O'Brien Gold Project
NI 43-101 Technical Report & Preliminary Economic Assessment
Quebec, Canada

Effective Date: June 27, 2025

Prepared for:
Radisson Mining Resources Inc.
50 Rue du Petit-Canada
Rouyn-Noranda, QC, J0Y 1C0

Prepared by:
Ausenco Engineering Canada ULC
11 King St. West, 15th Floor
Toronto, Ontario, M5H 4C7

List of Competent Persons:
Renee Barrette, P.Eng., ing., Ausenco Engineering Canada ULC
Luke Evans, M.Sc., P.Eng., ing., SLR Consulting (Canada) Ltd.
Hugo Latulippe, P.Eng., ing., BBA Inc.
Marc R. Beauvais, P.Eng., ing., Norda Stelo Inc.

CERTIFICATE OF QUALIFIED PERSON

Renee Barrette, ing.

I, Renee Barrette, ing., being employed as a Principal Metallurgist with Ausenco Engineering Canada ULC (Ausenco), with an office address of Suite 1550 - 11 King St West, Toronto, ON, M5H 4C7, do certify that:

1. This certificate applies to the technical report titled "O'Brien Gold Project NI 43-101 Technical Report & Preliminary Economic Assessment in Quebec, Canada," with an Effective Date of June 27, 2025 (the "Technical Report").
2. I graduated from Laurentian University in 2001 with a Bachelor of Applied Science degree in Extractive Metallurgical Engineering.
3. I am a professional engineer registered with the OIQ (No. 6019759).
4. I have practiced my profession for 24 years. I have been directly involved in the development, design, operation, and commissioning of mineral processing plants, focusing on Gold, Base Metals and other PGM projects, both domestic and internationally. To name a few specific examples, I have completed a due diligence review of a gold deposit with 4.1M Gold reserves near Val-d'Or, Quebec and completed design reviews to commissioning on a 17,000 mtpd Base Metals Project at a Sudbury, Ontario.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the project site on October 7, 2024.
7. I am responsible for Sections 1.1, 1.12, 1.15, 1.16, 1.17, 1.19, 1.20, 1.22, 2.1, 2.2, 2.3, 2.4.1, 2.5, 2.6.2 2.7, 2.8, 3.1, 3.2, 13, 17, 18.1 to 18.3.2, 18.3.3, 18.6, 18.8, 18.10, 19, 21.1, 21.2.1, 21.2.2, 21.2.3, 21.2.5, 21.2.6, 21.2.7.1.1, 21.2.7.2, 21.2.8 to 21.2.10, 21.3.1, 21.3.3 to 21.3.6, 22, 24, 25.1, 25.7, 25.9, 25.10, 25.13 to 25.14, 25.15, 25.16.1.2 to 25.16.1.4, 25.16.2 to 25.16.5, 26.1, 26.4, 26.5, and coauthor on 27 of the Technical Report.
8. I am independent of Radisson Mining Resources (Radisson) as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the O'Brien Gold Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: August 20, 2025

/signed/

Renee Barrette, ing.

CERTIFICATE OF QUALIFIED PERSON
Luke Evans, M.Sc., P.Eng., ing.

I, Luke Evans, M.Sc., P.Eng., ing., being employed as a Global Technical Director – Geology Group Leader, and Principal Geologist with SLR Consulting (Canada) Ltd, of Suite 501, 55 University Ave., Toronto, ON, M5J 2H7, do certify that:

1. This certificate applies to the technical report titled "O'Brien Gold Project NI 43-101 Technical Report & Preliminary Economic Assessment in Quebec, Canada," with an Effective Date of June 27, 2025 (the "Technical Report").
2. I am a graduate of University of Toronto, Ontario, Canada, in 1983 with a Bachelor of Science (Applied) degree in Geological Engineering and Queen's University, Kingston, Ontario, Canada, in 1986 with a Master of Science degree in Mineral Exploration.
3. I am registered as a Professional Engineer in the Province of Ontario (Reg. #90345885) and as a Professional Engineer in the Province of Quebec (Reg. # 105567). I have worked as a professional geologist for a total of 42 years since my graduation. My relevant experience for the purpose of the Technical Report is:
   - Geological engineer specializing in resource and reserve estimates, audits, technical assistance, and training since 1995.
   - Review and report as a consultant on numerous exploration and mining projects around the world for due diligence and regulatory requirements.
   - Senior project geologist in charge of exploration programs at several gold and base metal mines in Quebec.
   - Project geologist at a gold mine in Quebec in charge of exploration and definition drilling.
   - Project geologist in charge of sampling and mapping programs at gold and base metal properties in Ontario, Canada.
4. I have read the definition of "qualified person" set out in National Instrument 43-101 (NI 43-101) and certify that by reason of my education, affiliation with a professional association (as defined in NI 43-101) and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of NI 43-101.
5. I visited the O'Brien Project on October 12, 2022. I previously visited the O'Brien Project on April 30, 1998, for Roscoe Postle Associates Inc. (RPA) as well as September 27, 2006, for Scott Wilson RPA Inc.
6. I am responsible for Sections 1.2 to 1.11, 1.13, 1.21, 1.22, 2.3, 2.4.2, 2.6.1, 3.1, 4 to 12 except 4.5.2, 14, 23, 25.2 to 25.6, 26.1, 26.2, and coauthor on 27 of the Technical Report.
7. I am independent of Radisson Mining Resources (Radisson) as independence is defined in Section 1.5 of NI 43-101.
8. I was the author of a technical report for the O'Brien Project in 1998, 2007, 2013, and 2023
9. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: August 20, 2025

/signed/

Luke Evans, M.Sc., P.Eng., ing.

CERTIFICATE OF QUALIFIED PERSON
Hugo Latulippe, P.Eng., B.Sc.A.

I, Hugo Latulippe, P.Eng., being employed as a Principal Mining Engineer with BBA Inc., with an office address of 990 de l'Église Road, Office 590, Québec, QC, G1V 3V7, do certify that:

1. This certificate applies to the technical report titled "O'Brien Gold Project NI 43-101 Technical Report & Preliminary Economic Assessment in Quebec, Canada," with an Effective Date of June 27, 2025 (the "Technical Report").
2. I graduated from Laval University in 2001 with a Bachelor degree in Mining and Mineralogy Engineering.
3. I am a professional engineer registered with the Ordre des ingénieurs du Québec (No. 126558), with the Professional Engineers Ontario (No. 100520994), and Engineers and Geoscientists British Columbia (No. 209460).
4. I have practiced my profession for 24 years and I began as a mining engineer in underground mines in Abitibi and then worked in open pit operations in James Bay and New-Caledonia. I acquired solid experience in mining operations before working on the development of three projects. I have been involved in mining studies since 2012.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I have not visited the project site.
7. I am responsible for Sections 1.16, 1.18, 1.16, 2.3, 3.1, 4.5.2, 18.3.3, 18.9, 20, 21.2.2, 21.2.3, 21.2.7.1.1, 21.2.7.1.2, 25.11, 25.12, 25.13, 25.16.1.5, 26.6, and coauthor on 27 of the Technical Report.
8. I am independent of Radisson Mining Resources (Radisson) as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the O'Brien Gold Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: August 20, 2025

/signed/

Hugo Latulippe, P.Eng., B.Sc.A

CERTIFICATE OF QUALIFIED PERSON

Marc R. Beauvais, P.Eng.

I, Marc R. Beauvais, P.Eng., being employed as a senior mining engineer with Norda Stelo Inc., an Engineering Consulting Firm, 560, 3e Avenue, Val-d'Or, Québec, Canada, J9P 1S4, do certify that:

1. This certificate applies to the technical report titled "O'Brien Gold Project NI 43-101 Technical Report & Preliminary Economic Assessment in Quebec, Canada," with an Effective Date of June 27, 2025 (the "Technical Report").
2. I graduated from Laval University in 1991 with a B.Sc. in Mining Engineering.
3. I am a professional engineer registered with the Ordre des ingénieurs du Québec (No. 126558), with the Professional Engineers Ontario (No. 100520994), and Engineers and Geoscientists British Columbia (No. 209460)..
4. I have practiced my profession for 34 years with experience in mining operation, construction and management. I have experience in gold, base metals and diamonds. I founded and operated my own consulting firm (Promine Consultant Inc.) from 2001 to 2005. I have been a Business Associate of Genivar Inc. from 2005 to 2009. I have been assigned to various projects owned by foreign mining companies in Azerbaijan, Colombia, Peru, Philippines, Kazakhstan, and Tanzania between 1999 to 2010. In 2012, I founded and managed Minrail Inc, which developed a patented, fully integrated mining system designed specifically to extract the mineralized material from shallow-dipping deposits in underground mines. I have multiple specializations in computer modelling, mine planning and construction.
5. I have read the definition of "Qualified Person" set out in the National Instrument 43-101 Standards of Disclosure for Mineral Projects ("NI 43-101") and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfill the requirements to be a "Qualified Person" for those sections of the Technical Report that I am responsible for preparing.
6. I visited the project site on September 23, 2024 for 1 day.
7. I am responsible for Sections 1.14, 1.16, 1.22, 2.3, 2.4.3, 3.1, 15, 16, 18.3.3, 18.4, 18.5, 18.7, 21.2.2, 21.2.4, 21.2.10.1, 21.2.11, 21.3.1, 21.3.2, 25.8, 25.13, 25.14, 25.16.1.1, 25.16.2.1, 26.1, 26.3, and coauthor on 27 of the Technical Report.
8. I am independent of Radisson Mining Resources (Radisson) as independence is defined in Section 1.5 of NI 43-101.
9. I have not been previously involved with the O'Brien Gold Project.
10. I have read NI 43-101 and the sections of the Technical Report for which I am responsible have been prepared in compliance with that Instrument. As of the effective date of the Technical Report, to the best of my knowledge, information and belief, the sections of the Technical Report for which I am responsible contain all scientific and technical information that is required to be disclosed to make those sections of the Technical Report not misleading.

Dated: August 20, 2025

/signed/

Marc R. Beauvais, P.Eng.

Disclaimer

This report was prepared as a National Instrument 43-101 Technical Report for Radisson Mining Resources Inc. (Radisson) by Ausenco Engineering Canada ULC (Ausenco), SLR Consulting Ltd. (SLR), BBA Inc. (BBA), and Norda Stelo Inc. (Norda Stelo), collectively the Report Authors. The quality of information, conclusions, and estimates contained herein is consistent with the level of effort involved in the Report Authors' services, based on (i) information available at the time of preparation, (ii) data supplied by outside sources, and (iii) the assumptions, conditions, and qualifications set forth in this report. This report is intended for use by Radisson subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposed legislated under Canadian provincial and territorial securities law, any other uses of this report by any third party are at that party's sole risk.

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1 SUMMARY

1.1 Introduction

Radisson Mining Resources (Radisson) commissioned Ausenco Engineering Canada ULC (Ausenco) to compile a preliminary economic assessment (PEA) of the O'Brien Gold Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design for metallurgy and recovery methods, a cost estimate for process plant modifications related to toll milling, and mine site electrical infrastructure. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- Norda Stelo Inc. (Norda Stelo), designed the underground mining methods, surface infrastructure related to waste rock storage and mineralized material storage, and prepared the mine capital and operating costs.
- SLR Consulting (Canada) Ltd. (SLR) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, and mineral resource estimate.
- BBA Inc. (BBA) completed the work related to mine site water management and environmental, permitting, and social considerations.

1.2 Property Description and Location

The O'Brien Gold Project is in the Abitibi region in northwestern Québec, approximately 1 km north of the town of Cadillac. Gravel roads provide access to the project from provincial Highway 117. It is approximately 50 km east of the town of Rouyn-Noranda, Québec; 30 km by road west of the town of Malartic, and 55 km by road west of the town of Val d'Or.

1.3 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The O'Brien Gold Project consists of a contiguous block of 146 exploration claims that cover an area of 7,137.74 hectares (ha) and one mining concession that covers an area of 288.19 ha. Radisson has a 100% interest in the project, which is comprised of three former properties named O'Brien, Kewagama, and Thompson-Cadillac (previously called New Alger).

Following are the details of royalties held by third parties on the O'Brien Gold Project:

- O'Brien: $1 million cash payment in the event of commercial production

- Kewagama: 2% net smelter return (NSR) royalty
- Thompson-Cadillac/New Alger:
    - 3% NSR on certain mining claims replacing the old mining concession known as CM240-PTA
    - 2% NSR on certain mining claims in the southern portion of the property
    - 1% NSR on certain mining claims in the southern portion of the property

1.4 Accessibility, Climate, Local Resources, Infrastructure and Physiography

On the O'Brien property, several historical mines (O'Brien, Thompson-Cadillac, and Kewagama) were in intermittent operation between 1925 and 1981. Most of the surface infrastructure has been dismantled; only the garage and the mill building of the O'Brien mine have been preserved and are used today for Radisson's exploration activities.

Radisson also has a core shack with an adjoining core sawing room and exploration offices on the O'Brien mine site. An orphan tailings storage facility with a footprint of 4 ha and a polishing basin are located directly north of the old mill.

1.5 History

The first claims for the project were staked in 1924 by the O'Brien Company Ltd, which later became O'Brien Gold Mines Ltd. O'Brien Gold Mines Ltd. began mining the property in 1925 with the sinking of the first shaft and commencement of underground development. Continuous exploration facilitated resource growth and a 90 ton per day amalgamation mill was constructed in 1933 to support operations. Between the start of mining and 1939, roasting and cyaniding facilities were added, and the capacity of the mills was increased to 150 short tons per day. The O'Brien mine sold crude arsenic from 1940 to 1950 to Deloro Smelting and Refining. With rising costs eroding profits and reserves declining, the mine closed in 1956.

Abandoned since 1956, the O'Brien mine was acquired, invested in, and/or sold by several companies including Darius Gold Mines (1969), Goldfield Mining Consolidated (1977), and Sulpetro Minerals (1981), all of whom conducted exploration and reassessment activities, and in some cases, development, extraction, and construction activities, including the building of a 200 ton per day capacity mill by Sulpetro Minerals. Following this intermittent activity, the O'Brien mine was shut down permanently in 1981 and allowed to flood in 1985.

During that period, intermittent exploration and development activities were occurring at the adjacent Kewagama and Thompson-Cadillac mines. These historical mines are located on the project claim area and were less extensively explored, developed, and mined than the O'Brien mine.

In 1986, Sulpetro reorganized into Novamin, and began a drilling and geophysical exploration program. In 1989, Breakwater acquired Novamin, continued exploration drilling at the O'Brien mine property, and released a preliminary resource estimate.

In 1992, Breakwater and Radisson began negotiations, and in 1994, signed a deal whereby Radisson could earn a 50% interest in the property. Through to 1998, Radisson completed exploration work, whereupon it purchased 100% of the rights to the property as well as the existing infrastructure.

1.6 Geology and Mineralization

The O'Brien Gold Project is located along the Cadillac-Larder Lake Fault Zone (CLLFZ), in the southeastern part of the Cadillac Mining Camp (CMC), Québec. Approximately 40 gold deposits, which have produced over 60 million ounces (Moz) of gold since the early 20th century, are associated with this major structure and its subsidiary faults.

The CMC covers a 25 km long stretch of the CLLFZ, from the former Mouska mine in the west to the former Lapa-Cadillac mine to the east. Within the CMC, the CLLFZ runs along an east-west axis and separates the Pontiac metasedimentary subprovince to the south from the Abitibi volcano-sedimentary subprovince to the north. The CMC is underlain by rocks of the Southern Volcanic Zone of the Abitibi subprovince, which are intruded by Proterozoic diabase dykes.

The project straddles the Piché Group volcanic rocks and CLLFZ. The Piché Group is a relatively thin band of interlayered mafic volcanic rocks, conglomerates, and porphyric andesitic sills. With a few significant exceptions like the Vintage Zone hosted in the Cadillac Group, the Piché Group hosts most of the gold mineralisation occurrences on the property.

Gold production at the historical O'Brien mine came from a few quartz veins, mostly hosted by the O'Brien mine conglomerate and the northern porphyric andesitic sill. Approximately 95% of the O'Brien mineralized material came from four veins (No. 1, No. 4, No. 9, and "F") in the eastern part of the mine. The veins contained high-grade shoots that occasionally yielded considerable amounts of visible gold.

Mineralization is currently defined within two areas of the project site: (1) O'Brien East, which is host to Zone 36 East located east of the historical O'Brien mine, and Kewagama (at-depth extension of the historical Kewagama mine), and (2) O'Brien West, which is the extension of the historical Thompson-Cadillac mine.

Within the Zone 36 East area, host to the majority of the current mineral resources at the project, and Thompson-Cadillac, the main mineralized structures ("veins") are generally narrow, ranging in true thickness from several centimetres up to 7 m, and are similar in character to what was mined at O'Brien in that mineralization is hosted in narrow, near vertical, high-grade shoots within more laterally extensive quartz veins oriented sub-parallel to lithology. Gold-bearing veins occur in different lithologies of the Piché Group, the Pontiac Group, and the Cadillac Group.

In the Kewagama area, the gold mineralization occurs in rocks of the Piché Group, but present as a series of smaller veins instead of bigger single vein as within the O'Brien deposit.

1.7 Deposit Types

The O'Brien deposit is a greenstone-hosted quartz-carbonate vein deposit with valuable amounts of gold. Greenstone-hosted quartz-carbonate vein deposits are a subtype of lode gold deposits and are also known as mesothermal and orogenic gold deposits.

1.8 Exploration

Since acquisition, Radisson has continued exploration work on the property with drilling, trenching, and geophysical programs, and through advancing the resource with multiple mineral resource estimates over the continuing years.

In 2022, Radisson embarked on an extensive exploration campaign on the southern New Alger area (south of Highway 117) of the O'Brien Mine property, focusing in the Pontiac Group sedimentary units. The work was separated into three distinct phases and was carried out between January and October 2022. The work consisted of a compilation/planning phase, a prospecting and sampling phase, and a trenching phase.

In 2024, Radisson began drilling deep holes beneath the O'Brien mine and has succeeded in intercepting mineralization. Additional drilling is warranted.

1.9 Drilling

A summary of drilling at the project is presented in Table 1-1. From the 1930s to 2025, approximately 1,207 diamond drill holes totalling 374,461.8 m have been drilled.

Table 1-1: Summary of Validated Drill Holes

Mine	Time Period	No. of Holes	Length (m)
O'Brien	Historical (1930s – 1993)	226	34,882
	1994 – 2025	584	273,016.6
Thompson- Cadillac	Historical (1930s – 1993)	64	6,619
	1994 – 2022	70	15,845
Kewagama	Historical (1930s – 1993)	203	16,253
	1994 – 2025	60	27,846.2
Total	**-**	**1,207**	**374,461.8**

Source: Radisson Leapfrog project.

1.10 Sampling Preparation and Security

Before 1995, the O'Brien mine used its internal laboratory for assaying. Between 1995 and 2025, a number of laboratories were used, such as Chimitec Ltd. (Chimitec) in Val d'Or, Québec; XRAL Laboratories (XRAL) in Rouyn-Noranda, Québec; Techni-Lab Inc. (Techni-Lab) in Ste-Germaine, Québec; Laboratoire Expert Inc. (Laboratoire Expert) in Rouyn-Noranda, Québec; Swastika Laboratories Ltd. (Swastika) in Swastika, Ontario; ALS Minerals (ALS) in Val d'Or, Québec; and SGS Canada Inc. (SGS) in Val d'Or, Québec.

Except for the internal O'Brien mine laboratory, all the laboratories are independent of Radisson. Commercial laboratories Swastika, ALS, and SGS are accredited to the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 9001:2008 standards for quality management and to ISO/IEC 17025:2005 for all relevant procedures. Accreditation of the other laboratories is unknown.

Sample preparation and analysis procedures have remained consistent over time, despite changes to the primary laboratory employed. Laboratories have generally employed a standard approach whereby samples are crushed and pulverized prior to gold analysis by fire assay (with AAS or gravimetric finish), with or without follow-up of metallic screen gold analysis on selected high-grade samples.

In the QP's opinion, the sample preparation, analytical protocols, QA/QC, and security procedures are acceptable for the purposes of mineral resource estimation.

1.11 Data Verification

The QP visited the property on October 12, 2022. While on site, the QP held discussions with site personnel and inspected selected core intercepts from several drill holes and compared them against recorded lithological logging and assay results. In addition, data collection and QA/QC procedures were reviewed.

The QP regards the geological and mineralization interpretations used to support mineral resource estimation consistent with the drill core, and that the Radisson geologists to have a good understanding of the geology and mineralization.

The QP reviewed the drill hole database for the project in Leapfrog software, and conducted a standard review of import errors and visual checks. The QP noted a discrepancy between available underground survey information and expected underground collar locations. Radisson was able to subsequently verify and update the underground working locations to bring the drill collars and workings into agreement. No other significant errors were discovered in the database.

A program of database verification was carried out by means of spot checking a random selection of drill holes from a subset of the master drill hole database representing drill holes which intersected the gold mineralization wireframes. Verification activities primarily focused on comparison of assay values contained within the digital database against those values contained within the certificates reported from the assaying laboratories. Comparison of drill hole collar elevations with the topographical surface, as well as available underground survey information, was also completed.

In addition, 22 drill holes that intersected the mineralization wireframes were selected for validation, representing approximately 9% of the drill holes drilled between 2019 and 2022. No discrepancies were found.

The QP is of the opinion that database verification procedures for the O'Brien Gold Project comply with industry standards and are adequate for the purposes of mineral resource estimation.

1.12 Mineral Processing and Metallurgical Testwork

Radisson engaged SGS and Ausenco in 2024 to carry out a testwork program to determine the optimum flowsheet and to develop process design criteria for the potential processing of O'Brien deposit material at the IAMGOLD Westwood complex.

Between 2017 and 2019 there were multiple testwork programs carried out on the property on behalf of Radisson by Dundee Sustainable Technologies, Centre Technologique des Résidue Industriels, and SGS. These testwork programs reviewed multiple gold recovery methods with recoveries ranging from 63% to 94%.

The 2024 testwork program was conducted in two phases to determine the optimum flowsheet, and then to optimize recoveries and processing costs. The selected process flowsheet is based on the latest testwork results and was determined to be gravity concentration, flotation, regrind of flotation concentrate, and cyanidation of flotation concentrate and flotation tails to extract gold. An average gold recovery of 90% was achieved with an 82 µm primary grind size (P_{80}) through the selected process.

1.13 Mineral Resource Estimate

A mineral resource estimate for the O'Brien Gold Project, effective March 2, 2023, was developed for the Zone 36 East, Kewagama, and New Alger (now Thompson-Cadillac) deposits and was prepared by SLR using available drill hole sample data as of January 24, 2023. The March 2023 mineral resource estimate was based on 1,079 drill hole collars representing 325,509 m of drilling, as well as 120,352 assay samples, and was prepared in accordance with CIM (2014) definitions. The 2025 mineral resource estimate is based on reporting the March 2023 block model using a new cut-off grade developed from new metal price and cost assumptions.

A summary of the updated mineral resources, effective May 6, 2025, for the O'Brien Gold Project is presented in Table 1-2. Indicated mineral resources are estimated to total 2.20 million tonnes (Mt) at a grade of 8.22 g/t Au, containing 582 thousand ounces (koz) Au. Inferred mineral resources are estimated to total 6.67 Mt at a grade of 4.35 g/t Au, containing 932 koz Au.

Table 1-2: O'Brien Project Mineral Resource Estimate – May 6, 2025

Class	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	2,204	8.22	582
Inferred	6,671	4.35	932

Notes: **1.** CIM (2014) definitions were followed for mineral resources. **2.** Mineral resources are reported above a cut-off grade of 2.2 g/t Au based on a C$172.5/t operating cost. **3.** Mineral resources are estimated using a long-term gold price of US$2,000/oz Au, a US$/C$ exchange rate of 1:1.33, and a metallurgical recovery of 90%. **4.** Wireframes were modelled at a minimum width of 1.2 m. **5.** Bulk density varies by deposit and lithology and ranges from 2.00 to 2.82 t/m^3. **6.** Mineral resources that are not mineral reserves do not have demonstrated economic viability. **7.** Numbers may not add due to rounding.

Wireframes representing vein structures and incorporating a minimum thickness of 1.2 m were prepared in Leapfrog Geo software by Radisson geologists and reviewed and adopted by the QP. Block model estimates were completed by SLR using Leapfrog Edge software using full-length capped composites, and a multi-pass, inverse distance cubed (ID^3), interpolation approach designed to capture the narrow, subvertical mineralization shoots within the vein structures.

Blocks were classified using a novel automated approach which considered local drill hole spacing, composite density, and block-grade continuity. Indicated mineral resources were defined where there were contiguous blocks above 1.0 g/t Au, when the contiguous block group contained two or more economic composites and an overall drill hole spacing of 50 m or less was achieved. All other estimated blocks within wireframes were classified as inferred resources. Final block classification groupings were reviewed, and manual adjustments were made to ensure cohesive classification shapes.

All blocks above the cut-off-grade of 2.2 g/t Au have been included within the mineral resource estimate, and existing mine workings have been excluded from the O'Brien mineral resource estimate. Underground constraining shapes were not used to report the mineral resource, but the full width compositing, minimum thickness application to wireframe building, and classification approaches taken in tandem have ensured that there is no selective reporting bias and that the criteria have been met for the mineral resources meeting reasonable prospects for eventual economic extraction (RPEEE) in an underground mining scenario.

Wireframe and block model validation procedures—including confirmation of wireframe to block volume; statistical comparisons with block model and nearest neighbour (NN) estimates; swath plots; and visual reviews in 3D, longitudinal, cross-section, and plan views—were completed.

Drilling from January 2023 to May 2025 within the O'Brien area has validated the 2023 mineral resource and will not significantly impact the global numbers for that footprint. New drilling outside the O'Brien mineral resources, particularly at depth, appears promising for additional mineral resources, although further drilling is required before estimating mineral resources at depth.

The only modifications to the mineral resource model between 2023 and 2025 were a lower cut-off grade, reflecting updated cost and metal price assumptions as outlined below, and a reduction in the vertical block size from 10 to 5 metres to enhance flexibility in mine design.

1.14 Mining Methods

The O'Brien Gold Project is designed as an underground mining operation targeting subvertical, narrow, mineralized veins through the application of longitudinal long-hole retreat mining. This method was selected based on the geometry and continuity of the deposit, offering an efficient and safe extraction approach while minimizing dilution. The project comprises two main zones—O'Brien East and Kewagama—which have their own portals, declines, and ventilation networks, but are interconnected by an underground drift to optimize material handling and logistics.

Access to the mineralized zones is provided via ramps with level spacing set at 25 meters. To support stope stability, cemented rockfill will be used in strategic areas, with a higher cement content above sill pillars.

The mining plan is underpinned by geotechnical studies conducted by Golder Associates Ltd. (now WSP Global Inc.) in 2020 and 2021, which established four geomechanical domains and provided the input parameters for stope and crown pillar design. Mine infrastructure includes a centralized crusher and vertical conveyor system located in the O'Brien East zone that is designed to transport up to 1,400 tonnes of mineralized material per day from Level 300 to the surface. Prior to commissioning the conveyor, a fleet of 20-tonne trucks will be used to haul material to the surface.

Dewatering is handled through a system of secondary sump pumps and a main GEHO pump, with a total design capacity of 2,814 m^3/day. Ventilation has been designed to operate independently in each zone, using a network of fresh air raises and surface-mounted main fans equipped with silencers and variable frequency drives. The system provides up to 498 kcfm for O'Brien East and 273 kcfm for Kewagama.

Mine services include a 25 kV underground electrical distribution network, a long-term evolution (LTE) communication system with fibre-optic backbone, and compressed air supplied via surface compressors.

Commercial production is scheduled to begin at the start of the second quarter of Year 2 following a 15-month pre-production period during which development and limited stoping will supply 197,000 tonnes of mineralized material at a grade of 3.49 g/t Au. Commercial production is expected to begin once 70% of the average steady-state output is reached. Over commercial production, average annual gold production is forecast at approximately 65,000 ounces.

1.15 Recovery Methods

The potential for processing mineralized material from O'Brien at a local existing process plant was considered. In September 2024, Radisson and IAMGOLD entered into a Memorandum of Understanding (MOU) titled "Memorandum of Understanding Between Radisson Mining Resources Inc. and IAMGOLD Corporation" (September 6, 2024) to assess the design criteria for processing mined material at the Doyon mill, part of IAMGOLD's Westwood complex (collectively "Westwood" or "Westwood complex"). In this report, the Westwood carbon-in-pulp mill is the location where gold will be recovered from feed material and processed into doré. The process flowsheet was therefore developed based on the existing process plant facilities at the Westwood complex with some limited modifications to increase project economics.

As described in Section 13, a testwork program accompanying the preliminary economic analysis indicated that a gravity/flotation/carbon-in-leach/carbon-in-pulp circuit would be most suitable for project economics, based on a design gold grade of 4.9 to 6.9 g/t, with an average overall gold recovery of 87.5%. The selected flowsheet would require utilizing the existing gravity circuit at the Westwood complex and modifying the process plant by adding a flotation and regrind circuit. The other equipment required to process the feed already exists and is part of Westwood's current operation.

Process plant production at Westwood was calculated at 3,000 t/d based on a toll milling campaign basis, where the facility alternates between processing O'Brien project feed and IAMGOLD's Westwood feed.

1.16 Project Infrastructure

The mine site infrastructure required at O'Brien includes civil infrastructure, site facilities/buildings, water management infrastructure, waste rock and process feed material storage facilities, and electrical services. Buildings located at the mine site will include the following:

- mine dry and administration office
- cemented rockfill building
- water treatment building
- mined material temporary stockpile building.

Site facilities have been located based on proximity to the portal locations. A site layout plan is provided in Figure 1-1.

Figure 1-1: Infrastructure Layout Plan

Source: Ausenco (2025).

1.17 Market Studies and Contracts

This PEA establishes criteria for the development of O'Brien based on processing and tailings management at an existing off-site facility, namely IAMGOLD's Westwood complex, under a toll milling arrangement. A milling assessment for the processing of O'Brien mined material at the Westwood complex was conducted under the auspices of a September 2024 MOU between Radisson and IAMGOLD. The MOU is non-binding and non-exclusive and contains no specific terms around potential commercial arrangements between the parties. IAMGOLD has not independently confirmed the processing assumptions, metallurgical results and/or cost assumptions assumed in this study.

This PEA assumes that doré bars will be produced. The market for doré is well-established and accessible to new producers. The doré bars will be refined in a certified North American refinery and the gold will be sold on the spot market.

No market studies have been conducted by Radisson or its consultants on the doré that will be produced for the O'Brien Gold Project. Gold is a freely traded commodity on the world market and there is a steady demand from numerous buyers. Terms and conditions included as part of the sales contracts are expected to be typical for this commodity. Gold is bought and sold on many markets, and it is not difficult to obtain a market price at any time. The gold market is liquid, with many buyers and sellers active at any given time.

1.18 Environmental, Permitting and Social Considerations

This section summarizes the baseline environmental conditions of the O'Brien Gold Project area, based on existing inventories, public databases, and baseline studies commissioned by Radisson and conducted by SNC Lavalin in 2019. Since mineral processing will occur off-site, no tailings or processing infrastructure will be located at the project site.

1.18.1 Environmental Considerations

Regional conditions for air quality are influenced by nearby industrial activity (e.g., the LaRonde Mine), and a site-specific monitoring program is currently underway. The site is located approximately 400 m from the nearest residence in a rural zone. A noise modelling study will be conducted as part of the project development phases. The project lies within the Blake River watershed, and local drainage patterns have been characterized.

There are no protected areas on site; nearby features include heronries and biological refuges. Twenty-nine wetlands (90.3 ha) were identified, most of which are considered to have medium or low ecological value. No species at risk were observed in wetlands or recorded within 15 km of the site according to the CDPNQ database. However, regional occurrences of species such as the bank swallow and smooth green snake have been documented.

Further environmental and social studies will be completed to support the forthcoming impact assessment process.

1.18.2 Closure and Reclamation Considerations

Closure and rehabilitation planning will be conducted in collaboration with stakeholders. Key closure measures will include:

- plugging underground infrastructure (ramps, raises, conveyor)
- revegetating the project footprint, waste rock pile, and stockpile areas
- demolishing and removing all surface buildings and infrastructure
- managing dismantling materials according to reduction, reuse, recycling, and disposal principles
- conducting land characterization to identify and remediate contamination, as per applicable environmental regulations
- scarifying access roads and restoring natural drainage
- dismantling industrial water treatment systems when no longer required
- breaching water management ponds and revegetating those areas
- restoring natural hydrology where appropriate.

An environmental monitoring program will be implemented to confirm the success of the reclamation activities.

1.18.3 Permitting Considerations

The project is subject to Quebec's *Environment Quality Act* ("EQA," c. Q-2), which states that all mining developments, including the additions to, and alterations or modifications of, existing mining developments are automatically subject to assessment and review. The O'Brien Gold Project will therefore be subject to the same provincial process.

