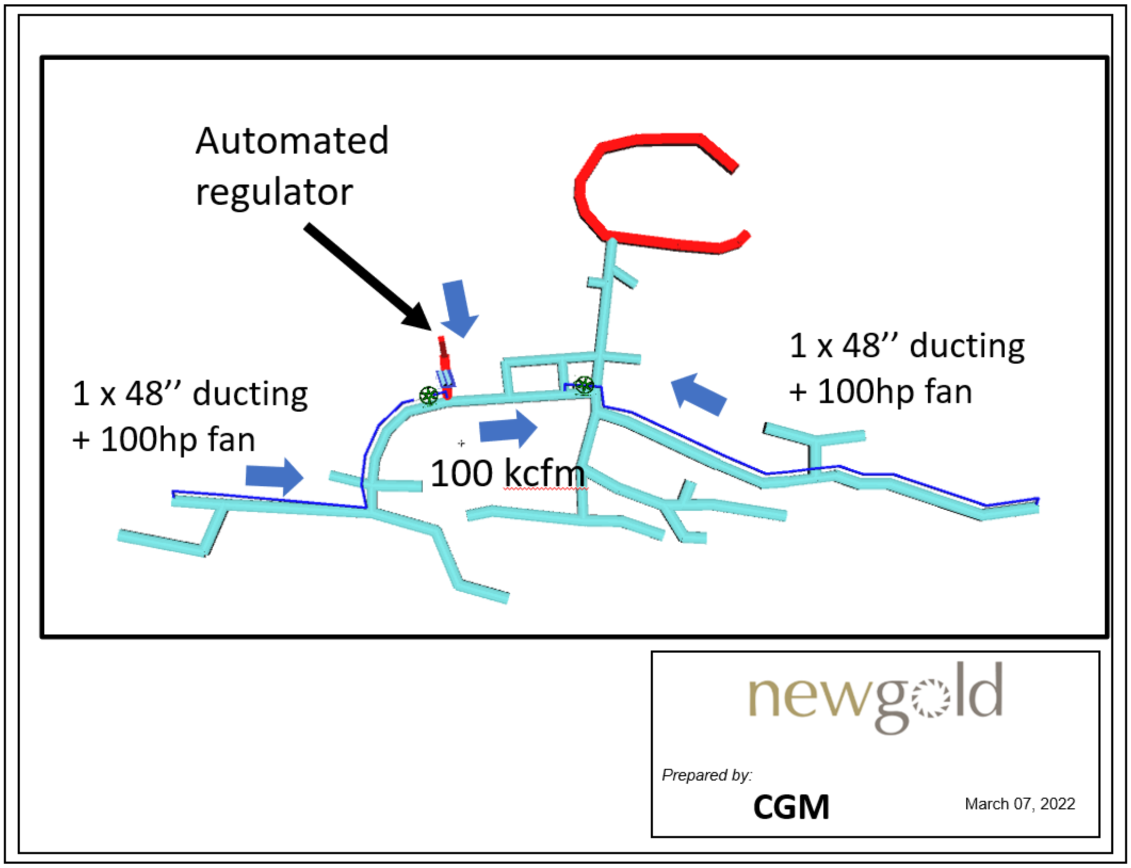

Figure 16.30 – Typical Auxiliary Ventilation Arrangement for a Level when the air is taken from the Internal Raise

Main fans and heating system

The ventilation will consist of two 4.5-m diameter intake raises (East and West) and exhaust by the ramp when the mine is in full production. The East fans will supply 500,000 cfm with 2 fans with motor (Model AFN SO 15 1200 2157 Arrangement 8. 700HP each). The West fans will supply 800,000 cfm with 2x Fan with motor (Model AFN SO 15 1500 2500 Arrangement 8. 2000HP each).

The East fan raise will also serve as a secondary egress with all the ground support required, and construction must be completed before commissioning the fan installation.

Both will have burners. For 800 000 CFM, six 13' long burners total is required. For the 500 000 CFM duty point, four 13' long burners are recommended to possibly defer CAPEX expenditures and reuse the equipment at the different duty points.

Intrepid Zone

The main fan system at Intrepid is already installed at surface (refer to Item 0). The basis of the ventilation network is straightforward. Each level is connected to the main ventilation raise network by ventilation accesses. Ventilation walls and doors are installed where needed (every active level). Additional fans and booster fans are used to ventilate the production drifts. Fans are moved and reused as levels become inactive.

The system at full capacity will generate 420,000 cfm to allow production in the lower level and satisfactory ventilation of all active levels.

Figure 16.31 presents an overview of the Intrepid ventilation network.

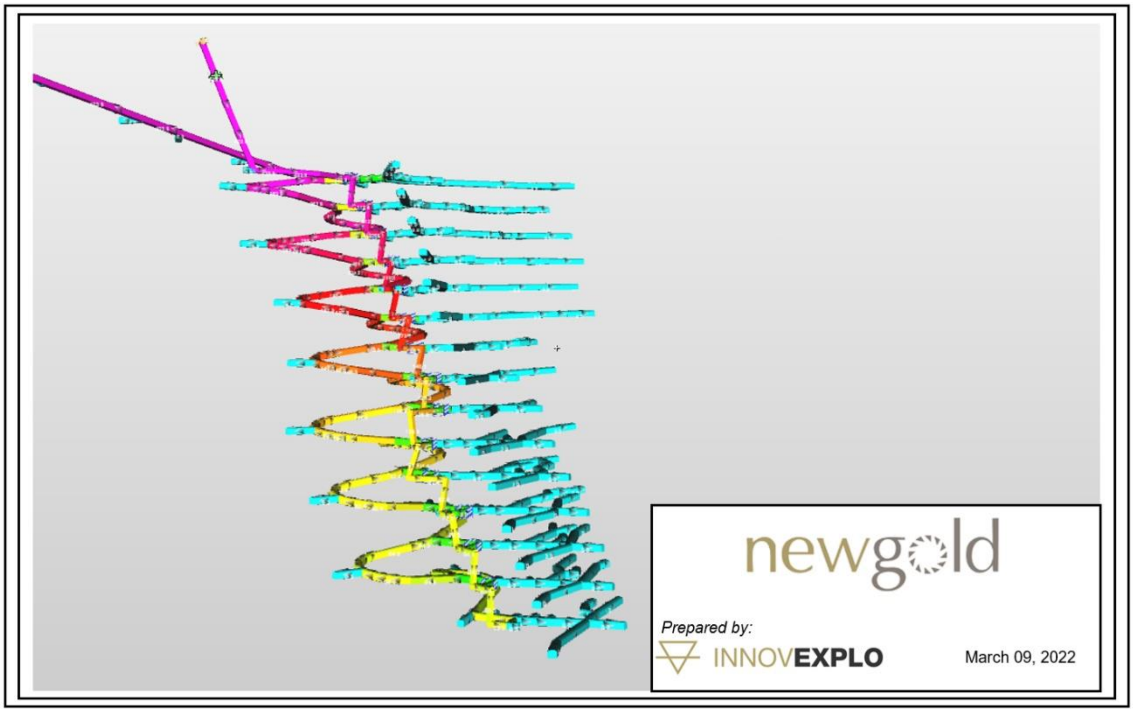

Figure 16.31 – Ventilation Network Overview (Intrepid)

16.3.11 Underground mine equipment selection and fleet requirement

The required operational quantities for all major and critical equipment (jumbo, cable bolter, production drills, LHDs, trucks, etc.) were estimated during the planning process. Yearly operation hours have been estimated for all other secondary services equipment based on typical operation and current mine scheduling requirements. For secondary equipment, yearly operation hours range between 1,200 and 2,400 (20% to 50% utilization).

All equipment listed in this study should be acquired by Rainy River between 2022 and 2028. The cash flow does not include the equipment required before the underground production; it is assumed contractors will provide the required development fleet during pre-production.

The required mobile equipment fleet acquired by the owner for the UG Main Zones is presented in Table 16.28 by year.

The estimated necessary contractor fleet for the UG Main Zones is presented in

Table 16.29 by year.

Table 16.28 – Equipment Distribution for UG Main (Owner)

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Production Drill		0	1	2	2	3	3	2	3	2
Explosif Truck (Emulsion)		0	1	1	1	1	1	1	1	1
LHD Mucking		0	1	2	3	4	4	4	4	4
LHD Backfilling		0	1	1	2	3	2	2	3	2
Truck 50TM		0	2	5	5	6	6	7	9	7
Scissor Lift		1	1	1	1	1	1	1	1	1
Boom Truck		0	1	1	1	1	1	1	1	1
Personnel Carrier		0	1	1	1	1	1	1	1	1
Mechanical Truck		1	1	1	1	1	1	1	1	1
Deck Truck - CRF service vehicule		0	1	1	1	1	1	1	1	1
Fuel-Lube Truck		0	1	1	1	1	1	1	1	1
Underground Grader		0	1	1	1	1	1	1	1	1
Water Truck		0	1	1	1	1	1	1	1	1
Electric service vehicle		0	1	1	1	1	1	1	1	1
Light vehicle		3	6	12	12	12	12	12	12	10
Mine Rescue - Light vehicle		1	1	1	1	1	1	1	1	1
Mobile Air Compressor		1	3	3	3	3	3	3	3	3
Electric Lift		0	1	1	1	1	1	1	1	1
Excavator		1	1	1	1	1	1	1	1	1
Services LHD		1	1	1	1	1	1	1	1	1
Cassette Carrier CS3		1	1	1	1	1	1	1	1	1
Total - Equipment Distribution - Owner	0	10	29	40	42	46	45	45	49	43

Table 16.29 – Equipment Distribution for UG Main (Contractor)

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Jumbo - 2 booms		1	3	4	4	4	3	2	1	
Bolter		2	4	6	6	6	5	3	2	
Scissor Lift (Development)		1	3	4	4	4	3	2	1	
LHD (Development)		1	2	3	3	3	3	2	1	
Truck 50tm (Development)		1	2	2	2	2	2	1	1	
Explosif Truck (Anfo)		1	2	2	2	2	2	1	1	
Raise Bore		1	1	1	0	0	1	0	0	
ITH Drill (Drop Raise & V-30)		0	0	1	1	1	1	1	1	1
Mechanic Service vehicule		1	1	1	1	1	1	1	1	
Electric service vehicule		1	1	1	1	1	1	1	1	
Light vehicle		1	2	2	2	2	2	2	2	
Total - Equipment Distribution - Contractor	0	11	21	27	26	26	24	16	11	1

The Intrepid Zone, already in production, will continue with the existing fleet. Equipment will eventually be shared between the two zones to accommodate the varying production rate required per year.

The estimated mobile equipment fleet by year for the Intrepid Zone is presented in Table 16.30.

Table 16.30 – Equipment Distribution for Intrepid Zone

Description	2022	2023	2024	2025	2026	2027	2028
Production Drill	1	2	2	1	1	1	1
Explosif Truck (Emulsion)	1	2	3	2	1	1	1
LHD Mucking	1	2	3	2	1	1	1
Jumbo	1	2	3	2	1	1	1
Bolter	1	2	3	2	1	1	1
Truck 40TM	1	2	2	1	1	1	1
Scissor Lift	1	2	2	1	1	1	1
Boom Truck	1	1	1	1	1	1	1
Personnel Carrier	1	3	3	2	2	2	1
Underground Grader	1	1	1	1	1	1	1
Electric Lift	1	1	1	1	1	1	1
Face Charger (Anfo)	1	2	3	2	1	1	1
Cassette Carrier CS3	1	1	1	1	1	1	1
Total - Intrepid - Equipment Distribution	**13**	**23**	**28**	**19**	**14**	**14**	**13**

16.3.12 Mine personnel

Mine personnel are split between three areas: technical services, maintenance and supervision (mechanical and electrical), and underground operations (construction, development, and production).

Operators and maintenance personnel generally work on a 7-7 schedule. This results in four crews alternating days and nights, when necessary. The electrical and mechanical supervisors will alternate day and night shifts at times; a supervisor or senior employee will always be present to oversee the shifts. Additional supervisors, technicians and some specific workers will work Monday to Friday on a 5-2/4-3 schedule, day shifts only.

In addition, all pre-production and development will be completed by a contractor. The costs associated with the use of a contractor have been summarized in Item 21

16.3.12.1 Mine operations, services and construction

The operators include those required for the major and secondary equipment, as well as blasters. Underground supervision includes a supervisor, trainer, and some operators related to service tasks. The list of underground operation, services and construction personnel required over the life of the mine is presented in Table 16.31.

Table 16.31 – Mine Personnel - UG Main Zones - Operations and Services

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Support service		0	3	4	4	4	4	4	4	3
Support service - CRF		0	3	4	4	4	4	4	4	3
Support service - CS3 Cassette		2	2	2	2	2	2	2	2	2
Support service - Mens Carrier/Water truck		2	2	2	2	2	2	2	2	2
Support service - Production		0	1	2	2	2	2	2	2	2
Grader operator		0	3	4	4	4	4	4	4	3
Construction/Shotcrete miner		0	3	4	4	4	4	4	4	3
Shotcrete construction		0	0	0	0	0	0	0	0	0
Production drill operator (JN)		0	1	5	7	10	9	8	10	6
Production drill operator (J)		0	0	0	0	0	0	0	0	0
Blaster Production		0	2	4	4	4	4	4	4	4
Cable drill operator		0	0	0	0	0	0	0	0	0
Cable installer		0	0	0	0	0	0	0	0	0
Scoop operator - Mucking		0	2	7	12	14	14	14	16	13
Truck operator - Hauling		0	8	19	20	21	22	28	34	25
Truck operator - Backfilling		0	0	0	0	0	0	0	0	0
Scoop operator - Backfilling		0	0	3	6	8	8	7	9	5
Total UG Main Zones - Maintenance & Operations	0	3	29	60	71	79	79	83	95	71

16.3.12.2 Supervision and maintenance personnel

Maintenance staff includes mechanics and electricians for the underground mine; the crew includes a full operational team able to fulfil preventive and unplanned maintenance. A list of underground maintenance personnel required over the life of the mine is presented in Table 16.32.

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Table 16.32 – Mine Personnel - UG Main Zones - Supervision & Maintenance

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Maintenance superintendant		0	0	0	1	1	1	1	1	1
Surface supervisor		0	0	0	2	2	2	2	2	2
Mechanics supervisor		1	4	4	4	4	4	4	4	3
Maintenance planning supervisor		0	1	2	2	2	2	2	2	2
Maintenance planner mechanic/electric		0	2	2	2	2	2	2	2	2
Reliability technician		0	1	2	2	2	2	2	2	2
Mobile mechanic		0	0	0	0	0	0	0	0	0
Senior mechanic			4	8	10	12	12	12	14	10
Field mechanic		1	2	4	5	6	6	6	7	5
Electromechanic			2	4	5	6	6	6	7	5
Junior mechanic			1	2	3	3	3	3	4	3
Welder		0	0	1	2	2	2	2	2	2
Fuel & Lube attendant		0	3	4	4	4	4	4	4	3
Fixe mechanic surface		0	0	0	2	2	2	2	2	2
Fixe mechanic underground		0	1	2	2	2	2	2	2	2
Loader operator		0	0	0	0	0	0	0	0	0
Maintenance assistant superintendant		1	1	1	1	1	1	1	1	1
Electrical supervisor		0	1	2	2	2	2	2	2	2
Instrumentation technician		0	1	2	2	2	2	2	2	2
Electrician		0	2	4	4	4	4	4	4	3
Electrician (J/N)		2	4	4	4	4	4	4	4	3
Automatisation/Communication specialist		0	1	2	2	2	2	2	2	2
Electrician construction		0	0	0	0	0	0	0	0	0
Mine superintendant		0	0	0	1	1	1	1	1	1
Mine assistant superintendant		0	1	1	1	1	1	1	1	1
Mine Captain		2	2	2	2	2	2	2	2	2
Supervisors		2	3	6	8	8	8	8	8	6
Production technician		0	0	0	0	0	0	0	0	0
Mine trainer		0	2	4	4	4	4	4	4	3

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Total UG Main Zones - Maintenance & Supervision	0	10	38	60	75	81	80	81	86	66

16.3.12.3 Technical services

Most of the staff in technical services work at the mine site office during the day, with weekends off (5-2 schedule). A list of technical services personnel required over the life of the mine is shown in Table 16.33.

Table 16.33 – Mine Personnel – UG Main Zones – Technical Services

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Senior geologist		0	0	0	1	1	1	1	1	1
Operations geologists		0	2	2	2	2	2	2	2	2
Exploration geologist		0	0	0	0	0	0	0	0	0
Geology – data integration technician		0	0	1	1	1	1	1	1	1
Geology field technicians		0	0	2	2	2	2	2	2	2
Technical services superintendent		0	0	0	1	1	1	1	1	1
Senior planning engineer		1	1	1	1	1	1	1	1	1
Planning engineer		0	0	1	1	1	1	1	1	1
Planning technician		0	1	1	1	1	1	1	1	1
Drill and blast engineer		0	0	1	1	1	1	1	1	1
Drill and blast technicians		0	1	2	2	2	2	2	2	2
Project engineer		0	1	1	1	1	1	1	1	1
Project technician		0	1	1	1	1	1	1	1	1
Ventilation technician		0	0	1	1	1	1	1	1	1
Surveyors		1	3	4	4	4	4	4	4	3
Rock mechanic engineer		0	0	1	1	1	1	1	1	1
Ground control technician		0	1	1	1	1	1	1	1	1
Total UG Main Zones – Technical Services	**0**	**3**	**11**	**18**	**22**	**22**	**22**	**22**	**22**	**17**

It is assumed that the required technical team for the UG Main Zones would also be able to manage the Intrepid Zone. Table 16.34 presents an estimate of the allocated technical resources specific to the Intrepid Zone.

Table 16.34 – Mine Personnel – Intrepid Zone – Technical Services

Description	2022*	2023*	2024*	2025	2026	2027	2028	2029	2030	2031
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			1	3	3	3	3			
Mine geologists				3	3	3	3			
Ground control technician				1	1	1	1			
Ground control engineer			0	1	1	1	1			
Total Intrepid - Technical Services	0	0	3	14	14	14	14	0	0	0

Note: *Mine technical staff covered by open pit personnel from 2020–2024.

16.3.12.4 Contractors

All contractors will work on a 7-7 schedule, or whichever schedule is the most suitable to achieve the given production/development targets. A list (to be confirmed by the contractor) of personnel required over the life of the mine is shown in Table 16.35.

Table 16.35 – Mine Personnel – UG Main Zones – Contractors

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Jumbo operators		3	10	16	15	16	12	7	3	
Bolter operators		5	15	23	23	23	17	10	4	
Development workers		3	10	16	15	16	12	7	3	
Scoop operators - Dev		2	8	12	12	12	9	5	2	
Truck operators - Dev		1	4	6	6	6	5	3	1	
Blaster development workers		2	5	8	8	8	6	3	1	
Raisebore operators/long-hole drillers		0	2	4	3	3	3	2	0	
Cable stopes workers		0	0	1	1	1	1	1	1	
ITH V-30 drillers			0	2	2	2	2	3	3	
Mobile mechanics - Lateral		3	11	18	17	18	13	8	3	
Mobile mechanics - Vertical		2	2	2	2	2	2	2	0	
Mine captains		2	2	2	2	2	2	2	2	
Supervisors		2	4	4	4	4	4	4	4	
Clerks		0	2	2	2	2	2	2	0	
Total UG Main Zones – Contractor	0	26	76	115	113	113	89	58	28	0

Intrepid will continue its production plan with a full contractor development and production crew. The estimated contractor personnel list for both Operations and Maintenance are presented in Table 16.36 and Table 16.37.