It should not be subject to the federal impact assessment examination procedure (*Impact Assessment Act* (S.C. 2019, c. 28, s. 1), Physical Activities Regulations (SOR/2019-285). However, the project will be subject to the Metal and Diamond Mining Effluent Regulations (*Fisheries Act*).

1.18.4 Social Considerations

The O'Brien Gold Project is located in the northwestern portion of the Abitibi-Témiscamingue region within the jurisdiction of the City of Rouyn-Noranda. The project lies just over 1 km from the city of Cadillac's urban perimeter.

The project is on the traditional territory of the Abitibiwinni First Nation (Pikogan), located ~45 km northeast. The community is governed by the Algonquin Anishinabeg Nation Tribal Council and participates in the Pikogan Agreement, which outlines consultation and accommodation procedures.

The project is located ~75 km north of Anishinabeg First Nation (Long Point) situated on the unceded territory of Anishnabe Aki. This Algonquin community comprises approximately 800 members, with about half residing in Winneway, located in the Abitibi-Témiscamingue region of Western Quebec. The Long Point First Nation Council asserts ancestral land claims in the Preissac area.

While Indigenous communities continue to use the land for traditional purposes (hunting, trapping, fishing), no traditional traplines have been identified within the project boundaries.

All mining claims associated with the project are located on public land.

The area surrounding Preissac Lake (~3.5 km from site) is a popular recreational area with outfitters, campgrounds, and snowmobile/ATV trail networks. The Trans-Québec snowmobile Trail 83 crosses the Radisson property.

As per the Pikogan Agreement, mining activities within the project territory require consultation with the Abitibiwinni First Nation.

Radisson will initiate a consultation process involving key stakeholders including Indigenous communities, local governments, land users, and businesses as next steps in the project development.

1.19 Capital and Operating Cost

1.19.1 Capital Cost Estimate

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q1 2025 Canadian dollars based on Ausenco's in-house database of projects and studies, as well as experience from similar operations.

The capital cost estimate includes underground mining, surface infrastructure at the O'Brien site, modifications at the Westwood process plant, project indirect costs, project delivery, Owner's costs, and contingency. The capital cost summary is presented in Table 1-3. The total initial capital cost for the O'Brien Gold Project is C$174.8 million; and life-of-mine sustaining costs are C$172.7 million. O'Brien mine closure costs are estimated at C$5.3 million, and salvage credits are C$3.1 million. Note: closure costs and salvage credits are not included in Table 1-3.

Table 1-3: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
1000	Mining	93.2	168.7	261.9
3000	Process Plant	20.5	-	20.5
5000	On-Site Infrastructure	15.7	-	15.7
6000	Off-Site Infrastructure	8.2	-	8.2
	Total Directs	**137.7**	**168.7**	**306.7**
7000	Project Indirects and Delivery	13.7	-	13.7
8000	Owner's Cost	3.9	-	3.9
	Total Indirects	**17.6**	-	**17.6**
9000	Contingency	19.6	4.0	23.6
	Contingency	**19.6**	**4.0**	**23.6**
	Project Totals	**174.8**	**172.7**	**347.5**

Source: Ausenco (2025).

1.19.2 Operating Cost Estimate

The operating cost estimate is presented in Q2 2025 Canadian dollars (currency abbreviation: CAD; symbol: C$). The estimate was developed to have an accuracy of ±50%. The estimate includes mining, O'Brien site water treatment, Westwood processing, a cost allowance for tailings, and general and administration (G&A) costs.

Processing operating costs, including both variable and fixed costs, have been developed based on per tonne milled basis. For processing operating costs that are variable in nature (such as reagents, power), costs have been developed on a per tonne basis and are independent of the fraction of utilization of the Westwood complex by Radisson.

Processing operating costs that are fixed in nature were estimated on an annual basis and then divided by a nominal average annual throughput of 1.095 Mt/a (3,000 t/d) to determine a cost per tonne of milled material. The cost per tonne of milled material was then multiplied by the yearly throughput in the mine plan to allow for the consideration of the percent utilization of the Westwood complex by the O'Brien Gold Project. For example, if the O'Brien Gold Project provided 0.438 Mt of mineralized material for processing in a year at the Westwood complex, then 40% (1.095 Mt /0.438 Mt = 40%) of these fixed costs have been attributed to the project for that year. This method applies to fixed costs such as labour, maintenance, and vehicle costs and allows for an equitable attribution of costs between Westwood and O'Brien during the use of the Westwood complex with multiple feeds.

Operating costs were estimated based on process plant operating costs plus an additional toll milling fee. The toll milling fee is intended to represent a profit margin beyond expected operational and capital costs, not to offset costs incurred by IAMGOLD.

The overall life-of-mine operating cost, excluding toll milling fees, is $661 million over 11 years, or an average of $144.44/t milled. The overall life-of-mine operating costs, including toll milling fees is $747 million over 11 years, or an average of $163.38/t milled. Table 1-4 provides a summary of the project operating costs.

Table 1-4: Operating Cost Summary

Cost Area	Life-of-Mine Total ($M)	Milled Cost ($/t)	% of Operating Cost Subtotal	% of Total Costs, with Toll
Mining	346	75.66	52	46
Process	173	37.71	26	23
G&A	142	31.06	22	19
Operating Cost Subtotal	661	144.44	100	N/A
Toll Milling Fees	87	18.94	N/A	12
Total with Toll Fees	**747**	**163.38**	**N/A**	**100**
Off-Site Costs, Refining & Transport	6	-	-	-
Royalties	10	-	-	-
Total Cash Costs	**763**	**-**	**-**	**-**

Source: Ausenco (2025).

1.20 Economic Analysis

The 2025 PEA is preliminary in nature and is partly based on inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves. There is no certainty that the 2025 PEA based on these mineral resources will be realized. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

The results of the economic analyses represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

The project was evaluated using a discounted cash flow analysis based on a 5% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; refining and transport costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections. Cash flows were taken to occur at the endpoint of each period. The economic analysis also used the following assumptions:

The economic analysis also used the following assumptions:

- a pre-production period of 21 months
- a mine life of 11 years
- 100% ownership
- capital cost funded with 100% equity (no financing cost assumed)
- cash flows that are discounted to the start of construction using an end-period discounting convention
- metal products that are sold in the same year they are produced
- project revenue derived from the sale of gold doré
- no contractual arrangements for refining exist.

The pre-tax NPV discounted at 5% is $782.5 million; the IRR is 65.1%; and payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is $532.4 million; the IRR is 47.6%; and payback period is 2.0 years. The project is cash positive post-tax at gold prices above US$1,260/oz.

A summary of project economics is shown in Table 1-5.

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the project using the following variables: gold price, exchange rate, initial capital costs, operating costs, and mill head grades.

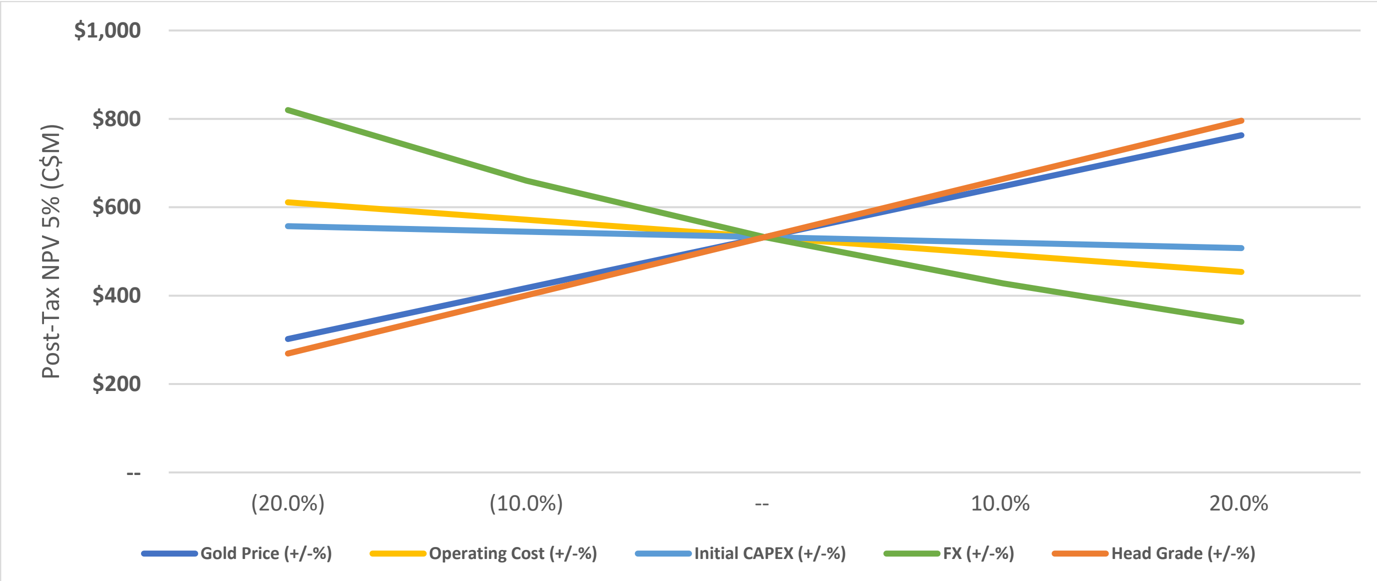
As presented in Figure 1-2 and Figure 1-3, the sensitivity analysis showed that the project is most sensitive to changes in gold price, head grade, and foreign exchange.

Table 1-5: Economic Analysis Summary

Description	Unit	Life-of-Mine Total / Average
General Assumptions		
Discount Rate	%	5.0
Gold Price	US$/oz	2,550
Exchange Rate	CAD/USD	0.73
Mine Life	years	11.0
Total Overburden and Waste Tonnes Mined	kt	3,314
Total Mill Feed Tonnes	kt	4,575
Production		
Mill Head Grade (Au)	g/t	5.03
Mill Recovery Rate	%	87.5
Total Mill Ounces Recovered	koz	647
Total Average Annual Production	koz	59
Transport, Refining, Royalties		
Gold Payable	%	99.95
Refining & Transport Cost	US$/oz Au	6.25
NSR Royalty	%	2.0
Operating Costs		
Mining Cost	C$/t mined	75.66
Processing Cost	C$/t milled	37.71
G&A Cost	C$/t milled	31.06
Toll Milling	C$/t milled	18.94
Total Operating Cost	C$/t milled	163.38
Cash Costs and All-In Sustaining Costs		
Cash Costs*	US$/oz Au	861.06
All-In Sustaining Cost (AISC)**	US$/oz Au	1,058.54
Capital Expenditures		
Initial Capital Cost	C$M	175
Sustaining Capital Cost	C$M	173
Closure Capital Cost	C$M	5
Salvage Value	C$M	(3)
Economics		
Pre-Tax NPV @ 5%	C$M	782.5
Pre-Tax IRR	%	65.1
Pre-Tax Payback	years	1.4
Post-Tax NPV @ 5%	C$M	532.4
Post-Tax IRR	%	47.6
Post-Tax Payback	years	2.0

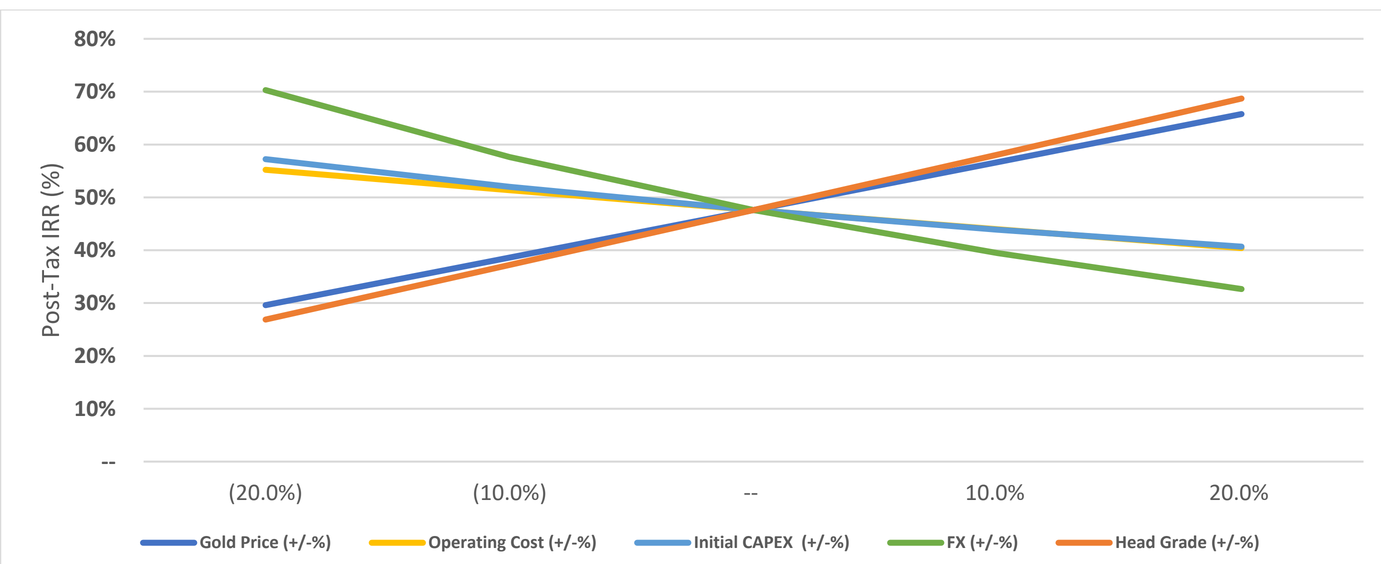
Notes: * Cash costs consist of mining costs, processing costs, mine-level G&A, refining and transport charges and royalties. ** AISC includes cash costs plus sustaining capital, closure costs, and salvage value. Source: Ausenco (2025).

Figure 1-2: Post-Tax NPV5% Sensitivity Results



Source: Ausenco (2025).

Figure 1-3: Post-Tax IRR Sensitivity Results



Source: Ausenco (2025).

1.21 Adjacent Properties

The O'Brien gold deposit is located near several producing gold deposits, including Agnico Eagle's LaRonde mine and IAMGOLD's Westwood gold mine.

1.22 Recommendations

The work carried out to date has justified the continued exploration and development of the project. Work for the next project phase, which is described in Section 26 and summarized in Table 1-6, is recommended to advance the project to a pre-feasibility study level of engineering, with additional drilling and metallurgical testwork and an improved mineral resource estimate. The estimated cost of the recommended program is $16.6 million.

Table 1-6: Phase 1 Recommended Work Program and Budget

Program Component	Estimated Total Cost ($k)
Drilling	
Resource Expansion at Depth (35,000 m)	7,000
Phase 1 Mineral Resource Infill (25,000 m)	5,000
Drilling Contingency (10%)	1,200
Subtotal	13,200
Mining	
Perform a Prefeasibility-Level Hydrogeological Study	350
Perform a Prefeasibility-Level Geomechanical Study on the O'Brien and Kewagama Zones, as well as the Surface Crown Pillar	500
Perform a Groundwater Flow Study	100
Subtotal	950
Metallurgy	
Sample Preparation, Storage, Head Assays	42
Hardness Determination	81
Open Circuit Flotation Testwork	276
Locked Cycle Tests	180
Mineralogy / Liberation Analysis	100
Test Management / Supervision / Reporting	150
Metallurgy Contingency	124
Subtotal	954
Pre-Feasibility Study	**1,000**
Environmental Studies and Community Engagement	**500**
Grand Total	**16,604**

2 INTRODUCTION & TERMS OF REFERENCE

2.1 Introduction

Radisson Mining Resources (Radisson) commissioned Ausenco Engineering Canada ULC (Ausenco) to compile a preliminary economic assessment (PEA) of the O'Brien Gold Project. The PEA was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and the requirements of Form 43-101 F1.

The responsibilities of the engineering consultants and firms who are providing qualified persons are as follows:

- Ausenco managed and coordinated the work related to the report. Ausenco developed the PEA-level design for metallurgy and recovery methods, a cost estimate for process plant modifications related to toll milling, and mine site electrical infrastructure. Ausenco also compiled the overall cost estimate and completed the economic analysis.
- Norda Stelo Inc. (Norda Stelo) designed the underground mining methods, surface infrastructure related to waste rock storage and mineralized material storage, and prepared the mine capital and operating costs.
- SLR Consulting (Canada) Ltd. (SLR) completed the work related to property description, accessibility, local resources, geological setting, deposit type, exploration work, drilling, exploration works, sample preparation and analysis, data verification, and mineral resource estimate.
- BBA Inc. (BBA) completed the work related to mine site water management and environmental, permitting, and social considerations.

2.2 Terms of Reference

The report supports disclosures by Radisson in a news release dated July 9, 2025, entitled, "Radisson Announces Positive Preliminary Economic Assessment for O'Brien Gold Project."

All measurement units used in this report are International System units (SI) and all currencies are expressed in Canadian dollars (C$ or CAD) unless otherwise stated.

The O'Brien Gold Project consists of a mine site and currently assumes the mineralized material would be processed at the nearby IAMGOLD Westwood complex. A milling assessment for the processing of O'Brien mined material at the Westwood complex was conducted under the auspices of a September 2024 MOU between Radisson and IAMGOLD. The MOU is non-binding and non-exclusive and contains no specific terms around potential commercial arrangements between the parties. IAMGOLD has not independently confirmed the processing assumptions, metallurgical results and/or cost assumptions assumed in this study.

The deposit and mine are located at the O'Brien site. The process plant and tailings management facility to process the material are at the Westwood complex and would be owned and operated by IAMGOLD, with Radisson paying a toll milling fee for processing the gold.

2.3 Qualified Persons

The qualified persons for the report are listed in Table 2-1. By virtue of their education, experience, and professional associations, they meet the standard of a qualified person as defined in the NI 43-10 guidelines.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of Issuer
Renee Barrette	P.Eng., ing.	Principal Metallurgist	Ausenco Engineering Canada ULC	Yes
Luke Evans	P.Eng., ing.	Global Technical Director	SLR Consulting (Canada) Ltd.	Yes
Hugo Latulippe	P.Eng., ing.	Principal Mining Engineer	BBA Inc.	Yes
Marc R. Beauvais	P.Eng., ing.	Senior Mining Engineer	Norda Stelo Inc.	Yes

Source: Ausenco (2025).

2.4 Site Visits and Scope of Personal Inspection

2.4.1 Renee Barrette

Renee Barrette visited the site on October 7, 2024 to review the current exploration activities and facilities at the O'Brien mine, although there was limited activity to observe. Facilities or processes viewed included the old mill, garage, core shack, electrical input and main switchroom, potable water supply, core sample processing areas, QA/QC processing and general site security.

2.4.2 Luke Evans

Luke Evans visited the project site on October 12, 2022. He previously visited the O'Brien Gold Project on April 30, 1998, for Roscoe Postle Associates Inc. (RPA) as well as on September 27, 2006, for Scott Wilson RPA Inc.

2.4.3 Marc R. Beauvais

Marc R. Beauvais visited the O'Brien property on September 23, 2024. The aim was to assess the condition of the overburden composition near the proposed portal and potential mine pad locations, and review the general conditions of the land, the existing surface infrastructure, and the main site accesses.

2.5 Effective Dates

The effective date of this technical report is June 27, 2025. This report also has the following significant dates:

- Mineral resource estimate: May 6, 2025
- Financial analysis: June 27, 2025.

2.6 Information Sources and References

2.6.1 Property Description

For Section 4, Property Description, as well as the property location and relevant portions of Section 1, Summary, the QPs completely relied upon ownership information provided by Radisson. The QP did not research property title,mineral rights, or royalty agreements for the project and does not express an opinion as to the ownership status of the property.

2.6.2 Metallurgy

The QPs have reviewed upon historical metallurgical data provided in the WSP document "Revue Des Données Métallurgiques Projet O'Brien." The document provides a summary of historical metallurgical data as discussed in Section 13. The source data was unavailable to confirm the conclusions obtained from the document. However, the recovery model in this report does not utilize the historical metallurgical data, and is based on the 2024 testwork program.

2.7 Previous Technical Reports

The O'Brien Project has been the subject of previous technical reports under Radisson including the following:

SLR Consulting (Canada Ltd).; (April 14, 2023). Technical Report on the Obrien Project, Northwestern Québec, Canada Report for NI 43-101.

Kenneth Williamson; (August 29, 2019). NI 43-101 Technical Report and Mineral Resource Estimate for the O'Brien Project, Abitibi, Québec.

InnovExplo Inc; (May 3, 2018). NI 43-101 Technical Report and Mineral Resource Estimate for the O'Brien Project, Abitibi, Québec.

InnovExplo Inc; (April 10, 2015). Technical Report for the Obrien Project, Abitibi, Québec.

RPA Inc; (October 21, 2013). Technical Report On The O'brien Project Mineral Resource Estimate, Québec, Canada.

2.8 Currency, Units, Abbreviations and Definitions

All units of measurement in this report are metric and all currencies are expressed in Canadian dollars (symbol: C$ or currency: CAD) unless otherwise stated. Contained gold metal is expressed as troy ounces (oz), where 1 oz = 31.1035 g. All material tonnes are expressed as dry tonnes (t) unless stated otherwise. A list of abbreviations and acronyms is provided in Table 2-2, and units of measurement used in the report are listed in Table 2-3.

Table 2-2: Abbreviations and Acronyms

Abbreviation	Description
AA	atomic absorption spectroscopy
Au	gold
Az	azimuth
BIF	banded iron formation
BWi	bond ball mill work index
CAD:USD	Canadian-American exchange rate
CIM	Canadian Institute of Mining, Metallurgy and Petroleum
CIM Definition Standards	CIM Definition Standards for Mineral Resources and Mineral Reserves 2014
CIP	carbon in pulp
CoG	cut-off grade
CRM	certified reference material
CWi	Bond crusher work index
DCIP	direct current resistivity and induced polarization
DDH	diamond drill hole
ECCC	Environment and Climate Change Canada
E-GRG	extended gravity recoverable gold
EM	electromagnetic
FA	fire assay
FET	federal excise tax
FS	feasibility study
G&A	general and administration
GPR	gross production royalty
GQCV	greenstone-hosted quartz-carbonate vein deposits
GRAV	gravimetric finish method
ICP	inductively coupled plasma
ICP-OES	inductively coupled plasma - optical emission spectrometry
ID2	inverse distance squared
ID3	inverse distance cubed
IOCG	iron oxide copper gold
IP	induced polarization
IRGS	intrusion-related gold system

Abbreviation	Description
ISO	International Organization for Standardization
koz	thousand ounces
LIDAR	light detection and ranging
LUP	land use permit
MCF	mechanized cut and fill
MELCCFP	Ministère de l'Environnement, de la lutte contre les changements climatiques, de la Faune et des Parcs
MRE	mineral resource estimate
MRNF	Ministère des Ressources naturelles et des Forêts
NAD 83	North American Datum of 1983
NI 43-101	National Instrument 43-101 (Regulation 43-101 in Quebec)
NN	nearest neighbour
NRCan	Natural Resources Canada
NSR	net smelter return
NTS	national topographic system
OK	ordinary kriging
PEA	preliminary economic assessment
PFS	prefeasibility study
PGE	platinum group elements
QA/QC	quality assurance/quality control
QP	qualified person (as defined in National Instrument 43-101)
RBQ	Régie du Bâtiment du Québec
RCM	Regional County Municipality
ROM	run of mine
RQD	rock quality designation
SAG	semi-autogenous grinding
SCC	Standards Council of Canada
SD	standard deviation
S_{d}-BWI	micro hardness or bond ball mill work index on SAG ground material
SEDEX	sedimentary exhalative deposits
SG	specific gravity
SQ	Sureté du Québec
TMF	tailings management facility
UG	underground
UTM	Universal Transverse Mercator coordinate system
UV	ultraviolet
VLF-EM	very low frequency electromagnetic
VMS	volcanogenic massive sulphide

Table 2-3: Units of Measurement

Abbreviation	Description
%	percent
% solids	percent solids by weight
CAD	Canadian dollar (currency)
C$	Canadian dollar (as symbol)
$/t	dollars per metric ton
°	angular degree
°C	degree Celsius
µm	micron (micrometre)
cm	centimetre
cm^3	cubic centimetre
ft	foot (12 inches)
g	gram
g/cm^3	gram per cubic centimetre
g/L	gram per litre
g/t	gram per metric ton (tonne)
h	hour (60 minutes)
ha	hectare
kg	kilogram
kg/t	kilogram per tonne
km	kilometre
km^2	square kilometre
kW	kilowatt
kWh/t	kilowatt-hour per tonne
L	litre
lb	pound
m, m^2, m^3	meter, square meter, cubic meter
M	million
Ma	million years (annum)
masl	meters above mean sea level
mm	millimetre
MMBTUH	million British thermal unit hours
Moz	million (troy) ounces
Mt	million tonnes
MW	megawatt
oz	troy ounce
oz/t	ounce (troy) per tonne
oz/ton	ounce (troy) per short ton (2,000 lbs)

Abbreviation	Description
ppb	parts per billion
ppm	parts per million
t	metric tonne (1,000 kg)
ton	short ton (2,000 lbs)
t/d	tonnes per day
USD	US dollars (currency)
US$	US dollar (as symbol)

3 RELIANCE ON OTHER EXPERTS

3.1 Introduction

The authors have reviewed, within the scope of their technical expertise, all the available information presented to them by others; however, they cannot guarantee its accuracy and completeness. The authors reserve the right, but will not be obligated to, revise the technical report and its conclusions if additional information becomes known to them subsequent to the effective date of this report.

This technical report has been prepared by qualified persons (QPs). The information, conclusions, opinions, and estimates contained herein are based on:

- information available at the time of preparation of this technical report
- assumptions, conditions, and qualifications as set forth in this technical report.

3.2 Legal Agreements

Memorandum of Understanding between Radisson Mining Resources Inc and IAMGOLD Corporation, as disclosed in the news release entitled, “Radisson announces Memorandum of Understanding with IAMGOLD on Milling Assessment for O’Brien Gold Project” September 9, 2024.

The QP’s have fully relied on this information in Sections 1.15, 1.16, 17, 18.6, 18.8, 19, 25.9, 25.10.2, 25.16.1.3, 25.16.2.3, and 25.6.2.5.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Location

The O'Brien Gold Project is located in the Abitibi region in northwestern Québec, approximately 1 km north of the town of Cadillac at geographic coordinates 48°14'00"N 78°23'30"W. Gravel roads provide access to the project from provincial Highway 117. It is approximately 50 km east of the town of Rouyn-Noranda, Québec; 30 km by road west of the town of Malartic, and 55 km by road west of the town of Val d'Or. The project's location is shown in Figure 4-1.

4.2 Mineral Rights

In Canada, natural resources fall under provincial jurisdiction. In the Province of Québec, the management of mineral resources and the granting of exploration and mining rights for mineral substances and their use are regulated by the *Québec Mining Act*, which is administered by the Ministère de l'Énergie et des Ressources Naturelles (MERN). Mineral rights are owned by the Crown and are distinct from surface rights. In Québec, a mining lease (BM) is initially granted for a 20-year period but can be renewed for additional 10-year periods. A mining concession (CM) is a grandfathered mining claim sub-category for mining claims staked before January 1, 1966, and reviewed alongside evidence to the satisfaction of the minister of reasonable indications of a mineral deposit which can be economically developed. Exploration claims (CDCs) may be obtained by map designation via GEOSTIM Plus or by land staking in designated areas and grant the holder exclusive rights to search for mineral substances in the public domain, except sand, gravel, clay, and other loose deposits, on the land subjected to the claim. The term of an exploration claim is two years, which can be renewed indefinitely provided the claim holder meets the conditions stipulated in the *Mining Act*. These conditions extend to the carrying out of exploration work, the nature and amount of which is established by regulation. Claim fees are indexed automatically to reflect the annual change in the Consumer Price Index for Québec, currently at 1.26%.

4.3 Land Tenure

The O'Brien Gold Project consists of a contiguous block of 146 exploration claims (CDC) and one mining concession (called the 240-PTA block), listed in Table 4-1. The exploration claims cover an area of 7,137.74 ha and the mining concession has an area of 288.19 ha (Figure 4-2). Overall, the project covers an area of 7,425.93 ha. Radisson has a 100% interest in the project, which is comprised of three former properties named O'Brien, Kewagama, and Thompson-Cadillac (previously called New Alger).

4.4 Surface Rights

The mining claims included for O'Brien are located on Crown land. Radisson has the first right to acquire the surface rights to O'Brien by taking them to the mining lease status. Under Québec Mining Legislation, the owner of the mining rights can make use of the timber on the leased property by paying a nominal fee if such timber is deemed to be of commercial value. Radisson currently has surface rights to one area via annually renewable leases, which are in good standing.