Table 16.36 – Mine Personnel – Intrepid Zone – Operations Contractor

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
Mine shift supervisors	4	4	4	4	4	8	8	8	8	
Jumbo operators	4	4	4	8	8	16	12	4	4	
Long-hole drill operators	4	4	4	4	4	8	8	8	4	
Scoop operators	2	2	2	2	8	16	16	16	4	
Truck drivers	2	2	2	2	8	12	12	12	4	
Trainer (equipment and safety)	1	1	1	1	1	1	1	1	1	
Diamond drillers	2	2	2	2	2	2	2	2	-	
Blasters	6	6	6	6	12	24	18	12	12	
Bolters and ground support	4	4	4	8	8	16	12	4	4	
Grader operator	1	1	1	1	1	1	1	1	1	
General labourers	3	3	3	4	8	8	8	8	4	
Services	4	4	4	4	8	12	12	4	4	
Total Intrepid - Operations (Contractor)	37	37	37	46	72	124	110	80	50	0

Table 16.37 – Mine Personnel – Intrepid Zone – Maintenance Contractor

Description	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031
UG warehouse workers	2	2	2	2	2	2	2	2	2	
Mechanical foreman	1	1	1	1	1	1	1	1	1	
Lead hand mechanic	1	1	1	1	1	1	1	1	1	
Welders	1	1	1	1	3	3	3	1	1	
Mechanics	8	8	8	8	8	16	16	12	12	
Electrical foreman	1	1	1	1	1	1	1	1	1	
Electricians	4	4	4	8	8	8	8	4	4	
Labourers	1	1	1	4	4	4	4	4	4	
Total Intrepid - Maintenance (Contractor)	19	19	19	26	28	36	36	26	26	0

16.4 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 70.2 Mt of ore grading 1.24 g/t gold and 3.11 g/t silver, totaling 2,799 koz of contained gold and 7,022 koz of contained silver. The mine-to-mill schedule is shown in Table 16.38 and the mill feed by source in Table 16.39.

Table 16.38 – Mine-to-Mill Schedule

Year	Tonnes (t)	Gold (g/t)	Silver (g/t)	Contained Gold (oz)	Contained Silver (oz)
2022	9,463,416	0.97	2.4	294,761	726,742
2023	9,855,000	0.97	3.0	306,749	954,765
2024	9,855,000	1.10	2.9	348,195	922,174
2025	9,855,000	1.14	3.0	360,803	949,478
2026	9,855,000	1.15	3.0	363,408	946,209
2027	9,855,000	1.15	3.2	365,232	1,005,945
2028	7,000,652	1.31	3.6	295,582	802,173
2029	1,643,071	3.10	6.4	163,874	335,833
2030	1,625,515	3.45	4.3	180,075	223,783
2031	1,212,232	3.07	4.0	119,611	155,068
Total	70,219,886	1.24	3.1	2,798,288	7,022,170

Table 16.39 – Mill Feed by Source

Description	Open Pit (incl. stockpiles)	Underground	Total
2022	9,316,450	146,966	**9,463,416**
2023	9,543,879	311,121	**9,855,000**
2024	9,195,029	659,971	**9,855,000**
2025	8,404,106	1,450,894	**9,855,000**
2026	7,992,969	1,862,031	**9,855,000**
2027	7,894,511	1,960,489	**9,855,000**
2028	5,216,358	1,784,294	**7,000,652**
2029		1,643,071	**1,643,071**
2030		1,625,515	**1,625,515**
2031		1,212,232	**1,212,232**
LOM Total :	**57,563,302**	**12,656,584**	**70,219,886**

17 RECOVERY METHODS

17.1 Process description

The original Rainy River processing plant was nominally designed to process 7.7 Mtpa, or 21,000 tpd from the open pit and underground mines. The target production was originally 19,500 tpd from the open pit mine and 1,500 tpd from the underground mine. The process plant commenced ore processing in September 2017 and commercial production in mid-October 2017. Rainy River was able to achieve daily plant throughputs of 21,817 tpd (7.96 Mtpa) in 2019. Throughput was consistently increased to 24,161 tpd (8.82 Mtpa) and 25,341 tpd (9.25 Mtpa) in 2020 and 2021, respectively. Throughput is programed to achieve approximately 26,550 tpd (9.69 Mtpa) in 2022 with the LOM throughput averaging approximately 27,000 tpd (9.86 Mtpa).

Figure 17.1 shows the historical ore processed from 2017 through to 2021 and the forecasted throughput in 2022.

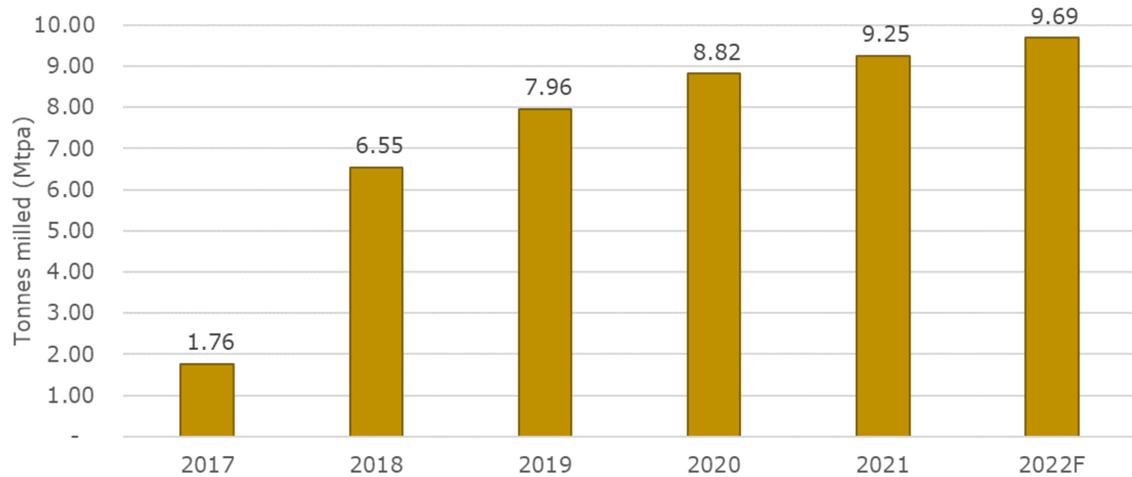


Figure 17.1 – Ore processed

The Rainy River processing plant has two main mineral processing buildings:

- Primary Crushing Building; and
- Main Process Plant.

A general building site layout representing the electrical substation, process plant, stockpile and the primary crusher is presented in Figure 17.2.

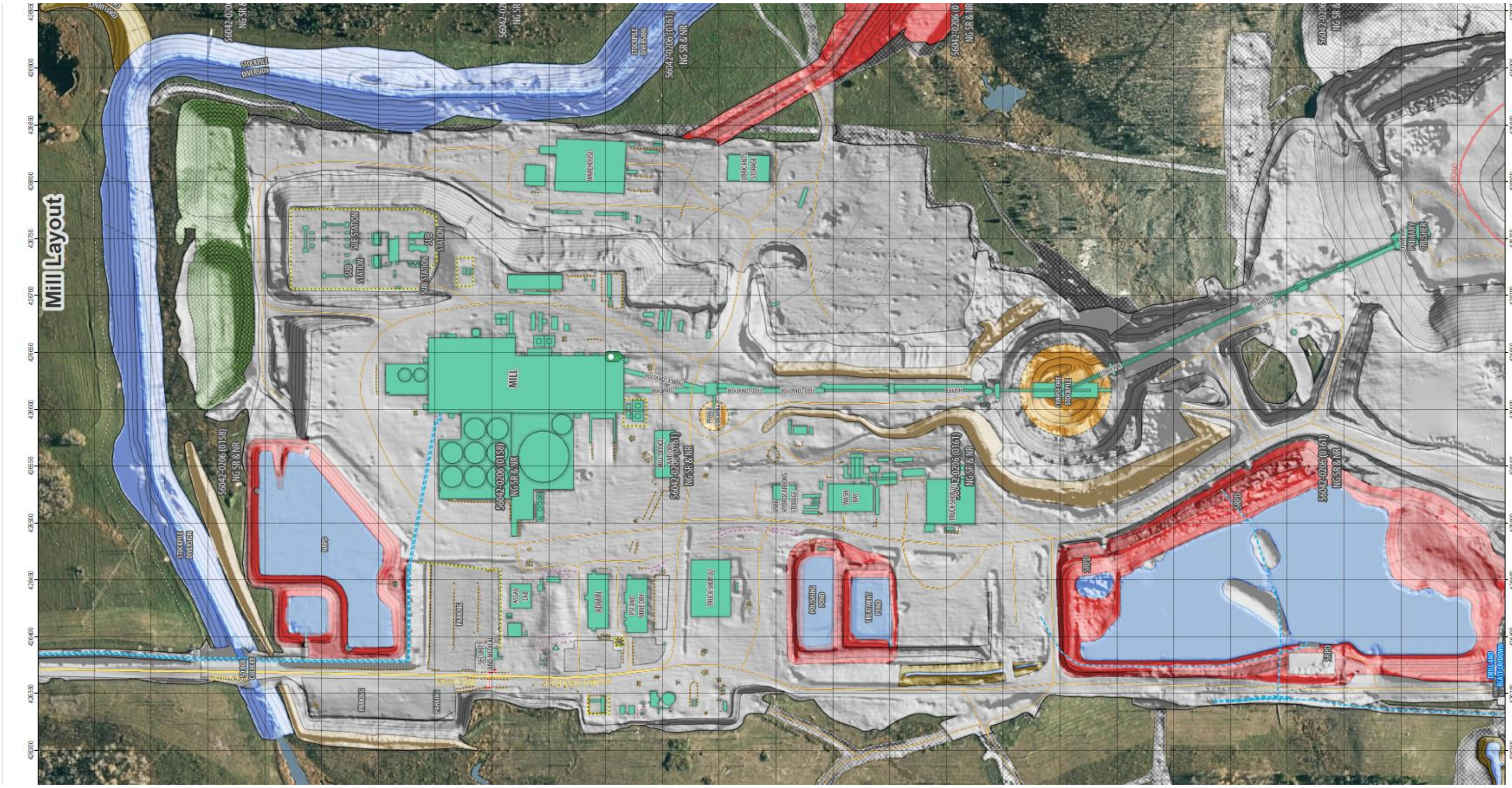
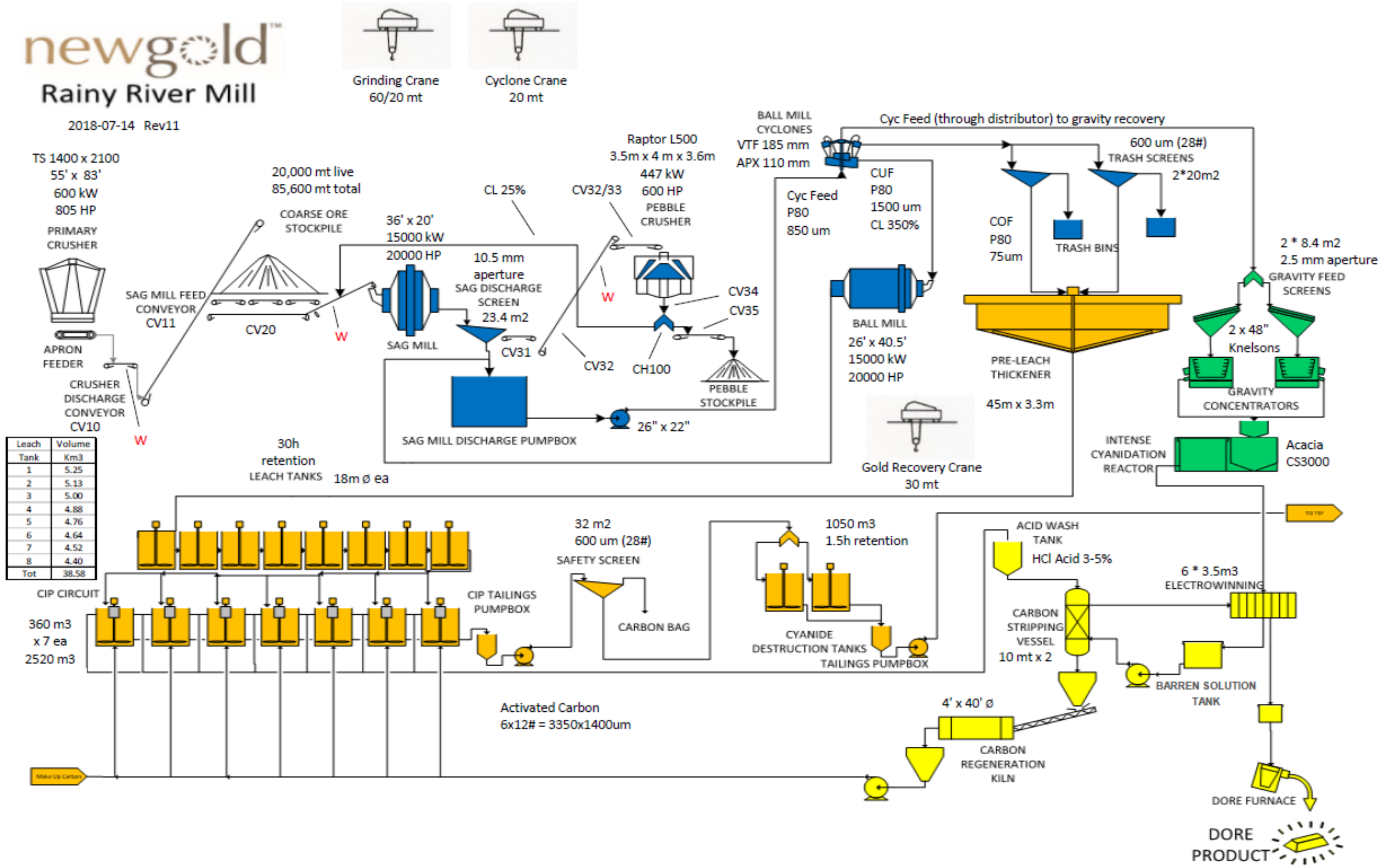


Figure 17.2 – General processing area and buildings site layout

Figure 17.3 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



Leach Tank	Volume Km3
1	5.25
2	5.13
3	5.00
4	4.88
5	4.76
6	4.64
7	4.52
8	4.40
Tot	38.58

Source: New Gold 2019.

Figure 17.3 – Site flowsheet

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 Primary 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 truckloads. 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 an F_{100} feed size of 1,050 mm, an F_{80} of 550 mm and an operating availability of 65%. The crusher operates with an open side setting of 100 mm - 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 truckloads. 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 85,690 t and a live capacity of 19,007 t. Ore is drawn from the coarse ore stockpile by three apron feeders. The designed feed rate of each apron feeder is 476 tph. The apron feeders discharge onto the 1,372 mm wide SAG mill feed conveyor, CV20, which is installed in a single reclaim tunnel beneath the stockpile. CV20 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 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 65 mm pebble ports. The open area per grate is 0.4 m^2 and total open area is 7.2 m^2. The SAG mill currently operates with a 10% - 13% (v/v) ball charge made up with 140 mm balls and a total mill charge volume of 25% v/v. The maximum design ball charge is 16% (v/v) with a maximum design mill charge volume of 30% (v/v). The mill is currently operating at 76% to 80% of critical speed to achieve a production rate of 1,000 – 1,400 tph.

The mill discharge is fitted with a single deck horizontal vibrating screen with 9.5 mm openings to remove oversized pebble, ball chips and tramp steel.

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, CV20, and recycled to the mill or fed to a bypass conveyor, CV35, 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 SAG mill is operated with a slurry density of approximately 70% - 73% solids (w/w) and discharges into the cyclone feed pump box, where it is diluted to approximately 50% – 53% w/w 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. The cyclones are operated with feed pressure of approximately 105 kPa – 135 kPa.

To improve the classification performance of cyclones, the cyclone geometry was modified in Q4, 2021. Modification in cyclone geometry and changing the vortex finder/ Apex combination from 185mm/ 110mm to 230mm/ 140mm, resulted in an increase in the cyclone capacity by approximately 20% - 25%, and reduced the required quantity of operating cyclones by about 20% - 25%. Maintaining the same feed pressure allowed the target cyclone overflow P_{80} to be kept.

17.1.5 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 an F_{80} of 2,800 µm and the target product size is P_{80} of 75 µm. The mill operates at 75% of critical speed to achieve a production rate of 1,000 – 1,400 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.

17.1.6 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 maximum 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. 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 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 leach aid 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 20 m^2 trash 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 into the feed well of a 45 m diameter by 3.3 m high pre-leach thickener.

The thickener underflow density is controlled to 55% - 60% 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 55% - 60% 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 with total slurry volume of 38,550 m^3 for a total retention time of 24 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 1.5 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 or 55.5 g/l. The average carbon transfer rate is once per day. The total transfer and refill time is approximately 3 hours. The average carbon loading is 1,360 g/t Au and 2,990 g/t Ag. 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 a 30 m^2 carbon safety screen with 600 µm openings 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 600minutes 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-tpd 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.

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 tonnes 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 tonnes 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 meters 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. Cyanide destruction is performed by sulfur dioxide and air (SO_2/air) to lower concentration of CN_{TOTAL} at discharge of cyanide destruction tanks to below 5 ppm. The process involves the addition of SO_2 to destroy the cyanide, lime to neutralize the sulfuric acid (H_2SO_4) that is formed as by-product, and copper (as copper sulfate), which acts as a catalyst in the reaction. The discharge from the cyanide destruction vessels pumps to a tailings pond by a pipeline.

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 Tailings management area (TMA). When depositing tailings along the North Dam, a booster pump station was installed in 4th quarter of 2021 to allow for sustained flow rates to achieve 27,000 tpd nominal production. The booster pump station was 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 32,000 tpd.

17.1.15.2 Water management pond

The water management pond (WMP) is designed to hold 5 million cubic meters (Mm^3) of water. This 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. Average consumption of sodium cyanide is approximately 0.25 kg/t.

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. Average consumption of lime is approximately 0.73 kg/t.

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 (SO2) 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 SO2 system enter the process plant. Average consumption of SO2 is in the 0.25 kg/t – 0.35 kg/t range.

17.1.16.5 Copper sulphate

Copper sulfate (CuSO4.5H2O) 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. Average consumption of copper sulfate is 50 g/t.

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. Average consumption of activated carbon is approximately 30 g/t.

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. Average consumption of antiscalant is approximately 17 g/t.

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. Average consumption of flocculant is approximately 15 g/t.

17.1.16.9 Sodium metabisulphite

Sodium metabisulphite is only used as a back-up reagent to sulfur 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.

17.1.17 Auxiliary systems

17.1.17.1 Compressed air

Instrument and plant air compressors are provided for each area of the plant. Table 17.1 shows a list of the compressors and their capacities.

Table 17.1 – 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 Process control

Control of process equipment is done via a Delta V control system. Site possesses a real-time expert system platform and has developed advanced process control (APC) systems for the reclaim apron feeders, SAG mill, ball mill, cyclone feed pump box, and thickener. Site uses PARCview software for trending and data extraction.

17.1.19 Plant specific energy

Figure 17.4 shows the specific energy usage for the SAG mill, ball mill, and total site for the 2-years period from 1 January 2020 to 31 December 2021.

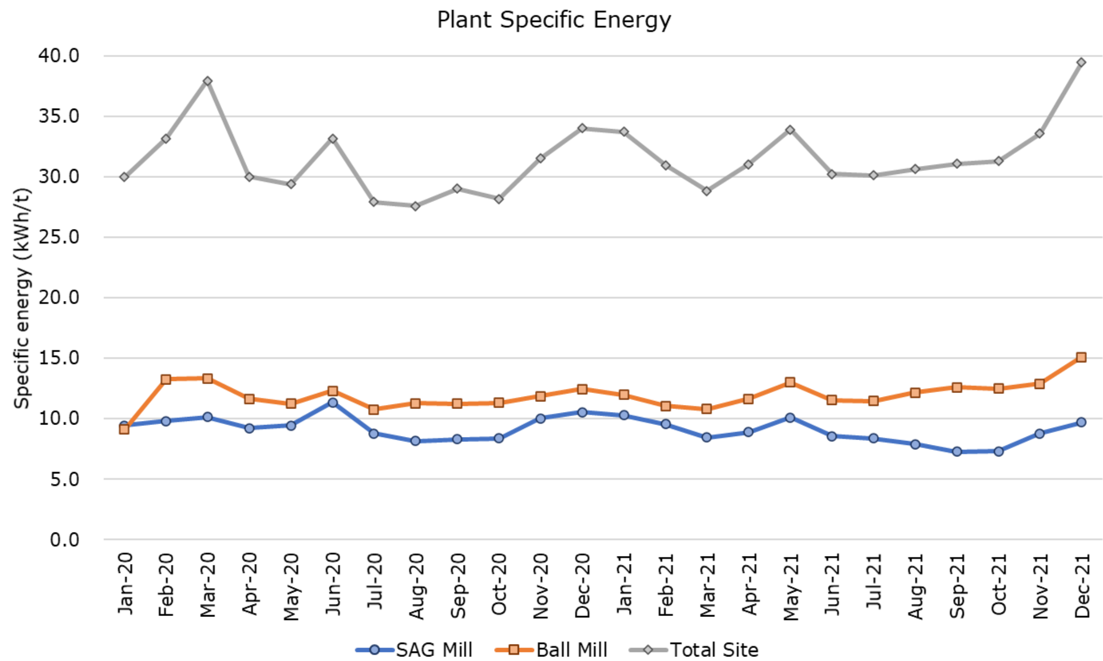


Figure 17.4 – Mill energy usage from January 2020 to December 2021

The actual SAG mill specific energy usage for the 24-month period was 9.1 kWh/t against the design specific energy of 15.8 kWh/t. The actual ball mill specific energy usage for the 24-month period was 11.9 kWh/t against the design specific energy of 15.8 kWh/t. 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 year of 2020 - 2021 are presented in Plant debottlenecking and expansion projects

The following items summarize the projects planned for the increase in plant throughput capacity.

17.1.21 Crushed ore stockpile

Dust from the crushed ore stockpile has been identified as an environmental and health concern. Solutions to the dust problem include installation of dry fog system and dust control curtains at discharge point of coarse ore stockpile feed conveyor, CV 11. This modification has been planned to be installed in 2nd and 3rd quarter of 2022. Rainy River expects these measures should largely remove the dust issue.

17.2 Plant debottlenecking and expansion projects

The following items summarize the projects planned for the increase in plant throughput capacity.

17.2.1 Crushed ore stockpile

Dust from the crushed ore stockpile has been identified as an environmental and health concern. Solutions to the dust problem include installation of dry fog system and dust control curtains at discharge point of coarse ore stockpile feed conveyor, CV 11. This modification has been planned to be installed in 2nd and 3rd quarter of 2022. Rainy River expects these measures should largely remove the dust issue.

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Table 17.2 – Rainy River processing plant operating parameters

Crusher Metrics	Units	Q1-20	Q2-20	Q3-20	Q4-20	Q1-21	Q2-21	Q3-21	Q4-21
Tonnes Crushed	t	1,739,605	2,010,914	2,479,099	2,490,504	2,434,373	2,354,013	2,248,840	2,262,208
Availability	%	92.23	80.81	90.99	89.49	91.18	89.34	82.97	89.26
Operating Time	%	57.66	63.95	74.77	73.44	77.80	73.42	69.70	75.41
Feed Rate	tph	1,382	1,440	1,502	1,535	1,449	1,468	1,461	1,358
Mill Metrics									
Tonnes Milled	t	1,678,120	2,173,124	2,483,849	2,483,911	2,367,088	2,306,780	2,322,510	2,253,302
Availability	%	90.82	89.83	90.32	93.97	89.08	88.33	90.76	93.56
Operating Time	%	72.63	85.17	87.33	88.51	85.54	86.62	89.51	92.25
Milling Rate	tph	1,058	1,168	1,288	1,270	1,282	1,219	1,175	1,106
Tonnes Milled per Day	tpd	18,441	23,880	26,998	26,999	26,301	25,349	25,245	24,492
Gold Metrics									
Gold Head Grade	g/t	1.00	0.79	0.92	0.93	0.82	0.79	0.93	1.00
Gravity Gold Recovery	%	6.41	0.41	0.00	0.00	0.00	0.60	14.68	23.84
Gravity Gold Recovered	oz	3,139	202	0	0	0	311	8,998	15,934
Overall Gold Recovery	%	90.38	89.21	89.46	89.81	89.47	87.40	88.60	92.10
Overall Gold Recovered	oz	48,967	49,374	65,705	66,760	55,944	51,650	61,282	66,843
Gold Poured	oz	50,381	48,800	63,004	66,734	54,656	52,901	58,557	68,364
Silver Metrics									
Silver Head Grade	g/t	1.68	1.93	2.73	3.24	3.19	3.73	3.56	3.09
Gravity Silver Recovery	%	2.25	0.13	0.00	0.00	0.00	0.20	2.81	4.35

Crusher Metrics	Units	Q1-20	Q2-20	Q3-20	Q4-20	Q1-21	Q2-21	Q3-21	Q4-21
Gravity Silver Recovered	oz	1,265	92	0	0	0	330	4,623	6,323
Overall Silver Recovery	%	62.28	52.21	51.92	50.21	57.06	59.92	61.74	65.06
Overall Silver Recovered	oz	56,297	70,333	113,305	129,962	138,339	165,839	164,261	145,501
Silver Poured	oz	61,265	70,394	102,814	127,390	133,730	162,879	160,461	153,394

17.3 Reducing throughput of the Rainy River process plant

New Gold engaged Halyard to investigate the implications of significantly reducing the throughput of their Rainy River (RR) process plant and quantify a range of possible scenarios. The current plant capacity is rated at approximately 25 to 27 ktpd and the investigation considered the impacts of reconfiguring it to operate at 4 to 5 ktpd. This would accommodate ore from underground mining only when the ore from the open pit and waste ore stockpiles is depleted.

In order to address this scenario and arrive at an optimum “fit-for-purpose” solution, Halyard proposed to split the project into two phases:

- Phase 1 - Project definition and consideration of all alternatives.
- Phase 2 – Cost estimation to PFS level for selected.

Phase 1 is further split in two sub-phases:

- Phase 1a - Comminution
- Phase 1b - Gold Recovery (downstream)

Phase 1 evaluated six options. Options 2a and b, as well as 3 were carried through to Phase 2 for further evaluation. The final recommended option is 2b.

17.3.1 Phase 1

The complete list of options under consideration for Phase 1 are listed in Table 17.3. Equipment sizing for Phase 1a is given in the Orway report, and each option is displayed on a schematic flowsheet.

After review, it was determined that the downstream plant considered under Phase 1b could easily be modified to operate effectively under any of the proposed Options, 1 through 6.

Table 17.3 – Options for Phase 1

Option No.	Category	Details
Option 1	Batching	Do nothing to plant capacity
Option 2	Batching	Turn Down SAG & Ball
Option 3	Batching	Remove BM, Turn Down SAG
Option 4	Small Plant	New Smaller Jaw & SAG
Option 5	Small Plant	New Jaw, SAG, Ball
Option 6	Small Plant	New Jaw, Sec. &Tert. Crush, Ball

After consideration of capital and operating costs as well as practical operating issues, batching options, 2 and 3 were selected as the preferred options to proceed to Phase 2. No small plant options were selected.

17.3.2 Phase 2

The selected Options (2a, 2b and 3) were considered in further detail in Phase 2, where the advantages and disadvantages of each option were considered. A high-level summary is outlined in Table 17.4.

Table 17.4 – Advantages/Disadvantages

Option no.	Description	Disadvantages	Advantages
#2a	SAG and ball both "turned down". Operation 14 days/month	This option involves minimal changes. However, it involves many stop/starts and operation in winter would be difficult, particularly regarding the stockpile.	This option involves minimal changes. Lowest capital cost.
#2b	SAG and ball both "turned down". Operation 6 months/year, warm weather operation	Difficulties with holding personnel during the extended "off" period.	This option involves minimal changes and a continuous, warm weather operation has obvious advantages. Lowest capital cost.
#3a	SAG mill operated as a single stage, no ball mill. Operation almost continuous.	Highest operating cost. Highest capital cost.	This is a practical option, but apart from almost year-round, continuous operation holds no advantages over option 2.

Project economics are an important driver and given that Option 3 has the highest capital and operating costs with no obvious advantages, the logical final selection is Option 2. Within the accuracy of this study, the capital costs of Options 2a and 2b are identical and the operating costs are similar. Scheduling of the labour force needs further consideration. The advantages of warm weather operation would outweigh the potential difficulties of a year-round, stop/start operation. Option 2b is therefore recommended for further evaluation.

17.4 Phase 1 Equipment Sizing

The complete list of options under consideration are described in Table 17.3. Equipment sizing is given in the Orway report, which is appended to the Halyard PFS and focusses on Phase 1a, Comminution. In the PFS, each option is displayed on a schematic flowsheet.

17.4.1 Options

The Options are described under two sub-headings: Comminution (1a) and Downstream (1b).

17.4.1.1 Comminution (1a)

During preliminary discussions, the six options were suggested for analysis. Options 1, 2 and 3 were based on "batching" of ore through the existing plant. Options, 4, 5 and 6 were based on design and construction of a new smaller plant. Options 1, 2 and 3 each had two sub-options, termed "a" and "b". These referred to different scheduling of the period of operation and are presented as days/month or months /year in Table 17.5.

Table 17.5 – Phase 1a Comminution - (Options)

Option no.	Primary crusher/stockpile	Grinding	Operating time
Batching			
#1a	"Base Case". Existing primary crusher and stockpile	Existing SAG and ball mill (no turn-down)	6 days/month
#1b	As above	As above	2.5 months/a
#2a	Existing primary crusher and stockpile	Existing SAG and ball mill (turned down)	14 days/month
#2b	As above	As above	6 months/a
#3a	Jaw crusher and existing stockpile	Single-stage AG/SAG (existing modified)	25 days/month
#3b	As above	As above	10 months/a
Small Plant			
#4	Jaw and stockpile (new)	Single-stage AG/SAG (new)	30 days/month
#5	Jaw and stockpile (new)	SAG and ball mill (new)	30 days/month
#6	Jaw and stockpile (new)	Secondary/tertiary crusher and ball mill (new)	30 days/month

Note: The Jaw Crusher referred to in Options 3, 4, 5 and 6 is a new but existing semi-portable jaw crusher provided by RR.

17.4.1.2 Downstream (1b)

After review, it was determined that the downstream plant could be easily modified to operate effectively under any of the proposed Options, 1 through 6. The tonnage will be reduced to approximately 20% of design and the gold production will be approximately 50% of design.

The changes are summarized in the Halyard PFS. The lower instantaneous throughput in Options 3, 4, 5 and 6 will be accommodated by smaller cyclone feed, thickener u/f and tailings pumps and pipelines. The lower gold production in all options can be accommodated by transferring less carbon and employing fewer carbon strips. Also, Options 3, 4, 5 and 6 will require the use of only 6 leach tanks, 6 CIP (Carbon in Pulp) tanks and a single cyanide destruction reactor.

17.4.2 Phase 1 Recommendations

The following Phase 1 recommendations, were made after discussions between with New Gold and Halyard staff:

- Either of Options, 2a, 2b or 3a are the preferred "batching" options.
- No "small plant" options were selected.

17.5 Phase 2 Equipement Sizing

Following discussion with the New Gold team, Options 2a, 2b and 3 were selected for further in-depth analysis in Phase 2:

17.5.1 Description Option 2a and 2b

Refer to 'H21241-PFD-002 Rev-A, Crushing and Grinding Option-2'.

For both these options, there is no major change to the major items of equipment; the existing primary crusher, stockpile, SAG and ball mills are retained. The two mills, however, are slowed down using the existing variable speed drives and the ball loads are reduced.

As described in the Orway report, the instantaneous throughput in both options is 478 t/h, (10,548 t/d).

Therefore, the mill runs only 47% of the time. This could be 14 days/month or almost 6 months/year.

Due to the reduced throughput, the cyclone feed pump, pipeline and cyclones are replaced, as are the preleach thickener u/f pumps, tailings pumps and pipelines. Only four leach tanks and one cyanide destruction tank are used while carbon transfer, elution, electrowinning and smelting rates are reduced.

17.6 Description Option 3

Refer to 'H21241-PFD-002 Rev-B, Crushing and Grinding Option-3'.

This option uses a new, but existing semi-portable jaw crusher provided by RR, new fine ore stockpile, existing SAG, existing trash screens, gravity circuit and pre-leach thickener. However, the single-stage SAG mill operates at reduced speed and with a lower ball load. The reduced speed in the SAG mill is achieved using the existing variable speed drive. The ball mill can be removed for sale.

As described in the Orway report, the instantaneous throughput is 274 t/h, (6,050 t/d). Therefore, the mill runs only 83% of the time. This could be 25 days/month or 10 months/year.

Due to the reduced throughput, the cyclone feed pump, pipeline and cyclones are replaced, as are the preleach thickener u/f pumps, tailings pumps and pipelines. Only four leach tanks and one cyanide destruction tank are used while carbon transfer, elution, electrowinning and smelting rates are reduced.

17.7 Conclusion and recommendations

Phase 1 identified all possible practical options and carried out a preliminary evaluation of each one. After review of the preliminary capital and operating costs and consideration of the various practical operating considerations, it was decided to move forward to Phase 2 with Options 2a, 2b and 3a. These options both involve relatively minor modifications to the existing plant. No small plant options were selected, mainly due to the higher capital costs. Option 1 was discarded mainly due to the short operating period which would involve multiple stop/starts compared with Options 2 and 3.

Phase 2 refined the preliminary capital and operating costs developed in Phase 1 for Options 2a, 2b and 3a. During this process, the practicality of the various operating schedules was considered in more detail, particularly regarding winter operation. The three options are fairly similar in that they both involve relatively minor plant modifications. However, they have different operating schedules. As explained in the previous items, Option 2 involves a “turn-down” of both the SAG and ball mill, while Option 3 involves a “turndown” of the SAG mill only, which operates as a single-stage grinding unit. Options 2a and 2b both operate close to 50% of the time at almost 500 t/h. However, option 2a operates 14 days per month, throughout the year, while option 2b operates for a single 6 month period during the warmer weather.

Project economics are an important driver and given that Option 3 has the highest capital and operating costs with no obvious advantages, the logical final selection is Option 2. Within the accuracy of this study, the capital costs of Options 2a and 2b are identical and the operating costs are similar. It seems that although scheduling of the labour force needs further consideration, the advantages of a warm weather operation would outweigh the potential difficulties of a year-round, stop/start operation. Option 2b is therefore recommended for further evaluation.

The practicalities and costs of manning at the reduced level with contract labour should be investigated further.

Another important operating cost item is power costs. Preliminary discussions should be held with the utility to define a new power supply contract.

The selected option, 2b is shown on the following schematic PFD. Only the comminution circuit is shown as the downstream plant is essentially unchanged.