Figure 4-1: Location Map

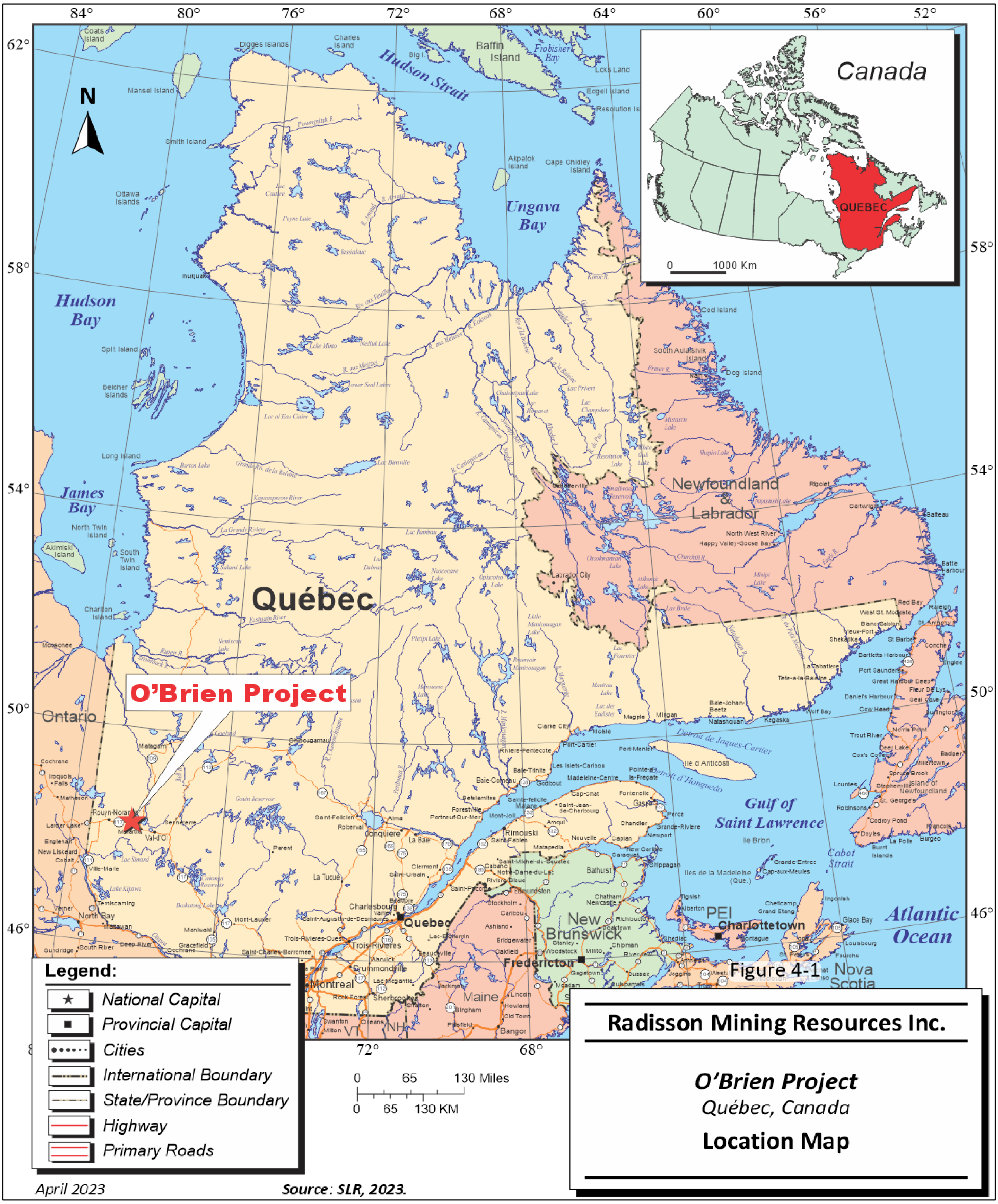


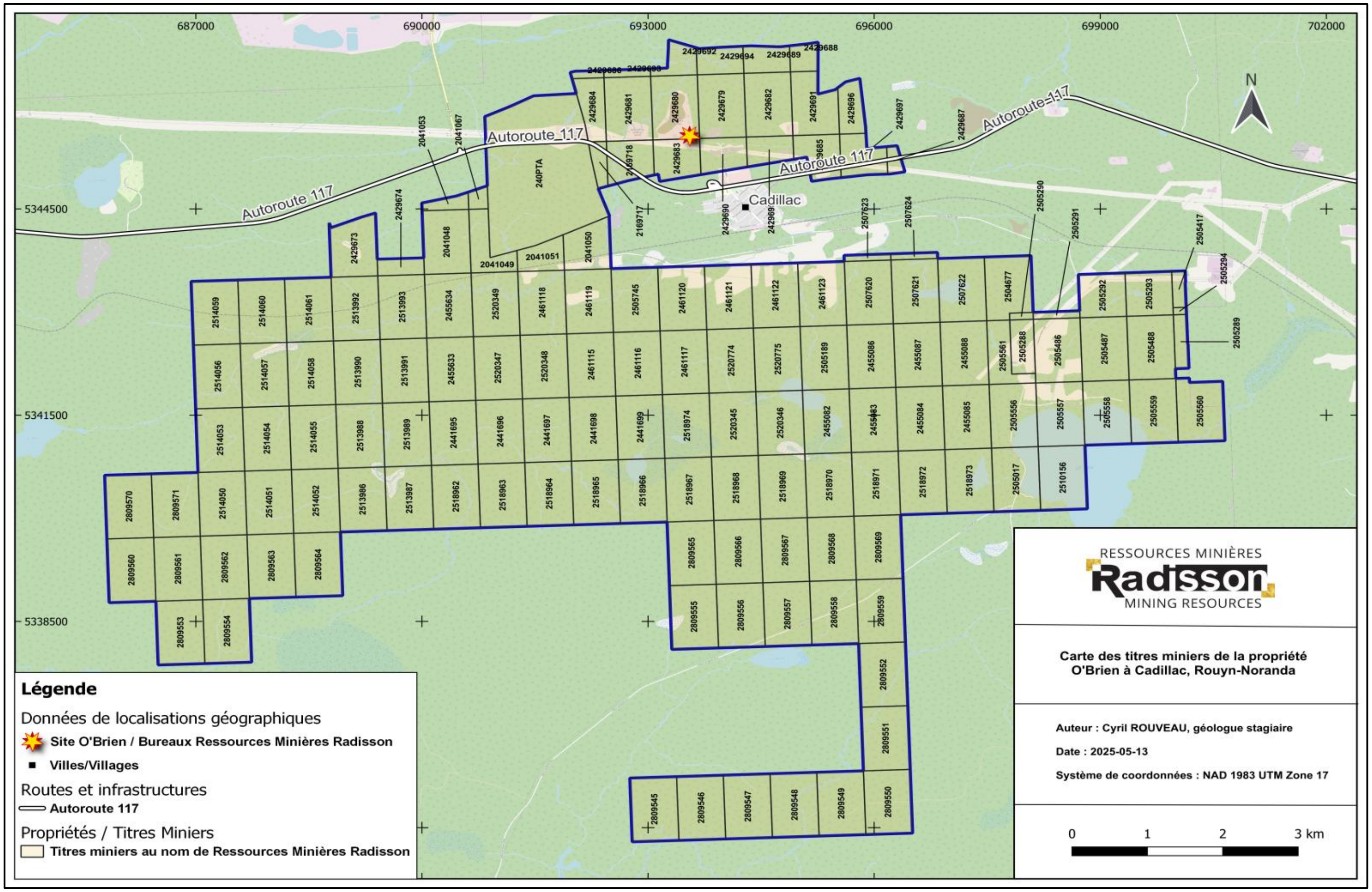
Table 4-1: Mineral Tenure Table

Type	Title	NTS Sheet	Status	Area (ha)	Registration Date	Expiration Date	Holder
CDC	2169717	SNRC 32D01	Active	12.67	2008-08-07	2025-08-06	Radisson Mining Resources Inc. 100%
CDC	2169718	SNRC 32D01	Active	35.61	2008-08-07	2025-08-06	Radisson Mining Resources Inc. 100%
CDC	2429679	SNRC 32D01	Active	57.37	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429680	SNRC 32D01	Active	57.37	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429681	SNRC 32D01	Active	57.37	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429682	SNRC 32D01	Active	57.37	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429683	SNRC 32D01	Active	34.65	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429684	SNRC 32D01	Active	29.92	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429685	SNRC 32D01	Active	33.92	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429686	SNRC 32D01	Active	4.57	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429687	SNRC 32D01	Active	7.27	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429688	SNRC 32D01	Active	14.76	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429689	SNRC 32D01	Active	23.71	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429690	SNRC 32D01	Active	29.69	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429691	SNRC 32D01	Active	49.52	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429692	SNRC 32D01	Active	19.99	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429693	SNRC 32D01	Active	6.65	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429694	SNRC 32D01	Active	24.02	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429695	SNRC 32D01	Active	24.12	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429696	SNRC 32D01	Active	24.75	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CDC	2429697	SNRC 32D01	Active	32.13	2015-07-30	2026-03-01	Radisson Mining Resources Inc. 100%
CM	240PTA	SNRC 32D01	Active	288.19	1928-11-29	Note 1	Radisson Mining Resources Inc. 100%
CDC	2461115	SNRC 32D01	Active	57.40	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461116	SNRC 32D01	Active	57.40	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461117	SNRC 32D01	Active	57.40	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461118	SNRC 32D01	Active	57.39	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461119	SNRC 32D01	Active	57.39	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461120	SNRC 32D01	Active	57.39	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461121	SNRC 32D01	Active	57.39	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461122	SNRC 32D01	Active	57.40	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2461123	SNRC 32D01	Active	57.40	2016-09-01	2025-08-31	Radisson Mining Resources Inc. 100%
CDC	2041048	SNRC 32D01	Active	57.38	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2041049	SNRC 32D01	Active	33.41	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2041050	SNRC 32D01	Active	39.65	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2041051	SNRC 32D01	Active	24.86	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2041053	SNRC 32D01	Active	10.72	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2041067	SNRC 32D01	Active	7.56	2006-12-13	2025-12-12	Radisson Mining Resources Inc. 100%
CDC	2429673	SNRC 32D01	Active	50.67	2015-07-16	2026-01-26	Radisson Mining Resources Inc. 100%
CDC	2429674	SNRC 32D01	Active	14.94	2015-07-16	2026-01-26	Radisson Mining Resources Inc. 100%
CDC	2504677	SNRC 32D01	Active	54.27	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505017	SNRC 32D01	Active	57.42	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505189	SNRC 32D01	Active	57.40	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505288	SNRC 32D01	Active	24.68	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505289	SNRC 32D01	Active	13.83	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505290	SNRC 32D01	Active	3.13	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505291	SNRC 32D01	Active	6.27	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505292	SNRC 32D01	Active	39.04	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505293	SNRC 32D01	Active	39.15	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505294	SNRC 32D01	Active	1.80	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505417	SNRC 32D01	Active	9.27	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505486	SNRC 32D01	Active	57.41	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505487	SNRC 32D01	Active	57.41	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505488	SNRC 32D01	Active	57.41	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505556	SNRC 32D01	Active	57.42	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505557	SNRC 32D01	Active	57.42	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505558	SNRC 32D01	Active	57.42	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505559	SNRC 32D01	Active	57.42	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505560	SNRC 32D01	Active	54.59	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505561	SNRC 32D01	Active	32.72	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2505745	SNRC 32D01	Active	57.39	2017-11-20	2026-11-19	Radisson Mining Resources Inc. 100%
CDC	2507620	SNRC 32D01	Active	57.40	2017-12-08	2026-12-07	Radisson Mining Resources Inc. 100%
CDC	2507621	SNRC 32D01	Active	57.40	2017-12-08	2026-12-07	Radisson Mining Resources Inc. 100%
CDC	2507622	SNRC 32D01	Active	57.40	2017-12-08	2026-12-07	Radisson Mining Resources Inc. 100%
CDC	2507623	SNRC 32D01	Active	5.52	2017-12-08	2026-12-07	Radisson Mining Resources Inc. 100%
CDC	2507624	SNRC 32D01	Active	5.54	2017-12-08	2026-12-07	Radisson Mining Resources Inc. 100%
CDC	2809545	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809546	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809547	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809548	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809549	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809550	SNRC 32D01	Active	57.47	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809551	SNRC 32D01	Active	57.46	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809552	SNRC 32D01	Active	57.45	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809553	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809554	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809555	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%

Type	Title	NTS Sheet	Status	Area (ha)	Registration Date	Expiration Date	Holder
CDC	2809556	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809557	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809558	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809559	SNRC 32D01	Active	57.44	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809560	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809561	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809562	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809563	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809564	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809565	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809566	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809567	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809568	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809569	SNRC 32D01	Active	57.43	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809570	SNRC 32D01	Active	57.42	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2809571	SNRC 32D01	Active	57.42	2023-12-13	2026-12-12	Radisson Mining Resources Inc. 100%
CDC	2510156	SNRC 32D01	Active	57.42	2018-01-22	2027-01-21	Radisson Mining Resources Inc. 100%
CDC	2513986	SNRC 32D01	Active	57.42	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513987	SNRC 32D01	Active	57.42	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513988	SNRC 32D01	Active	57.41	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513989	SNRC 32D01	Active	57.41	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513990	SNRC 32D01	Active	57.40	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513991	SNRC 32D01	Active	57.40	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513992	SNRC 32D01	Active	57.39	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2513993	SNRC 32D01	Active	57.39	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514050	SNRC 32D01	Active	57.42	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514051	SNRC 32D01	Active	57.42	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514052	SNRC 32D01	Active	57.42	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514053	SNRC 32D01	Active	57.41	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514054	SNRC 32D01	Active	57.41	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514055	SNRC 32D01	Active	57.41	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514056	SNRC 32D01	Active	57.40	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514057	SNRC 32D01	Active	57.40	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514058	SNRC 32D01	Active	57.40	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514059	SNRC 32D01	Active	57.39	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514060	SNRC 32D01	Active	57.39	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2514061	SNRC 32D01	Active	57.39	2018-03-06	2027-03-05	Radisson Mining Resources Inc. 100%
CDC	2441695	SNRC 32D01	Active	57.41	2016-04-18	2027-04-17	Radisson Mining Resources Inc. 100%
CDC	2441696	SNRC 32D01	Active	57.41	2016-04-18	2027-04-17	Radisson Mining Resources Inc. 100%
CDC	2441697	SNRC 32D01	Active	57.41	2016-04-18	2027-04-17	Radisson Mining Resources Inc. 100%
CDC	2441698	SNRC 32D01	Active	57.41	2016-04-18	2027-04-17	Radisson Mining Resources Inc. 100%
CDC	2441699	SNRC 32D01	Active	57.41	2016-04-18	2027-04-17	Radisson Mining Resources Inc. 100%
CDC	2518962	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518963	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518964	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518965	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518966	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518967	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518968	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518969	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518970	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518971	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518972	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518973	SNRC 32D01	Active	57.42	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2518974	SNRC 32D01	Active	57.41	2018-05-31	2027-05-30	Radisson Mining Resources Inc. 100%
CDC	2520345	SNRC 32D01	Active	57.41	2018-07-05	2027-07-04	Radisson Mining Resources Inc. 100%
CDC	2520346	SNRC 32D01	Active	57.41	2018-07-05	2027-07-04	Radisson Mining Resources Inc. 100%
CDC	2520347	SNRC 32D01	Active	57.40	2018-07-05	2027-07-04	Radisson Mining Resources Inc. 100%
CDC	2520348	SNRC 32D01	Active	57.40	2018-07-05	2027-07-04	Radisson Mining Resources Inc. 100%
CDC	2520349	SNRC 32D01	Active	57.39	2018-07-05	2027-07-04	Radisson Mining Resources Inc. 100%
CDC	2520774	SNRC 32D01	Active	57.40	2018-07-16	2027-07-15	Radisson Mining Resources Inc. 100%
CDC	2520775	SNRC 32D01	Active	57.40	2018-07-16	2027-07-15	Radisson Mining Resources Inc. 100%
CDC	2455082	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455083	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455084	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455085	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455086	SNRC 32D01	Active	57.40	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455087	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455088	SNRC 32D01	Active	57.41	2016-07-27	2027-07-26	Radisson Mining Resources Inc. 100%
CDC	2455633	SNRC 32D01	Active	57.40	2016-07-28	2027-07-27	Radisson Mining Resources Inc. 100%
CDC	2455634	SNRC 32D01	Active	57.39	2016-07-28	2027-07-27	Radisson Mining Resources Inc. 100%
Total				**7,425.93**			

Note: Includes exploration claims and mining concession. Renewal payment for exploration claims 2169717 and 2169718 was made in July 2025 and confirmation of renewal is expected in September 2025.

Figure 4-2: Mineral Tenure Plan



Source: Radisson (2025).

4.5 Encumbrances

4.5.1 Urban Perimeter

Part of the project is subject to regulations respecting an “urban perimeter” (Claim 2429690 on Figure 4-2) or an “area dedicated to vacationing.” These areas, as documented in GESTIM, fall under “Exploration Prohibited” (see Bill 70, 2013, chapter 32, section 124). This area is very minor and does not impact the planned exploration drilling on the property.

The O’Brien Gold Project only includes mining rights obtained before December 10, 2013. Exploration is permitted on mining rights which overlap the urban perimeter and the area dedicated to vacationing, until mining-incompatible territories are determined by the regional county municipality (“MRC” in French). If a claim overlaps a mining-incompatible territory, exploration will still be permitted on the overlapping claim, but renewal of such claim will only be permitted if work is performed on the claim during any term occurring after the determination of the mining-incompatible territory (section 61 of the *Mining Act*).

4.5.2 Environment

Radisson is presently exempted by the MERN of all liabilities associated with the onsite historical tailings. However, should a decision to use the same area for future tailings be made, Radisson would acquire all liabilities for past and present tailings storage facilities.

In 1956, 8,928 metal barrels containing arsenic trioxide were stored underground at the O’Brien mine. The barrels were placed in an underground opening on Level 1500 of the mine (15-G-West and 15-F-West drifts) and the entrance was sealed.

The mine was reactivated and pumped out in 1972, but no information about the barrels is available for that period. In 1981, Darius Gold Mines, then owner of the O’Brien mine, got the concrete walls from the 1500 ft level demolished, as a potential buyer for the arsenic trioxide had been found. Later that year, the potential buyer withdrew.

In 1985, waterproof and reinforced concrete plugs (2.3 m wide) at the entrance of the underground opening containing the barrels on Level 1500 were reconstructed prior to the second flooding in 1985 by Sulpetro Minerals Ltd. The storage site has not been visited since then. In 1989, GERLED, a Quebec Government entity with the mandate to catalogue and monitor all known dangerous waste material sites in the province, categorized this storage site as a class 1 dangerous waste material site.

Radisson addressed this through a basic hydrogeological study (Fournier and Leblanc, 2017). The study concluded that it was not possible to determine limiting impacts of such an underground storage facility on the economic potential of the 2018 MRE (Beausoleil, 2018). To update and acquire more data and information, Radisson carried out an extensive hydrogeological study in 2018 and 2019 and has since been monitoring the groundwater quality bi-annually. In addition, a water treatment plant to dewater, keep underground workings dry, and allow the arsenic to be treated has already been designed.

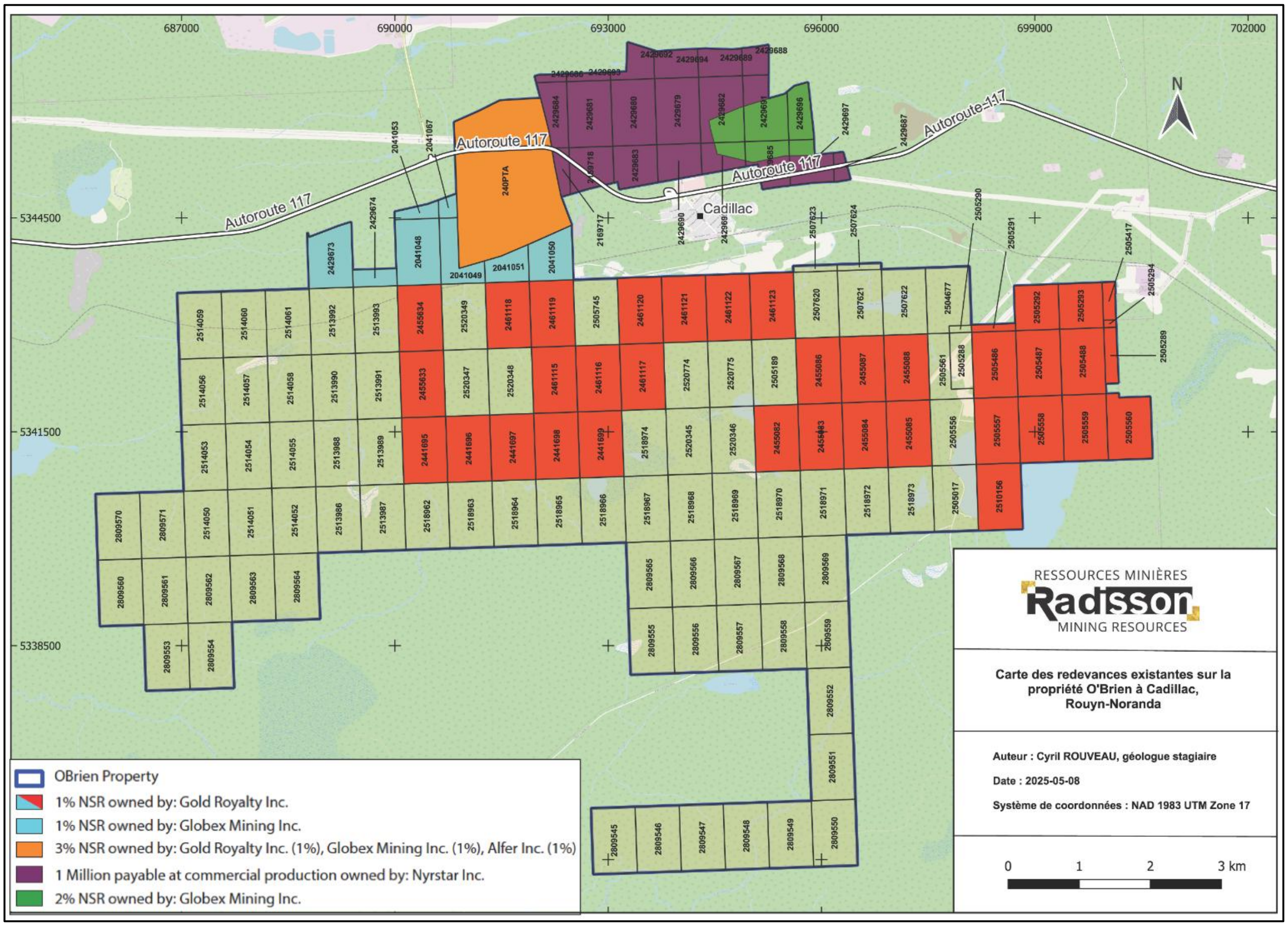
The QP is not aware of any other encumbrances on the O’Brien Gold Project. The QP is not aware of any other significant factors and risks that may affect access, title, or the right or ability to perform the proposed work at the O’Brien Gold Project, nor is the QP aware of any associated environmental liabilities.

4.6 Royalties

Radisson has a 100% interest in the entire O’Brien Gold Project.

Below are the details of royalties (Figure 4-3) and contractual obligations held by third parties on the mining titles forming the O’Brien, Kewagama, and Thompson-Cadillac (previously called New Alger) properties combined under the O’Brien Gold Project.

Figure 4-3: Map of O’Brien Royalties



Source: Radisson (2025).

- O’Brien – C$1 million in payment in the event of commercial production
- Kewagama – 2% net smelter return (NSR) royalty
- Thompson-Cadillac (previously called New Alger)
  - 3% NSR on certain mining claims replacing the old mining concession known as CM240-PTA
  - 2% NSR on certain mining claims in the southern portion of the property
  - 1% NSR on certain mining claims in the southern portion of the property
  - C$1.5 million contingent payment related to the New Alger property shall be payable on the earliest of:

    (i) a change of control of the corporation

    (ii) the declaration by the corporation of commercial production of the project, and;

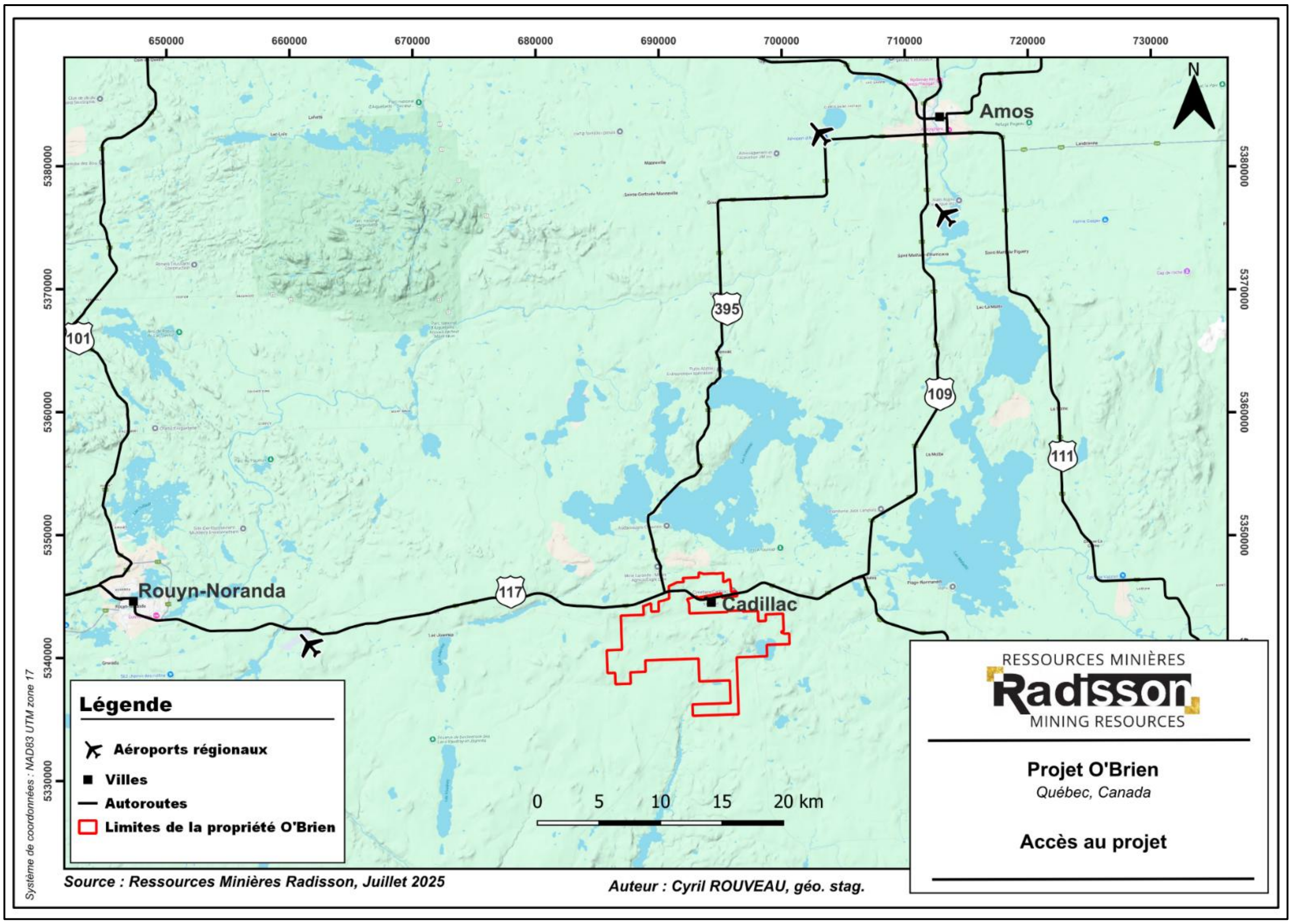
    (iii) a sale of the project for proceeds of more than $40,000,000.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE & PHYSIOGRAPHY

5.1 Accessibility

The O'Brien Gold Project is located in the Abitibi-Témiscamingue administrative region, and current exploration activities are north of the Municipality of Cadillac. The project site can easily be accessed via provincial Highway 117, which is the Trans-Canada Highway Northern Route, as illustrated in Figure 5-1. Access to the project is available from Highway 117 via well-maintained gravel roads year-round.

Figure 5-1: Project Access



Source: Radisson (2025).

5.2 Climate

The Abitibi region experiences a typical continental climate characterized by cold, dry, winters and mild, humid, summers. According to Environment Canada, the nearest meteorological station (Mont Brun, located 28 km northwest of the property) records average temperatures of 16.7°C in July and -17.9°C in January (1981 to 2010). On average, the annual precipitation is 985 mm, including 280 cm of snow and 704.9 mm of rain. Snow accumulates on the ground from October to May with increased activity during the months of November to March. Climatic conditions do not prevent exploration activities from being carried out, but may require adjustments for some, such as drilling in marshy areas.

5.3 Local Resources and Infrastructure

The O’Brien Gold Project is located 45 km from the cities of Rouyn-Noranda and Val d'Or, two cities with economies based in part on mining ensuring a skilled local workforce is available for the project. Additionally, there is sufficient land to support the development of the mine. This region is associated with several active mines and numerous exploration companies. The closest concentrator is that of Agnico-Eagle’s LaRonde mine, which is 7 km by road to the west of the project. A full range of infrastructure, service, and experienced human resources companies are available in most nearby communities including in Val d'Or, Malartic, and Rouyn-Noranda.

Regional airports in the cities of Rouyn-Noranda and Val d'Or provide regular connection with the main centres of the province (Montreal and Quebec City). In addition, the property is connected to a high-voltage powerline and is intersected from east to west by Highway 117 and the Trans-Canada Railway.

Several historical mines (O'Brien, Thompson-Cadillac, and Kewagama) on the O'Brien property were in operation between 1925 and 1981. Most of the surface infrastructure has been dismantled; only the garage and the factory of the O'Brien mine have been preserved and are used today for Radisson’s exploration activities.

Radisson has a core shack with an adjoining core sawing room and exploration offices on the O’Brien site. The industrial lease also includes a large storage area for mineralized material and waste rock.

5.4 Physiography

The topography of the O’Brien Gold Project is generally flat and characterized by rounded reliefs and elevation differences of up to 50 m. The altitude varies between 320 metres above sea level (masl) on the site of the former Thompson-Cadillac mine and 395 masl near Lake Héva. Low topographical areas are associated with the presence of swamps, ponds, and small lakes. Most of the land is poorly drained, except for areas covered by quaternary fluvial deposits.

The project is located at the border between the transition forest and the boreal forest. The dominant species are black spruce, balsam fir, and larch. Locally, stands of white birch and poplar have established themselves in old logging areas. A vast majority of the territory is in various states of growth after recent forestry work.

6 HISTORY

This section provides an overview of historical work carried out on the former O'Brien mine and Kewagama mine properties. The descriptions have been taken from Williamson (2019), Beausoleil (2018), and Evans (2007), and were reviewed and updated by the QP.

6.1 Ownership, Exploration, and Development History

6.1.1 Former O'Brien Mine Property

6.1.1.1 O'Brien Gold Mines Ltd – 1924 to 1957

1924: Claims were staked in 1924 by Austin Dumont and W. Herweston from M.J. O'Brien Company Ltd., and No. 1 vein was discovered by prospecting.

1925: No. 1 shaft was sunk to a depth of 110 ft and underground development commenced.

1926-1929: Diamond drilling was carried out, comprising 12 holes for 6,000 ft. Five principal veins (Nos. 1 through 5) were interpreted and delineated by surface diamond drilling and underground work. High-grade mineralization milled during 1928 amounted to several hundred ounces.

1929: Stoping commences on the most easterly shoot.

1930: No. 2 shaft (which became the main shaft) was sunk to a depth of 300 ft. Levels were established at depths of 100, 200, and 300 ft.

1932-1933: An amalgamation mill, with a capacity of 90 t/d, was built in 1932 and began operating. While in operation, the mill was processing about 75 to 80 t/d.

1934: The No. 2 shaft was extended from 300 ft to 500 ft deep, and the 400 ft and 500 ft levels developed. As of July 1934, the mine had produced 38,730 tonnes of mineralized material, averaging 15.43 g/t Au.

1935: No. 2 shaft reached a depth of 1,035 ft, and stations were established at approximately 625, 750, 875 and 1,000 ft.

Roasting and cyaniding facilities were added.

Production from September 9, 1934, to October 5, 1935 was given as 26,662 tonnes of mineralization averaging 9.19 g/t Au. Of this, 66.12% was recovered as bullion, and 26.12% was saved in concentrates for re-processing by the new addition to the mill.

1937-1939: The milling capacity was increased to 150 t/d.

No. 3 shaft was sunk to a depth of 1,500 ft, with stations were established at 125 ft intervals.

1940: Crude arsenic was sold to Deloro Smelting & Refining Company in Deloro, Ontario, with production sales continuing until 1950.

1941: No. 2 shaft was converted to skips with a mineralization transfer system at the 2,125 ft level and production stoping changed to inclined cut and fill in the deeper levels.

The sinking of the internal No. 4 shaft began in 1941.

1942-1949: Steady production peaked in 1942 at 63,096 tonnes milled, averaging 12.79 g/t Au, and reserves were at their highest at 218,648 tonnes, averaging 12.14 g/t Au. Reserves slowly declined between 1942 and 1949 and fell rapidly thereafter.

No. 4 shaft was completed in July, 1949 at a depth of 3,480 ft.

1952: Rising costs eroded profits to a break-even point and mineralization reserves declined to a two-year supply.

Leads to a new high-grade mineralization were considered to be exhausted on the development levels, and the most favourable prospecting ground was considered to be at depth.

The last commercial crude arsenic shipment was made between 1951 and 1952 to Belgium.

1954: Seven underground drill holes totalling 4,000 ft were drilled between depths of 3,450 and 4,000 ft, and results reported in 1954 demonstrated continuity of the No. 1 vein, although gold values could not support shaft sinking or a continuing operation.

1956-1958: O'Brien mine was closed. Surface facilities were cleaned to recover accumulated gold. The mine closed because of rising operating costs, lower grades, and the fixed price of gold at US$35/oz.

The O'Brien mine produced 6,313 tonnes of crude arsenic, 5,176 tonnes of which were sold. The remaining stockpile, which contained an estimated 1,150 tonnes of crude arsenic (arsenic trioxide), was stored in 8,938 barrels west of the No. 3 shaft on the 1500 ft level in the 15-G-West and 15-F-West drifts. Drift entries were sealed with concrete plugs about 1.2 m wide. The mine was flooded thereafter.

Between 1926 and 1956, 35,700 ft of underground development (mainly drifts, but also crosscuts, raises, and shafts) was constructed, and close to 6,200 m from surface and over 54,000 m underground was drilled. Overall production shows 587,120 oz of gold, produced from 1,197,147 tonnes milled with an average grade of 15.26 g/t Au. Recoveries averaged 96.0%. Historical production data are summarized in Table 6-1 in Section 6.3.

6.1.1.2 Darius Gold Mines Inc – 1969 to 1981

Abandoned since its closure in 1956, the O'Brien mine was acquired by A. N. Ferris and the property was renamed the "Ferris property." The property was re-evaluated, and surface stripping was carried out.

1972-1973: Darius Gold Mines Inc. was created and initiated an exploration and reassessment program at the former O'Brien mine.

A brief study on the tailings from the former O'Brien mine was carried out to ascertain the form of the contained gold and the amount that might be recoverable by further treatment.

Darius also dewatered the mine down to the Level 9 (1400 ft) and began a sampling program that lasted for several years.

1974: Darius carried out an underground bulk sampling program composed of many samples.

A dump ramp was built on the west side of the headframe, and one mucking machine and four 1-ton cars were purchased. Track was installed from the cage, and carloads were dumped one at a time directly into the truck. Between February and April 1974, 171 tonnes was extracted from the 375 ft level in the F and G veins.

1975: At the end of February 1975, 2,500 tonnes averaging 3.14 g/t Au were extracted during the bulk sampling program at the O'Brien mine.

A total of 2,406 linear feet of drift backs were sampled on the 275, 500, 625, 750 and 875 ft levels, and 523 ft of drifting and 422 ft of raising (three raises) were completed.

Seventeen underground holes (74-1 to 74-11, D-16, D-18, D-19, D-21, D-24, and D-25) were drilled for 2,985 ft.

1976: Thirty-two underground holes were drilled for 4,275 ft.

Following the underground drilling campaign, Robert E. Schaaf carried out a mineral inventory compilation on veins No. 1 S, No. 1 N, F9 and H-4-14.

1977: In October 1977, Goldfield Mining Consolidated acquired a 51% interest in the property for US$4,635,000 to commit to making the mine operational and exploring adjacent properties.

Additional restoration work and bulk sampling were performed. Among other projects, Darius built a mill with a capacity of 200 short tons per day, which could be increased to 500 short tons per day. The mill was completed on July 1, 1978, for about C$3,000,000.

1978: A total of 11,018 tonnes grading 1.07 g/t Au were milled in the new mill. The source of the mineralization was primarily from drifting.

1979: Surface drilling, comprising 24 holes for 3,979.8 m, was performed to test unexplored areas.

A total of 36,106 tonnes grading 3.04 g/t Au were milled in the new mill. The mineralization was produced from small stopes.