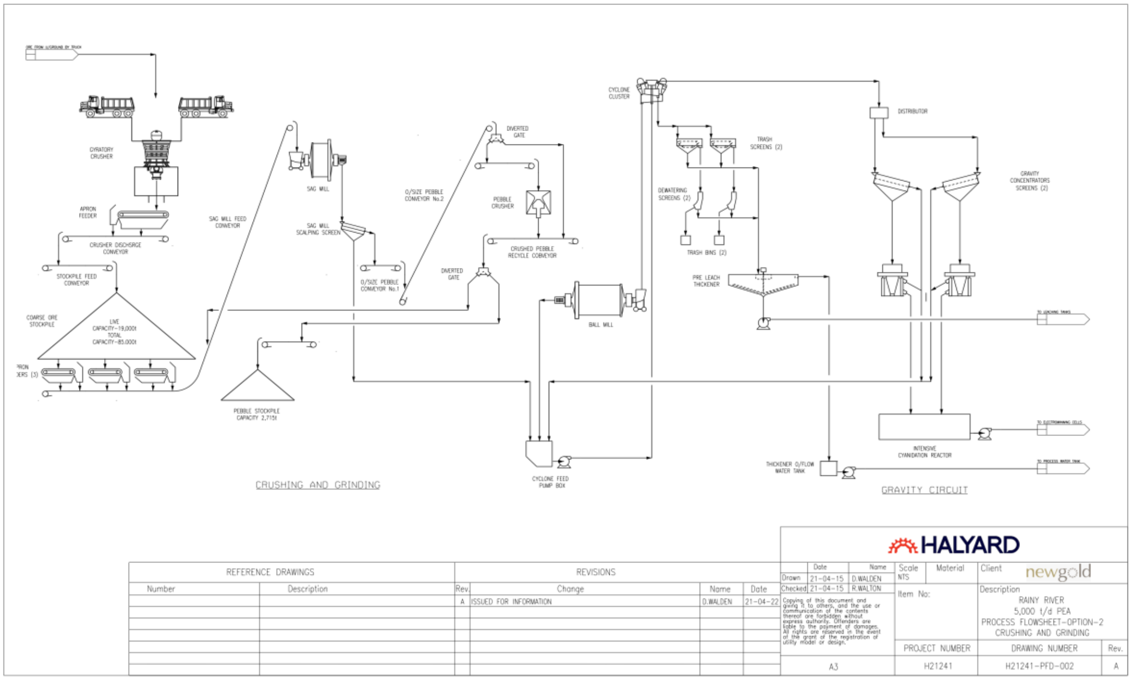
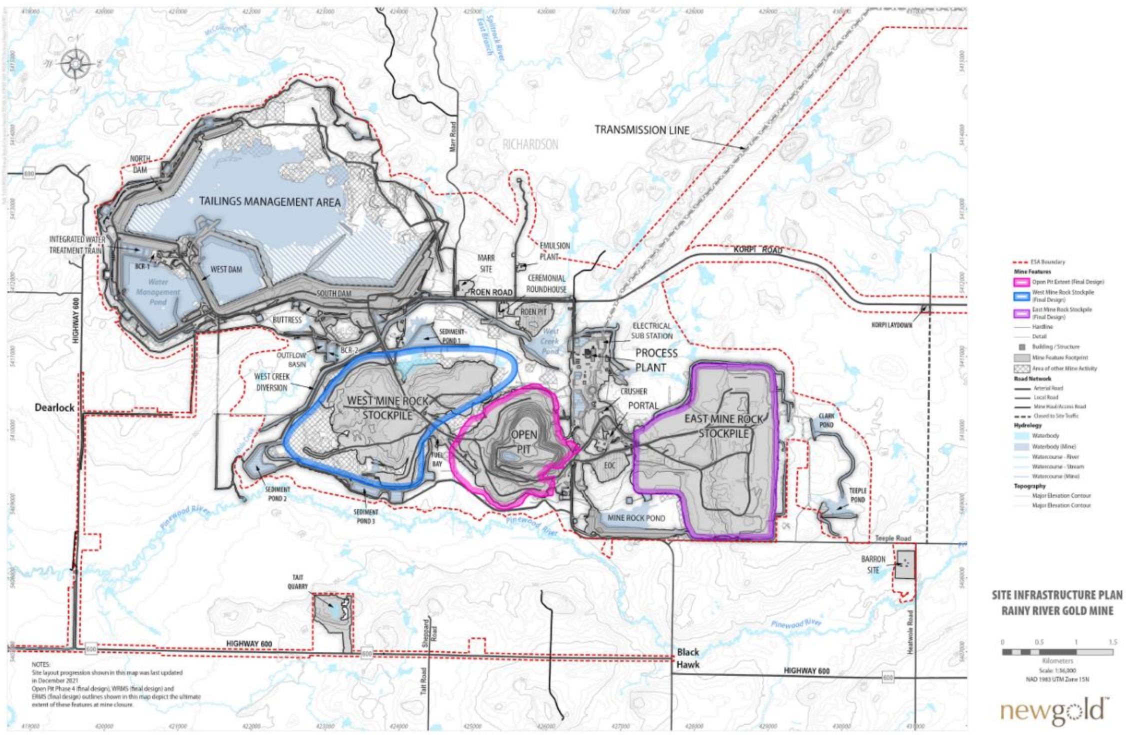


Figure 17.5 – H21241-PFD-002 Rev-A, Crushing and Grinding Option-2

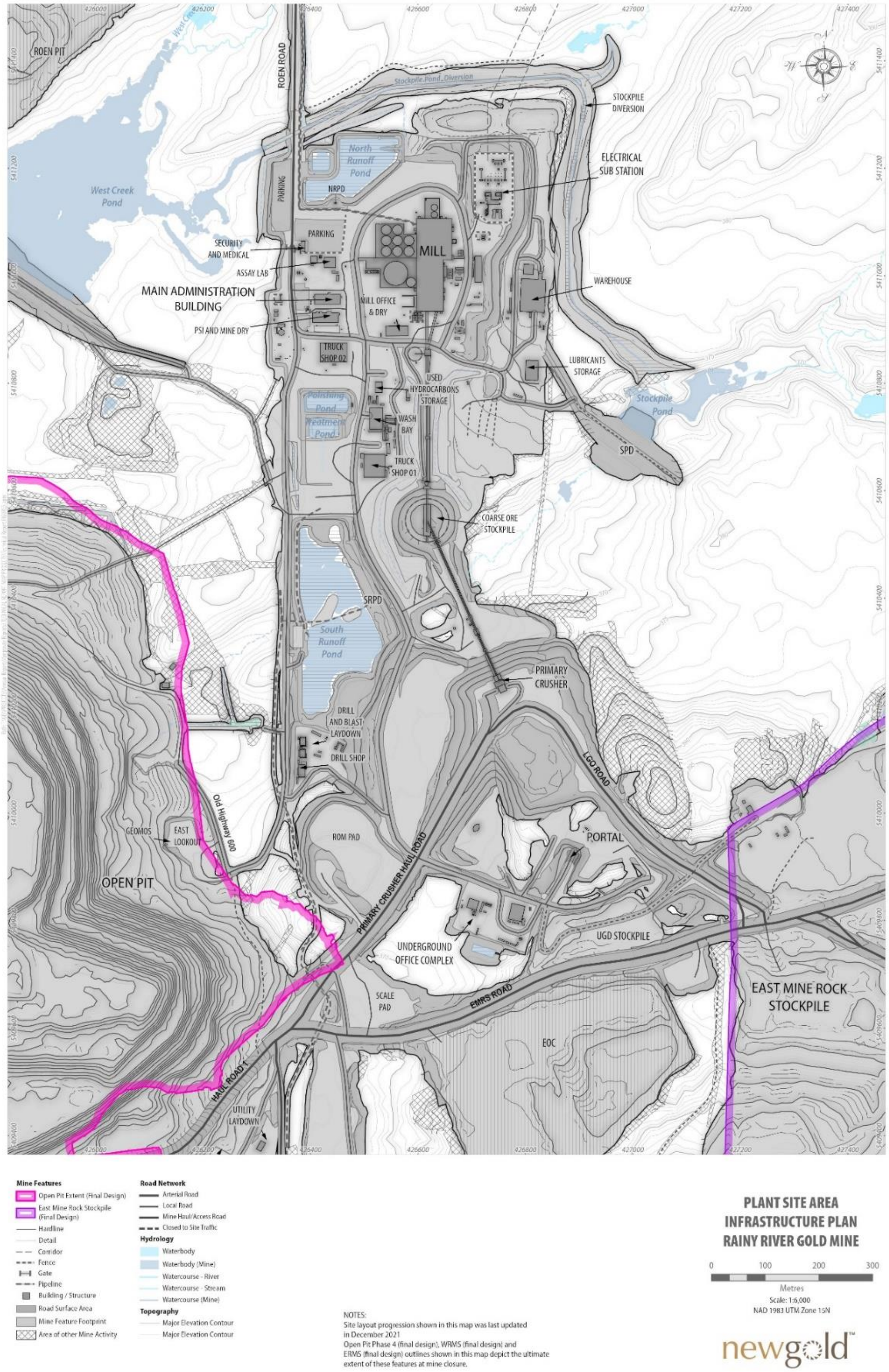
18 PROJECT INFRASTRUCTURE

This Item 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.



Source: New Gold 2020.

Figure 18.1 – General site plan



Source: New Gold 2020.

Figure 18.2 – Detailed site plan

18.1 Primary access roads

The mine site access 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 downstream rockfill 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 is a 1,500 m^2 heated and insulated fabric covered steel structure building with a concrete floor that includes three service bays and is located south of the existing Mine Dry. Truck shop 2 provides additional service bay capacity to support the ongoing maintenance of the mine fleet and includes a 50 t crane, and compressed air and lubricant distribution systems.

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 consisted of two 75,000 L double walled storage tanks until 2020 when it was expanded by the additional two further 75,000 L double wall storage tanks. The expanded total storage capacity of 300,000 L of diesel fuel provides 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

The 2,800 m^2warehouse facility is located at the upper laydown to the east of the process plant. The warehouse / supply chain offices are located adjacent to the warehouse on the north side, consisting of 11 offices, a single meeting room as well as a kitchen and bathroom facilities.

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) warehouses new lubricants.

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 used for temporary storage of used hydrocarbons and includes a double walled waste oil tank.

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 dry and office building is located near the southwest corner of the process plant. The building consists 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.

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. The camp consists of 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 located on the south side of Roen Road and west of the Roen Pit. The roundhouse provides a place for gatherings and traditional Indigenous ceremonies.

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 provided by a 13.8 kV line routed 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 Tailings Management Area (TMA) is located northwest of the open pit and plant site. As part of the original mine startup, the TMA was divided into three independent cells for tailings deposition: TMA Cell 1, TMA Cell 2, and TMA Cell 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 were constructed to 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.

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, respectively. As the TMA perimeter dams (TMA North, South, and West Dams) are raised above the crest elevations of TMA Cell 1 and TMA Cell 2 dams, the internal dams would be covered by tailings forming a single impoundment area. As of 2021, TMA Cell 2 dam is entirely buried in tailings. Following the completion of the TMA Stage 3 raise in 2021, and the planned breaching of the TMA Cell 1 dam in Winter/Spring 2022, all three TMA Cells will be combined into a single contiguous cell.

The Water Management Pond (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 North Dam, TMA West Dam (Dams 4 and 5), and TMA South Dam will be raised in stages during the mine life to an ultimate elevation of 379.1 m. The TMA West Dam (Dam 4) was constructed to the Stage 1 starter dam elevation of 366.5 m by July 2017. The TMA West Dam (Dam 5) and a portion of the TMA South Dam (from approximate Sta. 0+000 m to 0+800 m) were constructed to a Stage 2 crest elevation of 371.5 m by September 2017. Stage 1 starter dam construction of the TMA South Dam (from approximate Sta. 0+800 m to 3+200 m) and TMA North Dam to a crest elevation of 366.5 m was completed in the 2018 construction season. Stage 2 construction of the TMA North Dam, TMA West Dam (Dam 4), and TMA South Dam to a crest elevation of 371.5 m was completed by September 2020. Stage 3 construction of the TMA North Dam, TMA West Dam (Dam 4 and Dam 5), and TMA South Dam to a crest elevation of 373.6 m was substantially completed by December 2021.

Tailings deposition in TMA Cell 1 commenced in November 2017 with placement into TMA Cell 2 beginning in May 2018. Tailings placement into TMA Cell 3 began in May 2019. Generally, the tailings deposition strategy is to establish tailings beaches upstream of the perimeter dams (i.e., TMA North Dam, TMA West Dam [Dams 4 and 5], and TMA South Dam), while maintaining a pond around the fixed reclaim located at TMA Cell 2. The main depositional constraints for tailings placement are not burying or blocking the fixed reclaim; and not blocking the spillway channel situated at the TMA North Dam.

The Stage 3 raise TMA emergency spillway is located at TMA North Dam Sta. 0+950 m. To allow for the staged construction of spillway raises and to realize schedule efficiencies, New Gold chose to alternate the spillway location for each raise between approximate TMA North Dam Sta. 0+850 m and Sta. 0+950 m. For the Stage 4 raise, the TMA emergency spillway is planned to be located at TMA North Dam Sta. 0+850 m.

Considering the estimated ultimate crest elevation and the requirement to maintain containment within the Environmental Site Assessment boundary, a flood protection berm is required at a low topographic area located toward the northeast of the TMA, as shown in Figure 18.3. The proposed TMA flood protection berm will be approximately 600 m long with a maximum height of approximately 1.5 m (Figure 18.3).

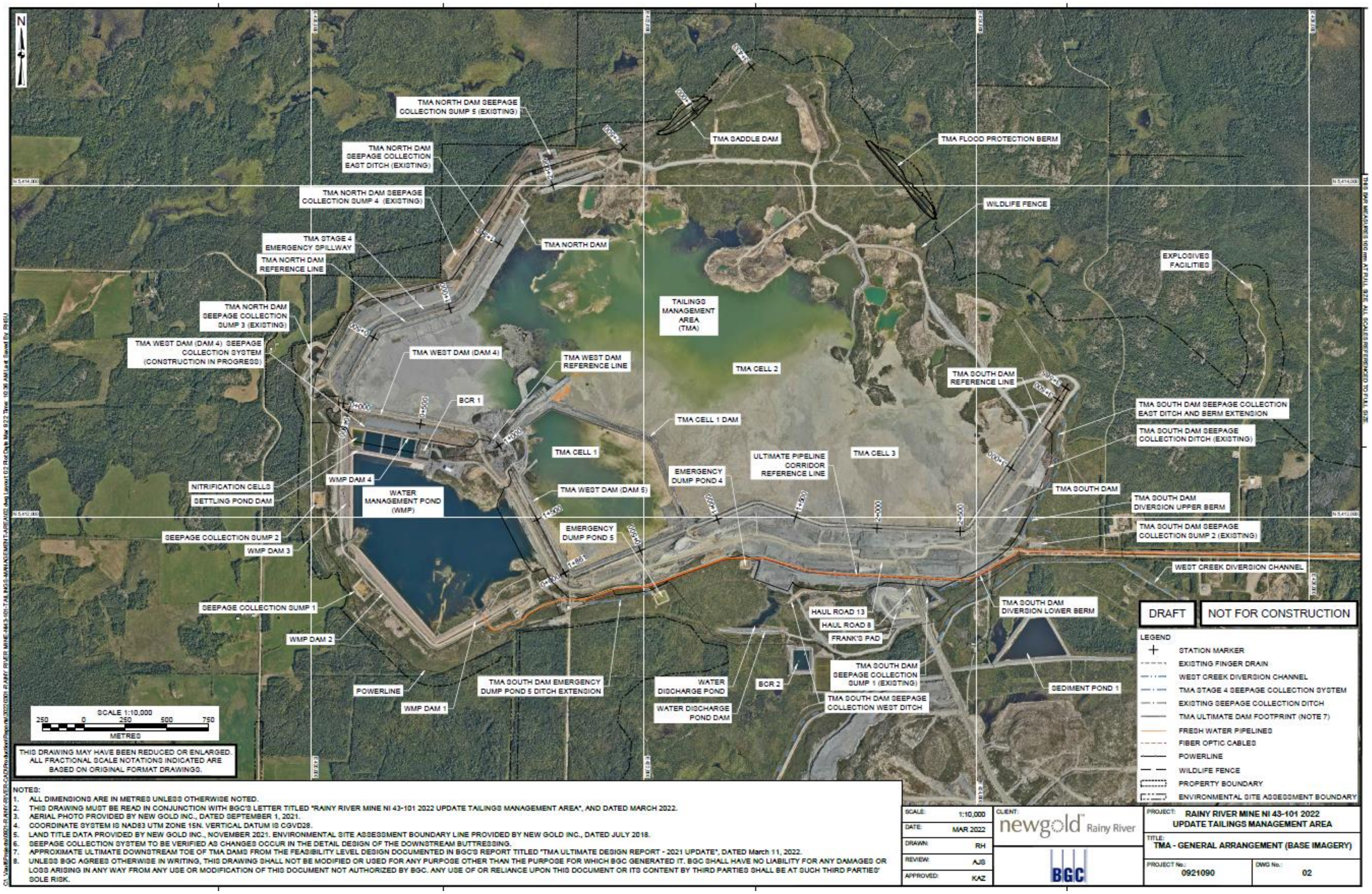


Figure 18.3 – 2021 TMA General Arrangement (Base Imagery) (BGC, March 14, 2022)

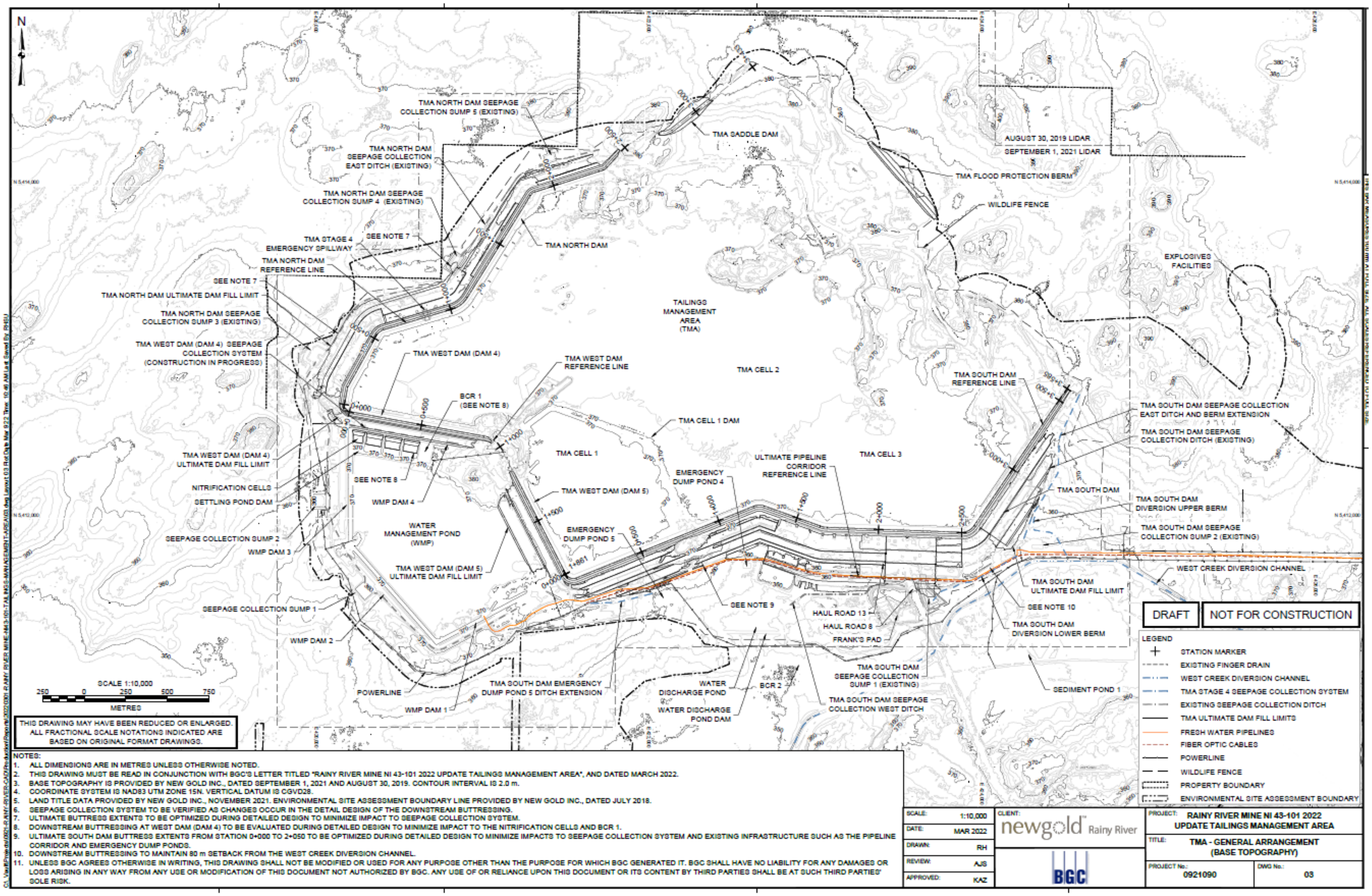


Figure 18.4 – 2021 TMA General Arrangement (Base Topography) (BGC, March 14, 2022)

18.7.2 TMA Design

The TMA is designed to provide sufficient containment for the projected tailings storage requirements and operational pond volume. The Environmental Design Flood (EDF) is to be stored below the TMA emergency spillway invert level (also referred as the EDF Level or EDFL) and the TMA emergency spillway is designed to pass Inflow Design Flood (IDF).