1980: Surface drilling, comprising 33 holes for 4,995.5 m, was carried out to test unexplored areas.

A total of 33,706 tonnes grading 3.73 g/t Au were milled in the new mill. The mineralization was produced from small stopes.

1981: The mine was closed at the end of August, and the mill ceased activity in October.

Between 1974 and 1981, 10,852.4 oz Au were produced from 128,373 tonnes milled, averaging 2.63 g/t Au. It is estimated that 47,587 tonnes averaging 2.79 g/t Au were milled on site. Recoveries averaged 70.0%. Production data is summarized in Table 6-2 in Section 6.3.

During the year, Darius believed it had a buyer for the crude arsenic that had been stored on the 1500 ft level since 1956. The concrete wall from the 1500 ft level was bolted. Later, the potential buyer withdrew.

6.1.1.3 Sulpetro Minerals / Novamin Resources / Breakwater Resources – 1981 to 1986

1981: In December, Sulpetro Minerals (Sulpetro) bought the property for C$2,800,000 for the purpose of treating mineralization from its adjoining Kewagama mine to the east (known as the Kewagama Division). The O'Brien property was renamed O'Brien Division.

Sulpetro tried unsuccessfully to find other buyers for the crude arsenic stored on the 1500 ft level.

1982: Some Kewagama material was processed in the O'Brien mill, while efforts were being deployed, including rebuilding the gravity circuit, to improve the gold recovery rate.

The mine went on standby and was allowed to flood to a 1,500 ft depth after November 1982.

1985: In April, new waterproof and reinforced concrete plugs (2.3 m wide) were installed at the entrance of the underground opening containing the crude arsenic.

In August, the Ministry allowed the mine to be flooded.

The surface infrastructure was kept, but all electrical equipment was removed from the No. 2 shaft.

1985-1986: In January 1986, Sulpetro was reorganized into Novamin.

Magnetometric (49.5 line-km) and very low frequency (VLF) electromagnetic (49.5 line-km) surveys were conducted over the property; this included a limited number of induced polarization (IP) (4.9 line-km) surveys.

1986-1987: Surface drilling was done in the No. 3 shaft area, extending the No. 2 and No. 4 vein structures towards the New Alger property boundary. A first campaign of eight drill holes totalling 1,999.8 m was drilled, followed by an additional eight holes totalling 2,185 m during a second campaign.

Zone 36 East, which is series of gold-bearing quartz echelon veins similar in nature and character to the mined structures of the O'Brien mine, was discovered. This area now hosts most of the current mineral resources.

1988: Novamin drilled eight additional holes on Zone 36 East for 2,198.5 m.

1989: Breakwater completed the acquisition of Novamin and continued drilling on the property. A total of 24 holes were drilled on Zone 36 East, totalling 7,832.1 m.

1992: Negotiations began between Breakwater and Radisson.

1994: On October 24, 1994, a deal was signed whereby Radisson could earn a 50% interest in the O'Brien property.

1998: Following exploration activities on site by Radisson, Radisson purchased 100% of the rights to the O'Brien property as well as all the infrastructure. It also acquired the adjacent Kewagama property.

6.1.2 Former Kewagama Property

6.1.2.1 Kewagama Gold Mines Ltd – 1928 to 1980

1928: Activity on the property commenced in 1928 with trenching and diamond drilling by Cartier Malartic Gold Mines.

1931: Eight of the present claims were acquired by Canadian Gold Operators Ltd. (Canadian Gold).

1932-1933: Considerable development was carried out by Canadian Gold, including diamond drilling (10 holes aggregating about 5,000 ft), sinking a two-compartment shaft to 125 ft, and carrying out approximately 1,500 ft of lateral work (drifts and crosscuts) at the 125 ft level. The shaft is 4,800 ft east of the O'Brien No. 2 shaft. The work indicated that the geological and structural conditions of the Kewagama property are similar to those of the O'Brien property.

Exploration revealed the presence of several gold-bearing quartz veins. Four veins (Nos. 1, 6, 7, and 8) were developed and investigated. Although the limited amount of drifting done on these veins did not establish ore shoots, it did reveal encouraging gold values.

The property was shut down in April 1933.

1934-1935: The underground workings were flooded.

1936: Kewagama Gold was created from the acquisition of Canadian Gold by Ventures Ltd.

1937-1938: The shaft was deepened to 524 ft with three compartments, and new levels were established at 250, 375, and 500 ft. A winze was developed 400 ft east of the shaft from the 500 ft level to the 700 ft level, and new sublevels were established at 550, 600, and 700 ft. Lateral developments were carried out on four levels from the shaft, and three sublevels from the winze. A total of 12,600 ft of drilling was drilled.

Although interesting gold assays were obtained from the material encountered, especially on the lower levels, commercial grade mineralization was not present in sufficient quantity to ensure a profitable venture.

1939: All operations were suspended in early 1939 due to the restrictions on gold mining with the outbreak of World War II.

1940: A total of 2,470 tonnes of stockpiled development mineralization, having an average grade of 9.9 g/t Au, was processed at the neighbouring Thompson-Cadillac Mill, from which 790.7 oz of gold were recovered.

1947: A magnetometer survey was completed over the Piché Group (Cadillac Shear Zone) and the Cadillac Formation north of the shear to determine whether the gold mineralization of the neighbouring Wood-Central and Pandora properties to the east continued onto the Kewagama property.

1964: Falconbridge Nickel Mines, the successor to Ventures Ltd, initiated a surface drilling program in 1964, partially for assessment work. Four holes totalling 981.7 ft were drilled approximately 50 ft apart to trace the upward extension of the Winze Zone that had been partially developed from the 500 ft level from 1937 to 1939.

1973-1974: Surface exploration was renewed by Kewagama Gold under the direction of Derry, Michener & Booth, Geological Consultants. A program of overburden (basal till) sampling for gold was conducted along the 2,800 ft strike length of the favourable Cadillac Belt of rocks extending east of the 1964 Falconbridge drill holes and north of the Cadillac Shear, to explore the iron formation environment that had been productive on the neighbouring Wood-Central and Pandora properties to the east.

Diamond drilling 400 ft east of the shaft consisting of 13 holes for 3,149 ft followed, and the results were considered encouraging and worthy of underground investigation.

1976: Management control of the company was acquired by A. N. Ferris of Cadillac, Québec.

1977: The mine site was cleared of bush and levelled.

1978: A temporary mining plant/service building, hoist room, headframe, mine dry, and machine shop were constructed.

1979-1980: The hoist was operative in early 1979, and the mine was dewatered and secured in May. Underground workings were inspected, followed by sampling and planning. The company rehabilitated and sank approximately 200 ft deeper, cut a station on the 700 ft level, and drove 800 ft of drift.

On November 12, 1980, an agreement was signed with St-Joseph Explorations Ltd. (later Sulpetro Minerals Ltd.). Considering strong gold prices and the excellent outlook, St-Joseph Explorations decided to continue exploring the Kewagama property.

6.1.2.2 Sulpetro Minerals / Novamin Resources / Breakwater Resources – 1981 to 1998

1981: Sulpetro deepened the shaft to 1,150 ft. Ore and waste passes were driven from the Level 7 to Level 4. Thirty-one surface holes were drilled for 4,789.8 m. Geophysical surveys (magnetic, VLF, IP) were carried out on the Kewagama property. Five holes were drilled to test a coincident magnetic and IP anomaly between lines 3+20E and 4+00E. The result was the discovery of the West IP Zone.

1982: Development continued along Levels 6 and 7, and the Winze Zone was mined out, producing 11,340 tonnes averaging 3.03 g/t Au. Production also continued from the Q, R, and S veins until operations were suspended in November 1982.

1988: Four surface diamond drill holes totalling 1,005.8 m were drilled by Novamin to test the Piché Group "mine horizon" lithologies between the O'Brien and Kewagama property boundaries at the westernmost end of the 500 ft level in the Kewagama underground workings. These holes intersected favourable lithologies that could host ore-grade gold mineralization laterally and at depth.

1994: On July 25, the wooden Kewagama shaft was struck by lightning and burned down.

1995: Breakwater re-activated exploration activities on the Kewagama property and established new surveyed gridlines spaced 100 m apart, with a cumulative length of 16 km.

As a first step, historical work was compiled to better understand the geological setting and assess the economic potential of the Kewagama property. Consequently, geological mapping was conducted to study the lithological and structural controls on gold distribution and to build a geological map of the Kewagama property.

1999: Radisson became 100% owner of the Kewagama property in 1999.

6.1.3 Former Thompson-Cadillac Mine

1924: The property was first staked by E. J. Thompson during the same gold rush that discovered the O'Brien mine.

1920s: The mine opened and production commenced under Thompson Cadillac Mines Ltd.

1934: Thompson Cadillac Mines Ltd. was delisted, and the Ontario Securities Commission investigated the management. Ownership was changed to the Thompson Cadillac Mining Corporation.

1939: The Thompson Cadillac Mining Corporation declared bankruptcy. The mine ceased production after 21,000 oz of gold was mined.

2006: A joint venture was established between Cadillac Ventures and Renforth Resources Inc. (Renforth).

2013: Renforth acquired full ownership of the New Alger property, which included the Thompson-Cadillac mine plus a land package to the south of Highway 117.

2020: Renforth completed a mineral resource estimate at Thompson-Cadillac which was referred to as "New Alger." Renforth sold the New Alger property, including the former Thompson-Cadillac mine, to Radisson for C$4,340,000 in securities and cash payments.

6.2 Historical and Previous Resource Estimates

These resources listed in this section are historical or previous in nature and should not be relied upon. It is unlikely that they conform to current NI 43-101 requirements or follow CIM (2014) Definition Standards, and they have not been verified to determine their relevance or reliability. They are included in this section for illustrative purposes only and should not be disclosed out of context. The QP has not completed sufficient work to classify the historical estimate as current mineral resources or mineral reserves, and the issuer is not treating the historical estimate as current mineral resources or mineral reserves.

6.2.1 Breakwater

In 1989, Breakwater estimated the Zone 36 East resource at 249,746 tonnes averaging 8.23 g/t Au using a cut-off grade of 3.4 g/t Au and totalling 66,071 oz. This inventory was developed using the following parameters: a 7.6 m lateral and 45.7 m vertical maximum zone of influence from each intersect; a grade-thickness cut-off of 3.4 g/t Au over 1.2 m, with combined individually cut grades diluted to 1.2 m if necessary, and zero values assigned to missing samples. High-grade capping was established at 34.3 g/t Au. Neither gold price nor exchange rate was mentioned in Breakwater's report.

6.2.2 Radisson

In 1994, Radisson estimated the Zone 36 East at 489,277 tonnes averaging 7.20 g/t Au using a cut-off grade of 3.4 g/t Au for a total of 113,260 oz. This inventory was developed using a 7.6 m and 45.7 m vertical maximum zone of influence from each pierce point. Individually cut assays were established at 34.3 g/t Au. Specific gravity was fixed at 2.67. A 3.4 g/t Au over 1.2 m true thickness cut-off was used. Neither gold price nor exchange rate was mentioned in Radisson's related report.

In 1996, Radisson re-estimated resources to 1,270,000 tonnes at an average grade of 6.9 g/t Au for a total of 281,740 oz. Of this total, 735,600 tonnes averaging 7.2 g/t Au for a total of 170,280 oz were in Zone 36 East. This inventory was developed using a 7.6 m and 45.7 m vertical maximum zone of influence from each pierce point. Assays were capped at 34.3 g/t Au. Specific gravity was fixed at 2.67. A 3.4 g/t Au over 1.2 m true thickness was used. Gold price and exchange rate were not reported.

In 1998, Radisson commissioned Roscoe Postle Associates Inc. (RPA) to update the gold resource estimate in Zone 36 East in the O'Brien mine. As at April 30, 1998, using a cut-off grade of 5.1 g/t Au, RPA estimated that indicated resources down to a depth of 610 m below surface amounted to 348,365 tonnes at a 9.9 g/t Au cut to 68.5 g/t Au (14.5 g/t Au uncut), for a total of 111,000 contained oz Au (162,000 oz Au uncut). Inferred resources to the same depth amounted to 15,422 tonnes at 18.6 g/t Au cut to 68.5 g/t Au (19.8 g/t Au uncut) for a total of 9,000 contained oz Au (10,000 oz Au uncut). The specific gravity was set at 2.67 g/cm^3. The price of gold was US$300/oz with a CAD:USD exchange rate of 1.444.

In 2007, Radisson once again commissioned RPA to update the gold resource estimate in Zone 36 East. Indicated resources were estimated at 251,295 tonnes at an average grade of 12.3 g/t Au, for a total of 97,000 oz Au. RPA estimated Inferred resources totalled 165,110 tonnes at an average grade of 9.9 g/t Au for a total of 54,000 oz Au. The resources were estimated using a conventional 2D longitudinal block resource estimation methodology; a horizontal thickness for Indicated resources ranging from 1.2 to 2.7 m with an average of 1.4 m; a gold price of US$575/oz Au; a USD:CAD exchange rate of 0.87; a gold recovery of 90%; a specific gravity of 2.67; and a selected capping level of 68.5 g/t Au.

In 2013, RPA again estimated the resources of Zone 36 East as follows: indicated resources of 508,032 tonnes at an average cut grade of 6.5 g/t Au for a total of 106,000 contained ounces, and inferred resources of 287,582 tonnes at an average cut grade of 7.29 g/t Au for a total of 67,000 contained ounces. The resources were estimated using a block model in GEMCOM software; a minimum horizontal width of approximately 1.8 m; a gold price of US$1,600/oz Au; a

USD:CAD exchange rate of 1.0; a gold recovery of 90%; a specific gravity of 2.67; and a selected capping level of 51.9 g/t Au.

In 2015, InnovExplo completed a mineral resource estimate on Zone 36 East and the Kewagama areas (Richard et al., 2015). The estimate was completed by Pierre-Luc Richard, P.Geo., M.Sc. and Alain Carrier, P.Geo., M.Sc., and the effective date of the estimate was April 10, 2015. It was estimated using a block model in GEMCOM software; a minimum true thickness of 1.5 m; a cut-off grade of 3.5 g/t Au (based on a gold price of US$1,200/oz; a USD:CAD exchange rate of 1.20; a processing recovery of 92.5% and mining dilution of 15%; a fixed density of 2.67; and high grade capping of 65 g/t Au for zones in the Western sector, 30 g/t Au for the Eastern sector, 3.5 g/t Au for the Western dilution zone, and 4.0 g/t Au for the Eastern dilution zone.

In 2018, InnovExplo completed a mineral resource estimate on the O'Brien Gold Project, which covered Zone 36 East, Vintage, and the Kewagama areas (Beausoleil, 2018). The 2018 mineral resource estimate was completed by Christine Beausoleil, P.Geo., with an effective date of March 20, 2018. InnovExplo estimated that the indicated resources amounted to 1,125,447 tonnes at an average grade of 6.45 g/t Au for a total of 233,491 oz Au, and inferred resources amounted to 1,157,021 tonnes at a grade of 5.22 g/t Au for 194,084 oz Au. Resources were estimated using GEMCOM block modelling software using the inverse distance squared (ID^2) interpolation method; a cut-off-grade of 3.5 g/t Au based on a gold price of US$1,300/oz Au; a USD:CAD exchange rate of 1.3; a processing recovery of 87.4%; a fixed density of 2.75; and high-grade capping of 30 g/t Au.

In 2019, Kenneth Williamson 3DGeo-Solution (KW3DS) completed a mineral resource estimate on the O'Brien Gold Project based on 14,000 m of new drilling and an updated litho-structural interpretation of the deposit (Williamson, 2019). The mineral resource estimate was completed by Kenneth Williamson, P.Geo., M.Sc., and the effective date of the estimate was July 15, 2019. KW3DS estimated that the indicated resources amounted to 649,700 tonnes at an average grade of 9.48 g/t Au for 289,400 oz Au, and inferred resources amounted to 617,400 tonnes at 7.31 g/t Au for 145,000 oz Au. It was estimated using a block model in GEOVIA GEMS using the ID^2 interpolation method; a cut-off grade of 5.0 g/t Au based on a gold price of US$1,350/oz Au; a USD:CAD exchange rate of 1.3; a processing recovery of 87.4%; a fixed density of 2.82; and a high-grade capping of 60 g/t Au.

6.3 Past Production

Historical gold production at the O'Brien Mine is presented in Tables 6-1 and Table 6-2. No production information is available for the Kewagama mine.

The Thompson-Cadillac mine produced 21,000 Au of gold from the 1920s to the 1930s.

Table 6-1: Gold Production from the O'Brien Mine, 1926-1957

Year	Mined (Hoist)	Milled (t)	Milled Grade (g/t Au)	Recovered (oz Au)	Development (t)	Development (g/t Au)	Stopes (t)	Stopes (g/t Au)
1926 - 1932	-	1,574	94.50	4,782.0	-	-	-	-
1933	-	13,481	10.97	4,755.0	-	-	-	-
1934	-	24,796	9.57	7,626.0	-	-	-	-
1935	-	26,662	6.07	5,200.9	-	-	-	-
1936	-	24,497	18.89	14,875.6	-	-	-	-
1937	-	33,897	33.84	36,879.5	-	-	-	-
1938	50,912	50,902	24.61	40,280.2	23,037	12.00	27,875	32.57
1939	52,516	61,286	19.05	37,538.7	22,606	7.89	29,711	34.59
1940	61,286	61,563	14.40	28,494.2	13,808	10.90	45,746	16.77
1941	62,757	62,730	12.52	25,257.4	3,468	7.34	53,534	14.40
1942	63,066	63,086	12.79	25,947.0	9,306	11.38	53,760	13.78
1943	62,882	62,701	13.04	26,285.2	3,346	8.64	59,536	13.92
1944	50,552	50,652	16.00	26,049.0	2,875	10.80	47,677	17.11
1945	44,810	44,918	17.98	25,964.2	6,718	14.47	38,092	19.34
1946	45,748	45,784	15.54	22,868.2	4,129	9.60	41,620	16.80
1947	48,053	48,048	14.95	23,092.4	3,200	9.02	44,853	16.05
1948	49,600	49,699	17.09	27,308.5	6,173	7.89	43,427	19.27
1949	52,890	52,702	15.89	26,920.5	3,771	9.02	49,119	17.18
1950	60,550	60,686	14.49	28,266.9	5,197	8.88	55,353	15.77
1951	59,139	59,139	14.66	27,870.9	3,509	8.13	55,630	15.77
1952	61,393	61,393	13.02	25,705.7	2,631	11.69	58,762	13.71
1953	58,088	58,088	12.84	23,973.6	1,420	8.88	56,668	13.44
1954	62,879	62,879	12.74	25,752.5	1,761	10.22	61,118	13.37
1955	63,616	63,616	11.37	23,251.7	1,328	8.23	62,287	11.97
1956	52,012	52,370	11.94	20,099.6	351	7.61	51,661	11.04
1957	-	-	-	2,074.4	-	-	-	-
Total	**1,062,749**	**1,197,149**	**15.26**	**587,120**	**11,8634**	**10.07**	**936,429**	**16.17**

Table 6-2: Gold Production from the O'Brien Mine, 1974-1981

Year	Milled (t)	Milled Grade (g/t Au)	Recovered (oz Au)
1974-1975	2,500	3.14	252.4
1978	11,266	0.78	282.6
1979	36,114	2.48	2,875.7
1980	33,388	3.15	3,381.2
1981	45,105*	2.79*	4,060.4[1]
Total	**128,373**	**2.63**	**10,852.3**

Note: * Estimated data. Source: Beausoleil, 2018.

7 GEOLOGICAL SETTING AND MINERALIZATION

This section is a slightly modified version of the regional geology description provided in the technical report by Beausoleil (2018), and updated by Williamson (2019) and references therein. Th QP has reviewed and compared Beausoleil's and Williamson's geological description to other such accounts in publicly available documents and considers it accurate to the best of its knowledge.

7.1 Regional Geology

7.1.1 Archean Superior Province

The Archean Superior Province (Figure 7-1) forms the core of the North American continent and is surrounded by provinces of Paleoproterozoic age to the west, north, and east, and the Grenville Province of Mesoproterozoic age to the southeast. Tectonic stability has prevailed since approximately 2.6 Ga in large parts of the Superior Province. Proterozoic and younger activity is limited to rifting of the margins, emplacement of numerous mafic dyke swarms, compressional reactivation, large-scale rotation at approximately 1.9 Ga, and failed rifting at approximately 1.1 Ga. Except for the northwest and northeast Superior margins that were pervasively deformed and metamorphosed at 1.9 to 1.8 Ga, the craton has escaped ductile deformation.

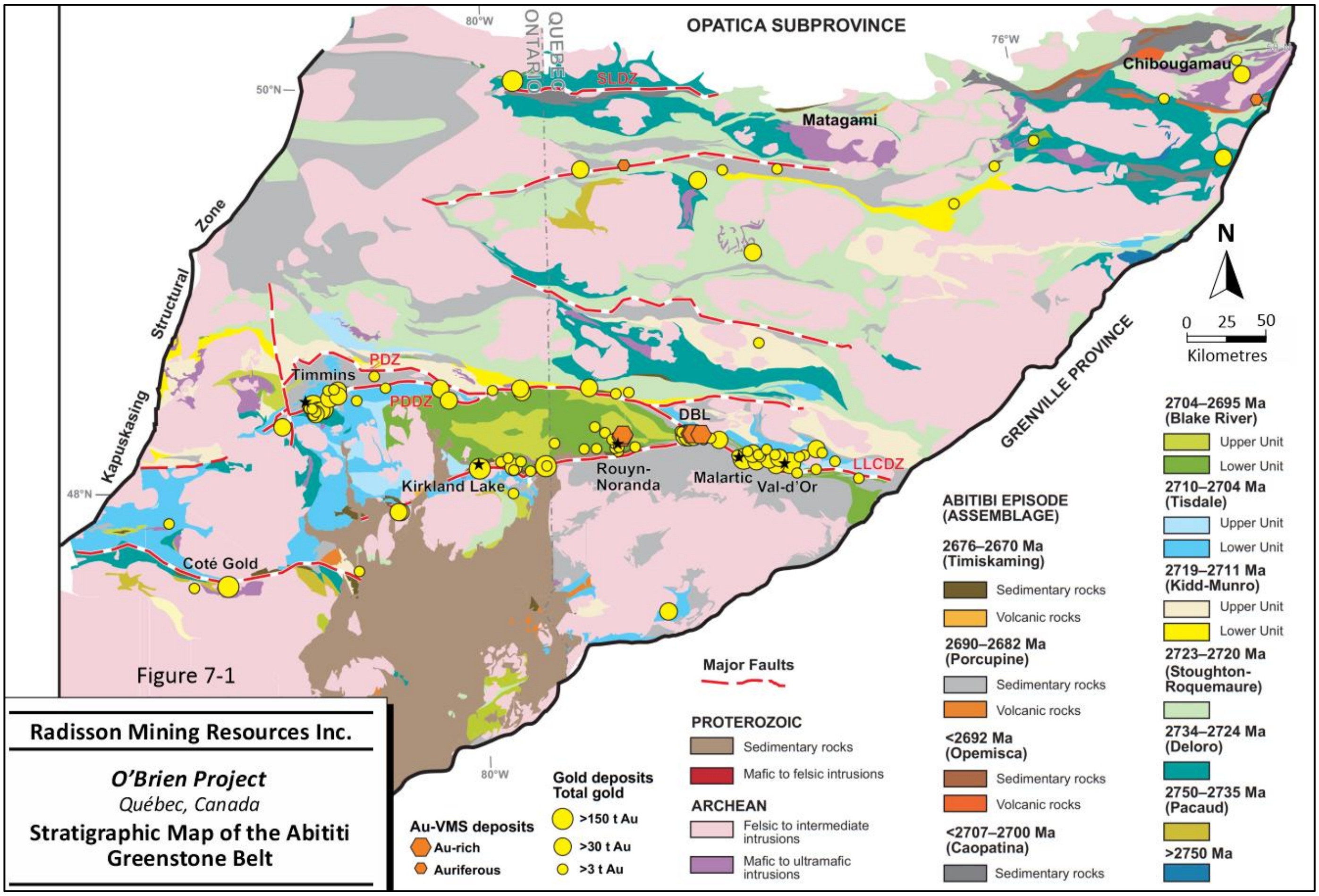
A first-order feature of the Superior Province is its linear subprovinces, or "terranes," of distinctive lithological and structural character, accentuated by subparallel boundary faults. Trends are generally east-west in the south, west-northwest in the northwest, and northwest in the northeast. The term "terrane" is used in the sense of a geological domain with a distinct geological history prior to its amalgamation into the Superior Province during the 2.72 Ga to 2.68 Ga assembly events, and a "superterrane" shows evidence for internal amalgamation of terranes prior to the Neoarchean assembly. "Domains" are defined as distinct regions within a terrane or superterrane.

7.1.2 Abitibi Subprovince

The Abitibi Subprovince, commonly designated as the Abitibi Greenstone Belt, is in the southern portion of the Superior Province (Figure 7-1). It is bounded to the west by the Kapuskasing Structural Zone and to the east by the Grenville Province. To the north, the Abitibi Subprovince is in structural contact with the plutonic Opatica Subprovince. The southern boundary of the Abitibi greenstone belt is marked by the Cadillac-Larder Lake Fault Zone (CLLFZ), a major structural break marking the contact with the younger metasedimentary rocks of the Pontiac Subprovince.

Thurston et al. (2008) presented the first geochronologically constrained stratigraphic and/or lithotectonic map. According to Thurston et al. (2008), Superior Province greenstone belts consist of mainly volcanic units unconformably overlain by largely sedimentary Timiskaming-style assemblages, and field and geochronological data indicate that the Abitibi Greenstone Belt developed autochthonously.

Figure 7-1: Stratigraphic Map of the Abitibi Greenstone Belt



Source: Modified from Gosselin & Dubé (2005) and Mercier-Langevin et al. (2011).

As suggested by Thurston et al. (2008), the Abitibi Greenstone Belt can be subdivided into seven discrete volcanic stratigraphic episodes based on groupings of numerous U-Pb zircon ages. These seven volcanic episodes, listed from oldest to youngest, are as follows:

1. Pre-2,750 Ma volcanic episode
2. Pacaud Assemblage (2,750-2,735 Ma)
3. Deloro Assemblage (2,734-2,724 Ma)
4. Stoughton-Roquemaure Assemblage (2,723-2,720 Ma)
5. Kidd-Munro Assemblage (2,719-2,711 Ma)
6. Tisdale Assemblage (2,710-2,704 Ma)
7. Blake River Assemblage (2,704-2,695 Ma).

The Abitibi Subprovince (or Abitibi Greenstone Belt) is composed of east-trending synclines largely composed of volcanic rocks and intervening domes cored by synvolcanic and/or syntectonic plutonic rocks (gabbro-diorite, tonalite and granite) alternating with east-trending bands of turbiditic wackes (MERQ-OGS, 1984; Ayer et al., 2002a; Daigneault et al., 2004; Goutier and Melançon, 2007). Most of the volcanic and sedimentary strata dip vertically and are generally separated by abrupt, east-trending and southeast-trending faults with variable dip. Some of these faults display evidence for overprinting deformation events including early thrusting, later strike-slip, and extension events (Goutier, 1997; Benn and Peschler, 2005; Bateman et al., 2008).

Two ages of unconformable successor basins occur: early, widely distributed Porcupine-style basins of fine-grained clastic rocks, followed by Timiskaming-style basins of coarser clastic and minor volcanic rocks which are largely proximal to major strike-slip faults, such as the Porcupine-Destor Fault Zone, the Cadillac–Larder Lake Fault Zone (CLLFZ), and other similar faults in the northern Abitibi Greenstone Belt (Ayer et al., 2002a; Goutier and Melançon, 2007).

In addition, the Abitibi Greenstone Belt is cut by numerous late-tectonic plutons from syenite and gabbro to granite with lesser dykes of lamprophyre and carbonatite. The metamorphic grade in the greenstone belt displays greenschist to sub-greenschist facies (Jolly, 1978; Powell et al., 1993; Dimroth et al., 1983; Benn et al., 1994) except around plutons where amphibolite grade prevails (Joly, 1978).

The Abitibi Greenstone Belt is known for hosting significant number of gold and base metal deposits, including the giant Kidd Creek massive sulphide deposit (Hannington et al., 1999) and the large gold camps of Ontario and Québec (Robert and Poulsen, 1997; Poulsen et al., 2000).

The O'Brien Gold Project is located along the CLLFZ and is one of the numerous gold deposits that are associated with this major structure and subsidiary faults.

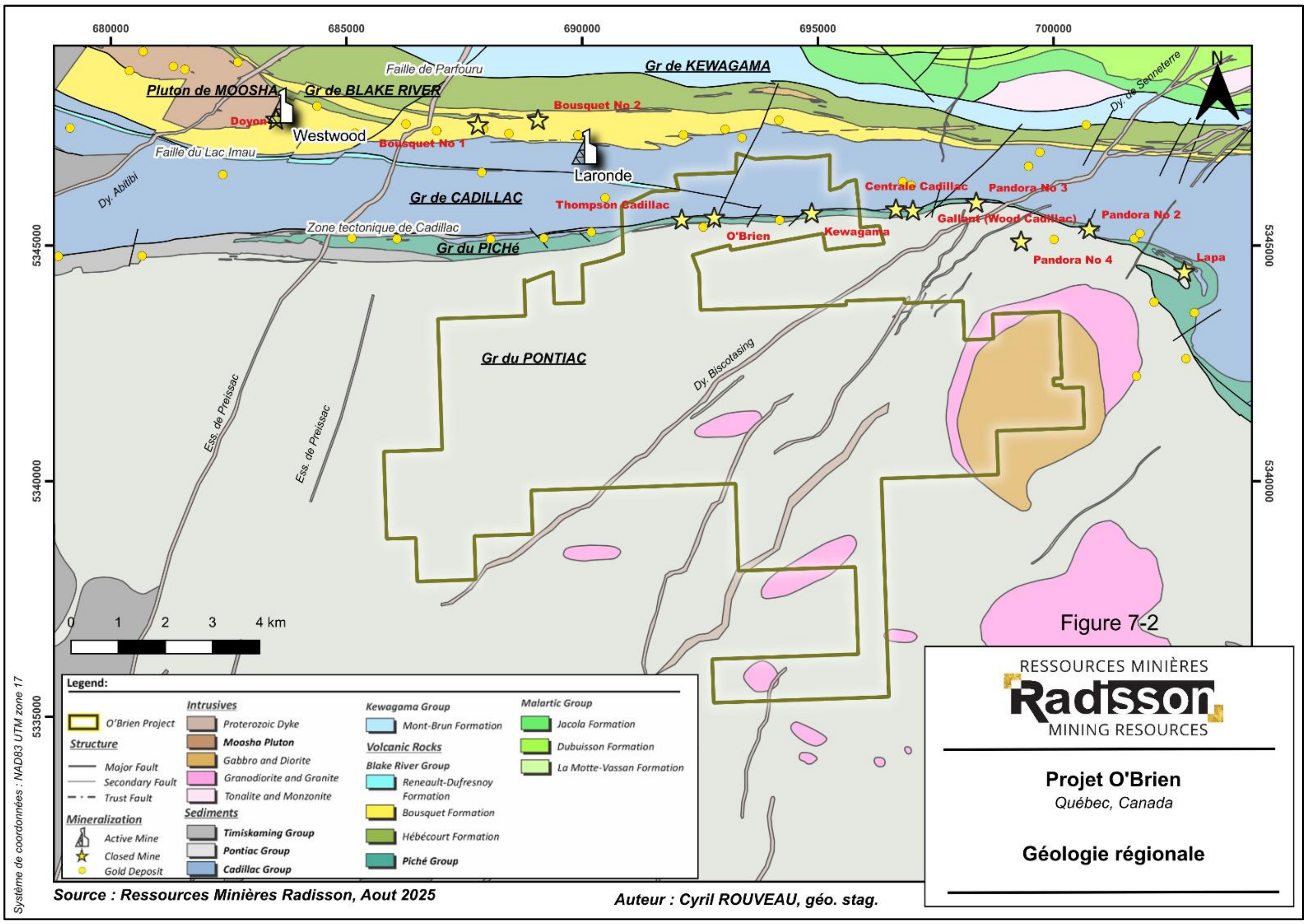
7.1.3 Cadillac Mining Group

The Cadillac Mining Camp covers a 25 km long stretch of the CLLFZ, from the former Mouska mine in the west to the former Lapa-Cadillac mine in the east. Within the CMC, the CLLFZ runs along an east-west axis and separates the Pontiac metasedimentary Subprovince to the south from the Abitibi volcano-sedimentary Subprovince to the north. The CMC is underlain by rocks of the Southern Volcanic Zone of the Abitibi Subprovince intruded by Proterozoic diabase dykes. From north to south, the following six major lithological units (groups) are observed: Malartic, Kewagama, Blake River, Cadillac, Piché, and Pontiac (Figure 7-2).