New Gold prepares a water balance model to predict TMA pond volumes, which is a key input provided to BGC for estimating TMA storage and dam raise schedule. Tailings properties were interpreted by BGC based on observed conditions measured by LiDAR and bathymetric surveys as well as mill throughput tonnages are provided by New Gold. The IDF and freeboard requirements are determined by BGC in accordance with Canadian Dam Association guidelines. The EDF and maximum normal operating water level (MNOWL) are operational criteria selected by New Gold.

18.7.2.1 Tailings management planning

The Tailings Management Plan (BGC, October 1, 2021) used historical tonnage records from mill start-up on August 9, 2017 to April 1, 2021 as provided by New Gold on April 20, 2021. Forecasted tailings production tonnages to the end-of-mine (EOM) are based on the updated life of mine (LOM) plan as provided by New Gold on April 20, 2021. LOM cumulative tailings tonnage provided to BGC was estimated to be 93.4 Mt. BGC estimated an average dry settled density of 1.35 t/m^3, a beach above water (BAW) slope = 0.50%, and beach below water (BBW) slope = 0.90%. In addition, BGC assumed the 99th percentile pond to be contained below the spillway invert elevation.

Table 18.1 provides a summary of the current dam raise schedule based on the tailings deposition modeling completed by BGC (October 1, 2021).

Table 18.1 – TMA dam raise schedule

Year	Dam Crest Elevation (m)	Raise Height (m)	Spillway Invert Elevation (m)	Dams to be Raised
2022	375.1	1.5	373.3	TMA Perimeter Dams(1)
2023	376.6	1.5	374.8	TMA Perimeter Dams
2024	377.9	1.3	376.1	TMA Perimeter Dams
2025	379.1	1.2	377.3	TMA Perimeter Dams

Note:

1. TMA perimeter dams include the TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam.

18.7.3 Foundation Characterization and Geotechnical Parameters

The following item provides a brief summary of the TMA foundation characterization and geotechnical parameters which are documented in BGC (October 29, 2021).

Given the variability of foundation conditions along the TMA dams, geotechnical Design Zones were defined in areas with similar topographic and foundation conditions (see Item 2.2.2.2).

18.7.3.1 Surficial Geology

The surficial geology at the Rainy River Mine consists of glacial sediments deposited during advance and retreat of the Laurentide Ice Sheet during the Late-Wisconsinan, between approximately 20,000 and 11,500 years before present (Bajc, 2001). Glacial advance and retreat led to the deposition of fine-grained glaciolacustrine soils and glacial (till) deposits. The typical stratigraphic units (from oldest to youngest) observed in the TMA and Water Management Dam foundations are listed below.

- The Whiteshell Till (WST) generally comprises a dense, granular lodgement till deposited by Labradorean ice advancing from northeast to southwest at the start of the Late-Wisconsinan age. WST sediments were derived from igneous and metamorphic rocks of the Canadian Shield.
- The Wylie Formation (WYL) generally comprises interbedded silt and clay deposited in a glaciolacustrine environment present during the retreat of the Labradorean ice. The clay fraction was generally sourced from advancing Keewatin ice.
- The Whitemouth Lake Till (WML) generally comprises a massive, dark grey, high plastic silty clay lodgement till with trace amounts of sand and gravel. Following retreat of Labradorean ice, the WML was deposited by Keewatin ice advancing from west to east. WML sediments were derived from soft sedimentary rocks of the Williston Basin, which occupies a portion of southern Manitoba. The till was also partly derived from Wylie Formation sediments. Bajc (2001), and BGC have noted that the WML contains sheared and softened zones attributed to glacial deposition processes.
- The Brenna Formation (BRE) comprises variable silt and fine sand to higher plasticity silts and clays deposited in a glaciolacustrine environment present during the retreat of the Keewatin ice. The sediment is sourced from Keewatin ice eroding to emergent land. BRE sediments varied as the lake basin became more distal to the ice front during progressive retreat of the Keewatin ice. The depositional environments also varied laterally across the site, depending on proximity to the lake shoreline.
- The Poplar River Formation comprises glaciofluvial sands and gravels deposited in a fluvial environment present during a low period in the lake level. The features associated with the Poplar River Formation include erosional surfaces, boulder lags, paleosols, and channel-fills.

- The Sherack Formation comprises variable silt and clay sediments deposited in a glaciolacustrine environment present during a re-advance of the Labradorean ice. The shoreline elevation is estimated at approximately 350 m to 352 m, suggesting a near shore environment with subaerial exposure to the north and west of the site (i.e., near the TMA).
- Organics (peat/topsoil)

18.7.3.2 Geotechnical Design Zones

The TMA North Dam, TMA West Dam, and TMA South Dam were divided into geotechnical Design Zones having similar stratigraphic and topographic conditions anticipated to exhibit similar geotechnical behavior. This allows design optimization by isolating unfavorable foundation conditions which require relatively large downstream stabilizing buttresses from areas with more favorable foundation conditions.

Unfavourable foundation conditions are encountered in low-lying topographic areas with tall dam heights and thick surficial soils including lightly overconsolidated to normally consolidated[1] overburden soils at depth. Favourable foundation conditions pertain to overburden soils typically encountered in high topographic areas which are relatively thin and moderately to heavily overconsolidated[1] and having lower dam heights (5 m to 20 m).

The shear strength and PWP conditions for lightly overconsolidated to normally consolidated overburden soils are generally lower, and as a result unfavourable for stability. Based on reviewing the PWP data measured from vibrating wire piezometers (VWPs) installed within the TMA dam footprints (BGC, September 8, 2021), relatively high excess PWPs and low rates of PWP dissipation are observed in low-lying topographic areas, which is attributed to the lower degree of overconsolidation and exacerbated by higher embankments (10 m to 22 m) across the localized valleys.

Table 18.2 summarizes the geotechnical Design Zones for each dam, including the maximum dam height and foundation soil thickness. Further detail regarding the selection and basis for each Design Zone is documented in (BGC, October 29, 2021).

Table 18.2 – Summary of geotechnical Design Zones

TMA Dam	Design Zone	Sta. From (m)	Sta. To (m)	Max. Dam Height(3) at Ultimate (m)	Max. Foundation Soil Thickness(1) (m)	Unfavourable Foundation Conditions(2)
TMA North Dam	Zone 1	0+045	0+600	16	35	Yes
	Zone 2	0+600	0+975	14	45	Yes
	Zone 3	0+975	1+250	12	20	
	Zone 4	1+250	1+860	10	25	
	Zone 5	1+860	2+540	9	25	

[1] The consolidated conditions of the overburden soils are defined by the overconsolidation ratio (OCR). OCRs ranging from <1 to 2 correspond to normally consolidated conditions. OCRs ranging from 2 to 4 correspond to lightly overconsolidated conditions. OCRs ranging from 4 to 10 correspond to moderately overconsolidated conditions and OCRs >10 correspond to heavily overconsolidated conditions.

TMA Dam	Design Zone	Sta. From (m)	Sta. To (m)	Max. Dam Height[3] at Ultimate (m)	Max. Foundation Soil Thickness[1] (m)	Unfavourable Foundation Conditions[2]
TMA West Dam	Dam 4	0+015	0+980	16	30	Yes
	Dam 5	0+980	1+861	16	30	Yes
TMA South Dam	Zone 1	0+000	0+815	18	15	
	Zone 1A	0+815	1+400	16	-	
	Zone 2	1+400	1+600	26	35	Yes
	Zone 3	1+600	1+900	24	30	
	Zone 4	1+900	2+050	22	35	Yes
	Zone 5	2+050	2+350	24	40	Yes
	Zone 5B	2+350	2+500	24	28	Yes
	Zone 6	2+500	2+600	24	30	
	Zone 7	2+600	3+000	22	30	
	Zone 8	3+000	3+585	14	-	

Notes:

1. Foundation soil thickness generally based on refusal depths of cone penetration tests (CPTs), test pits, and boreholes completed within the dam footprint. TMA South Dam Design Zone 1A and 8 foundation thickness not interpreted due to limited site investigation information.
2. For Design Zones with unfavorable foundation conditions, additional stability analyses will be completed to meet an EOC FOSmin = 1.3 considering lower-bound shear strengths.
3. Height is defined as the elevation difference between the lowest extent of the downstream buttress and the TMA Ultimate crest elevation.

18.7.4 Ultimate Footprint

The feasibility level Ultimate Design Report (BGC, January 28, 2022) was also intended to identify areas where the ultimate footprint of the TMA dams might be constrained, such as the Rainy River mine property boundary, environmental site assessment boundary, and critical mine infrastructure (e.g., biochemical reactor (BCR) #1, and the West Creek Diversion Channel (WCDC)).

As part of the Ultimate Design Report (BGC, January 28, 2022), stability modelling was completed for selected geotechnical Design Zones for the TMA South Dam, TMA West Dam (Dams 4 and 5), and TMA North Dam at the ultimate crest elevation of 379.1 m. At the time of preparing the report New Gold requested that BGC assume the design transitions from a low permeability clay core to a geosynthetic liner system for the 2023 TMA Stage 5 raise. The proposed feasibility design consists of transitioning from a clay core centerline raise to a downstream raise with the geosynthetic liners placed on 3H:1V upstream slope. As noted, New Gold wishes to maintain the current clay core design with downstream filters and downstream rockfill shell following a centerline raise design concept.

The TMA ultimate dams were designed to satisfy the end of construction static, pseudo-static, and post-earthquake stability criteria. The stability criteria, loading conditions, geotechnical parameters and analysis methodologies are consistent with the design basis (BGC, January 12, 2022b). As noted above, stability modeling was completed on the geosynthetic liner system and not on the revised clay core cross section. Further, the stability modeling undertaken for the geosynthetic liner system was only for the ultimate crest elevation and not completed for the interim dam raises. For updating the estimated material quantities, the estimated downstream buttressing required at the ultimate crest elevation for the geosynthetic liner system was assumed to be adequate to support the revised clay core cross section. Changes to the ultimate design assumptions should be captured in the annual dam raise designs and in future updates of the ultimate design report, which may require upstream and downstream buttressing to be modified to satisfy required factors of safety. Results of stability modeling for the ultimate design with the geosynthetic liner system are presented in the Ultimate Design Report (BGC, January 28, 2022).

The minimum ultimate buttress requirements to satisfy the downstream stability of the TMA Dams are summarized in BGC (January 28, 2022) with the TMA ultimate footprint shown Figure 18.3 and additional ultimate buttressing shown in Figure 18.4. Note, the presented results are intended as guidance for planning purposes as detailed designs will be required for future dam raises at the TMA. Based on the design assumptions and stability analyses completed for the TMA ultimate design, the following areas have been identified as having downstream impacts and will need to be addressed during detailed design. Detailed design and planning, including updating the stability analyses, should be completed to refine the buttressing requirements considering changes to the design basis (i.e., clay core rather than geosynthetic liner), any additional site investigation and align with updated non-acid generating (NAG) rockfill quantities and fill placement schedule.

- The TMA North Dam ultimate footprint extends to the property boundary near Sta. 1+000 m and encroaches into the existing seepage collection system Sump 4 near Sta. 1+200 m and into the seepage collection system ditch from approximately Sta. 0+300 m to Sta. 0+550 m. Although infilling along portions of the seepage collection system is anticipated, the pre-load recommendations for the ultimate crest elevation provided by BGC (BGC, January 12, 2022a) indicated to not infill the seepage collection system at this time; however, stability should be assessed during future design stages to assess whether infilling of the seepage collection system is necessary.
- The ultimate footprint for the TMA South Dam is adjacent to the WCDC between approximately Sta. 2+500 m and Sta. 2+600 m, which is based on estimated PWP dissipation. The PWP dissipation assumptions will need to be verified in future detailed design stages and buttressing updated as required. If PWPs do not dissipate as expected, additional buttress fills may be required that will encroach on the 80 m setback from the WCDC.
- The current ultimate pipeline corridor will be impacted by the TMA South Dam buttress fill from approximately Sta. 0+300 m to Sta. 2+350 m due to the slope of the pipeline corridor. Detailed design and planning should be completed to refine downstream fills or to address potentially raising the pipeline corridor to the required elevation.
- For the TMA West Dam (Dam 4), stability modeling indicated the ultimate dam footprint would need to extend into the downstream nitrification cells or BCR #1 of the water treatment system to meet the minimum required FOS. The infilling of the nitrification cells or the BCR #1 was not required for the pre-load recommendations provided by BGC (January 12, 2022a), PWP dissipation in this area should be monitored, and buttressing requirements refined during

future detailed design stages. If the extent of the buttressing cannot be modified to avoid placement of buttress material into the nitrification cells or BCR #1, the design cross section should be re-evaluated.

- Downstream buttressing to satisfy the TMA West Dam (Dam 5) stability for the ultimate crest elevation is predicted to extend into the WMP pond. The reduced WMP storage volume and the constructability of placing buttress fills sub-aqueously into the WMP will need to be considered.
- Slope stability was not assessed for the TMA Saddle Dam and the TMA Flood Protection Berm, which will need to be completed during future detailed designs. Additional site investigation to support detailed design of the TMA Flood Protection Berm footprint is required as there is currently no site investigation for this proposed structure.

Refer to the Ultimate Design Report (BGC, January 28, 2022) for a discussion of all identified risks and opportunities for the TMA design, including design basis, closure, seepage collection and analyses

18.7.5 Construction Material Quantities

The estimated construction material quantities are based on the stability assessments completed for the feasibility level Ultimate Design Report (BGC, January 28, 2022). LOM quantities were then updated to account for the clay core design (BGC, March 14, 2022). However, no additional stability assessments were completed and buttressing requirements were assumed to be consistent with the Ultimate Design Report (BGC, January 28, 2022).

The estimated construction quantities for 2023, 2024, and 2025 presented are a Class 4 estimate with accuracy ranging from -30% to +50% % (AACE International, August 7, 2020). The estimated construction quantities for 2022 are based on the 2022 TMA Stage 4 design (BGC, January 12, 2022a), and are considered to be a Class 1 estimate with accuracy ranging from -5% to +15% (AACE International, August 7, 2020).

The estimated material take-offs (MTOs) for construction materials are summarized in Table 18.1. Detailed assumptions for the development of the estimates are provided in BGC (March 14, 2022). The key assumptions and limitations related to the MTOs are listed below:

- Clearing and grubbing areas have been estimated based on the TMA ultimate footprint. This was assumed to occur in 2022 for the 2022 pre-load placement, and in 2023 for the construction of the TMA Saddle Dam and TMA Flood Protection Berm.
- The estimated volumes do not account for settlement that will occur during operations or after the final dam raise or mine closure phase.
- Buttress construction based on the Pre-loading Design (BGC, January 12, 2022a) is assumed to be completed by November 2022.
- Subsurface conditions of the TMA Flood Protection Berm assumed the structure will be founded in soil and will not require tie-in to bedrock.
- The emergency spillway construction for operations assumes a similar or same design as the 2022 TMA Stage 4 design. BGC assumed the lock blocks will

not be reused, and that approximately 75% of Zone 7 (Cobbles and Boulders) and Zone 10 (Boulders) materials can be reused.

- Material quantity estimates for the closure spillway are not included.
- Construction of the TMA Saddle Dam and TMA Flood Protection Berm is assumed to start in 2023.
- The quantities provided are estimates based on neat line, straight measures, and compacted in place. Allowances for contingencies, over building, access ramps, or wastage are not considered.

New Gold provided annual volumes of available NAG rockfill for construction on September 24, 2021. Consistent with previous NAG supply estimates, BGC assumed 70% of the total NAG available will be suitable for construction. Based on volumes provided, BGC understands that after 2023, NAG rockfill availability will decrease significantly. A summary of the NAG rockfill (Zone 3 and Zone 3A) supply and requirements are provided below in Table 18.3.

Table 18.3 – Summary of TMA Dams Life of Mine Quantities – All Dams

Material Type	Units	Total (2022-2025)	Year				
			2022 Crest El. 375.1 m		2023 Crest El. 376.6 m	2024 Crest El. 377.9 m	2025 Crest El. 379.1 m
			Stage 4 Raise	Pre-Loading			
Dam - Earthworks							
Clearing and Grubbing	ha	6.3	0.4		5.5	0.2	0.2
Common Excavation	m^3	32,665	1,350		29,365	1,125	825
Bedrock Cleaning	m^2	6,585	2,040		4,220	145	180
Dental Concrete	m^3	783	222		516	20	25
Slush Grout	m^3	5,770	1,775		3,705	125	165
Zone 1: Core – Select Clay	m^3	312,500	80,400		113,700	65,800	52,600
Zone 2: Upstream Shell – Random Granular Fill	m^3	1,024,000	263,000		321,900	233,600	205,500
Zone 2A: Upstream Shell – Select Granular Fill	m^3	133,800	32,100		44,100	29,900	27,700
Zone 3: Downstream Shell – Clean Mine Rock	m^3	2,334,100	288,500	1,790,000	226,200	22,200	7,200
Zone 3A: Downstream Shell – Select Clean Mine Rock	m^3	209,700	52,800		62,300	49,000	45,600
Zone 4: Chimney Fine Filter	m^3	67,600	16,500		22,200	15,000	13,900
Zone 4A: Blanket Fine Filter	m^3	11,890	2,090		8,700	600	500
Zone 5: Transition/Drain – Processed Rock	m^3	77,100	20,100		27,000	15,500	14,500
Zone 7: Cobbles and Boulders – Spillway	m^3	1489	850		213	213	213
Zone 10: Boulders – Spillway	m^3	2450	1,400		350	350	350
Dam - Geosynthetics and Anchoring							
Non-Woven Geotextile	m^2	3200	800		800	800	800
Precast Concrete Lock Blocks	ea.	100	25		25	25	25
Seepage Collection System – Earthworks and Geosynthetics							
Clearing and Grubbing	ha	5.7	5.7				

Material Type	Units	Total (2022-2025)	Year				
			2022 Crest El. 375.1 m		2023 Crest El. 376.6 m	2024 Crest El. 377.9 m	2025 Crest El. 379.1 m
			Stage 4 Raise	Pre-Loading			
Common Excavation	m^3	14,100	14,100				
Zone 1: Core – Select Clay	m^3	11,100	11,100				
Zone 3: Downstream Shell – Clean Mine Rock	m^3	430	430				
Zone 3A: Downstream Shell – Select Clean Mine Rock	m^3	80	80				
Zone 7: Cobbles and Boulders	m^2	45	45				

Notes:

- The quantities presented herein are estimates only based on neat line, straight measures, and compacted in place. Allowances for contingencies, over building, access ramps, or wastage are not considered.
- Definitions: “ha” = hectares, “ea” = each.
- 2022 – Stage 4 Raise and Preloading refers to material quantities required as per BGC’s TMA Stage 4 Detail Design Rev. 1 Issued for Construction Drawings, dated January 12, 2022.
- TMA Life-of-Mine quantities are based on feasibility level design and will be refined based on detailed raise designs.