The Malartic Group is composed of ultramafic volcanic rocks (komatiites) and tholeiitic basalts (Trudel et al., 1992). The Kewagama Group contains wackes and pelitic rocks. The Blake River Group comprises the Hebecourt and Bousquet formations. The Hebecourt Formation is composed of massive and pillowed basalts, gabbro sills, and rhyolites of tholeiitic affinity. According to Lafrance et al. (2003), the Bousquet Formation includes a lower member and an upper member. The lower member is composed of an intermediate scoriaceous tuff; mafic, intermediate, and felsic volcanic rocks; and felsic and mafic subvolcanic intrusions. The upper member consists of massive felsic volcanic rocks and volcaniclastic units. Rocks of the lower member are tholeiitic to transitional, whereas those of the upper member show a transitional to calc-alkaline affinity (Lafrance et al., 2003). The Cadillac Group is composed of wackes, pelitic schists with bands of polymictic conglomerate and iron formation.

In the Cadillac area, the Piché Group is composed of volcanic rocks (tholeiitic basalts, porphyritic andesites analogous to the QFP sill, and calc-alkaline block tuffs) interbedded with conglomerates, wackes, graphitic schists, and pyritic cherts. Most of the mineralization in the southern part of the Cadillac mining camp are hosted in rocks of the Piché Group, which forms a thin band several tens of kilometres long that follows the trace of the CLLFZ (Figure 7-2).

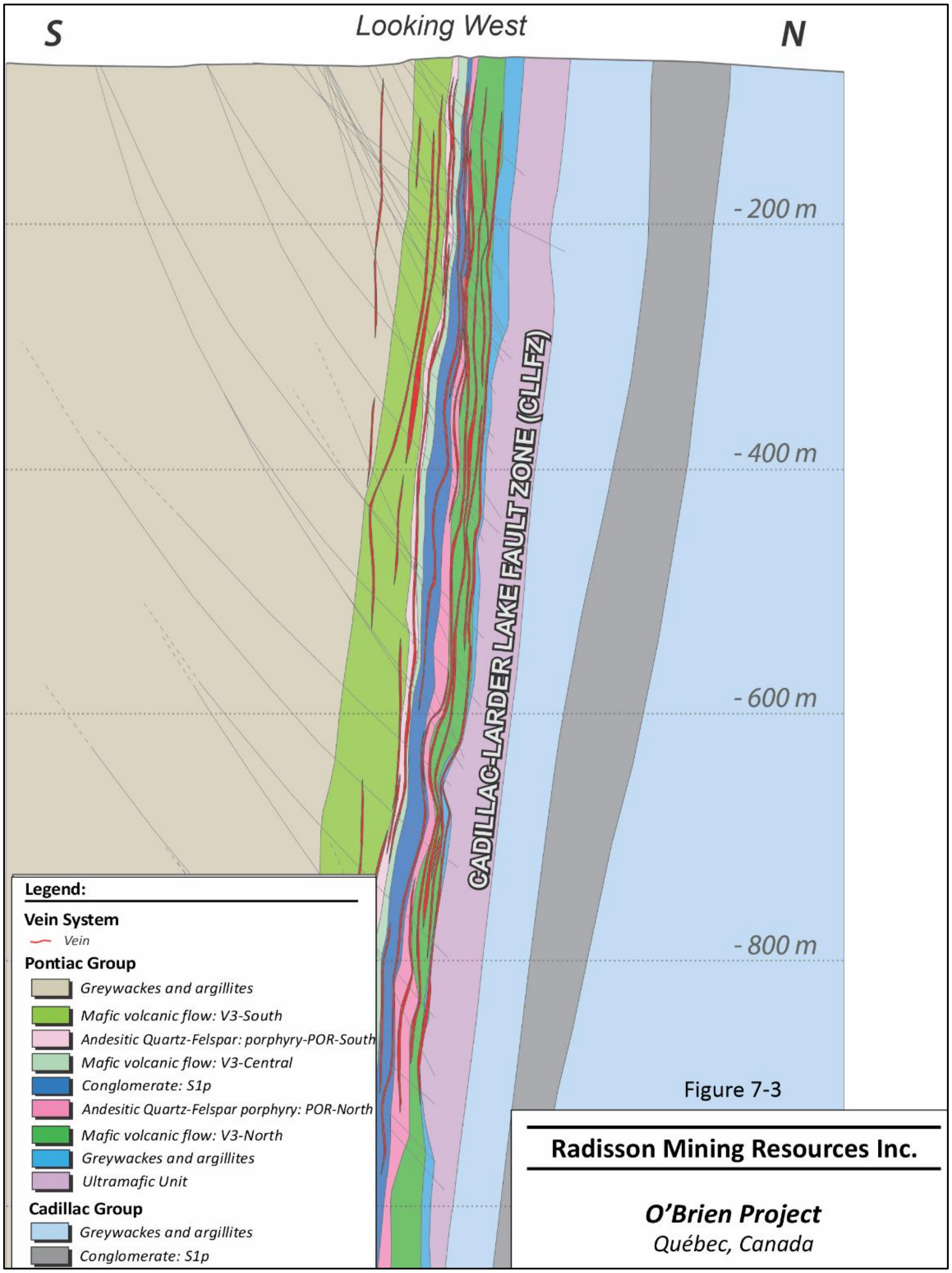
Figure 7-2: Regional Geology of the Cadillac Mining Camp



Source: Radisson (2025).

Sedimentary rocks, mainly wackes, of the Pontiac Group lie south of the CLLFZ. Volcanic and sedimentary rocks in the Cadillac area form a series of east-west-trending steeply dipping monoclonal panels. Volcanic and sedimentary sequences are separated by longitudinal faults parallel to lithological contacts such as the CLLFZ and Lac Imau faults (Figure 7-3).

Figure 7-3: Geology Section



Source: SLR (2023).

Intrusive rocks in the Cadillac area include mafic sills (gabbro and diorite) occurring in the Blake River and Piché groups, the synvolcanic Mooshla Pluton, composed of gabbro, quartz diorite, tonalite and trondhjemite, as well as north-south and northeast-southwest-trending Proterozoic diabase dykes.

North of the CLLFZ, regional metamorphism ranges from the greenschist facies to the upper greenschist facies, but the metamorphic grade increases south of the fault to reach the amphibolite facies.

7.2 Property Geology

This section is a slightly modified version of the property geology description provided in the technical report by Beausoleil (2018), updated by Williamson (2019) and references therein. The QP has reviewed and compared Beausoleil's and Williamson's geological description to other such accounts in publicly available documents and consider it to be an accurate description.

The project straddles the Piché Group volcanic rocks and CLLFZ that separate Pontiac Group metasedimentary rocks to the south from Cadillac Group metasedimentary rocks to the north.

Across the project, the CLLFZ shows a general east-west strike and dips steeply south at approximately 85°. On the property, the CLLFZ consists mainly of chlorite-talc-carbonate ultramafic schist, and ranges in thickness from 30 to 100 m in the mine area and narrows significantly to about 12 m wide to the east of Zone 36 East. The CLLFZ is in places closely associated to the Piché Group-Cadillac Group contact, but in most places, the fault is hosted by sedimentary rocks of the Cadillac Group (argillites, greywackes and, to a lesser extent, chert).

Most lithological contacts are sub-parallel to the CLLFZ. The main lithologies are described in the following subsections.

7.2.1 Cadillac Group

Found to the north, the Cadillac Group metasedimentary rocks are in the footwall of the CLLFZ and most of the mineralized zones; hence, most of the diamond drill holes did not intersect the Cadillac Group rocks. The limited drilling shows the presence of argillite, greywacke, some pebble conglomerate-like units, and some iron formation.

7.2.2 Piché Group

The Piché Group is a relatively thin band of interlayered mafic volcanic rocks, conglomerates, and porphyric andesitic sills. From north to south, the Piché Group stratigraphy is divided into the following units:

- northern volcanics: tuff and foliated basalts (with small quantities of argillite, greywacke, chert and massive to variably porphyritic basalt flows)
- northern porphyric andesitic sills
- polygenic matrix supported conglomerate ("mine conglomerate")
- central volcanics: tuff and foliated basalt

- southern porphyric andesitic sills
- southern volcanics: massive to well-foliated, locally pillowed basalts.

All the above lithologies generally strike east-west with more pronounced flexures locally. Schistosity is more developed in the central and northern volcanic units than the southern unit.

With a few significant exceptions like the Vintage Zone, the Piché Group is host of most of the gold mineralization occurrences on the property.

7.2.2.1 Andesite Porphyry

Previously referred to as the QFP unit, the southern and northern porphyric andesite sills are much alike. They are characterized by generally sharp transposed contacts, abundant feldspar phenocrysts ranging in size from 0.1 to 0.5 cm, and range in colour from greyish to buff-beige, set in an aphanitic to fine-grained matrix of intermediate composition. In general, the porphyric andesitic sills are intensely sheared and show a more or less brownish biotite and chlorite alteration. The porphyric andesitic sills are continuous horizontally and vertically across the whole property and are useful stratigraphic marker horizons. The north and south porphyric andesitic sills are thicker in the vicinity of the O'Brien mine.

7.2.2.2 Conglomerate

The O'Brien mine conglomerate is represented in Zone 36 East area by well-bedded greywacke and argillite with the sporadic presence (2% to 5%) of greyish granitic pebbles, greenish volcanics elongated pebbles and other components. The pebbles tend to be somewhat flattened, consistent with north-south compression. The conglomerate unit is another useful marker horizon.

7.2.2.3 Volcanic Rocks

The volcanic rocks consist mainly of mafic tuffs and flows. The volcanic rocks generally have tholeiitic signatures (Trudel et al., 1992). In general, the flows are fine-grained and exhibit greenschist facies mineral assemblages. The tuffs are of mafic composition and are abundant. The tuffs can be finely bedded to very schistose and may be the expression of deformed mafic flows. Locally present are massive to pillowed, fine-grained basalt or lesser amounts of gabbro and amphibolites.

7.2.2.4 Graphitic Schist and Argillite

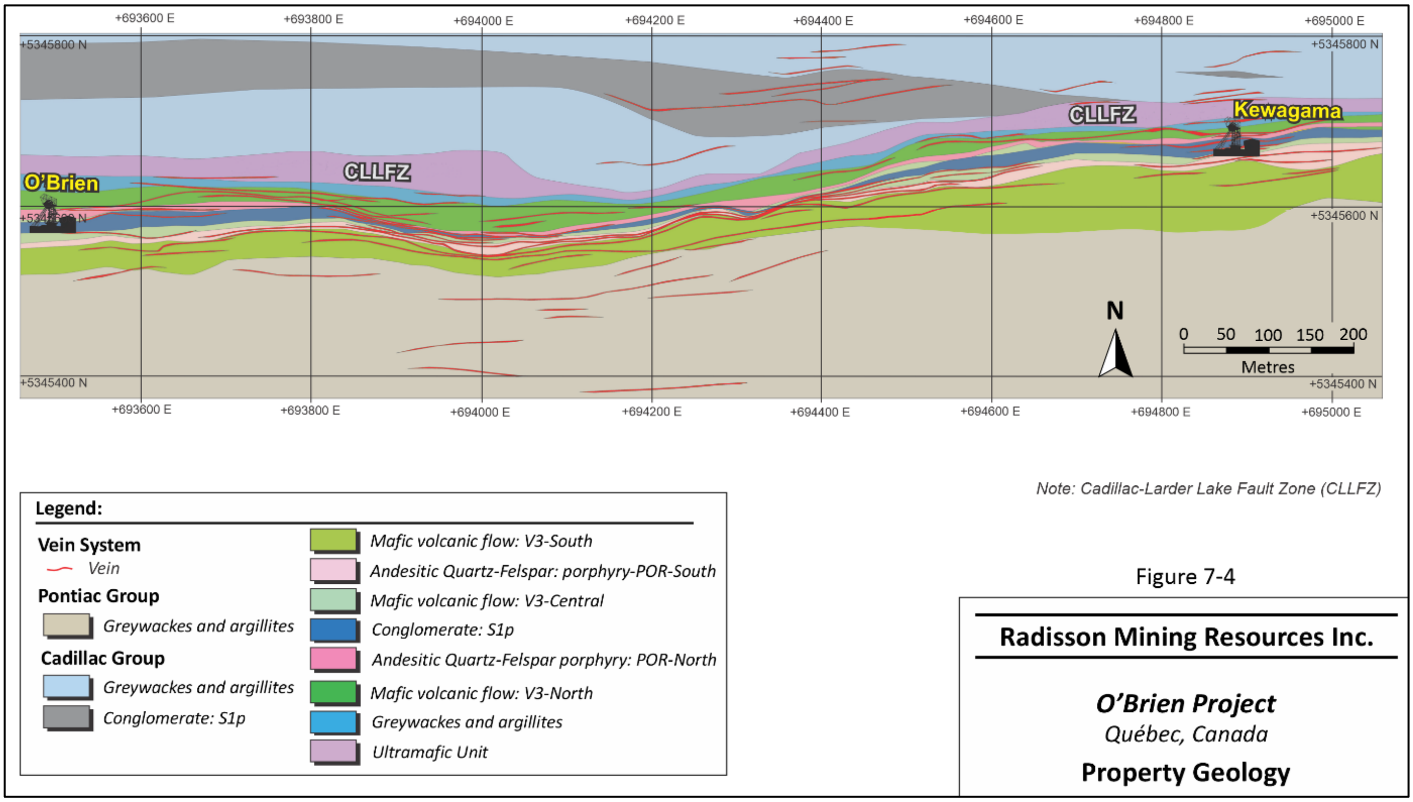
In places, thin layers of graphitic schist and argillite are hosted within the volcanics. These are highly sheared and deformed, characterized by tight folding, and often display breccias or slickensides with graphite. Pyrite is abundant, finely laminated and deformed.

7.2.3 Pontiac Group

The metasedimentary rocks of the Pontiac Group consist mainly of greywacke and some argillite, which is sometimes graphitic. In general, the sediments are well stratified. Some zones display weak biotitic alteration or chloritization.

Figure 7-4 illustrates the geology of the property.

Figure 7-4: Property Geology



Source: SLR (2023).

7.3 Mineralization

The following description of mineralization is mostly modified and summarized from Evans (2007), Beausoleil (2018) and Williamson (2019), and retains the references therein.

7.3.1 O'Brien Mine

Gold production at the O'Brien mine came from a few quartz veins mostly hosted by the O'Brien Mine conglomerate and the northern QFP dyke. Approximately 95% of the O'Brien ore came from three veins (No. 1, No. 4, No. 9 or "F") in the eastern part of the mine. The veins contained high-grade shoots that occasionally yielded considerable amounts of visible gold. The main veins generally strike from 083° to 098°, and dip steeply to the south (-84° to -90°). The stopes average 0.75 to 0.90 m wide. Gold mineralization extends vertically down to at least the 3450 ft level.

7.3.1.1 No. 1 Vein

The No. 1 vein was the most productive in terms of tonnage and occurs mainly in the conglomerate. This vein comprises No. 1 vein NE-SW (080° to 090° azimuth) and No. 1 vein NW-SE (090° to 095° azimuth).

No. 1 vein NE-SW extends from surface to at least the 3000 ft level and is over 500 ft in strike length. The richest and most productive portion of this vein was from an ore shoot 15 to 60 m long that plunges approximately 85° to the east from about the 750 ft level down to at least the 3000 ft level, at its intersection with vein No. 1 NW-SE, at the conglomerate hangingwall contact. A second moderate-grade shoot, about 15 to 45 m long, plunges about 60° to the east from about the 1000 ft level to the 2500 ft level.

Vein No. 1 NW-SE extends from the ~750 ft level to at least the 3450 ft level, and ranges in horizontal length from approximately 15 to 180 m. Higher grade shoots plunging about 85° to the east seem to be controlled by vein intersections and vein folds. Both veins average 30 cm thick (Mills, 1950).

7.3.1.2 No. 4 Vein

The No. 4 vein is spatially associated with the north porphyric andesitic sills. It extends from surface down to at least the 3450 ft level and has a 1,000 ft strike length. It averages 30 cm in thickness (Blais, 1954). Approximately 50% of the gold produced came from this vein. This was due to an exceptionally high-grade ore shoot, measuring only 9 to 15 m horizontally, but extending for 190 m from the 500 ft level down to the 1125 ft level.

7.3.1.3 No. 9 Vein

The No. 9 vein is located in the northern greywacke and volcanic units. This brown vein is rich in biotite and arsenopyrite. It is also wider than the others. The stopes were rarely less than 1.2 m wide and could reach 6 m in certain folded zones where visible gold was common. It was mined out from the 1250 ft level down to the 1375 ft level along a horizontal length of about 50 m.

7.3.2 Zone 36 East Area

Zone 36 makes up part of East O'Brien and hosts most of the current mineral resources. Within the area, the main mineralized structures (veins) are generally narrow, ranging in true thickness from several centimetres up to 7 m, but have good continuity both horizontally and vertically. Gold-bearing veins occur in different lithologies of the Piché and Pontiac groups. The veins cross the stratigraphy at low angles and are occasionally folded, particularly in volcanic and argillic host rocks. Generally, the veins strike east-west, dip steeply to the south and contain higher-grade shoots that plunge steeply to the east.

Often, the veins occur as a group of quartz veinlets scattered in a very sheared and altered zone that has no obvious main vein. Only very competent lithologies, like the conglomerate and the porphyric andesitic sills, host large veins. In some drill core, the quartz veinlets exhibit small tight folds (Bisson, 1995).

Gold grades vary considerably. The gold occurs mainly as fine to coarse free grains that are heterogeneously distributed, mainly in the quartz veins, and to a lesser extent, in the wall rock. Higher gold grades occur in short, steeply plunging shoots with a similar style to those mined at the O'Brien mine (Bisson, 1996).

7.3.3 Kewagama Area

In the Kewagama area, the gold mineralization occurs in rocks of the Piché Group to the south of the CLLFZ, which strikes east-west in this area and dips 80° to 85° to the south. North of the CLLFZ lies a considerable width of tuffs and agglomerates. Near the mine workings, the highly sheared rocks of the Piché Group have an aggregate width of 100 m to 130 m. The succession from north to the south is as follows: greenstone (15 to 25 m); north porphyric andesitic sill (3 to 10 m); conglomerate (12 to 25 m); greenstone and tuffs (3 to 7 m); south porphyric andesitic sill (3 to 9 m); and greenstone (about 60 m).

The only gold mineralization of particular interest disclosed by extensive underground workings is found in the winze, in a 25-foot raise above the winze and in the sublevels driven from the winze. These workings revealed an ore shoot with a vertical extent of 70 m and an east-west length of 4.5 to 25 m, in which irregular and discontinuous stringers of blue quartz carry free gold. Most of these veins are parallel and are contained within the north porphyric andesitic sill near its north margin, but some continue into the greenstone north of the porphyry. Individual veins are rarely more than 10 cm wide and 3 m long; occasionally, two or three are parallel to one another or overlap for part of their length. Some sections of these narrow veins are decidedly high grade, but in any stoping operation there would be considerable dilution.

The Kewagama ore shoot described above occurs in the same rocks as the high-grade shoot in the historical No. 4 vein mined at the O'Brien mine, and resembles it for its short lateral extent compared to vertical, and because it contains the same type of blue quartz and associated minerals. It differs from the O'Brien shoot in that it does not follow one definite fracture, instead consisting of a series of irregular overlapping stringers, and because it is of much lower grade.

7.3.4 Hydrothermal Alteration

The following description of hydrothermal alteration is mostly modified and summarized from Evans (2007) and Williamson (2019) and retains the references therein. The QP has reviewed and compared Evans' description of the hydrothermal alteration to other such accounts in publicly available documents and consider it accurate to the best of its knowledge.

Wallrock alteration ranges from several centimetres to over a metre thick, and is equally pervasive on both sides of the veins. The mineralized zones are usually comprised of a greater proportion of altered wallrock than actual veins. In general, the wallrock is well-foliated and has a distinctive dark brown to brownish grey colour due to intense biotite alteration. The brownish alteration is an easily recognizable indicator of potential gold-bearing mineralization. Biotite tends to occur as 1 to 2 mm thick layers of predominantly fine-grained biotite parallel to the foliation. On average, the mineralized zones contain about 5% biotite but can contain over 20% biotite.

Generally, zones of biotite alteration accompanied by silicification and sulphidation will yield gold values. Of all the sulphides, arsenopyrite is the most abundant and characteristic of the O’Brien mine. Arsenopyrite occurs mainly in intensely altered wallrock (0.1% to 1% based on inductively coupled plasma (ICP) analysis). The finer-grained and needle-like varieties of arsenopyrite are more likely to contain gold. Coarser-grained, euhedral rhombic arsenopyrite is less likely to contain gold (Bisson, personal communication, 1998).

Fine- to medium-grained, subhedral to euhedral pyrite is frequently observed generally overprinting the foliation (0.5% to 2%). Some pyrite is associated with gold-bearing zones (Hatch, 1998). Minor quantities of pyrrhotite and chalcopyrite are present in the mineralized zones (Bisson, 1995).

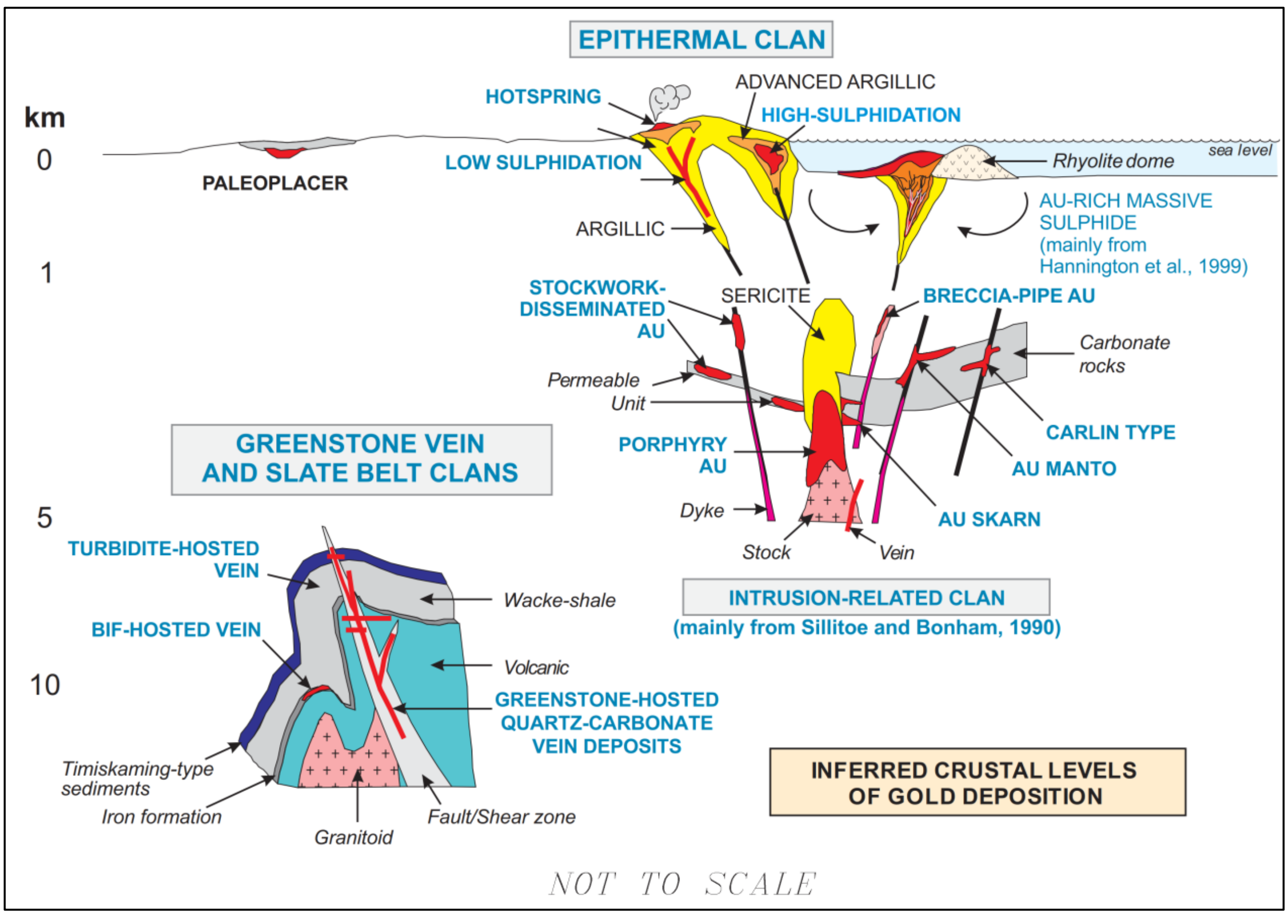
Carbonate alteration is mainly calcitic in micro-veinlets form, but it is also found frequently in all lithologies as more massive pervasive replacement. At times, iron carbonate veinlets are visible. Tourmaline is frequent but not always observed; it is generally found in small amounts in association with wallrock xenoliths.

8 DEPOSIT TYPES

This section is a slightly modified version of the mineral deposit type description provided in the technical report by Beausoleil (2018) and updated by Williamson (2019). The QP has reviewed and compared Beausoleil's geological description to other such accounts in publicly available documents and considers it accurate to the best of its knowledge.

Greenstone-hosted quartz-carbonate vein deposits occur as quartz and quartz-carbonate veins, with valuable amounts of gold and silver, in faults and shear zones located within deformed terranes of ancient to recent greenstone belts commonly metamorphosed at greenschist facies (Dubé and Gosselin, 2007). Greenstone-hosted quartz-carbonate vein deposits are a subtype of lode gold deposits (Poulsen et al., 2000) (Figure 8-1).

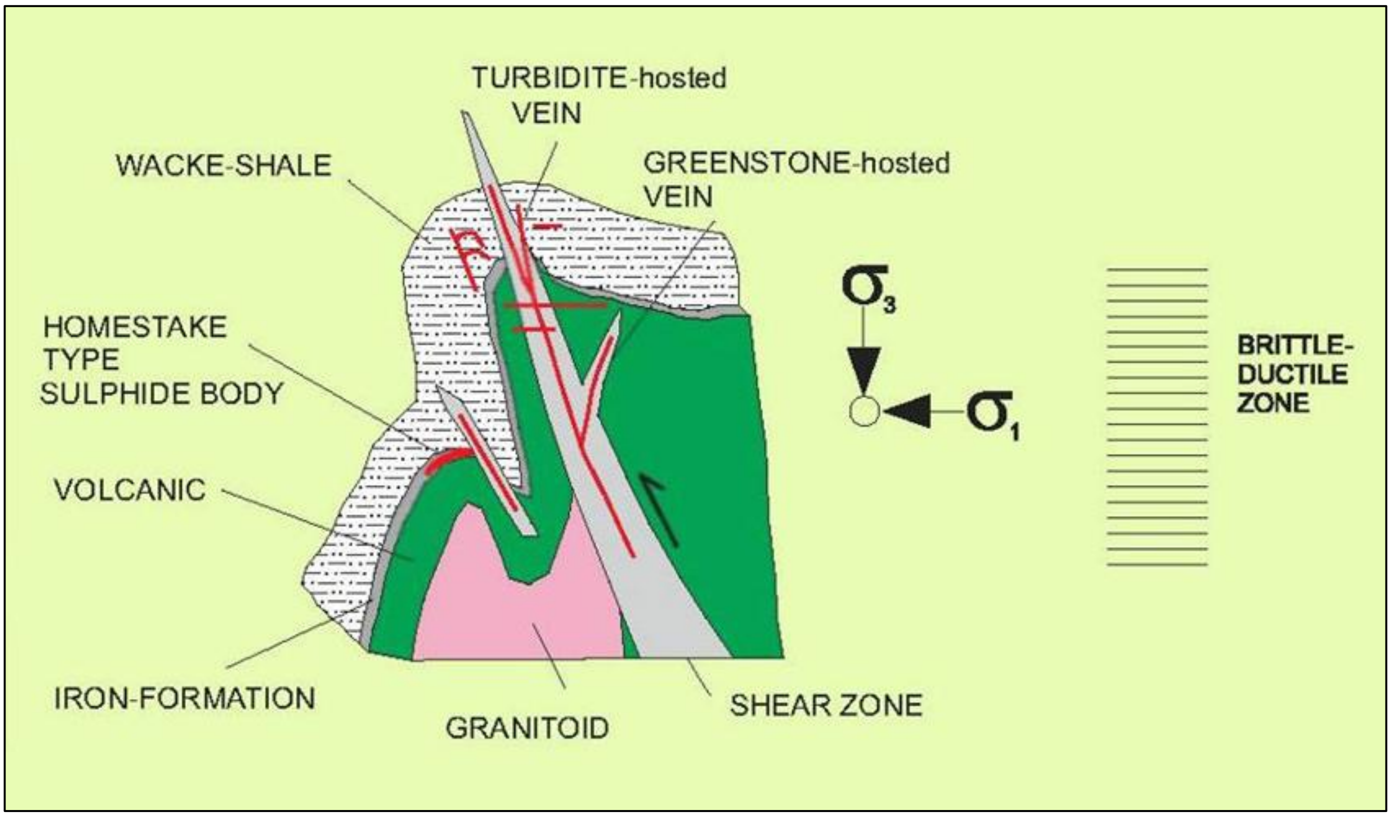
Figure 8-1: Inferred Crustal Levels Showing Different Types of Lode Gold Deposits and the Inferred Deposit Clan



Source: Modified from Poulsen et al., (2000) and from Dubé and Gosselin (2007).

They are also known as mesothermal and orogenic gold deposits. They consist of simple to complex networks of gold-bearing, laminated quartz-carbonate fault-fill veins in moderately to steeply dipping, compressional brittle-ductile shear zones and faults, with locally associated extensional veins and hydrothermal breccias. They can coexist regionally with iron formation-hosted vein and disseminated deposits, as well as with turbidite-hosted quartz-carbonate vein deposits (Figure 8-2). They are typically distributed along reverse-oblique crustal-scale major fault zones, commonly marking the convergent margins between major lithological boundaries such as volcano-plutonic and sedimentary domains. These major structures are characterized by different increments of strain, and consequently several generations of steeply dipping foliations and folds resulting in a fairly complex geological collisional setting.

Figure 8-2: Schematic Diagram Illustrating the Setting of Greenstone-Hosted Quartz-Carbonate Vein Deposits



Source : Modified from Poulsen et al. (2000) and Dubé and Gosselin (2007).

The crustal-scale faults are thought to represent the main hydrothermal pathways towards higher crustal level. However, the deposits are spatially and genetically associated with higher order compressional reverse-oblique to oblique brittle-ductile high-angle shear zones commonly located less than 5 km away and best-developed in the hangingwall of the major fault (Robert, 1990). Brittle faults may also be the main host to mineralization as illustrated by the Kirkland Lake Main Break; a brittle structure hosting the 25 Moz of gold in the Kirkland Lake deposit. The deposits formed typically late in the tectonic-metamorphic history of the greenstone belts (Groves et al., 2000) and the mineralization is syn- to late-deformation and typically post-peak greenschist facies and syn-peak amphibolite facies metamorphism (cf. Kerrich and Cassidy, 1994; Hagemann and Cassidy, 2000).

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

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

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

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

9 EXPLORATION

Radisson began conducting exploration activities at the property in 1994.

9.1 O'Brien Mine

1994: Radisson compiled historical data, reinterpreted the Zone 36 East, and drilled 12 holes totalling 3,998.4 m with the objective of increasing the Breakwater resource.

1996: Radisson drilled 31 holes totalling 11,962.8 m. The purpose was to increase the confidence level of the mineral inventory from the surface to 1,200 ft elevation and to demonstrate the presence of an extension of the veins at a vertical depth below 2,000 ft.

Mechanical stripping of 11 outcrops at 400 ft east of the No. 2 shaft was carried out to evaluate the gold potential of two gold-bearing structures located near the contact of the Piché and Pontiac groups. Some anomalous gold values were obtained from quartz veins in sedimentary rocks.

1997: Radisson drilled seven holes for 1,283 m, targeting the quartz veins associated with the contact zone between the Pontiac Group and Piché Group. Despite some economic grades, this drilling campaign was unsuccessful.

Radisson drilled an additional 23 holes for 4,555 m, targeting the Zone 36 East between sections 32E and 44E, from surface to a vertical depth of 230 m.

1998: Radisson contracted Roscoe Postle Associates Inc. (RPA) to independently estimate the resources and produce a pre-feasibility study to evaluate the viability of commercial production for the project. The study concluded that the project would not be profitable at a gold price of US$300/oz and a USD:CAD exchange rate of 1:1.444. The resources would have to increase; a better grade (than the cut grade of 6.9 g/t Au) would have to be confirmed; and a metallurgical recovery of at least 90% would need to be realized.

Two metallurgical tests were completed in two Canadian laboratories in 1998 on sulphide concentrates originating from Zone 36 East and Zone F. Two different processes were used: bioleaching at the BC Research Laboratory in Vancouver, British Columbia, and microwaves at the EMR Technology laboratory in Fredericton, New Brunswick. The objective was to reach 90% recovery for sulphide-related gold at a competitive processing cost. With direct cyanidation, the recovery barely reached 80%.