Table 18.4 – NAG Rockfill Supply and Requirements for TMA Construction

Description	Units[1]	2022	2023	2024	2025	Total
NAG Volume available for TMA Construction[2]	Mm^3	3.3	2.6	0.5	0.1	6.5
NAG Suitable for Construction[4]	Mm^3	2.3	1.8	0.3	0.1	4.5
NAG Volume required for TMA Construction[5]	Mm^3	2.1	0.3	0.1	0.1	2.6[6]

Notes:
1. Mm = megameter
2. LOM NAG tonnages provided by New Gold (email communication from Brent McFarlane, September 24, 2021). Quantities includes 20% bulking factor.
3. NAG volume based on a unit weight of 2.04 t/m^3.
4. Based on direction from New Gold, 70% of total NAG is assumed to be suitable for TMA construction.
5. Based on Zone 3 and Zone 3A rockfill required for annual raise construction and buttress pre-loading. This includes all rockfill required for Zone 3 and Zone 3A above the 2021 TMA Stage 3 Raise as-built configuration.
6. Total Zone 3 and Zone 3A required for the ultimate configuration and above the original ground surface based on the September 2021 Lidar surface.

Based on the NAG rock volumes and timing of availability provided by New Gold, BGC anticipates sufficient supply of NAG rockfill in 2022, but if changes are required, there is a risk of insufficient supply. This would delay the construction of the 2022 pre-load (BGC, January 12, 2022b), which is considered optional for placement in 2022. However, since pre-loading is intended to reduce overall fill quantities in the future, BGC recommends constructing as much as reasonable in 2022 to allow PWPs to dissipate. Extending the pre-load construction plan beyond 2022 may require additional volume of NAG rockfill to meet future stability requirements.

18.7.5.1 MTO Considerations

The following risks, opportunities, and recommendations have been identified based on the MTOs presented above:

- The TMA ultimate configuration, footprint, and quantities are provided for feasibility level design and will need to be updated during the detailed design for annual raises of the TMA, incorporation of the centerline, clay core raise, and as more data becomes available.
- NAG volumes provided in Table 18.4 indicates the available volume of NAG in 2022 is approximately equal to the amount required for construction. Based on discussions with New Gold, BGC understands that NAG quantities available for TMA construction may change from what was assumed for this TMA Ultimate Design Update.
- The additional buttressing shown on the Pre-load Drawings (BGC, January 12, 2022a) is considered optional, but if placed in 2022 would reduce the overall fill quantities required in the future.
- Extending the pre-loading plan beyond 2022 will extend the period of PWP dissipation which will likely require additional buttressing and additional NAG volumes compared to those presented.

- Based on the NAG rock volumes and timing of availability provided by New Gold, BGC anticipates that in 2025 New Gold will have a potential shortage of suitable NAG for construction. A plan to allocate sufficient NAG for dam construction should be developed to mitigate this risk.
- The TMA Dams are expected to continue to settle after the last crest raise in 2025. Material quantity requirements in 2025 will increase if the crest is overbuilt to accommodate this settlement. This may result in a shortage of suitable NAG for construction.
- The ultimate buttressing plan is considered a feasibility level design, which should be revisited depending on NAG availability, scheduling requirements, and detailed design stages.
- Additional materials will be required for constructing the closure spillway. These quantities will need to be considered and planned for once the closure spillway design is developed.

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. All systems in the water treatment train are operational from April through to November and is shut down during the winter months.

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 Low-Grade Ore (LGO)) and overburden waste is provided at two locations, the East Mine Rock Stockpile (EMRS) and the West Mine Rock Stockpile (WMRS) as shown in Figure 18.1.

At the end of year 2022, following mining of the open pit north lobe to elevation 70 m, an in-pit waste rock stockpile will be established. The conceptual level design of the in-pit waste stockpile provides an additional 22 Mt (about 10.0 Mm^3) of mine waste rock capacity. The implementation of the in-pit waste dump will allow additional time for foundation strength gain at the existing mine waste stockpiles and relieve strict scheduling requirements of PAG waste rock placement in the EMRS. Also, the TMA is planned to receive 3.1 Mm^3 of NAG and 1.4 Mm^3 of PAG.

The capacities of the EMRS, the WMRS, the in-pit waste dump and the TMA to receive mine waste rock (NAG and PAG) as well as overburden (OVB) and LGO are summarized in Table 18.5.

Table 18.5 – Capacities of Waste Storage Facilities

Structure	Available Capacity (Mm^3)				
	Overburden	NAG	PAG	LGO	Totals
EMRS	0.9	19.1		12.1	32.1
WMRS	13.6	11.9	-	-	25.5
TMA	-	3.1	1.4	-	4.5
In-pit Waste Stockpile	-	-	10.0	-	10.0
Totals	14.5	15.0	30.5	12.1	72.1

Note: The remaining quantities shown are as at the end of 2021

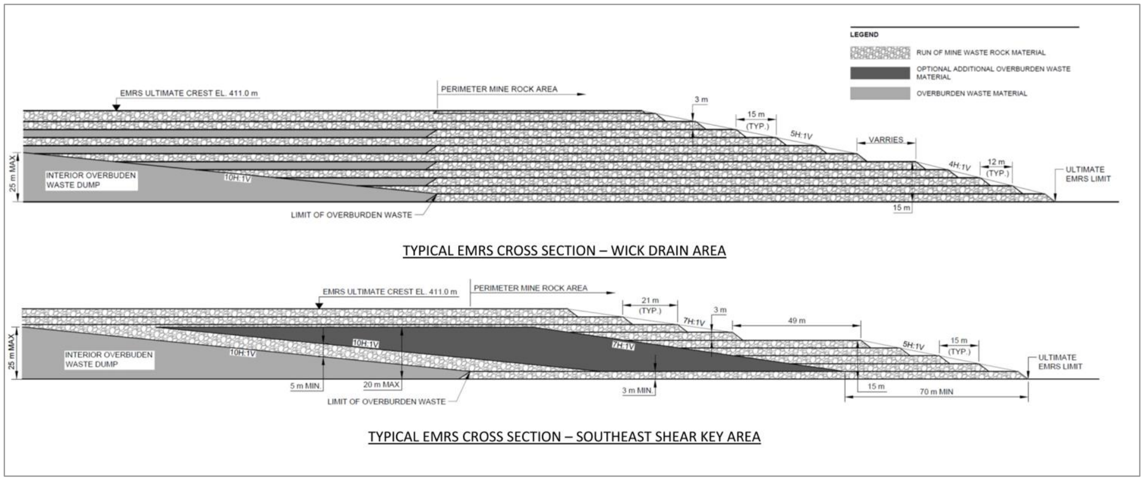
The EMRS and WMRS utilize an extensive geotechnical instrumentation monitoring and surveillance system consisting of slope inclinometers, vibrating wire piezometers and settlement plates to continually assess the performance of the structures. Trigger action response plans are in place for the instrumentation to inform mine operations should any abnormal measurements be detected so that appropriate actions can be taken to ensure safe operation of the facilities.

18.9.1 East Mine Rock Stockpile

The EMRS was designed 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 in the centre of the west side 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 3 m lifts of waste rock above the internal dump and within the south-east shear key area. Waste rock is being placed either internally above the overburden dump, or around the perimeter of the EMRS where it serves to buttress the internal overburden dump.

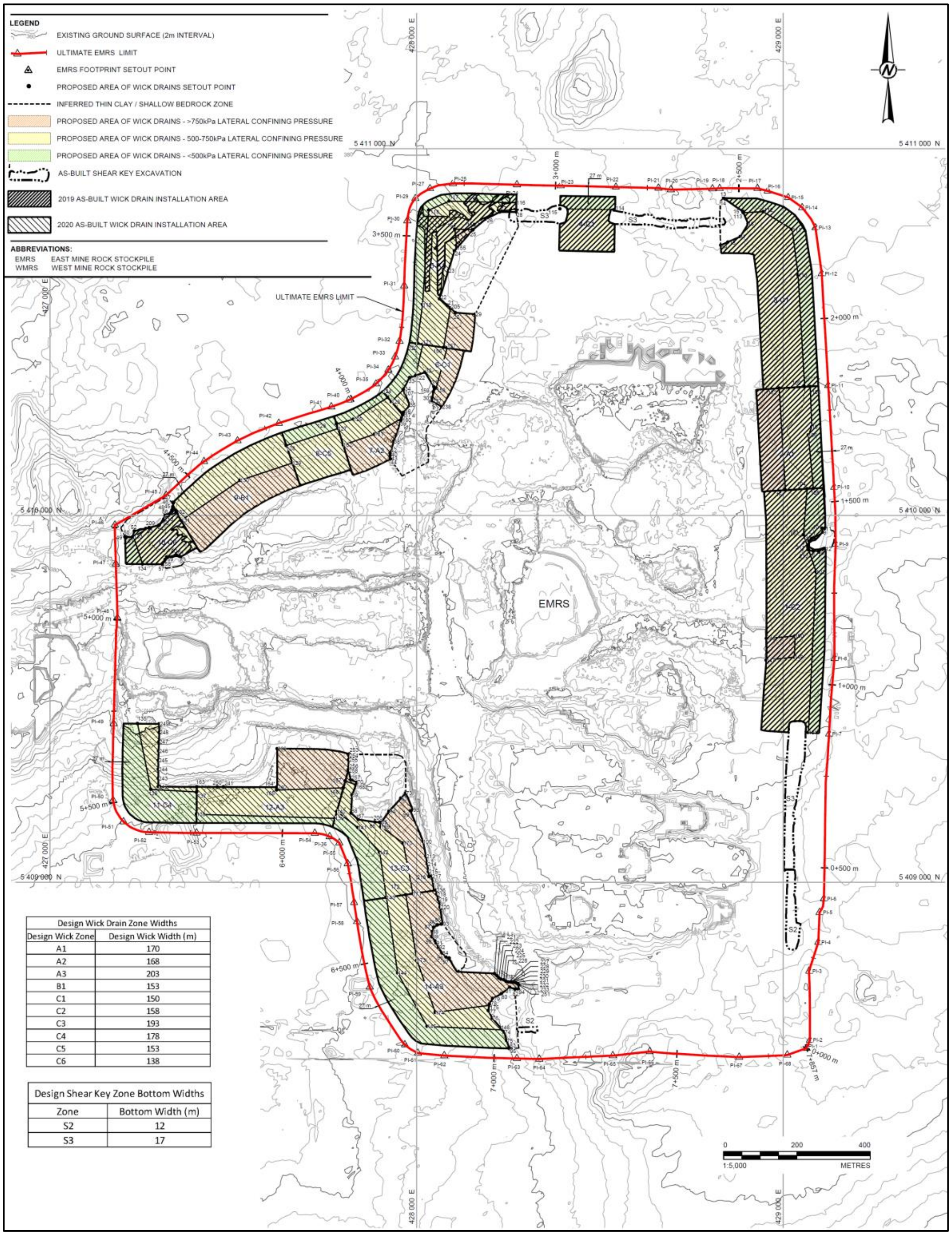
The EMRS typical section is provided in Figure 18.5. The EMRS design crest elevation varies, from a minimum elevation of 402 m in the east and the west, to a maximum elevation of 411 m in the north and the south. The EMRS design perimeter slopes vary from 4H:1V (Horizontal: Vertical) to 5H:1V below 15 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 LGOS area, a 95 m wide intermediate bench is required at elevation 376 m.

Ground improvement measures have been implemented to improve the shearing resistance of the foundation of the EMRS perimeter (as shown in Figure 18.6). EMRS ground improvement measures comprised constructing waste rock shear keys (in areas where foundation clay thicknesses are between 3 m and 8 m), and installing wick drains at 2 m spacing (in areas 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 porewater pressures 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.



Source: Golder 2020a.

Figure 18.5 – EMRS typical cross sections



Source: Golder 2020a.

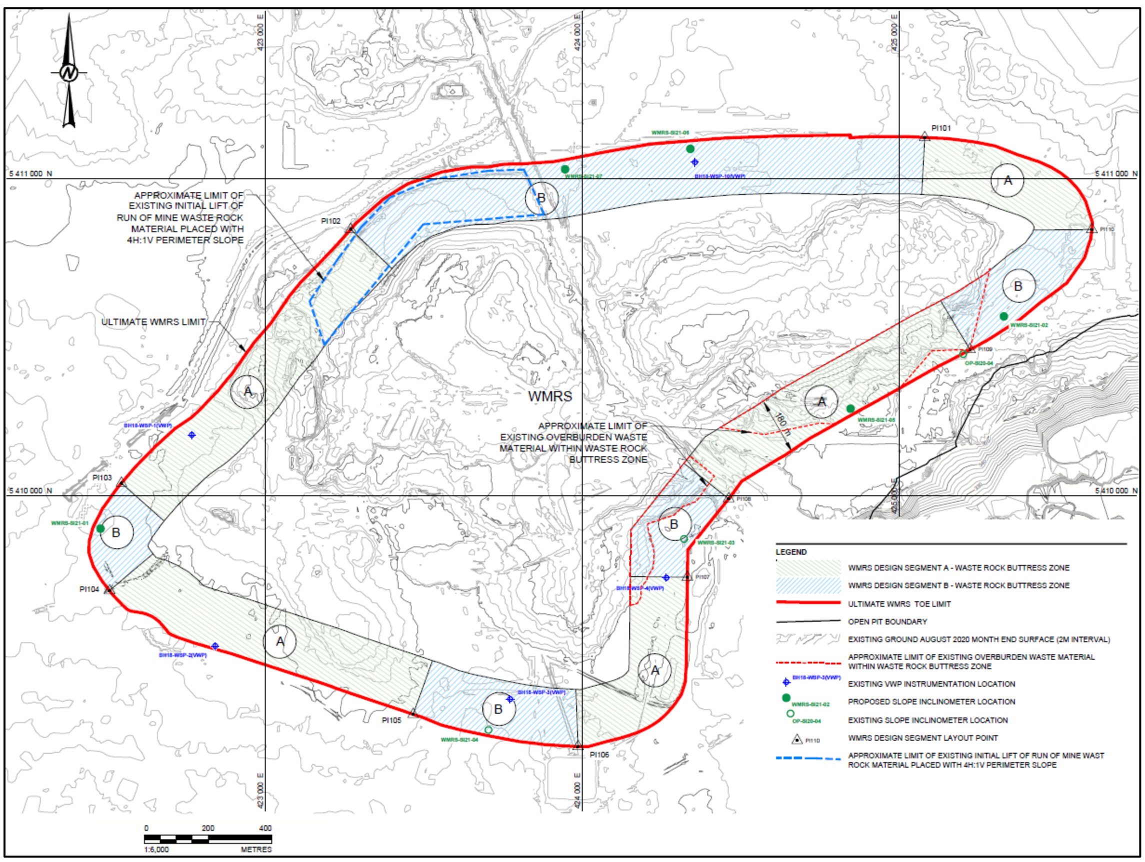
Figure 18.6 – EMRS ground improvement layout plan

18.9.2 West Mine Rock Stockpile

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 functions to buttress the overburden waste. The WMRS has been designed with a stockpile height and slope geometry which does not require foundation improvement measures.

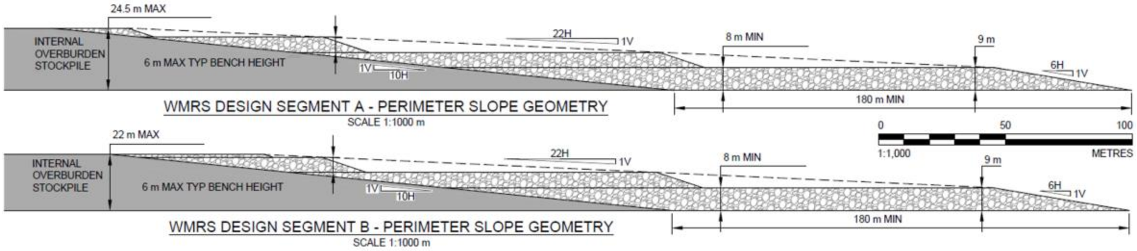
The WMRS geometry plan is provided in Figure 18.7. The WMRS has been designed to a maximum height of 24.5 m. The internal overburden dump may extend to the full height of the WMRS. The overburden dump will have a minimum 180 m setback from the ultimate WMRS perimeter, and it will be constructed with 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 are 6H:1V and 22H:1V slopes for heights below and above 9 m, respectively.

The design includes two typical design sections (Sections A and B on Figure 18.8), considering maximum foundation clay thicknesses of 26 m and 37 m, respectively. A revised design of the WMRS northeast area is currently ongoing due to the presence of a historical fill within this area.



Source: Golder 2021.

Figure 18.7 – WMRS plan view



Source: Golder 2021.

Figure 18.8 – WMRS design sections

19 METAL PRICES

Project economics have been assessed using the following metal prices:

- Gold price = $1,400/oz
- Silver price = $19/oz

According to the London Bullion Market Association (LBMA), the average daily PM Fix gold price for 2021 was $1,799 per troy ounce. The three-year and five-year rolling average prices through the end of December 2021 are $1,651 and $1,496 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 $1,151 and the high of $2,067 during the five-year period.

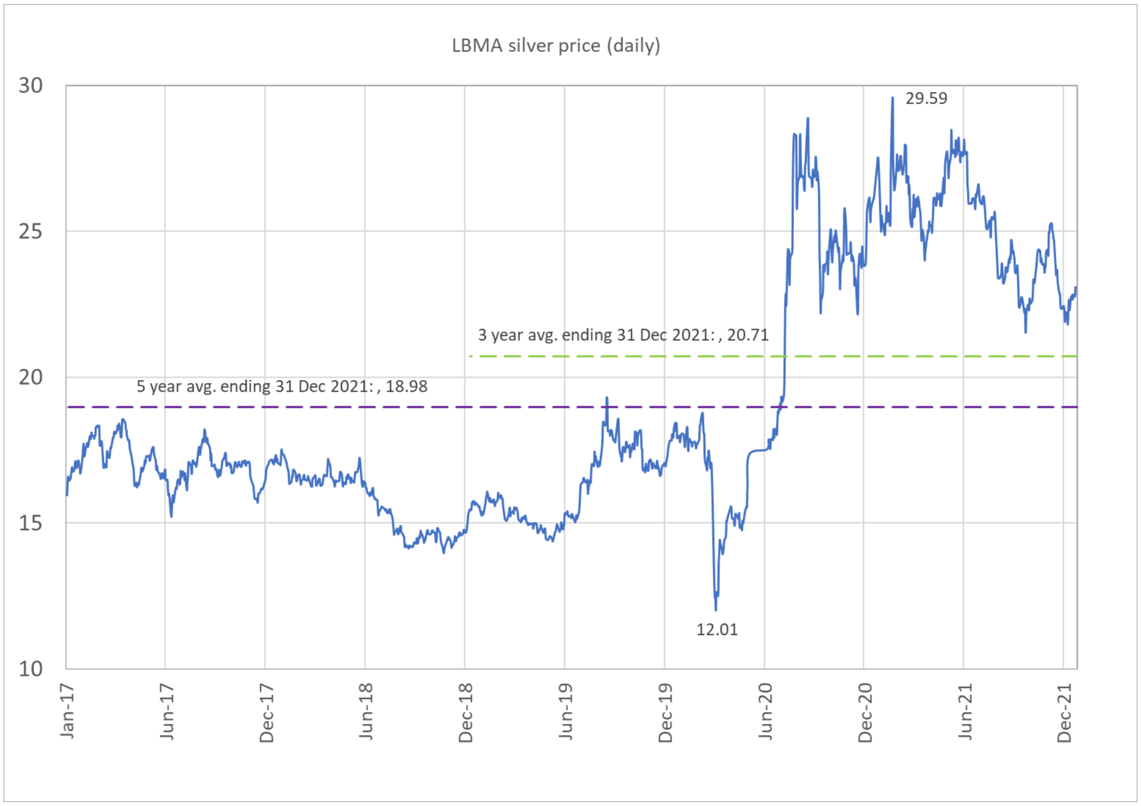
Source: kitco.com 2022.