Radisson drilled two holes for a total of 546.8 m on targets identified outside the known zones north of the Cadillac-Larder Lake Fault Zone (CLLFZ). The Vintage Zone, interpreted to be a network of horizontal gold-bearing quartz veins with free gold, was discovered.

Radisson drilled five more holes for 1,402 m to locate other gold-bearing veins north of the CLLFZ. Despite some interesting findings, nothing significant came out of the campaign.

2001: On August 24, 2001, Radisson signed an initial agreement with Rocmec Mining Inc. (Rocmec) concerning preliminary tests and the use of a new extraction technology applied to the gold-bearing quartz veins on the O'Brien property.

Rocmec drilled an initial series of thermal holes supervised by Radisson personnel. This work allowed 1.54 tonnes of gold-bearing quartz vein material to be extracted. The extracted sample was processed on a Deister table in the Radisson concentrator, on site in Cadillac. The gold in the batch totalled 35.245 grams, at a grade of 22.83 g/t Au. Recovery reached 77%. This work confirmed a high rate of recovery by gravimetry and an excellent grade for the smoky quartz veins in the former O'Brien mine.

2003: In the summer of 2003, a surface exploration program was carried out to verify the surface extraction potential of gold-bearing quartz veins in the O'Brien mine area approximately 900 ft east of the headframe, and the potential of the Zone 36 East veins. The O'Brien property was stripped to reveal new smoky quartz veins. The samples taken in the stripped zones did not yield economic grades.

Radisson drilled three holes for a total of 210.3 m. Two composite core samples drilled on the same zone, one from a vein and the other from its wall, were analysed at Laboratoire LTM in Val-d'Or. The test was intended to determine the content of the vein and the wall, as well as to verify the gold recovery ratio by gravimetric method. A content of 4.80 g/t Au was obtained for the vein with 63% recovery by gravity. The wall yielded 2.40 g/t Au gold and an equivalent recovery. On their own, these results could not justify a major surface bulk sample test, and it was decided to discontinue efforts.

Surface exploration efforts on the O'Brien property stopped.

2004: Radisson initiated an initial deep diamond drilling campaign to verify depth potential of the contact zone – type gold mineralization along the CLLFZ. One hole (OB04-01A), which was drilled under Zone 36 East and reached a length of 1,535 m, confirmed the continuity of Zone 36 East at depth, doubling its vertical extension.

2006: A high-resolution aeromagnetic, horizontal gradiometer and XDS-VLF-EM survey was carried out on the O'Brien and Kewagama properties in June 2006. The survey, which was the first phase of the 2006 exploration program, was conducted by Terraquest Ltd. with a flight line spacing of 50 m. Data from this survey was used to define drill targets north of the CLLFZ.

Radisson also carried out a litho-geochemical sampling program focusing on the talc-chlorite schists in drill core stored at the O'Brien mine site. The program's objective was to verify the presence of mineralization similar to the D Zone on the Wood/Pandora project.

Three holes were drilled on the No. 2 vein, Zone 36 East, and the North Zone, totalling 1,198 m.

2007: RPA estimated the mineral resources of Zone 36 East using the historical surface and underground drilling data available in April 2007.

RPA's study showed that the Zone 36 East mineralization was sensitive to cutting high gold assays, and the cut indicated average grade was approximately 36% lower than the uncut indicated average grade. Cutting high gold assays reduced the contained gold in the global resource by approximately 30% from the uncut figure.

An exploration program, with the purpose to test resource blocks identified in the 2007 Technical Report on the Zone 36 resources (Evans, 2007), was carried out by Radisson. It included 60.8 km of line cutting, 46.1 km of IP, and 2,053 m of diamond drilling in 15 holes (OB07-120 to OB07-134). The drilling program continued until March 2008.

Since the completion of the 2007 exploration program, exploration activities in the vicinity of the historical O'Brien mine have consisted solely of drilling, which is discussed in Section 10.

9.2 Kewagama Mine

After becoming the 100% owner of the Kewagama property in 1999, Radisson compiled existing data to assess the potential of existing gold showings. Radisson completed several small exploration drilling programs between 2003 and 2005.

In June 2006, a high-resolution aeromagnetic, horizontal gradiometer and XDS-VLF-EM survey was carried out on the O'Brien and Kewagama properties in June 2006. The survey, which was the first phase of the 2006 exploration program, was conducted by Terraquest Ltd. with a flight line spacing of 50 m. Data from this survey was used to define drill targets north of the CLLFZ. Radisson followed up with an exploration drilling program in 2006, which confirmed the potential for gold mineralization north of the CLLFZ (the North Zone). At the time, the North Zone extended for more than 300 m along strike, from section 43E to 53E.

During the fall of 2016, Abitibi Geophysics carried out an OreVision survey on the southern part of the project in Pointiac Group rocks (Beausoleil, 2018). A 43.35 km grid was surveyed and divided into 35 north-south lines with 100 m spacing. The survey configuration was a = 25 m and n = 1 to 30. A total of 21 polarized sources were identified. According to Dubois (2016) there is good potential for additional discoveries or extensions of mineralized zones based on the anomalies identified by OreVision.

Additional exploration drilling was carried out by Radisson in 2008 through 2022, as detailed in Section 10.

9.3 New Alger Area

This section summarizes information provided by the Radisson exploration team.

In 2022, Radisson embarked on an extensive exploration campaign on the New Alger area of the O'Brien Gold Project. This is the area of claims located south of Highway 117. The work was separated into three distinct phases—a compilation/planning phase, a prospecting and sampling phase, and a trench channel sampling phase—and was carried out between January and October 2022. These phases are described in the following subsections.

9.3.1 Phase 1 – Compilation/Planning

From January to April 2022, previous work and information on the regional and local geology were compiled, reviewed, and contextualized to reassess the discovery potential of the area. Results from the work highlighted the limited scope of the previous exploration, which was restricted mainly to prospecting and mapping campaigns during the 1920s and 1930s, and from 2013 to 2020. Very little drilling was completed in the southern part of the property, and the available drilling results were limited to within the 240-PTA mining concession.

The compilation of geological, structural, and geophysical information allowed favourable sectors to be targeted for gold mineralization, with a particular focus on knowledge from neighbouring deposits in the Cadillac mining camp and the Pontiac Subprovince.

The following types of gold mineralization are of specific focus for the New Alger project:

- gold mineralization emplaced coincident with S1 schistosity (i.e., shears parallel to the S1, or in the axial planes of folds emplaced in the Pontiac sediments)
- gold mineralization associated with second order faults
- mineralization associated with the emplacement of a syntectonic porphyritic felsic intrusive (intrusion related gold system).

9.3.2 Phase 2 – Prospecting and Sampling

Prospecting work consisted of geological reconnaissance and outcrop sampling. In total, 51 days of prospecting were necessary to cover the entire project. One to two teams, composed of a geologist or geological engineer and assisted by a technician or geology student, were employed to prospect the 54 km^2 of the New Alger area.

To facilitate the work and communication between teams, the territory to be covered was divided into 22 sectors that were delineated by roads or waterways. Prospecting work was organized in the form of traverses oriented north-south at 400 m spacing. Due to the property's ease of access by vehicle, an outcrop roadside mapping program was also carried out. During prospecting, teams carried a Beep Mat electromagnetic survey tool to assist with locating mineralization or conductive and/or magnetic horizons on surface.

During this phase, a prospecting-style sampling program of mineralized lithologies was carried out. A total of 228 samples were taken from outcrops and boulders. Samples were sent to the ALS laboratory in Rouyn-Noranda (ALS). Analysis for gold (Au-AA24) and base metals (ME-ICP61) were performed on all samples. In addition, whole rock analysis (ME-MS81d) was carried out on intrusive rocks of particular interest (felsic intrusives, rocks containing mineralization or alteration, etc.).

9.3.3 Phase 3 – Trench Channel Sampling

An analysis of the field data collected during the first two phases, combined with existing geophysical data, allowed Radisson's team to propose favourable targets for channel sampling work.

During August 2022, 22 channels were completed on outcrops with potential of being mineralized. Channels were cut in a north-south direction, ranging between 1 and 7.5 m in length, the total length of the excavated channels was 48 m, and 56 samples were collected. As with the prospecting phase, all samples were sent to ALS where they were analysed for gold and base metals; six samples were sent for whole rock analysis.

9.3.4 Till Sampling

In the summer of 2023, Radisson collected and analysed 114 glacial till surface samples on its New Alger property, with the work carried out by IOS Services Géoscientifiques. Among these, 14 samples were found to be anomalous, each containing between 10 and 45 pristine gold grains, which strongly suggests a nearby source of gold. These anomalous samples formed a distinct southward dispersion tail of gold grains in the western portion of the New Alger area. Based on this promising indicator, Radisson delineated a new gold target zone with a strike length exceeding 2 km. Given these results, Radisson staked an additional 15.5 km^2 of claims, expanding the total area of the New Alger claims.

9.3.5 Results and Recommendations

Analytical results confirmed the gold and base metal potential of the sector. Highlights included values of 7.33 g/t Au and 2.40 g/t Ag from an erratic block sample, and a value of 0.46% tungsten from an outcrop sample. Other than these results, several other analyses revealed the presence of copper, silver, nickel, and zinc.

The prospecting campaign also revealed the following mineralization-bearing lithologies and structures:

- felsic intrusive rocks showing epidote, silica, and sericite alterations, associated with shears mineralized with pyrite and other sulphides
- graphitic horizons associated with sheared quartz veins, locally showing 5% arsenopyrite
- breccia zones of unknown origin, spatially associated with mafic to ultramafic intrusive rocks and pyrite mineralization.

10 DRILLING

10.1 Introduction

A summary of drilling at the project is presented in Table 10-1.

Table 10-1: Summary of Validated Drill Holes

Mine	Time Period	No. of Holes	Length (m)
O'Brien	Historical (1930s-1993)	226	34,882.0
	1994-2025	584	273,016.6
Thompson- Cadillac	Historical (1930s-1993)	64	6,619.0
	1994-2022	70	15,845.0
Kewagama	Historical (1930s-1993)	203	16,253.0
	1994-2025	60	27,846.2
Total	**-**	**1,207**	**374,461.8**

Source: Radisson (2025).

10.2 1994 to 2019 Drilling

From 1994 to 2019 drilling campaigns were undertaken by Radisson over the property for a variety of purposes, including increasing mineral inventory at O'Brien (formerly referred to as Zone 36 East), Kewagama, and Thompson-Cadillac; increasing confidence in mineral resource estimation; vein extension testing; historical result confirmation; resource definition drilling; and mineralization continuity testing. All drilling was completed from surface using a diamond drill; in some cases, these were wedged to save drilling costs and time.

10.3 2019 to 2025 Drilling

Between 2019 and 2025, Radisson completed 176,570 m of surface diamond drilling, representing 437 holes, including 53 wedges, as summarized in Table 10-2. Ninety holes were abandoned due to strong deviation or stuck drilling equipment. All holes were drilled from surface with NQ (47.6 mm) diameter core. For deeper holes, Devico's DeviDrill directional drilling technology was used to control deviation, and smaller diameter (BQ) drilling was used for directional drilling and to intercept the target precisely.

Radisson acquired the New Alger area of the project in 2020 from Renforth. Renforth's 2019 and 2020 drilling campaigns into this area have not been included in these totals.

Table 10-2: Summary of Validated Drill Hole Details by Year from 2019 to 2025

Year	Hole Type	Number of Holes	Length (m)
2019	Diamond Drill Hole	21	12,902.6
	Wedge	4	539.6
	Abandoned	10	771.3
2020	Diamond Drill Hole	71	38,028.2
	Wedge	8	2,294.3
	Abandoned	22	2,525.5
2021	Diamond Drill Hole	107	49,861.1
	Extension	1	4.5
	Wedge	15	6,473.0
	Abandoned	24	1,689.1
2022	Diamond Drill Hole	6	4,137.0
	Extension	1	12.0
	Wedge	15	7,930.3
	Abandoned	3	450.0
2023	Diamond Drill Hole	11	5,214.0
	Extension	0	0.0
	Wedge	0	0.0
	Abandoned	4	83.4
2024	Diamond Drill Hole	57	28,555.6
	Extension	0	0.0
	Wedge	6	1,842.5
	Abandoned	21	1,709.1
2025	Diamond Drill Hole	14	7,019.0
	Extension	4	1,500.0
	Wedge	5	2,852.2
	Abandoned	6	179.0
Total	Diamond Drill Hole	287	145,717.5
	Extension	6	1,516.5
	Wedge	53	21,931.9
	Abandoned	90	7,404.4
	Total	**436**	**176,570.3**

Notes Excludes drill holes drilled by Renforth in the New Alger area. Extension, wedge and abandoned sub-categories are included in the diamond drill hole category. Source: Radisson (2025).

Rock quality designation (RQD) measurements were taken on most of the core collected. Digital photos were taken of all the drilled core.

Diamond drilling campaigns between 2019 and 2025 were carried out by the following drilling contractors:

2019: Forage SMP Inc (SMP), based in Val d'Or, Québec

2020: SMP and Spektra Drilling Canada Inc. (Spektra), headquartered in Toronto, Ontario

2021: Spektra; Major Drilling Group International Inc. (Major) from Moncton, New Brunswick; and Forage DCB (DCB) from Rouyn-Noranda, Québec

2022: Spektra and Major

2023-2025: Nordik Drilling, now part of RJLL Drilling.

Diamond drill holes were planned using cross-section, plan, and 3D views using Leapfrog Geo and Geotic Graph software. Radisson geologists and external consultants were involved in the targeting and follow-up phases of the drilling program.

Radisson geologists and technicians used a handheld Garmin GPS (models 64s and 66i) to survey in proposed holes. The collar locations of new drill holes were subsequently and systematically surveyed by professional surveyors (Corriveau J.L. & Associates Inc.) on a bi-annual basis. Once obtained, the reviewed collar positions are uploaded to the drill hole database and take precedence over the initial planned locations.

Between 2019 and 2021, single shot surveys of deviation were taken, starting slightly below the collar and at regular 30 m intervals thereafter. The drilling contractor handled the surveys, and information was transcribed and provided on paper to Radisson geologists. After successful completion of a drill hole, a multishot survey was taken over the full length of the hole at 3 m intervals. The REFLEX EZ-TRAC instrument was used to record azimuth and dip information. Multishot survey information was given preference over single-shot data in the survey database.

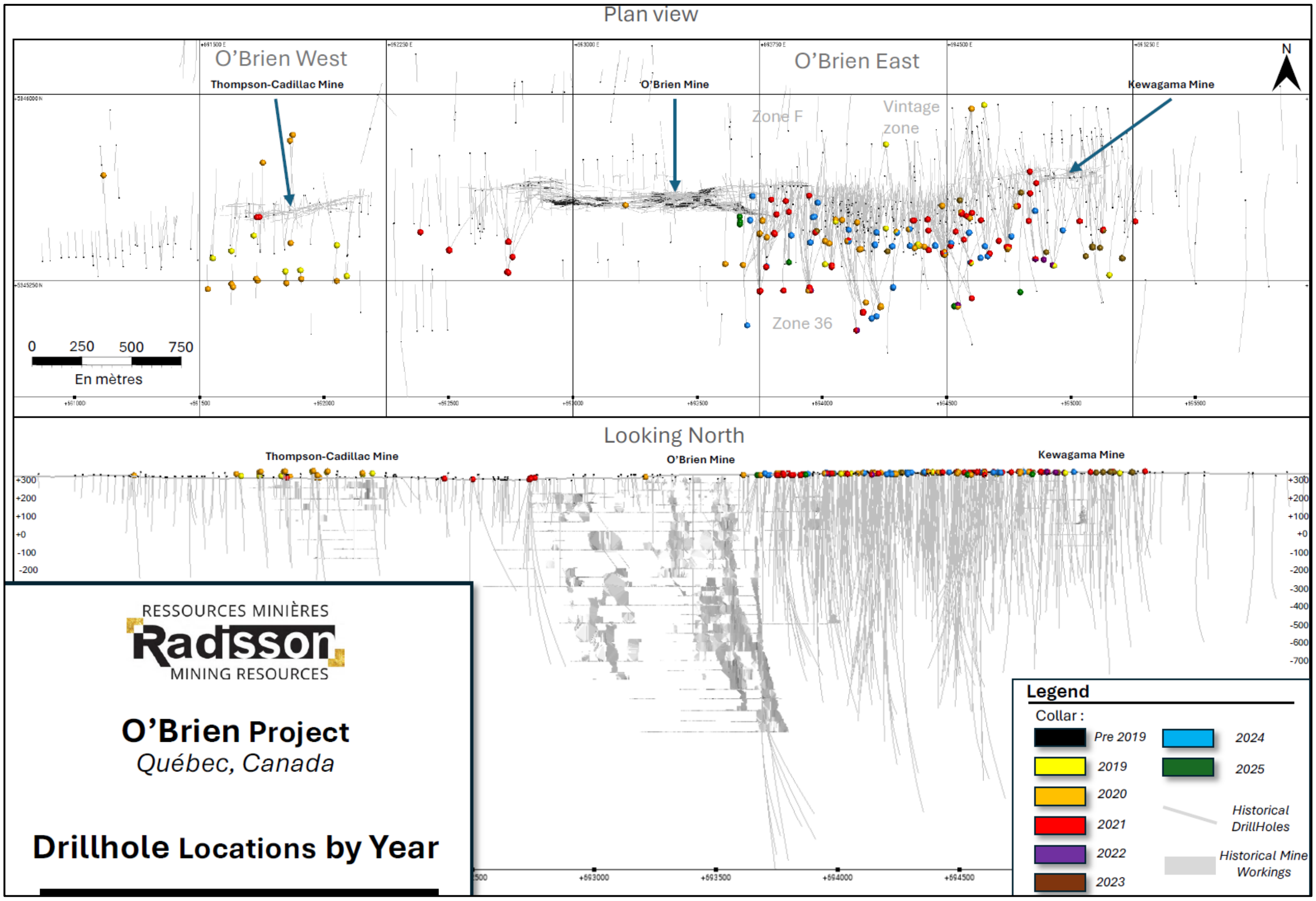
After 2021, Radisson began collecting single-shot survey information with a Gyro, starting below the collar, and collecting data at 18 m intervals. The instrument was handled by the drilling contractor, and data from the surveys were transferred directly to the Cloud (Imdex Hub IQ).

The number of holes that had to be abandoned between 2019 and 2020 due to unacceptable deviation, as well as the desire to accurately hit targets at depth, led Radisson to test Devico's controlled drilling technology starting in July 2021. Between July 2021 and May 2022, seven pilot holes and 31 wedges were drilled by Spektra and Major using this technology. The use of deviation control technology necessitated reducing the drilling diameter core to BQ (36.5 mm). Deviation control was only used in Pontiac sediments outside of mineralized areas. During the Devico drilling, surveys were taken using the DEVI-Gyro (survey instrument of Devico) every 3 m.

Casings were left in place, flagged, and capped. A metal tag identifying the hole was left on the cap for future reference.

Drill hole collars and traces for drilling completed between 2019 and 2025 are illustrated in Figure 10-3. Note: this figure includes the holes drilled by Renforth in the Thompson-Cadillac area in 2019 and 2020, which are excluded from the drill hole summary presented in Table 10-2.

Table 10-3: Drill Collar Location Plan



Source: Radisson (2025).

10.4 Core Logging Procedures

The following procedures were developed by Radisson and have evolved slightly over the years alongside evolving best practice guidelines and availability and the adoption of new software and technologies.

At the rig, the driller helper places the core into core boxes, marking off every three metres with wooden blocks. Once a core box is full, the helper wraps the box with fibre tape. At the start of each day, a Radisson technician brings the secured core boxes from the rig to the core shack facility.

In the core shack, Radisson employees remove the tape and place the boxes on the logging tables. The technicians rotate the core so that all the pieces slant one way, showing a cross-sectional view, along the strike of the main penetrative fabric observed in the core. They check that distances are correctly indicated on the wooden blocks placed every three metres. The core is then measured in each box and the boxes are labelled. RQD is measured either by geologists or by geological technicians. Any breakage under 10 cm is recorded.

The geologists use GeoticLog logging software. Lithological (principal and secondary lithologies), alteration, mineralization, veining, and structural characteristics of the core are entered into the database, along with geotechnical parameters including RQD.

Samples are selected by the geologists. Sample length is typically 1.0 m but may range from 0.2 m (minimum sampled length) to 2.0 m to honour lithological contacts defined by the geologist. Once all samples are marked on the core, photographs of the wet core are taken by either the geological technician or the geologist.

Once logged and labelled, the core is stored inside on racks until cut by the saw technician. The core of each selected interval is sawed in half using a typical table-feed circular rock saw. One half of the core and a sample tag are placed in a plastic bag for shipment to the laboratory, and the other half is returned to the core box as a reference sample. Sampled core is double-bagged and stacked in pallets or rice-bags prior to being transported to the laboratory. A tag bearing the sample number is left in the box at the end of the sampled interval. The core box is then taken to covered racks at the outdoor core storage area enclosed with secure fencing. The exact location of each hole in the outdoor core library is recorded in an Excel spreadsheet for future reference.

Complete core logging and sampling descriptions are exported into an Excel spreadsheet and sent to the geologist in charge of the project, to validate and sign the drill hole logs.

10.5 2018 to 2019 Relogging Program

During the first half of 2019, Radisson began relogging older holes to locate mineralization within specific intervals and to sample previously unsampled intervals. Over 521 samples representing 520 m of drill core have been added to the drill hole database used for the current mineral resource estimate because of these efforts.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Density Sample Preparation and Analysis

Following recommendations from Williamson (2019), Radisson continued the density sampling program. Density sampling has been conducted in-house by Radisson, at their on-site core shack.

The density sampling procedure can be summarized by the following:

- Radisson geologists select samples based on lithology, alteration, and mineralization.
- Selected samples are marked in the core box, and the interval is recorded.
- Dry samples are weighed and lowered into a water bath, whereupon the displacement of the water is measured.
- The density of the sample is obtained from the ration of the dry sample weight to the volume of displaced water using the formula $\rho = m \div V$ where ρ is equal to the density, m is the mass of the sample, and V is the volume of displaced water representing the volume of the sample.

Since March 2025, a new density measurement protocol has been followed that uses a hydrostatic balance to measure density by weight difference between material in the open air and immersed in water.

The QP is of the opinion that this density sampling method is adequate for the purpose of mineral resource estimation.

11.2 Gold Sample Preparation and Analysis

Before 1995, the O’Brien mine used its internal laboratory for assaying. Between 1995 and 2025, a number of laboratories were used, such as Chimitec Ltd. (Chimitec) in Val d’Or, Québec; XRAL Laboratories (XRAL) in Rouyn-Noranda, Québec; Techni-Lab Inc. (Techni-Lab) in Ste-Germaine, Québec; Laboratoire Expert Inc. (Laboratoire Expert) in Rouyn-Noranda, Québec; Swastika Laboratories Ltd. (Swastika) in Swastika, Ontario; ALS Minerals (ALS) in Val d’Or, Québec; and SGS Canada Inc. (SGS) in Val d'Or, Québec (Table 11-1).

Except for the internal O’Brien mine laboratory, all the laboratories are independent of Radisson. Commercial laboratories Swastika, ALS, and SGS are accredited to the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 9001:2008 standards for quality management and to ISO/IEC 17025:2005 for all relevant procedures. Accreditation of the other laboratories is unknown.

Sample preparation and analysis procedures have remained consistent over time, despite changes to the primary laboratory employed. Laboratories have generally employed a standard approach whereby samples are crushed and pulverized prior to gold analysis by fire assay (with AAS or gravimetric finish), with or without follow-up of metallic screen gold analysis on selected high-grade samples. Specific preparation and analytical techniques undertaken to test O’Brien Gold Project samples from 2019 to 2022 are described in detail in the following subsections.

Table 11-1: History of Laboratory Use

Year	Primary Laboratory	Secondary Laboratory
Historical	O'Brien Mine Laboratory	
1995-1996	Chimitec	
1997-1998	XRAL	Techni-Lab
2006-2007	Laboratoire Expert	
2007-2008	Techni-Lab	ALS Chemex
2008-2009	Laboratoire Expert	
2011-2012	Techni-Lab	
2012-2013	Laboratoire Expert	Techni-Lab
2014-2015	Techni-Lab	
2015-2017	Swastika	
	Techni-Lab	
2017-2018	Swastika	
2018-2025	ALS	Techni-Lab

11.2.1 ALS

For its 2019 to 2025 drilling, Radisson used ALS as its primary laboratory. ALS is independent of Radisson, and its Val-d'Or facilities are accredited to the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 9001:2008 standards, for all quality management and to ISO/IEC 17025:2005 for all relevant procedures.

The following analysis is undertaken at the ALS Val-d'Or facilities:

- Sample Preparation: PREP-31B. Samples are crushed to 70% less than 2 mm, riffle split to 1.0 kg, pulverized split to greater than 85% passing 75 µm.
- Gold Analysis: Au-AA24. A 30 g fire assay standard fusion method with AAS finish. The lower detection limit is 0.005 g/t Au, and the upper detection limit is 10 g/t Au. Each sample over 10 g/t Au is assayed by the Au-GRA22 combination fire assay and gravimetry method on 50 g of aliquot.
- Metallic Screen Gold Analysis: Au-SCR21. Samples with visible gold a submitted for metallic sieve analysis where the 1 kg pulp sample is screened to 106 µm. A 30 g fire assay standard fusion method with AAS finish is completed on the screen undersize as well as the entire oversize fraction.

Since January 2025, the following minor adjustments have been made to this protocol:

- Sample Preparation: PREP-33DN. Samples are crushed to 90% less than 2 mm, riffle split to 1.0 kg, pulverized split to greater than 95% passing 106 µm.
- Metallic Screen Gold Analysis: Au-SR24. Sample with visible gold is submitted for metallic sieve analysis where 1 kg pulp sample is screened to 100 µm. A 50 g fire assay standard fusion method with AAS finish is completed on the screen undersize as well as the entire oversize fraction. Since March 2025 a new protocol was implemented to send each all samples return values greater than 2.5 g/t Au from the Au-AA24 method to metallic sieve process.

11.2.2 Techni-Lab

For its 2018 to 2022 drilling, Radisson utilized Techni-Lab as its secondary laboratory. The following analysis is undertaken at the Techni-Lab Ste-Germaine facilities on selected duplicate pulp samples initially analysed at ALS:

- Sample Preparation: Samples are dried to 60°C and then crushed to 80% passing 8 mesh and split to 250 g using a Jones riffle splitter or rotary split. The subsample is pulverized to 90% passing 200 mesh.
- Gold Analysis: TMT-G5B. Core samples are analysed by fire assay with AA from 30 g pulps. The lower detection limit is 8 ppb.
- Gold Analysis: TMT-G5C. When assay results are higher than 5 g/t Au, core sample pulps are re-assayed by FA with gravimetric finish.
- Metallic Screen Gold Analysis: If visible gold is observed, the sample is sent for metallic sieve. In which case, the entire sample is pulverized and assayed.

11.3 Sample Security

Samples are handled and transported by Radisson personnel or contractors. Drill core is stored at the on-site core storage facility at the project site, the grounds of which are locked. The storage facilities are open on the sides and covered. A core storage map is maintained by Radisson. Sample pulps are catalogued, organized by number, and stored in locked shipping containers. Sample rejects are stored at site in rice bags. Drill hole logging and sample data are maintained in Geotic's Geoticlog software and are regularly backed up.

In the QP's opinion, the sample security procedures are acceptable for the purposes of mineral resource estimation.

11.4 Quality Assurance and Quality Control

Quality assurance (QA) consists of evidence that the assay data has been prepared to a degree of precision and accuracy within generally accepted limits for the sampling and analytical methods to support its use in a resource estimate. Quality control (QC) consists of procedures used to ensure that an adequate level of quality is maintained in the process of collecting, preparing, and assaying the exploration drilling samples. In general, QA/QC programs are designed to prevent or detect contamination and allow assaying (analytical), precision (repeatability), and accuracy to be quantified. In addition, a QA/QC program can disclose the overall sampling-assaying variability of the sampling method itself.

In the QP's opinion, the QA/QC programs carried out by Radisson are adequate and the assay results within the database are suitable for use in a mineral resource estimate.

11.4.1 Historical (Pre-1993) Procedures

There are no known records of QA/QC procedures and results of historical data.

11.4.2 1993 to 2017 Procedures

The QP reviewed QA/QC procedures and results compiled by Williamson (2019), de l'Etoile and Salmon (2013), and Evans (2007). Observations are summarized in Table 11-2.

Table 11-2: Summary of QA/QC Submissions and Results, 1993-2017

QA/QC Type	Years Inserted	Count	Summary of Results
Check Sample	1995-1997	300 (approx.)	Pulps submitted to Chimitec, XRAL, and Technilab. Evans (2007) notes XRAL assays within a window of 0.1 to 0.5 oz/ton Au show wide dispersal, low correlation coefficient (0.737), relative SD of 34% and a precision of ± 78% at a 90% confidence, indicating low reliability and low reproducibility, likely due to an elevated gold nugget effect. Evans notes that this variability in results adds a level of uncertainty to the mineral resource estimate.
Blank	2006	8	Blanks were submitted to Laboratoire Expert and Evans (2007) notes no unusual results found.
Standard	2006	8	Standards were submitted to Laboratoire Expert and Evans (2007) notes no unusual results found.
Blank	2007-2008	130	Blanks submitted to Technilab, 11% of which were above the threshold value of 0.05 g/t Au, with one sample identified as a mix-up.
Standard	2007-2008	128	Standards submitted to Technilab. Seven Rocklabs CRMs were used, ranging from 1 to 30 g/t Au. Evans (2007) notes difficulty in identifying the CRM material from Rocklabs. 8% of the standards submitted failed the ±3SD limit set, and Evans (2007) notes no drift observed in the results through time.
Check Sample	2008	151	Rejects submitted to secondary ALS Chemex. Evans (2007) notes low correlation coefficient of 0.57, and that the average grade of the samples submitted was 3.09 g/t Au and rejects was 2.71 g/t Au. Evans (2007) notes that these could be explained by reject material vs pulps, whether the two laboratories used the same procedures, and the probable high nugget effect.
Blanks	2011-2012	25	2011 samples submitted to Technilab, 2012 samples submitted to Laboratoires Expert. Only one blank reported above the threshold of 0.05 g/t Au.
Standard	2011-2012	88	2011 samples submitted to Technilab, 2012 samples submitted to Laboratoires Expert. Four different standards were used, from Rocklabs with certification sheets. 14% of the CRMs failed the ±3SD test. Evans (2007) notes high failure rate and recommends documentation and corrective action where these occur.
Blank	2015-2017	524	312 samples were submitted to Technilab, 201 samples submitted to Swastika. Two blanks from Technilab and five from Swastika failed the threshold (80 ppb for Technilab and 0.1 ppm for Swastika). Beausoleil (2018) recommends re-assaying batches where blanks failed.
Standard	2015-2017	544	5 CRMs ranging from 0.99 to 17.58 g/t Au were submitted to laboratories (330 to Technilab and 214 to Swastika). Nine samples failed the 3SD threshold, and Beausoleil (2018) notes that five can likely be attributed to insertion errors. They recommended batches containing failing standards be re-assayed.

11.4.3 2018 to 2025 Procedures

11.4.3.1 Certified Reference Material

Results of the regular submission of certified reference materials (CRMs or standards) are used to identify issues with specific sample batches, and biases associated with the primary assay laboratory (ALS). Radisson has sourced CRMs from OREAS North America Inc. (OREAS), of Sudbury, Ontario, and Rocklabs, of Aukland, New Zealand (Rocklabs). Results of the CRMs, including failure rates, defined as a gold value reporting more than three standard deviations (SDs) from the expected value, and warning rates, defined as gold values reporting more than two SDs, but less than three SDs from the expected values, were plotted in control charts.

Radisson's QA/QC program includes the regular insertion of standards into the sample shipments. Standards are inserted at a rate of approximately 1 per 20 samples. Eleven different CRMs were inserted at O'Brien from 2018 to 2025, totalling 3,158 individual samples, with an overall insertion rate of 3.86%. Radisson's policy for failing CRMs was to send the previous and subsequent 10 samples from the failing sample for re-assay if they intersected a mineralized zone or contained significant gold.

The QP reviewed the certificates of analysis for all CRMs used and they vary in grades from 0.90 to 12.39 g/t Au. Table 11-3 summarizes the technique used to assay the CRM material, expected values, standard deviation, and warning and failure rates of each CRM.

The QP notes that in 2020, four primary standards were in use. SG84, SG99, SP73, and SL108 show relatively high failure rates. Investigations by Radisson personnel uncovered some deficiencies in the way that samples were submitted, including likely cross-contamination in CRM preparation. In the context of the 2020 and 2021 data, the QP can attribute these results to sample contamination. Considering these discoveries, Radisson chose to re-assay the affected samples in line with their QA/QC policy, and switch to OREAS-supplied CRMs.

The QP recommends a pulp duplicate program be carried out to confirm the 2020 results.

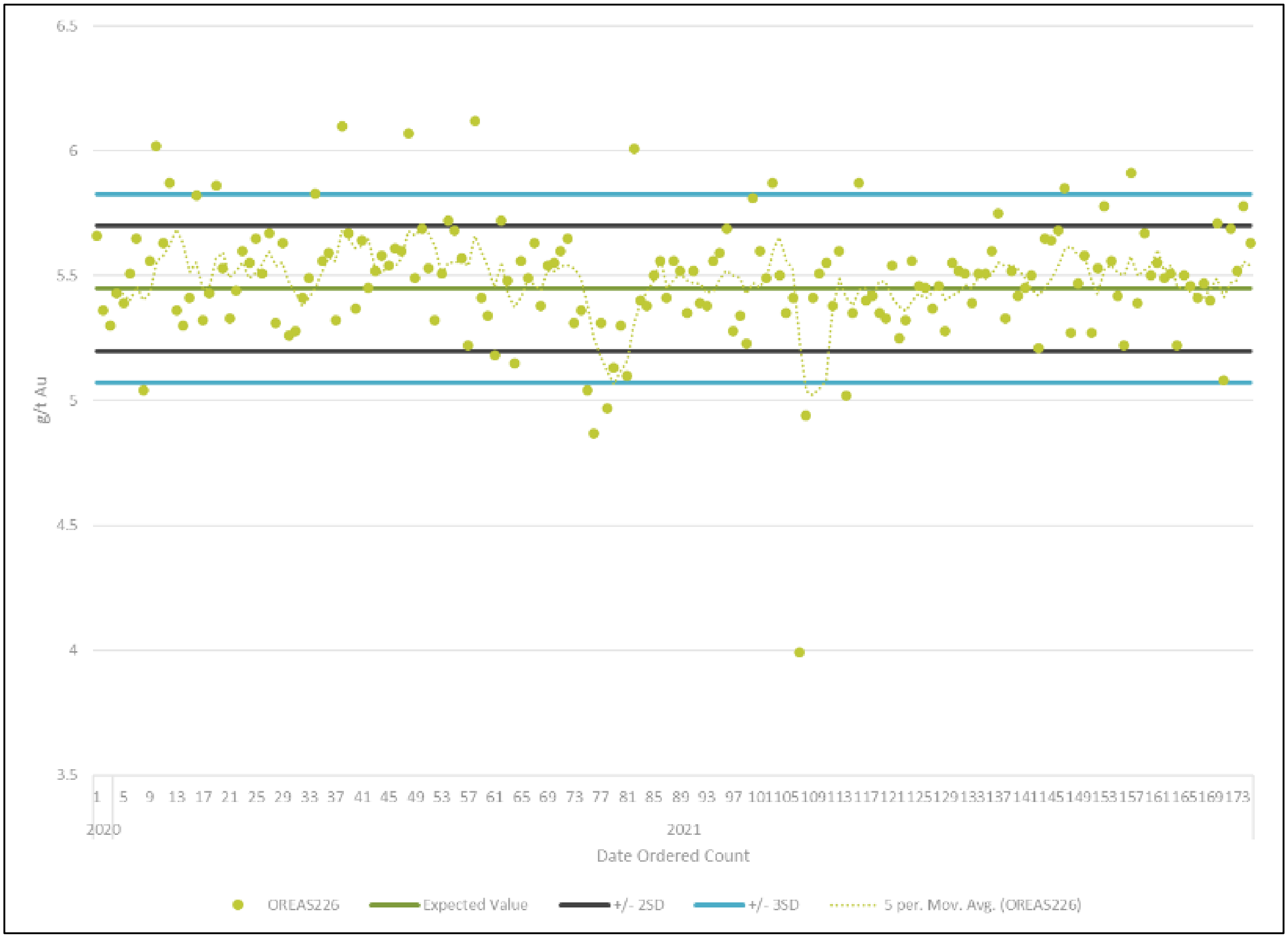
Table 11-3: Expected Values and Ranges of Selected Gold CRM

Standard	Year								Total	Grade		Assay Technique	Source	Warning Rate	Failing Rate
	2018	2019	2020	2021	2022	2023	2024	2025		(g/t Au)	(g/t Au)			>/< 2 SD (%)	>/< 3 SD(%)
OREAS 226	-	-	3	172	-	-	-	-	175	5.450	0.126	Fire assay	OREAS	22.3	12.6
OREAS 232	-	-	-	631	128	51	194	51	1055	0.902	0.023	Fire assay	OREAS	5.5	2.2
OREAS 240	-	-	-	275	126	34	13		448	5.510	0.139	Fire assay	OREAS	15.8	6.0
OREAS 240b							186	52	238	5.65	0.143	Fire assay	OREAS	3.8	3.4
OREAS 243	-	-	-	-	31	13	136	4	184	12.390	0.306	Fire assay /gravimetry	OREAS	12.5	5.4
SG84	60	49	72	-	-				181	1.026	0.025	Fire assay	Rocklabs	27.6	8.3
SG99	-	1	300	114	-				415	1.041	0.019	Fire assay	Rocklabs	31.9	13.1
SL108	-	17	302	-	-				319	5.744	0.138	Fire assay	Rocklabs	24.8	10.7
SL46	1	-	-	-	-				1	5.867	0.170	Fire assay	Rocklabs	0.0	0.0
SL76	49	19	-	-	-				68	5.960	0.192	Fire assay	Rocklabs	22.1	10.3
SP73	3	11	47	13	-				74	18.170	0.420	Fire assay	Rocklabs	31.9	15.3

Note: SD = Standard deviation.

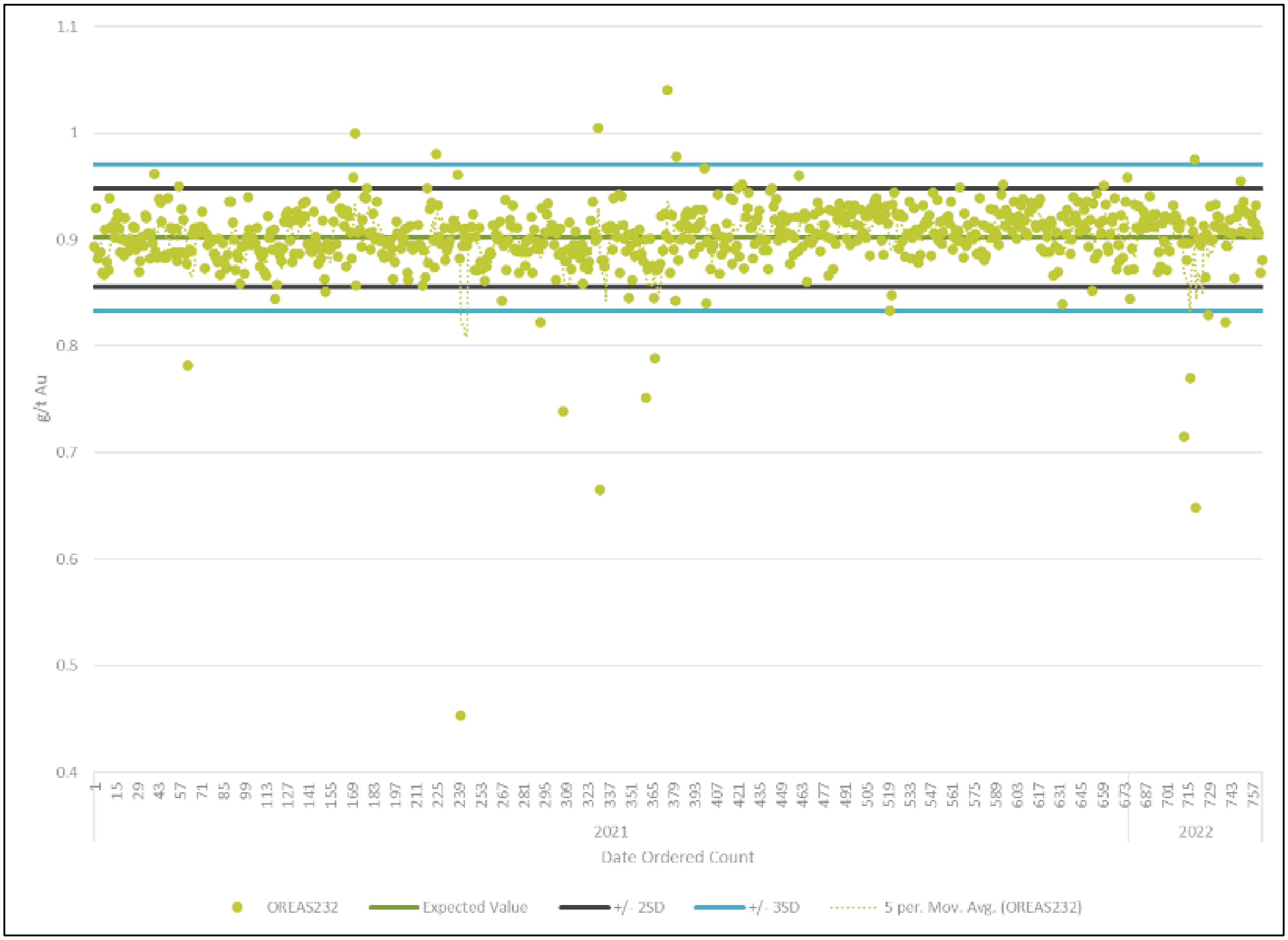
Results from OREAS 226 standard (Figure 11-1) represent average grades for O'Brien material. Results indicate moderate laboratory precision, with no clear grade bias at the grade range (5.45 g/t Au). Of the samples, 39 of the 175 (22.3%) CRMs were outside two SDs; however, only 22 (12.6%) of these were failures. Warnings and failures occurred equally above and below the grade range. OREAS 232 (Figure 11-2) represents low-grade material from O'Brien. In general, there is good accuracy and precision from ALS; however, the data show isolated failures, representing 2.5% of the submitted CRM.

Figure 11-1: Control Chart of CRM OREAS 226



Source: SLR (2023).

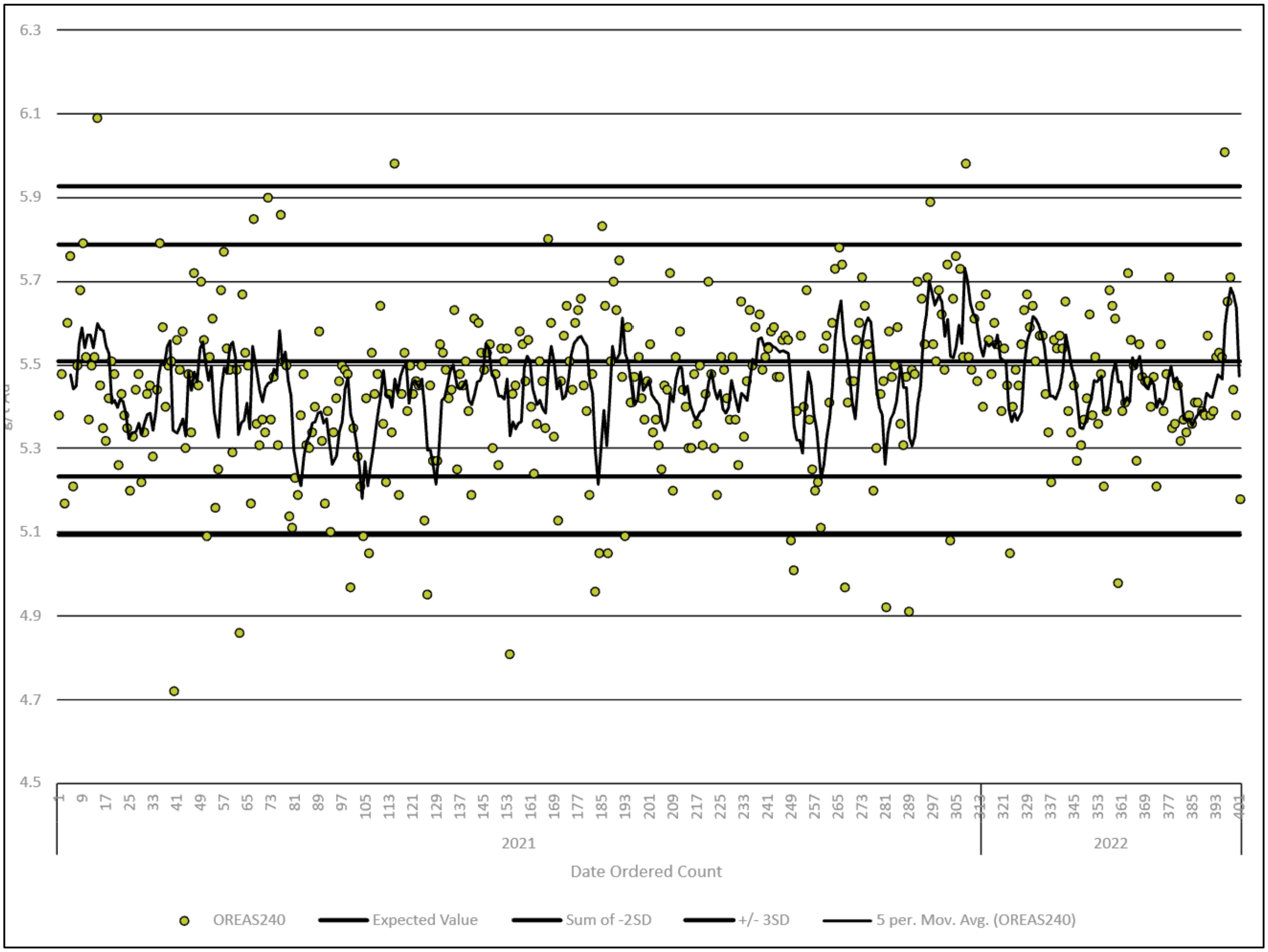
Figure 11-2: Control Chart of CRM OREAS 232



Source: SLR (2023).

OREAS 240 samples (Figure 11-3) represent average grades for O'Brien. Like the results from OREAS 226, these indicate moderate laboratory precision, with no clear grade bias at the grade range (5.51 g/t Au). Of the samples, 61 of the 401 (15%) CRMs were outside two SDs; however, 24 (6%) of these were failures. Warnings and failures occurred equally above and below the grade range.

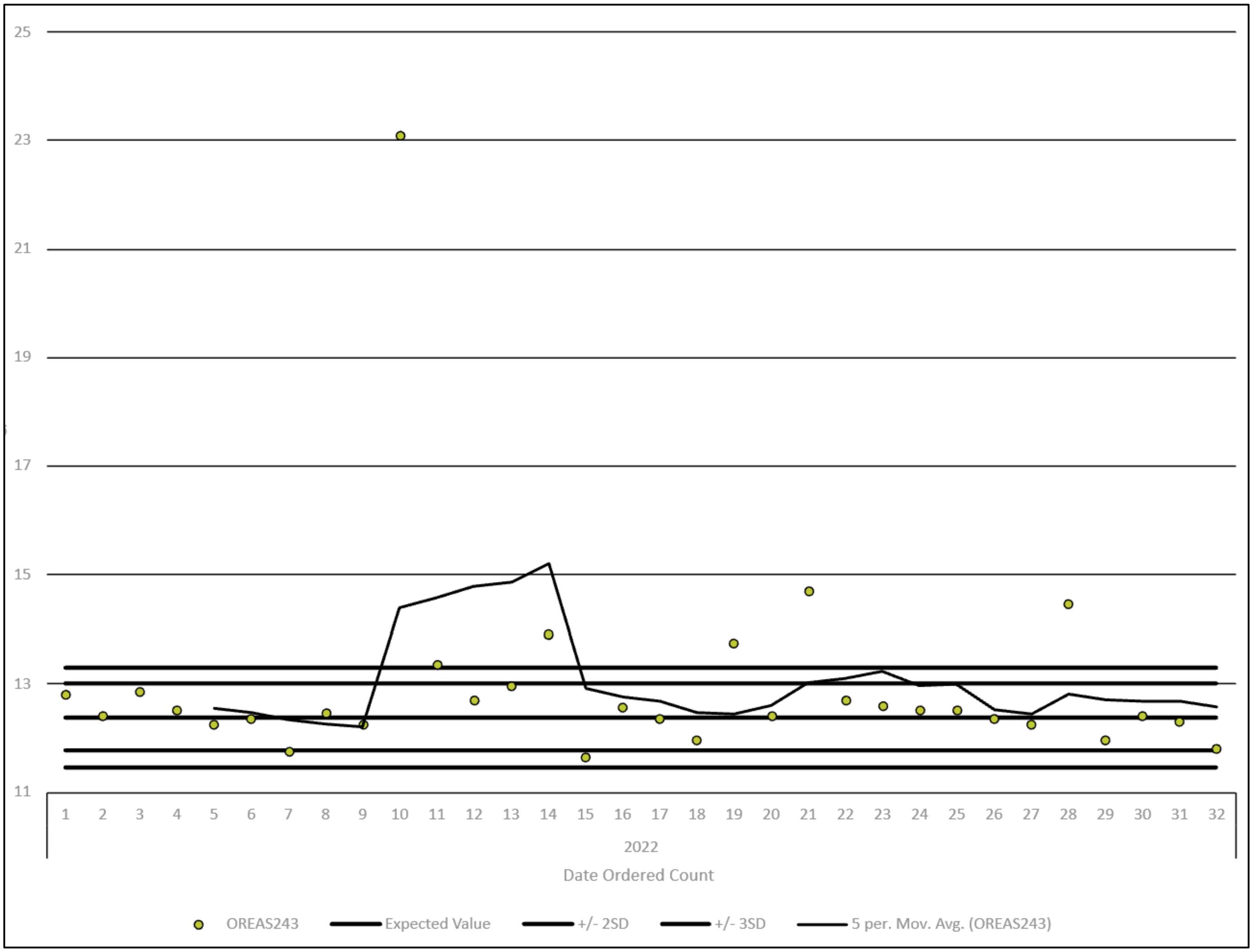
Figure 11-3: Control Chart of CRM OREAS 240



Source: SLR (2023).

OREAS 243 (Figure 11-4) represents high-grade material at O'Brien. In general, there is good accuracy and precision from ALS; however, the data indicate failures occurring with a high-grade bias at the grade range (12.39 g/t Au). Twenty-three out of 184 (12.5%) samples submitted were outside two SDs and 10 (5.4%) were failures, including one sample which reported 23.10 g/t Au, almost twice the expected value. OREAS 243 used 2023 to current have shown no bias.

Figure 11-4: Control Chart of CRM OREAS 243

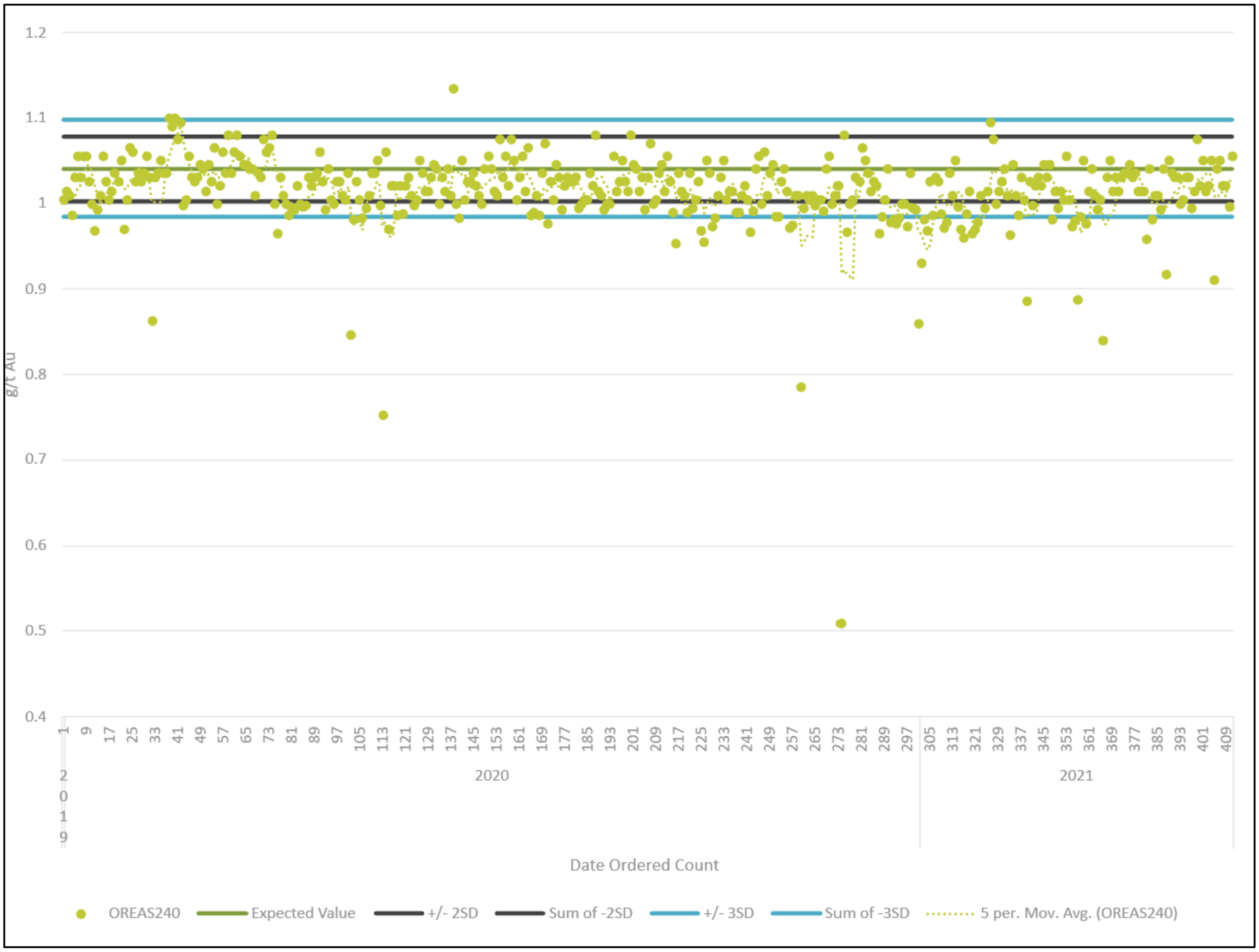


Source: SLR (2023).

SG99 (Figure 11-5) represents low grade material at O'Brien. In general, there is moderate accuracy and precision from ALS; however, the data indicate failures occurring with a low-grade bias at the grade range (1.041 g/t Au), with 131 (32%) samples outside two SDs and 54 (13%) being failures. This CRM is no longer in use at O'Brien.

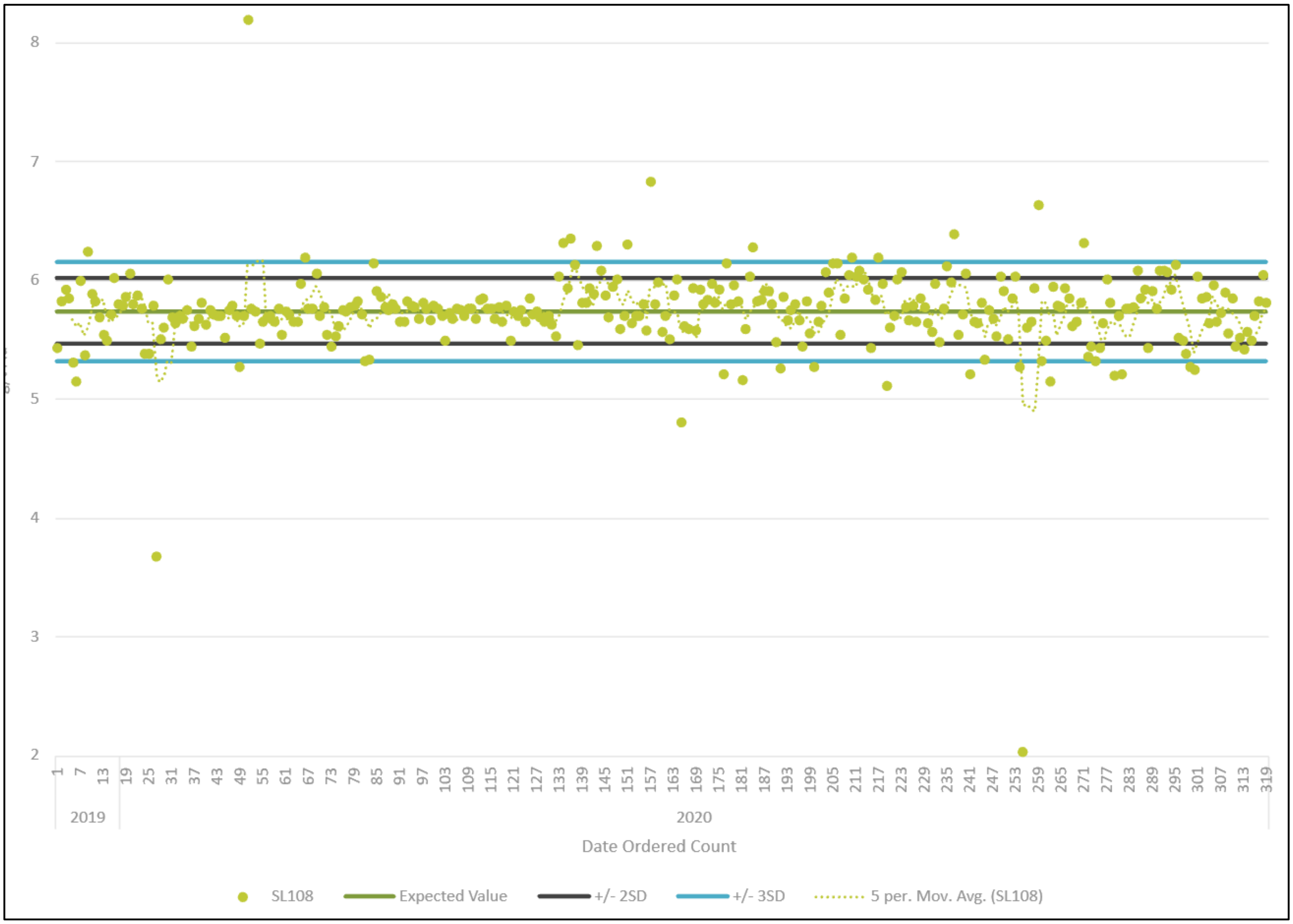
SL108 (Figure 11-6) represents average grade material at O'Brien. In general, there is good accuracy and precision from ALS in 2019 and the beginning of 2020, where the QP notes an increase in spread of the data points.

Figure 11-5: Control Chart of CRM SG99



Source: SLR (2023).

Figure 11-6: Control Chart of CRM SL108



Source: SLR (2023).

This may be attributed to the sample preparation issues evident in 2020, discussed previously in this section. There is no clear bias seen over the grade range (5.744 g/t Au). Over the two years, 79 (25%) samples were outside of two SDs and 34 (11%) were failures (10.7%). This CRM is no longer in use at O'Brien.

11.4.3.2 Blank Material

The regular submission of blank material is used to assess contamination during sample preparation and to identify sample numbering errors. Crushed quartzite was used as blank material. The QP prepared plotted charts of the assayed blank results against an error limit of five times the lower detection limit of the assay technique, or 0.015 g/t Au.

Radisson's quality control protocol indicates values above 10 times the detection limit are conditions for the batch to be re-assayed. None of the samples exceeded 10 times the detection limit.

Results (Table 11-4) indicate a negligible amount of sample contamination associated with the samples, with failure rates below 2% for all years after 2019.

Table 11-4: Expected Values and Ranges of Blank Material

Year	Number of Blanks	Detection Limit (g/t Au)	Detection Limit x5 (g/t Au)	Failing Blanks	Failure Rate (%)
2018	113	0.003	0.015	4	3.5
2019	94	0.003	0.015	1	1.1
2020	707	0.003	0.015	13	1.8
2021	1,287	0.003	0.015	12	0.9
2022	208	0.003	0.015	0	0.0
2023	96	0.003	0.015	0	0.0
2024	512	0.003	0.015	3	0.5
2025	34	0.003	0.015	0	0.0

11.4.3.3 Field, Coarse Reject and Pulp Duplicates

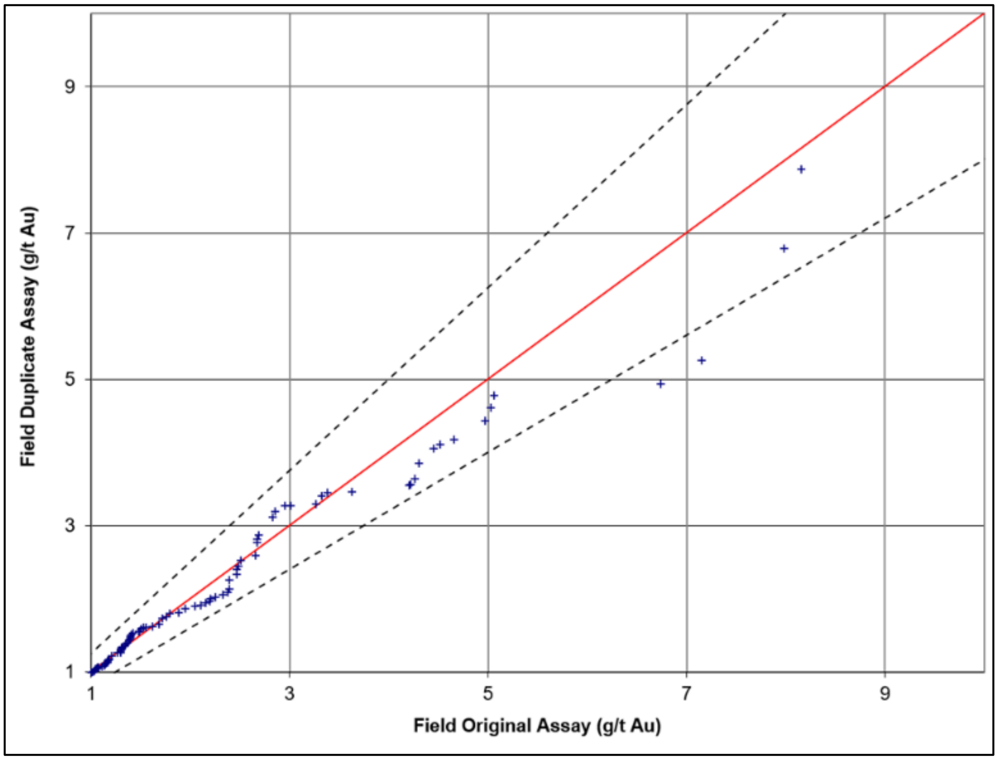
Duplicate samples help to monitor preparation, assay precision, and grade variability as a function of sample homogeneity and laboratory error. QA/QC protocols at O'Brien stipulate the inclusion of field duplicates at a rate of one duplicate every ten samples. Coarse rejects and pulp duplicates were part of the QA/QC procedure at O'Brien up to the 2019 mineral resource estimate (Williamson, 2019) but were then discontinued. The QP recommends that the protocol of inserting pulp duplicates be re-established in future work.

Field duplicates test the natural variability of the core sample, as well as all levels of error including core splitting, sample size reduction in the preparation laboratory, sub-sampling of the pulverized sample, and analytical error.

The QP analysed a dataset of field duplicate data representing the 2019 to 2025 drilling campaigns using basic statistics, scatter, and quantile-quantile plots for 2,278 sample pairs. The resulting correlation coefficient was 0.98. A quantile-quantile plot showing the results for the analysis of field duplicates is presented in Figure 11-7.

The QP is of the opinion that the dataset exhibits a comfortable level of homogeneity between duplicate pairs, expected in this mineralization environment.

Figure 11-7: Q-Q Plot of Field Duplicate Data



Source: SLR (2023).

11.4.3.4 Check Assays

Check assays, or the submittal of duplicate samples to a secondary laboratory, helps to monitor bias at the primary laboratory. The primary laboratory is ALS, with the secondary laboratory being SGS.

In 2021, Radisson sent 5% of pulps for that year to Technilab for re-assay; however, due to internal management and personnel changes, no additional work was done with the data.

The QP recommends that the check sampling program be continued, and that the results from the 2021 program be investigated.

11.4.4 Conclusions and Recommendations

The QP offers the following conclusions and recommendations regarding QA/QC data and results collected for the O'Brien Gold Project:

- The QA/QC program as designed and implemented by Radisson is adequate and the assay results within the database are acceptable for the purposes of mineral resource estimation.

- The results of the CRM program indicate good precision at low grades but show issues at the average and higher grade levels where OREAS 226, OREAS 240, and OREAS 243 reported failures. The QP recommends that these failures be investigated. Where the failure cannot be explained, the entire batch associated with the failing CRM should be re-assayed.
- The program of blank samples indicates low to no laboratory contamination.
- The results of the field duplicate program indicate comfortable homogeneity between paired duplicates, with a correlation coefficient of 0.98.
- The QP notes that the pulp duplicate and coarse reject protocol that was part of Radisson's QA/QC program in 2018 and 2019, was discontinued for unknown reasons after 2019. In early 2023, the QP recommended that it be re-established, particularly with respect to the issues highlighted above with the CRMs. Radisson re-established the protocol in Q3 2023.
- A low-grade bias is observed for CRM samples SG99 (1.041 g/t Au), in place from 2018 to 2022, and for samples SG84 (1.026 g/t Au), in place from 2017 to 2019.
- In 2020 particularly, the QP notes high warning and failure rates of the Rocklabs CRM material, attributed to sample preparation issues. As an additional verification check, the QP recommends that 1 in 20 pulp duplicates from the 2020 dataset be sent for re-assay with OREAS CRMs inserted at a rate of 1 in 10 samples.
- The QP recommends that the program of check assays be continued to help monitor bias at the primary laboratory.