Figure 19.1 – LBMA PM Fix gold price (daily)

According to LBMA, the average daily silver price for 2021 was $25.14 per troy ounce. The three-year and five-year rolling average prices through the end of December 2021 are $20.71 and $18.98 per troy ounce, respectively.

The volatility of the silver price over these periods can be seen illustrated in Source: kitco.com, 2022.

Figure 19.2, and shows the low of $12.01 and the high of $29.59 during the five-year period.



Source: kitco.com, 2022.

Figure 19.2 – LBMA silver price (daily)

Based on the undertaken review, the metal prices selected by New Gold are reasonable, particularly given the recent appreciation in metal prices.

19.1 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 296 k oz gold and 520 k oz silver over the period 2022 – 2028 and an annual average of 150 k oz of gold and 145 k oz of silver over the period 2029 to the end of the life of mine, for a total annual average of 252 k oz gold and 407 k oz of silver.

19.2 Contracts

19.2.1 Gold price option contracts

New Gold no longer participates in precious metals options contracts for Rainy River production. Metal production is thus sold at 'spot' prices.

19.2.2 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.

19.2.3 Other contracts

As of 31 December 2021, the main contracts involved with the mine are:

- Refining: Asahi Refining Canada Ltd. and Royal Canadian Mint
- Electricity: Independent Electricity System Operator
- Fuel supply: Shell
- TMA construction: currently in tender
- Tire supply: Michelin North America, Inc.
- Explosive supply: Dyno Nobel Canada Inc.
- Lubricant supply / support: Anokiigamig / Petro
- Sodium cyanide supply: Chemours
- TMA engineering: Currently in Tender
- Camp catering / housekeeping: Anokiigamig

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.

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 Sustainability and Safety Policy approved by the Board of Directors on March 8, 2021.

The Environmental Department is adequately staffed, and has accountabilities including water resource management, ambient air quality, wildlife monitoring, surface water, and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

From 2020 to 2021, the Rainy River Mine has recorded 22 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.

On July 31, 2020, the Impact Assessment Agency of Canada issued a Notice of Non-Compliance for New Gold's Rainy River Mine not compensating for the loss of fish habitat in accordance with the Metal and Diamond Mining Effluent Regulations by failing to achieve the success criteria of recreating functional fish habitat in the Stockpile Pond. Investigation into the fish habitat of Stockpile Pond was started in Q3 2020 and continued through 2021. Remediation measures are planned to be installed in Q3 and Q4 of 2022.

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 Plant. 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 and crusher operations, with less significant impacts from other site generated activities.

Background air quality data for particulate matter is reported from three on site ambient air monitoring stations.

20.2.2 Acoustics

Annual acoustic audits are performed at locations in and surrounding the mine site to ensure mine activities do not exceed applicable regulatory sound level criteria. Significant noise sources include equipment associated with construction activities at the TMA and mine rock stockpile development.

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). 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 (ECA) permit, updates to the hydrogeological model (i.e., groundwater flow model) are to occur every 3 years during mine operations. Wood PLC (Wood) provided the first update in the 2017. Klohn Crippen Berger Ltd. (KCB) developed the latest 1-D and 3-D transient groundwater model in 2020. The model has been reported to the regulator on March 2021 as part of the annual groundwater monitoring report. The next model update will be in 2023 as required by the permit.

Based on the 2020 assessment by KCB, the extent of the zone of influence (ZOI) has changed when compared to the previous steady state model predictions by Wood. Specifically, the ZOI extends further to west and southeast, and does not extend as far to the east and south. Site wide groundwater monitoring wells will continue monitoring groundwater levels and confirm any changes to the predicted drawdown cone from dewatering of the open pit.

20.2.5 Surface Water

Seventeen surface water monitoring stations are located both upstream and downstream of current plant and mine facilities, positioned accordingly along the Rainy River, 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 ECA permitted limits. 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, MRP, 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 Monitoring

Groundwater monitoring is regularly completed by site personnel at 45 monitoring wells and 3 VWP 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. The 2021 groundwater quality monitoring results are very similar to 2016 baseline results, indicating minimal change in conditions. 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 ECA permit conditions.

Constructed fish habitat monitoring is comprised of fish community surveys, fish habitat surveys, and associated reporting. Fish community and fish habitat surveys are conducted at the West Creek, Stockpile, Clark and Teeple Ponds and associated diversion. All constructed fish habitat and communities have met approved criteria except for Stockpile and Teeple Ponds and associated diversions. Further contingencies and monitoring will be completed in 2022.

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. Recent studies 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 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 has continued throughout operations until a mature vegetation community is established.

20.2.9 Wildlife

Bird monitoring studies suggest 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 monthly 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. 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.

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 were made 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, with exception of periodic dam raises, which are requested annualy. 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 almost fully implemented 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 2021, the mine work force was 23% 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 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 ENDM 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 ENDM 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. Once provided, it was filed by ENDM.

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 $123M, 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. The financial assurance provided to ENDM 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 $104M based on current disturbance as of 31 December 2021.

21 CAPITAL AND OPERATING COSTS

Capital and operating costs have been estimated by New Gold throughout their 2022 Budget and LOM planning process and have been reviewed by AMC and InnovExplo. All costs presented in this Item are presented in constant Q1-2022 US$, with no inflation or escalation factors considered. Where applicable a foreign exchange rate of C$:US$ of 1.25 was utilized.

Project capital mentioned in this item refers to growth capital for the expansion of the current mine.

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.

Total LOM capital costs are estimated to total $718M as summarized in Table 21.1. This excludes $104M in funds identified for progressive and final closure. Details for each category follow in this Item.

Table 21.1 – Capital Costs Summary

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
Open Pit	USD $M	89.9	86.9	8.6	2.0	3.9	1.6	0.0				192.9
Underground	USD $M	17.6	39.9	86.2	67.8	64.9	50.7	34.5	15.9	10.6	3.2	391.2
Process Plant	USD $M							1.3				1.3
TMA, Infrastructure & Other	USD $M	38.4	34.5	27.9	22.6							123.4
Capital Exploration	USD $M	1.7	1.1									2.8
Working capital	USD $M	0.7	-2	1.7	2.3	1.9	1.3	0.6				6.5
Grand Total	**USD $M**	**148.3**	**160.5**	**124.4**	**94.7**	**70.7**	**53.6**	**36.4**	**15.9**	**10.6**	**3.2**	**718.2**
Project Capital	USD $M	16.6	27.2	25.3	2.3							71.4
Sustaining Capital	USD $M	131.7	133.2	99.2	92.4	70.7	53.6	36.4	15.9	10.6	3.2	646.9
Grand Total	**USD $M**	**148.3**	**160.5**	**124.4**	**94.7**	**70.7**	**53.6**	**36.4**	**15.9**	**10.6**	**3.2**	**718.2**

Note: Totals may not add exactly due to rounding.

21.1.1 Open Pit Capital Costs

The open pit capital cost is estimated to total $193M as summarized in Table 21.2.

Table 21.2 – Open Pit Capital Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
Mobile Maintenance, Parts & Components	USD $M	20.6	23.3	8.6	2	3.9	1.6					60.0
Capital & Deferred Stripping	USD $M	65.5	63.6									129.1
Overburden Sloping	USD $M	3.8										3.8
Grand Total	**USD $M**	**89.9**	**86.9**	**8.6**	**2.0**	**3.9**	**1.6**	**0.0**				**192.9**

Note: Totals may not add 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: a replacement water truck, drill automation systems, dewatering pumps,, etc.
- Capitalized / deferred stripping costs associated with the extraction of 43 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.

21.1.2 Underground Capital Costs

The underground LOM capital cost is estimated to total $391M, inclusive of contingency, with $65M in project capital and $326M in sustaining capital, as summarized in Table 21.3.

The development cost and initial infrastructure costs for each zone is classified as project capital (non-sustaining) For simplification, when ore is realized, all infrastructure cost and continued development (with the exception of main decline ramp) is, thereafter, classified as sustaining capex.

Table 21.3 – Underground Capital Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
Intrepid- horizontal development	USD $M	10.2	11.6	7.5	9.7	10.0	0.0	0.0				48.9
Intrepid- vertical development	USD $M	0.6	0.6	0.4	0.5	0.6	0.0	0.0				2.7
Intrepid- infrastructure & equipment	USD $M	3.3	1.3	0.7	0.9	1.1	0.0	0.0				7.3
Intrepid - other	USD $M	3.4	0.1									3.5
Below Pit- horizontal development	USD $M		10.7	27.3	34.9	37.9	35.4	23.0	7.2	3.9	0.0	180.3
Below Pit- vertical development	USD $M		1.4	4.8	3.7	3.5	3.5	3.8	1.9	0.5	0.0	23.0
Below Pit- infrastructure & equipment	USD $M		9.0	39.5	18.2	11.8	11.8	7.7	6.8	6.2	3.2	114.2
Below Pit - other	USD $M	0.1	5.1	6.1	0.0	0.0	0.0	0.0	0.0	0.0	0	11.3
Underground	**USD $M**	**17.6**	**39.9**	**86.2**	**67.8**	**64.9**	**50.7**	**34.5**	**15.9**	**10.6**	**3.2**	**391.2**
Project Capital	USD $M	14.7	26.1	24.6								65.4
Sustaining	USD $M	2.9	13.8	61.6	67.8	64.9	50.7	34.5	15.9	10.6	3.2	325.8
Grand Total	**USD $M**	**17.6**	**41.0**	**86.2**	**67.8**	**64.9**	**50.7**	**34.5**	**15.9**	**10.6**	**3.2**	**391.2**

21.1.3 Process Capital Costs

The process capital costs are estimated to total USD$1.3M and relate to capital investment required to down-size the mill facility as summarized in Table 21.4.

Table 21.4 – Process Capital Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
Process	USD $M							1.3				1.3

21.1.4 Tailings Management Area and Infrastructure Capital Costs

The tailings management area and infrastructure capital cost are estimated to total $127M, as summarized in Table 21.5.

Table 21.5 – Tailings Management Area and Infrastructure Capital Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
TMA	USD $M	31.4	31.8	24.4	22.6							110.2
Stockpile Diversion Dam	USD $M	1.3										1.3
Other	USD $M	5.7	2.7	3.5								11.9
Grand Total	**USD $M**	**38.4**	**34.5**	**27.9**	**22.6**							**123.4**

Principal Tailings Management Area and Infrastructure capital costs include, but are not limited to the following principal items:

- TMA represents the expansion of the current tailings facility to accommodate the tailings generated from the processing of an additional 70 Mt of ore in the current mine plan via annual tailings dam raises.

The capital cost estimate is considered to be appropriate for process functions.

21.2 Operating Cost

Operating costs have been estimated using first principal 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 21.6. Estimated unit operating costs, plus the LOM average, are shown in Process Capital Costs.

Table 21.6 – Estimated Unit Operating Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Total
Open Pit	USD $M	98.9	101.9	118.2	43.7	24.1	24.5	16.0				427.3
Underground	USD $M	17.3	23.9	52.3	84.8	100.7	103.4	93.2	81.2	75.1	49.5	681.4
Process	USD $M	71.6	71.1	67.0	65.6	64.6	64.4	49.4	24.7	24.4	18.2	521.0
G&A	USD $M	32.6	29.0	28.2	26.4	26.1	24.9	24.2	17.2	17.3	15.1	240.9
Grand Total	**USD $M**	**220.6**	**225.8**	**265.8**	**220.4**	**215.5**	**217.2**	**182.9**	**123.0**	**116.8**	**82.8**	**1,871.3**

Table 21.7 – LOM Average

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Avg.
Open Pit	USD $/t mined	$2.94	$3.10	$3.63	$13.41							$4.18
Underground	USD $/t mined	$83.99	$70.69	$72.74	$55.64	$52.65	$52.68	$52.27	$49.41	$46.23	$40.97	$52.73
Total Mining (including rehandle)	USD $t/milled	$12.28	$12.76	$17.30	$13.03	$12.66	$12.98	$15.61	$49.40	$46.23	$40.83	$15.79
Process	USD $t/milled	$7.56	$7.21	$6.80	$6.66	$6.55	$6.53	$7.06	$15.03	$15.03	$14.98	$7.42
G&A	USD $/t milled	$3.47	$2.94	$2.86	$2.68	$2.65	$2.52	$3.46	$10.46	$10.61	$12.49	$3.44
Grand Total	**USD $/t milled**	**$23.31**	**$22.91**	**$26.97**	**$22.37**	**$21.86**	**$22.04**	**$26.13**	**$74.89**	**$71.88**	**$68.31**	**$26.65**

Table 21.8 shows expected metal production by year and additional information. Gold, silver and gold equivalent ounces produced are shown by period. Gold Equivalent ounces production is calculated from the value of the silver ounces produced, converted to gold ounces, and added to gold ounces produced. Also shown in the table are the direct operating expenses related to mining (OP and UG), processing and G&A. Other operating expenses include Royalties, inventory fluctuations and other costs.

OPEX per gold equivalent ounces is the summation of operating expenses divided by gold equivalent ounces. The sustaining capital is also shown by year, and when added to OPEX, forms the basis of the AISC per gold equivalent ounce calculation.

Table 21.8 – LOM Production & Operating Costs

Description	Unit	2022	2023	2024	2025	2026	2027	2028	2029	2030	2031	LOM Avg.
Gold Production	oz	261,981	271,129	309,927	321,489	323,199	324,816	262,850	155,525	171,465	122,230	252,461
Silver Production	oz	421,242	550,678	529,604	550,829	543,765	575,936	465,560	200,380	131,123	104,218	407,333
Gold Equivalent Production	oz	267,697	278,603	317,114	328,965	330,579	332,632	269,169	158,244	173,245	123,644	257,989
Operating Expenses (Mining, Processing, G&A)	US$ M	220.6	225.8	265.8	220.4	215.5	217.2	182.9	123.0	116.8	82.8	187.1
Other Operating Expenses	US$ M	6.7	9.8	-7.9	27.5	5.3	6.5	8.8	3.2	3.2	1.8	6.5
OPEX/Au. Eq. Oz	US$ / oz	$849	$846	$813	$754	$668	$672	$712	$798	$693	$651	749
Sustaining Capital	US$ M	$132	$133	$99	$92	$71	$54	$36	$16	$11	$3	65
AISC/Au. Eq. Oz	US$ / oz	$1,428	$1,410	$1,181	$1,093	$922	$884	$889	$898	$754	$676	1,047

Note:

- Gold Equivalent Production (oz) = gold production (oz) + [silver production (oz) x silver price ($/oz)] / gold price ($/oz)
- Where: Gold price used = US$ 1,400 / oz; Silver price used = $US 19 / oz

22 ECONOMIC ANALYSIS

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

23 ADJACENT PROPERTIES

There are no adjacent properties to report in this Item.

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 CONCLUSION

25.1 Introduction

The QPs have provided the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for this report

25.2 Geology

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

25.2.1 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.

25.2.2 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 19.2 Mt at grades of 2.50 g/t Au and 6.3 g/t Ag, containing 1,543 koz of gold and 3,894 koz of silver. Inferred Mineral Resources are estimated to total 2.5 Mt at grades of 2.37 g/t Au and 2.5 g/t Ag, containing 189 koz of gold and 196 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 extension.

25.3 Mining and Mineral Reserves

25.3.1 Open pit mining and Mineral Reserves

Open pit Proven and Probable Mineral Reserves, including stockpile, total 57.6 Mt grading 0.84 g/t gold and 2.1 g/t silver, containing 1,558 koz and 3,938 koz of gold and silver, respectively.

The mine planning resource model reconciliation provides a better overall prediction of tonnes, grade, and contained metal when compared to the regularized resource model. The reconciliation of the grade control model is relatively good compared to the mine planning resource model. However, mining and / or metal accounting practices during reconciliation appear to be impacting the DOM recorded metal mined from the deposit. New Gold has undertaken additional drilling in 2021, particularly in the East Lobe to improve model predictability and is focusing attention on mining practices for the open pit and metal accounting practices during reconciliation to improve results. Results of the drilling program are expected in Q1-2022 and according to New Gold will be included within an updated Mineral Resource model during 2022.

Modifying factors should be reviewed as new mining areas are exposed and additional reconciliation information is gathered to continue validating the model performance.

The current pit design 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 phase designs are reasonable; however, some further optimization should be undertaken to confirm the best pit bottom elevation upon which to transition from open pit mining to underground. This will permit additional pit limit optimization, particularly related to the south-west area of the final pit limit design.

The open pit overburden design slopes have been designed to meet or exceed slope stability criteria. 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 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).

Current IRA angles range from 37 degrees in the northeast domain of the north lobe (almost completed excavation) to 54 degrees in sectors of various other domains.

Currently, there are recommendations to perform blast trials to evaluate potential back-break and bench-scale rock hazards through the IMV prior to excavation in the southwest design sectors. Based on these trials there may be requirements to modify the design recommendations to improve performance and safety around the planned Phase 4 southwest ramps.

Approximately 145 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 processed generally to supplement excess processing capacity. After completion of open pit mining, the stockpile represents the principal ore feed to the mill, providing mill feed in excess of that possible from the underground operation. After the stockpiles are depleted, the mill will be re-sized to only accommodate underground mill feed for the duration of the mine plan.

Mine equipment requirements were developed from the annual mine production schedule, based on equipment availability, utilization, and equipment productivities. No further additional nor replacement open pit mine principal equipment fleet is considered for purchase during the remaining LOM plan. 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, with peak yearly vertical advance ranging from 7 to 11 benches per year in a single phase. Rainy River has achieved these rates of vertical advance in the past and will need to ensure maintenance of their short- to mid-range planning practices 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. No mining of the EOC is included in the current mine plan. However, NAG quantities being extracted from the mine after 2023 will be significantly reduced and it is recommended that New Gold review mitigating strategies to ensure sufficient quantities are available when required, 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.

25.3.2 Underground mining and Mineral Reserves

The underground Probable Mineral Reserves are estimated to be 12.7 Mt grading 3.05 g/t Au and 7.6 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 1.74 g/t gold equivalent for Phase 1 and 2.25 g/t gold equivalent for Phase 2, which is higher than the initially calculated breakeven COG of 1.7 g/t gold equivalent. The use of two COG allows an underground schedule to continue after pit processing is completed.

Access to the underground is via three portals (i.e. ODM Main Zone, 17 East Upper Zone, and Intrepid Zone). The underground will be a mechanized operation utilizing longitudinal long hole stoping for areas less than 20 m wide and transverse long hole stopes (ODM Main Zone only) where the ore body is thicker than 20m.

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.

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.

25.4 Process and metallurgy

Rainy River’s current focus (for the periods in which Open Pit ores are to be processed) is to operate at higher throughputs than the original plant design throughput. This produces a coarser grind size P80 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.

Dust from the crushed ore stockpile has been identified as an environmental and health concern. Solutions to the dust problem include installation of dry fog system and dust control curtains at discharge point of coarse ore stockpile feed conveyor, CV 11. This modification has been planned to be planned to be installed in 2nd and 3rd quarter of 2022. Rainy River expects these measures should largely remove the dust issue.

After Open Pit ores and Open Pit stockpiles materials are exhausted, the process plan will be downsized and operated on a 'batch' concept. This approach has been evaluated to be practical and achievable with minimal capital expenditure. Ongoing effort will be required to better specify operational details of the batch concept.

25.5 Infrastructure

Regarding general infrastructure, primary access roads, mine haul roads, truck shop, truck wash bay, fuel bays, explosive magazine and emulsion plant, warehousing, lubricant and fuel storage, principal buildings, assay lab, camp, ceremonial roundhouse, emergency power arrangements and communications facilities are all in place and appropriate to support ongoing mining operations.

The TMA and related water management structures are well described in Item 18. Tailings deposition in TMA Cell 1 commenced in November 2017 with placement into TMA Cell 2 beginning in May 2018. Tailings placement into TMA Cell 3 began in May 2019. Generally, the tailings deposition strategy is to establish tailings beaches upstream of the perimeter dams (i.e., TMA North Dam, TMA West Dam [Dams 4 and 5], and TMA South Dam), while maintaining a pond around the fixed reclaim located at TMA Cell 2. Since 2017, the dams have been constructed sequentially every year. The TMA is designed to provide sufficient containment for the projected tailings storage requirements and operational pond volume. The Environmental Design Flood (EDF) is to be stored below the TMA emergency spillway invert level (also referred as the EDF Level or EDFL) and the TMA emergency spillway is designed to pass Inflow Design Flood (IDF).

By 2025 the TMA is projected to have reached a crest elevation of 379.1m. The material quantities required for construction are well known, available, sufficient and the site teams are experienced in ongoing dam construction. TMA construction costs are well known and well managed. Construction costs for subsequent TMA storage to accommodate UG mined tonnage have been included in the Rainy River capital cost model and the UG cut-off grade calculations.

- 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).
- 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.

25.6 Environmental, Social, Community, and Reclamation / Closure

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 Sustainability and Safety Policy approved by the Board of Directors on March 8, 2021.

The Environmental Department is adequately staffed, and has accountabilities including water resource management, ambient air quality, wildlife monitoring, surface water, and groundwater monitoring using current staff and contracts several external consultants to conduct specialized work.

25.6.1 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 is a list of past studies and ongoing monitoring that is programmed to continue during operations:

- Meteorology and Air Quality
- Acoustics
- Geochemistry
- Hydrogeology
- Surface Water
- Groundwater Monitoring
- Aquatic Resources
- Vegetation Studies
- Wildlife
- Species at Risk and Critical Habitat
- Traditional Knowledge and Traditional Land Use (social license)
- Cultural Heritage
- Overall Environmental Sensitivities

Project Permitting: The mine has received all the permits and authorizations needed to construct major infrastructure and operate, with the exception of annual tailing dam raises.

25.6.2 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 2021, the mine work force was 23% Indigenous.

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.

25.6.3 Mine closure

The Rainy River Closure Plan, dated 22 January 2015, was filed by the ENDM 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 ENDM 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. Once provided, it was filed by ENDM.

The cost estimate for implementing project closure in the Environmental Assessment (EA) was estimated to be $118M, 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. The financial assurance provided to ENDM 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 $104M based on current disturbance as of 31 December 2021.

25.7 Risks and Mitigation

Area	Risks Description and Potential Impact	Mitigation Approach
Geology and Mineral Resources	Potential variability and grade continuity within the mineral resource due to local wide drilling spacing.	Future infill drilling program will reduce the spacing between samples informing the mineral resource.
	Reconciliation variance experienced within the East Lobe impacted the mine production profile during 2021 and may continue through the end of the open pit operation. Additionally possible variance on other areas of the open pit could further impact the operation.	Reverse circulation infill drilling program to support an updated block model, which will assess possible difference with the actual one and properly predict the ore material inventory.
	Potential to have underperforming or overperforming areas of the deposit against the resource estimate.	Reconciliation of the model, short term controls and production on 3-month basis. Updating the geological model based upon new mapping, observation, and new drilling data. Revise the classification of Mineral Resource by smoothing the outlines and removing the isolated blocks of Mineral Resources based upon single drill hole.
	Potential variance on the shape and distribution of the mineralization within the underground resources could impact the planned mine design	
Open Pit Mining	Mining practices and / or metal accounting practices during reconciliation appear to be impacting the DOM recorded metal mined from the deposit vs the polygon release model. Tonnage and metal is being lost between the polygon realease model and the DOM accounting.	Review procedures related to: (1) Polygon delineation to ensure polygon shapes support mining direction and minimum excavation widths; (2) Operating procedures are aligned to recover selectively higher-grade material; and (3) Reconciliation procedures account for the transfer of any residual polygon tonnes and grade to / from adjacent polygons.
	Currently, there are recommendations to perform blast trials to evaluate potential back-break and bench-scale rock hazards through the IMV prior to excavation in the southwest design sectors. Based on these trials there may be requirements to modify the design recommendations to improve performance and safety around the planned Phase 4 southwest ramps.	Use best-practice wall control practices in the blast trials to minimize wall damage and impact of any possible back-break. Continue using best practice on all wall control blasts.
	Localized loss of surface water control above the crest of the ODM shear zone area has occurred where the rock mass is more altered and weaker, leading to operational disruption at times in active lower benches and the pit bottom.	This zone along the West wall should be reviewed to identify any alteration / weathering influence or fallts which could contribute to the loss of surface water control. Where possible, the lower benches of the mine should not be advanced during freshnet periods and preferabley used as in-pit sumps for water collection and pumping.
	There is only a single-ramp access to the pit bottom which may impact future operations in the case of a geotechnical event on the Phase 4 south and west walls.	Undertake a risk assessment of the probability of such an event occurring and determine if a secondary ramp access is required.
Underground Mining	Uncertainties related to geomechanical knowledge could lead to delays in production, additional dilution, or additional support requirement.	Additional information will be gathered through future drilling campaigns and extension of the brittle-structural geology model to below the open pit; ground support and sequence will be validated with new information.
	Current Block Model is built for OP	The updated Block Model will need to have

Area	Risks Description and Potential Impact	Mitigation Approach
	optimization; cells are too big for proper UG optimization; discontinuities in the Block Model leads to uncertain stopes shapes	smaller sub cells suitable to UG optimization; new drilling will also improve the confidence in the Block Model
	All underground reserves are probable only – Risk of losing ore once proven	Diamond drilling necessary to promote indicated to measured resources
	Proximity with the active open pit could lead to excessive stressed in stopes and / or instability in the pit	Tight monitoring and geotechnical evaluation of horizontal and vertical pillars will allow to confirm and optimize the safety distance between excavations.
	If the mined out Open Pit is to be used as a waste storage area, the impact of the added mass of the waste and any water will need to be considered with regard to the stability of UG excavations	Geotechnical evaluation of horizontal and vertical pillars will allow to confirm and optimize the safety distance between excavations.
	Simultaneous open pit and underground production could lead to delays and / or safety concerns (blasting, traffic, etc.)	Adjustment to safety parameters regarding the simultaneous production will permit a safe exploitation; new procedures will be implemented for all unsafe or hazardous activities
	Important quantity of consumables needed, especially when only one portal is available, may cause traffic delays; Important quantity of backfill to be brought down from the North Pit waste reserve may create additional undesired traffic.	Traffic management will be optimized beforehand with optimization tools to validate the risk-level to congestion in main ramps; additional passing bays may be added in the final ramp design to alleviate the traffic load; prioritizing procedure to promote a preferential traffic direction between the two portals.
	Manpower shortage is a current risk for any new underground operation; the location of the project does not profit from a vast experienced underground worker's pool; Personnel operational readiness / capability risk during the transitioning process from open pit to underground.	Plan as soon as possible the contractor calls for tender to ensure numerous responses and more accurate quotes; plan for hiring experienced supervisors or key employees for the technical staff; training and formation of existing open pit staff will facilitate the crew build up by transitioning the OP staff to UG.
	Backfill operating costs may be higher if the ratio of CRF / RF is ultimately greater than estimated.	May affect specifically narrow stope, experiences and ground control will dictate the best compromise between high CRF volume and cost efficiency.
	Planned single heading advance and timeframe to reach the ventilation raises pose a moderate risk to the project schedule.	Priorities need to be maintained and monitored closely; optimize the development methods used by the contractor (mainly for the main ramp and ventilation raises access); prioritize faster methods and alternative tools to accelerate development rate (automation, critical ground support only, rapid cycling, longer round, etc.)
	In-pit rock stockpile in the North Lobe may present additional stress to the crown pillar, especially if combined with water accumulation in the North Lobe.	Evaluate and optimize the crown pillar dimensions considering the in-pit rock stockpile and potential water accumulation in the North Lobe.
	Backfill operation maynot be cost efficient or sustainable	CRF main component being rock and cement, a trade off study will be done to determine if another of binder could not be use to reduce operating cost. Pulling a control void in between the backfill block and the new block to be blasted could reduce all together the use of cement. Impact on ore lost and dilution will need to be done.
Tailing Management area	The TMA is now a mature operation. An experienced review Board monitors the operation. The Dams and foundations are highly instrumented. Amounts of processed ore	High levels of ongoing monitoring and review are required for this structure. It will be necessary to retain – in house – the key individuals to

Area	Risks Description and Potential Impact	Mitigation Approach
	from the UG operations are relatively small compared to that of the Open Pit.	maintain, monitor and build the TMA.
Other	The slope stability design of the EMRS assumed a rate of raise of about 9 m per year as this will allow sufficient time for the porewater pressures in the foundation to dissipate before more waste rock or overburden is placed. Depending on the mass flow from the open pit mine, there may be times when a faster rate of placement is required in certain areas.	This will require careful management of the mass flow to the EMRS as well as detailed stability analysis to verify faster placement can be accommodated in certain areas.
	Two portals are planned from the open pit. Design of the portals including any needed wider catch-benches directly above openings to install ground support/rock fall retainment is still required. There is a risk that the current pit design will need to be locally stepped in to accommodate future in-pit portals.	Detailed portal design and evaluation of the rock mass, structural geology, and predicted groundwater conditions are required. Establish a ground support and excavation strategy.
	Costs, metal prices and other variables change constantly. The Mine Operations team needs to further cultivate stability to manage the impact of external changes.	Mitigation measures include retention of key talent, better understanding of gold distribution in the deposit and reduction – where appropriate - of stockpiled materials.

25.8 Opportunities and Benefits

Opportunities Explanation	Benefits
Geomechanical detailed assessment; improve knowledge of ground conditions to validate and optimize design parameters, ground support and scheduling criteria.	Better estimation for dilution, recovery, support requirements, drilling pattern, infrastructure location, capital, and operating costs, etc.
Optimization of the number of trucks and LHDs for the UG; get a more accurate estimation of the number of trucks and LHDs required for the life of mine.	Ongoing UG fleet validation depending on ventilation and production centres availability will allow for an optimal amount of equipment for each phase of the project
Ventilation network optimization: should be reassessed in future phases to ensure optimal disposition, raises diameters and equipment selection.	Optimization to ensure sufficient airflow, while avoiding excessive pressure or airflow in airways.
Backfill process may benefit from a waste pass daylighting directly in the pit.	Will be considered to accelerate backfill material sent into the mine when the pit is finished; will reduce traffic in ramp and may improve backfill performance.
Numerous alternative mining methods such as 'Muckahi' (from Rhyolite Ressources) appear viable but require critical evaluation.	This is an UG mining transportation system which operates at a high angle – which reduces development costs - and offers potentially higher productivity with lower material handling cost.
All UG mining activities will require excellent geological control, through delineation / definition drilling. Lessons learned while mining "Intrepid' will be applied to the Main Zones'.	Definition drilling information will help to validate and reposition orebody for proper ore development and better stope design to optimize ore recovery. Several diamond drill bays are planned in various locations to provide proper drilling angle to the orebody.
High levels of cost control will be required to manage the UG mining areas.	Detailed knowledge of mining cost will enable the UG mining areas to be developed and sequenced in an 'optimal' manner.
Optimal sequencing of UG zones may rely on further trade-offs between long development excavations and shorter routes via portals.	Will depend on evolving development costs for long accessways vs. complexity, expense and inconvenience (to OP operations) of additional portals.
When OP activities cease, the milling capacity of the process plant will greatly exceed the ore tonnage from the UG mines. This creates an opportunity for 'toll milling' of materials from nearby properties.	This strategy is observed to be working well in British Columbia, simplifying many of the permitting arrangements (especially tailings) for smaller firms.
An extensive geotechnical instrumentation system has been installed in the foundations of the EMRS. In addition, records of refusal depths of the many wick drains have provided detailed spatial information about the base of the clay layers.	This information, together with ongoing piezometer readings provides a basis for engineering analysis that may indicate that the placement of mine rock can be accelerated in certain areas, while maintaining adequate stability.
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 somewhat reduce the storage capacity required for mine rock in the EMRS and WMRS. It will also reduce the haul distance and operating costs associated with the disposal of waste rock.
Geotechnical design sectors in the Northeast and Southeast continue to be well performing with good excavation performance.	Future slope design optimization maybe considered where geotechnical risks remain low.
An inpit dump is considered when North lobe will be fully excavated.	Haulage cost reduction with shorter haul from ODM main zone while mining towards pit bottomd

Opportunities Explanation	Benefits
The acceleration of Intrepid in comparison with the 2020 Tech report allow for the complete mining of Intrepid.	Mill all Intrepid while the stockpile is being process and benefit from reduce milling costs
Second inpit Portal is planned for March 2026	Possibility to pull into summer of 2025 depending on pit progression and productivity in the prior years
Due to the spare capacity in the downsized mill, toll milling of ore for third parties could be offered as a service.	Gain additional revenue tied in with the downsized mill (evaluation to confirm economic viability)
Ongoing improvement in mining practices	Improvement in identification and separation of DPO from LGO materials for improved segregation and sending to the mill increased quantities of better quality material preferentially.
Optimization of the open pit / underground transition level.	Increased overall mine economics by optimizing the combined open pit / underground econimics. This will result in the optimization of the open pit final limit design in addition to optimization of the underground mine design directly below the open pit.

26 RECOMMENDATIONS

26.1 Overall

The results presented herein demonstrate that the Rainy River Project is technically and economically viable. The main recommendation is linked to the current Block Model and the confidence in the geological and geotechnic knowledge for the current open pit, particularly the East Lobe, and also below the pit for the underground mining (mentioned several times in Items 25.7 & 25.8). Additional drilling needs to be conducted to confirm geological continuity and grade. Interpretation of the UG mine will also be extended from this new block model and based on smaller ore block and influence zone.

In addition to this, several trade-off studies are recommended, including the possibility for a third portal in the West lobe and the use of a backfill raise daylighting in the pit to optimize backfill operation.

Table 26.1 presents a summary of recommended tasks, detailed in the subsections that follow.

Table 26.1 – Recommended Tasks

Item	Item
26.1.1	Bench by Bench study
26.1.2	New Block Model (Interpretation of UG Mine)
26.1.3	Third Portal in West Lobe
26.1.4	Geotechnical Detailed Assessment
26.1.5	Ventilation Network Optimization
26.1.6	Feasibility for in-pit waste passes
26.1.7	Open Pit and Underground Mining Transition Optimization
26.1.8	Delineation Drilling to support UG operations planning
26.1.9	Reverse Circulation Drilling to support OP operations planning

26.1.1 Bench by bench study

During Q2 of 2022 a study will be performed on a bench-by-bench approach to understand and improve our mining practices. This includes a wide range of tasks but not limited to block model interpretation, blast pattern and sequencing, dig shape identification, equipment size per pit location, SOP reviews and operators training.

26.1.2 New block model (Interpretation of UG Mine)

A new block model will be created, and performance of this block model will be reviewed in light of previous year's production. Interpretation of the UG mine will also be extended from this new block model and based on smaller ore blocks and influence zone. Definition drilling information done in Intrepid during February and March of 2022 will be included.

Following the new block model for the UG mine, stope design will also be reviewed and re-interpreted with this new drilling.

26.1.3 Third portal in west lobe

The positioning of a third portal on the West side of the open pit will be analyzed to see if any upside can be achieved regarding the fast tracking the West zone. A long ventilation raise is planned in the NI 43-101 to allow for proper ventilation in the West end portion of the UG. The excavation of a third portal could alter this scenario by providing better air flow, another escape way and improve material movement.

26.1.4 Geotechnical detailed assessment

In parallel with the new block model, a full geotechnical analysis needs to be conducted to better understand the different parameters used in the UG project. To validate ground conditions, dilution, and recovery factors (and other parameters such as efficiencies of equipment, drilling patterns and location of major infrastructures), a full geomechanical program is to be established. This can be realized concurrently with the proposed geological drilling. This work should also consider extension of the brittle-structural geology model to below the open pit.

The stability and design of the portals that are planned from within the open pit require detailed assessment. This includes the local bench configuration requirements adjacent to portals, ground support designs, and interaction with the underground development plan.

Similarly, 3D stability analyses for the southwest wall are recommended to confirm the Phase 4 design conditions. This will incorporate monitoring data from new VWP installations that are planned in 2022. This work will support the long-term access considerations for the open pit and the portals.

26.1.5 Ventilation network optimization

Ventilation design could currently be optimized; some of the assumptions and ventilation phases could pose an operational risk if ventilation installations are not as efficient as theorized. Raise dimensions, air velocity, number of fans versus number of booster fans are all aspects that need to be reviewed and validated. Additional options and solutions can be looked at to ensure the feasibility of the ventilation design. For example, a third portal could significantly reduce the load on the ventilation network.

26.1.6 Feasibility for in-pit waste passes

To improve backfill performance and to reduce traffic in the main ramps, the option of a network of waste pass needs to be evaluated to validate the feasibility.

26.1.7 Open pit and underground transition optimization

The transition between open pit mining and underground mining could currently be optimized to maximize the economics of the deposit. Although the current transition depth is supported by highl-level analysis, no detailed analysis has been completed to-date.

26.1.8 Delineation Drilling to support UG operations planning

To support underground operations and planning, delineation drilling will be required.

26.1.9 Reverse Circulation Drilling to support OP operations planning

To support open pit operations and planning, reverse circulation drilling will be required.

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