12 DATA VERIFICATION

Data verification measures undertaken by the QP included reviewing the results of the previous data verification procedures and carrying out spot checks on all the data.

12.1 Williamson (2019)

In 2019, Kenneth Williamson 3D Geo-Solution independently verified the historical drill hole database. The work included the following activities:

- visiting the O'Brien Gold Project and reviewing a limited number of collar locations and selected core intervals, and discussing with Radisson geologists the core handling, assaying, density measurement, and QA/QC procedures
- reviewing the drill hole database, including:
  - investigating historical collar locations
  - validating 5% of the historical drill holes, through cross-check routines, looking at survey and assays
  - validating new drilling, focusing on collar surveying, down-hole survey checks, and assay cross-checks against certificates
- a resampling program, wherein 100 samples from six historical holes were quartered and re-assayed
- reviewing the logging, sampling, and assaying procedures
- validating mined-out voids in use.

12.2 SLR Site Verification Procedures

The QP visited the property on October 12, 2022. While on site, the QP held discussions with site personnel and inspected selected core intercepts from several drill holes and compared them against recorded lithology logging and assay results. In addition, the QP reviewed data collection and QA/QC procedures.

The QP regards the geological and mineralization interpretations used to support mineral resource estimation to be consistent with the drill core, and that the Radisson geologists have a good understanding of the geology and mineralization.

12.3 SLR Audit of the Drill Hole Database

The QP reviewed the drill hole database for the project in Leapfrog software, and conducted a standard review of import errors and visual checks. The QP noted a discrepancy between available underground survey information and expected underground collar locations. Radisson was able to subsequently verify and update the underground working locations to bring the drill collars and workings into agreement. No other significant errors were discovered in the database.

A program of database verification was carried out by means of spot checking a random selection of drill holes from a subset of the master drill hole database representing drill holes which intersected the gold mineralization wireframes. Verification activities primarily focused on comparison of assay values contained within the digital database against those values contained within the certificates reported from the assaying laboratories. Comparison of drill hole collar elevations with the topographical surface, as well as available underground survey information, was also completed.

In addition, 22 drill holes that intersected the mineralization wireframes were selected for validation, representing approximately 9% of the drill holes drilled between 2019 and 2022. No discrepancies were found.

The QP is of the opinion that database verification procedures for the O'Brien Gold Project comply with industry standards and are adequate for the purposes of mineral resource estimation.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

This PEA establishes criteria for the development of O'Brien based on processing and tailings management at an existing off-site facility, namely IAMGOLD's Westwood complex, under a toll milling arrangement. A milling assessment for the processing of O'Brien mined material at the Westwood complex was conducted under the auspices of a September 2024 MOU between Radisson and IAMGOLD. The MOU is non-binding and non-exclusive and contains no specific terms around potential commercial arrangements between the parties. IAMGOLD has not independently confirmed the processing assumptions, metallurgical results and/or cost assumptions assumed in this study.

The 2024 metallurgical testwork was performed by SGS on mineralized material from the O'Brien mine from three different spatial categories, the North, Central, and South zones of the deposit. The testwork started in 2024 and was completed in 2025. From these samples, there were four different lithological composites at different As/Au ratios and head grades. The purpose of these tests was to confirm the most suitable flowsheet to maximize the economic outcome for the O'Brien project, as well as to develop a flowsheet for the existing Westwood process plant using process design criteria that is specific to the O'Brien Gold Project.

13.2 Metallurgical Testwork

The metallurgical testwork summarized in Table 13-1 was referenced to develop the process plant design for this project phase.

Table 13-1: Metallurgical Testwork Summary

Year	Laboratory/Location	Testwork Performed
2017	Dundee Sustainable Technologies	Gold concentration – Knelson concentrators
2017		Arsenic extraction - pyrolysis
2017		Gold extraction – chlorination and cyanidation
2018	Centre Technologique des Résidus Industriels	Oxidized pyrite – environmental characterization
2018		Non-oxidized pyrite – environmental characterization
2018		Flotation reject – environmental characterization
2019	SGS Lakefield	Amount of recoverable gravity gold
2019		Gravity and flotation recoveries
2024	SGS Lakefield	Leaching kinetics
2024		Grind size variability
2024		Flowsheet optimization
2024		Pre-aeration influence

13.2.1 Legacy Testwork

The overview in this section of the historical testwork performed on mineralized material from the O'Brien deposit is summarized from a report published by SLR Consulting titled, "Technical Report on the O'Brien Project, Northwestern Québec, Canada," (April 14, 2023).

13.2.1.1 Dundee Sustainable Technologies (2017)

The first iteration of testwork completed for Radisson was performed by Dundee Sustainable Technologies (DST) and reported in their publication "Gold Concentration, Extraction and Arsenic Removal on Material from the O'Brien Deposit," (August 9, 2017). DST conducted laboratory and pilot-scale studies on a 5-tonne sample from the O'Brien deposit, focusing on gold concentration, arsenic extraction, and gold recovery.

A combination of gravity separation and flotation techniques were used to concentrate gold. The results show an achievable gold grade of 34,300 g/t Au for 13.4% of the total gold content using two Knelson concentrators. Gravity separation yielded an overall gold recovery of 47% at a grade of 1,138 g/t Au with a single Knelson concentrator, while cyanidation of the flotation circuits resulted in a 42% gold recovery and grade of 109 g/t Au.

Arsenic removal was successfully demonstrated at laboratory scale, achieving a 95% extraction rate. The method of pyrolysis in a tube furnace was employed to remove arsenic from the sulphide concentrate. This allows extraction of the arsenic in vaporous form, leaving the sulphide as pyrrhotite. Additionally, DST developed a vitrification process to sequester arsenic by integrating arsenic trioxide into a silica matrix, forming a stable glass which meet the standards outlined in the EPA's toxicity characteristic leaching procedure (TCLP) and synthetic precipitation leaching procedure (SPLP).

Gold extraction was evaluated using cyanidation and chlorination across different samples. Standard bottle roll cyanidation tests were conducted over a 48-hour leach period. Considering recovery losses from beneficiation, total gold extraction from untreated mineralized materials was calculated at 85.6%. Removal rates of 83.1% and 88.3% were obtained for concentrates subjected to pyrolysis and post-oxidation, respectively. Chlorination of the oxides concentrate achieved an 88.3% gold recovery, offering benefits of short reaction time and an environmentally sustainable closed-loop process.

13.2.1.2 Centre Technologique des Résidus Industriels

The Centre Technologique des Résidus Industriels (CTRI) performed another phase of testwork in 2018 for Radisson and summarized the results in a report titled, "Rapport Final Projet-107 – Caractérisation métallurgique et environementale," (August 21, 2018). CTRI conducted metallurgical and environmental characterization on three different types of samples: oxidized pyrite concentrate, non-oxidized pyrite concentrate, and a flotation reject.

Metallurgical studies involved laboratory cyanide leaching to assess leaching kinetics and overall gold recovery though cyanidation. Results obtained after 24 hours of reaction time indicated gold recovery rates of 73% to 76% for the pyrite concentrates, and 91% for the flotation reject. These results suggest that an overall gold recovery of 92% could be achieved through extended cyanide leaching.

Environmental characterization involved the TCLP and static tests to determine lixiviation rates and acid generation potential of the samples. TCLP test results indicated that potential lixiviation hazards may arise due to elevated concentrations of arsenic, calcium, and copper, and the static test results confirmed that both pyrite concentrates exhibit acid-generating potential, reinforcing the need for proper environmental management strategies.

13.2.1.3 SGS Mineral Services

The first metallurgical testwork conducted by SGS Mineral Services (SGS) for Radisson on the O'Brien Gold Project was between 2018 and 2019, as detailed in their report "Determination of the Gold Head Grade and Recovery from the O'Brien Project," (March 18, 2019). The primary objective of the testwork was to establish the gold head grade of eight composite samples derived from 120 core samples. These composites underwent gravity gold separation, followed by cyanide leaching of the gravity tailings. The gold head grade was then determined through back-calculation of these results.

To determine the amount of recoverable gravity gold, the composite samples were ground in a laboratory rod mill before being processed through a Knelson MD-3 concentrator and a Mozley C800 laboratory separator. The Mozley concentrate was fully assayed for gold content, leading to calculated recoveries from Knelson/Mozley gravity separation ranging between 30% to 74%. Bulk cyanide leach tests were subsequently performed on the eight gravity tailings composites, with gold extraction varying from 46% to 95% after 72 hours of retention. The combined gold recovery from gravity separation and cyanidation of the gravity tailings yielded results ranging from 63% to 94%.

Due to gold recoveries that were lower than expected, an alternative flowsheet incorporating gravity separation, flotation, regrinding, and cyanidation of the flotation concentrate was explored. The approach assumed that finer grinding would improve gold liberation and exposure, enhancing cyanide leaching efficiency. However, the modified flowsheet did not lead to improved gold recovery, indicating that further optimization of flotation and cyanidation conditions may be necessary.

13.3 2024 SGS Testwork

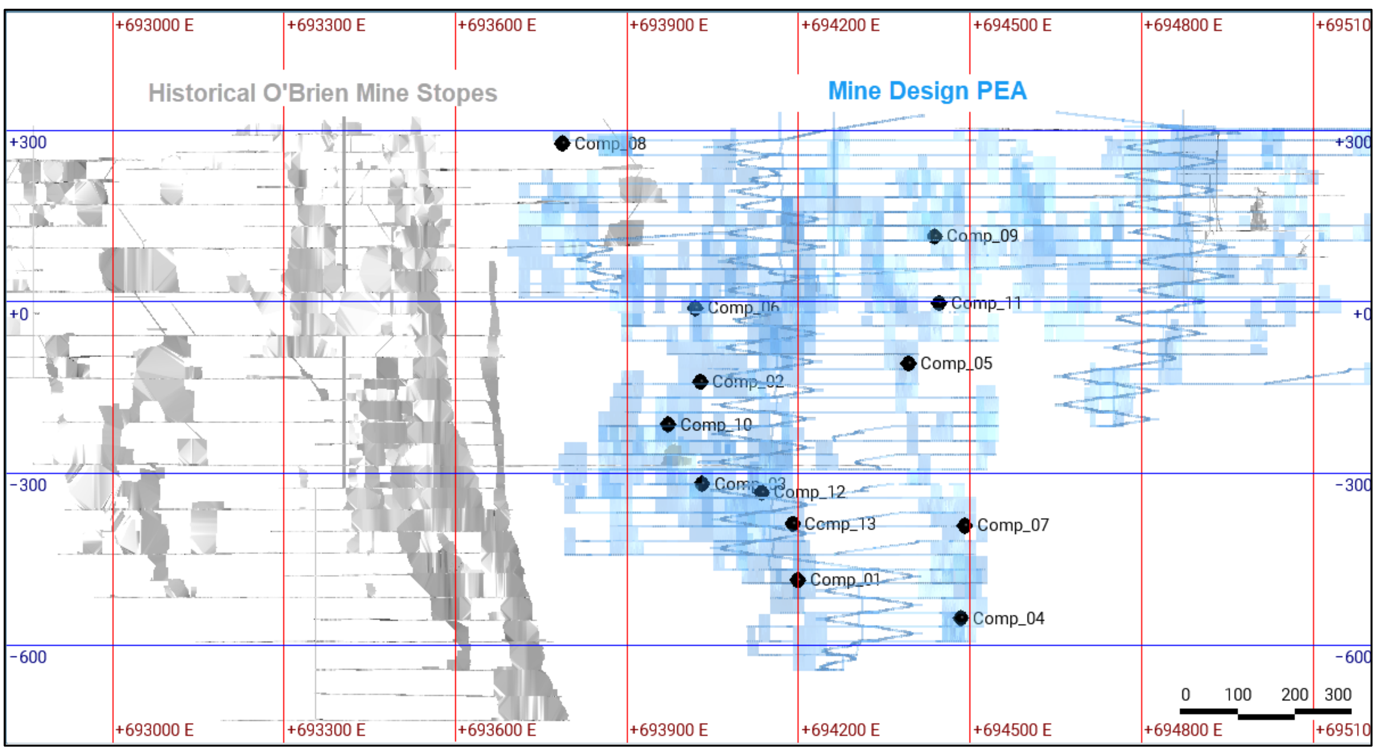
SGS conducted additional testwork in 2024 in two phases: Phase 1 was directed by Radisson, and Phase 2 was directed by Ausenco. Table 13-2 shows the samples selected for testing for both phases. Figure 13-1 shows the locations for the samples.

To direct the Phase 2 testwork, Ausenco reviewed the results of Phase 1 and the historical testwork to identify the proposed testwork methodology. The two phases are described in the following subsections.

Table 13-2: SGS Composite Sample Compositions

Composite	Hole ID	Vein Name	Lithology	From (m)	To (m)	Length (m)	Weight Distribution in Master Composite (%)
Comp_01	OB-20-181	CONG_01_A	POR-S	892.6	897.9	5.3	10.5
Comp_02	OB-20-125	V3-N_10_A	V3-N	516	519.3	3.3	5.4
Comp_03	OB-21-240	V3-N_01	V3-N	801.2	811.7	10.5	15.6
Comp_04	OB-22-308W7	V3-S_10_A	POR-S	960.3	964.3	4.0	6.6
Comp_05	OB-21-202	V3-S_11_A	V3-S/POR-S	465.3	473.3	8.0	11.5
Comp_06	OB-20-122	V3-N_12_A	V3-N	404	408	4.0	6.6
Comp_07	OB-20-156W2	V3-S_10_A	V3-S	748.6	756.6	8.0	13.1
Comp_08	OB-21-281	CONG_07	CONG	64	67	3.0	4.9
Comp_09	OB-17-052	POR-N_01	POR-N	245.5	248.5	3.0	4.1
Comp_10	OB-19-111W2	CONG_05	CONG/POR-N	608.1	611.3	3.2	5.2
Comp_11	OB-21-214	V3-S_07	V3-S	360.2	363.3	3.1	5.1
Comp_12	OB-19-101	POR-N_02_B	POR-N	788	792	4.0	6.6
Comp_13	OB-20-145	CONG_01_A	CONG	832	835	3.0	4.9

Figure 13-1: Locations for Composite Samples 1 to 13



13.3.1 2024 SGS Testwork – Phase 1

In early 2024, Radisson re-engaged SGS to carry out a testwork campaign on the O'Brien deposit to evaluate the optimum flowsheet for gold recovery, comparing a whole ore leaching process with gravity, flotation, regrind, and subsequent leaching.

Whole ore leaching is a process used in gold extraction where the entire material is subjected to a leaching solution, typically cyanide, to dissolve and recover gold. Gravity and flotation processes rely on density differences and sulphide association with gold, respectively.

The Phase 1 testwork included two sets of tests as summarized in Table 13-3.

Table 13-3: 2024 SGS Phase 1 Tests Summary

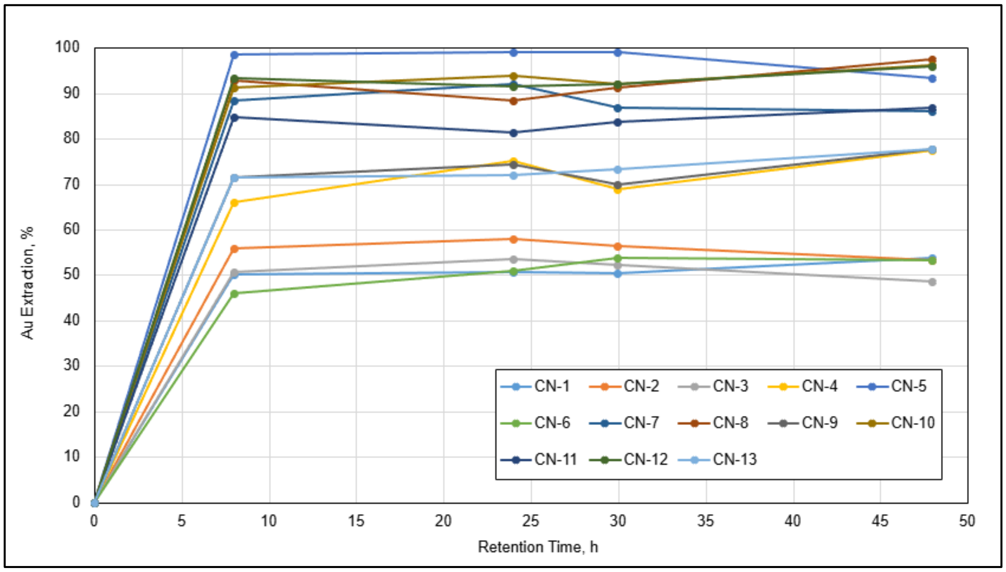
Flowsheet Description	Test Set 1 (Whole Ore Leach)	Test Set 2 (Gravity, Flotation, Leaching)
Whole Ore Leaching	X	-
Gravity Concentration	-	X
Leaching of Gravity Tails	-	-
Flotation of Gravity Tails	-	X
Flotation without Gravity	-	-
Leaching of Flotation Concentrate	-	X
Leaching of Flotation Tails	-	-
Master Composite Used	-	-
Number of Samples Processed	13	11

13.3.1.1 Phase 1 – Test Set 1 (Whole Ore Leach)

Thirteen composite samples underwent whole ore leaching tests to evaluate gold recovery efficiency with a varying feed grade of 2.6 to 6.7 g/t Au and an 80% passing product size P_{80} between 31 and 51 µm. Each sample was leached over a 48-hour period resulting in gold recoveries ranging from 53.4% to 98.0%.

As shown in Figure 13-2, all samples show limited additional recovery after 8 hours of leaching. Additional testwork is recommended for future study phases beyond the 2024 SGS testwork at more frequent intervals to confirm the leaching kinetics.

Figure 13-2: Kinetics of Whole Ore Leaching



13.3.1.2 Phase 1 – Test Set 2 (Gravity and Flotation)

Following the 13 whole ore leach tests, gravity and flotation tests were conducted. Eleven tests were processed by gravity concentration, followed by flotation of the gravity tails. Initially, no leaching was conducted. The gravity concentration stage recovered an average 45% of gold, while the gravity tailings flotation obtained an average gold recovery of 48.3%, leading to an overall gold recovery of 93.1% between the gravity concentrate and flotation concentrate. Phase 1 gravity and flotation results are summarized in Table 13-4.

Table 13-4: Phase 1 Gravity and Flotation Results

SGS Sample ID	Gravity Au Recovery (%)	Au Recovery in Flotation Concentrate (Based on Initial Gold, %)	Overall Au Recovery (%)
V3-N G3F1	6.9	81.6	88.5
V3-S/POR-S G5F2	11.2	75.1	86.3
POR-S G1F3	82.6	13.3	96.0
N3-N/POR-N G2/G9F4	19.0	70.4	89.4
POR-S G4F5	56.7	41.4	98.1
V3-N G6F6	22.4	66.7	89.1
V3-S G7F7	20.1	74.6	94.8
CONG/POR-N G10F8	64.3	32.1	96.4
V3-S G11F9	75.7	19.5	95.2
POR-N G12F10	97.2	0.7	97.9
CONG G13F11	37.3	55.5	92.7
Average	**44.9**	**48.3**	**93.1**

After the Phase 1 gravity and flotation tests, two concentrate composites were created by combining flotation concentrates from the tests listed in Table 13-4. The two flotation concentrate composites were reground to 11 µm (P_{80}) and 14 µm (P_{80}). The reground flotation concentrates and the flotation tails from samples F1 to F11 were then leached for 96 hours. The gold extraction percentage results are summarized in Table 13-5. Note: the gold extraction percentage listed in the table does not include recovery from gravity concentration and represents recovery only from the leaching tests, not overall recovery.

Table 13-5: Phase 1 Flotation Concentrate Leaching Results

Phase 1 SGS Sample ID	Sample Description	Grind Size (µm)	Au Extraction at 48 h (%)	Au Extraction at 96 h (%)
CN14 (Phase 1)	Flotation Concentrate	11	68.4	70.4
CN15 (Phase 1)	Flotation Concentrate	14	69.7	71.4
CN16 (Phase 1)	F1 to F11 Flotation Concentrate Tails	97	64.9	69.0

13.3.2 2024 SGS Testwork – Phase 2

In October 2024, a second phase of the 2024 SGS testwork was initiated by Ausenco and Radisson. The Phase 2 testwork was a continuation of the testwork completed in Phase 1. In addition to the gravity and flotation processes conducted in Phase 1, Phase 2 tested four different flowsheets as summarized in Table 13-6.

Table 13-6: 2024 SGS Phase 2 Tests

Description	Test Set 3 (Gravity & Gravity Tails Leaching)	Test Set 4 (Gravity, Gravity Tails Flotation)	Test 5 (Gravity, Flotation, Regrind, Separate Leaching of Flotation Concentrate & Tails)	Test 6 (Gravity, Flotation, Regrind, Combined Leaching of Flotation Concentrate & Tails)
Whole Ore Leaching	-	-	-	-
Gravity Concentration	X	X	X	X
Leaching of Gravity Tails	X	-	-	-
Flotation of Gravity Tails	-	X	X	X
Flotation without Gravity	-	-	-	-
Regrind of Flotation Concentrate	-	-	X	X
Separate Leaching of Flotation Concentrate	-	-	X	-
Separate Leaching of Flotation Tails	-	-	X	-
Combined Leaching			-	X
Master Composite Used	-	X	X	X
Total Number of Samples Processed	12	3	1	1

13.3.2.1 Phase 2 – Test Set 3 (Gravity and Gravity Tails Leaching)

The first iteration of testwork in Phase 2 tested the variability of grind size on leaching, comparing 40 and 60 µm P_{80} grind size, as well as the influence of pre-aeration of the concentrate before leaching on gold recovery.

Four composite samples underwent gravity concentration to evaluate gold recovery efficiency with a varying feed grade of 5.0 to 8.9 g/t Au at a P_{80} of 150 µm. Gravity recoveries ranged from 25.1% to 46.0%. The gravity concentrate tailings from the four composites were each divided and processed into three separate samples before leaching (for a total of 12 samples):

- 40 µm P_{80}, without pre-aeration
- 60 µm P_{80}, without pre-aeration
- 40 µm P_{80}, with pre-aeration.

Each sample was leached over a 56-hour period resulting in gold extraction ranging from 73.8% to 81.9%. When combined with the gravity concentrate recovery observed in the previous step, overall recoveries from test set 3 ranged from 80.4% to 90.2%. The cyanide consumption was significantly reduced with the additional step of pre-aeration, but a change in gold recovery was not clearly observed, indicating the potential for consuming less CN when pre-aeration is utilized. The results are summarized in Table 13-7.

Table 13-7: Gravity and Gravity Tails Leaching Test Results Summary

Composite & Sample ID	Gravity Au Recovery (%)	Regrind Size Prior to Leaching (P_{80}, µm)	Pre-aeration Time (h)	NaCN Consumption (kg/t of Feed)	Overall Au Recovery (%), Gravity + Leaching
POR G1CN7	46.0	42	4	0.29	90.2
POR G1CN8	46.0	42	-	1.21	89.4
POR G1CN9	46.0	59	-	0.86	88.9
CONG G2CN10	25.1	40	4	0.14	80.4
CONG G2N11	25.1	40	-	0.39	81.9
CONG G2CN12	25.1	60	-	0.39	80.5
V3 G3CN4	29.8	40	4	0.23	84.2
V3 G3CN5	29.8	40	-	0.85	85.4
V3 G3CN6	29.8	60	-	0.66	87.3
S3 G4CN1	41.2	40	4	0.39	87.8
S3 G4CN2	41.2	40	-	1.08	88.5
S3 G4CN3	41.2	60	-	0.91	86.4
Average	**35.5**	**N/A**	**N/A**		**85.9**

13.3.2.2 Phase 2 – Test Set 4 (Gravity and Flotation)

The following testwork was carried out to determine the recovery of gravity and gravity tails flotation concentrate with the effect of flotation feed grind size on recovery:

- Three tests were conducted on gravity concentrate tailings from a "master composite," with a grade of 6.27 g/t and a P_{80} ranging from 91 to 154 µm.
- The gravity tailings were then subjected to flotation and the combined gravity concentrate and flotation concentrate resulted in a range of recoveries from 90.9% to 92.7%.

The results are summarized in Table 13-8 and show a recovery improvement with finer grind size.

Table 13-8: Master Composite Gravity and Flotation Test Results Summary

Composite & Sample ID	Gravity Au Recovery (%)	Grind Size, Prior to Flotation (P_{80}, µm)	Overall Au Recovery (%), Gravity + Leaching
Master Comp G5F1	45.5	91	92.7
Master Comp G6F2	34.6	111	90.9
Master Comp G7F3	36.1	154	91.6
Average	**38.7**	**-**	**91.7**

13.3.2.3 Phase 2 – Test 5 (Gravity, Flotation, Regrind Separate Leaching of Flotation Concentrate and Tails)

The following testwork was carried out to determine the recovery of gravity separation, then flotation of gravity tails into flotation concentrate and flotation tails. The flotation concentrate was reground and leached, and the flotation tails were leached separately.

A single sample of master composite with a grade of 6.27 g/t was processed through this methodology with a primary grind of 82 µm (P_{80}). The recovery from gravity separation was 51.8%. Table 13-9 shows the results of flotation, with the flotation concentrate and flotation tails listed separately. Table 13-10 shows the results of the leaching of the flotation concentrate and tails.

Table 13-9: Master Composite Flotation Results Prior to Separate Leaching

Composite & Sample ID	Recovery in Flotation Concentrate (Based on Flotation Feed Au) (%)	Mass (g)	Mass as Percentage of Flotation Feed (%)	Au Grade (g/t)
Master Comp G8F4 (Flotation Concentrate)	87	199	10	41.1
Master Comp G8F4 (Flotation Tails)	13	1,794	90	0.68
Total	**100**	**1,993**	**100**	**N/A**

Table 13-10: Master Composite Leaching of Flotation Concentrate and Tails Results

Composite & Sample ID	Regrind Size Prior to Leaching (P_{80}, µm)	Pre-aeration Time (h)	NaCN Consumption (kg/t of feed)	Leaching Recovery (%)
Master Comp G8F4 (Flotation Concentrate)	15	8	11.4	90.2
Master Comp G8F4 (Flotation Tails)	85	0	0.2	78.9

The overall gold recovery from the 51.8% gravity separation. Adding the leach recovery from the flotation concentrate (90.2% recovery of 199 g at 41.1 g/t Au) and flotation tails (78.9% recovery of 1794 g at 0.69 g/t Au) results in an overall gold recovery of 90.6%.

13.3.2.4 Phase 2 – Test 6 (Gravity, Flotation, Combined Leaching of Flotation Concentrate and Tails)

The following testwork was carried out to determine the recovery of gravity separation, then gravity tails flotation into flotation concentrate and flotation tails. The flotation concentrate was reground and leached, and the flotation concentrate and tails were leached together. The same master composite as in Test 5, with a grade of 6.27 g/t, was also processed through this methodology with a primary grind of 82 µm (P_{80}). The recovery from gravity separation was 36%. Table 13-11 shows the results of flotation, with the flotation concentrate and flotation tails listed separately. Table 13-12 shows the results of leaching the combined flotation concentrate and tails.

Overall gold recovery from 36% gravity separation, followed by leach recovery from the flotation concentrate and tails (82.8% recovery of 64% of gravity tails), is 88.9%.

Table 13-11: Master Composite Flotation Results Prior to Combined Leaching

Composite & Sample ID	Recovery in Flotation Concentrate (Based on Flotation Feed Au) (%)	Mass (g)	Mass as Percentage of Flotation Feed (%)	Au Grade (g/t)
Master Comp G9F5 (Concentrate)	91.7	199	10	47.7
Master Comp G8F4 (Tails)	8.3	1794	90	0.48
Total	**100**	**1,993**	**100**	**N/A**

Table 13-12: Master Composite Combined Leaching of Flotation Concentrate and Tails Results

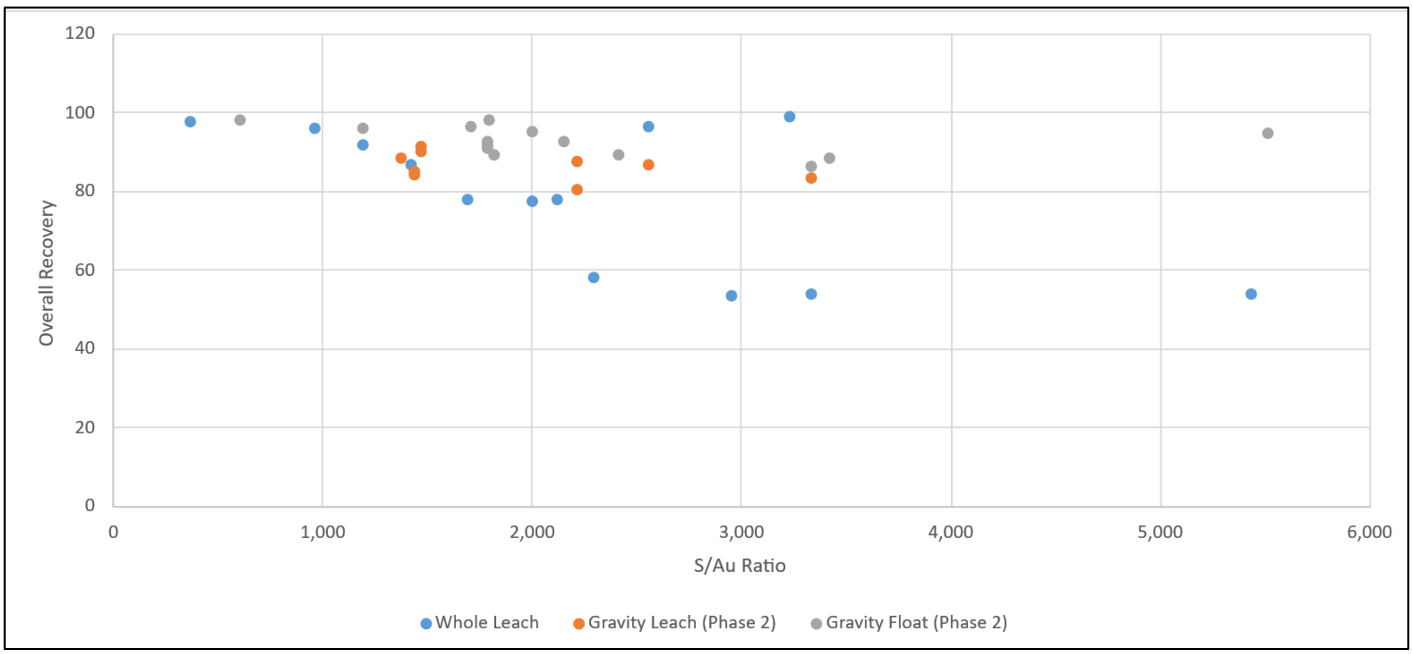
Composite & Sample ID	Regrind Size Prior to Leaching (P_{80}, µm)	Pre-aeration Time (h)	NaCN Consumption (kg/t of Feed)	Leaching Recovery (%)
Master Comp G9F5 Flotation Concentrate and Tails	15 (flotation concentrate) 82 (flotation tails)	0	3.2	82.8

13.4 Selected Flowsheet

The first phase of testwork did not include comminution, cyanide speciation, or soluble arsenic measurement, nor did it consider pre-aeration before leaching. While reviewing historical testwork, it was determined that previous work focused on feed material with a grind size of 40 µm (P_{80}) in alignment with the Westwood complex mill, but this was not exclusively determined optimal for the O'Brien deposit, and deleterious elements must be measured for detoxification circuit designs. Assays also indicated there may be pyrrhotite present in the feed, which can consume cyanide added during leaching without pre-aeration to initiate oxidization, resulting in a higher reagent cost. Further investigation in these areas was carried out during the Phase 1 and Phase 2 testwork, including per-aeration of samples, with lower cyanide consumption observed.

Leaching results from the first phase of testwork indicate a correlation to sulphur and/or arsenic models rather than solely gold grade values. This correlation is shown in Figure 13-3. As a result, sample selection for future testwork should account for a broader range of sulphur and arsenic concentrations, and it is recommended that samples be composited in a method to allow for a further understanding of the effect of sulphur and arsenic on the recovery model.

Figure 13-3: S/Au Ratio vs. Overall Au Recovery



Source: Ausenco (2025).

From the results of this testwork campaign, the proposed flowsheet for the project consists of gravity concentration followed by flotation of the gravity tails, then regrind of the flotation concentrate and combining with flotation tails. This is due to the consistency of the higher recoveries observed when regrinding flotation concentrate and is based on maximizing overall gold recovery by utilizing gravity separation and leaching of the flotation tails.

13.5 Lithology

The lithology of the project area was classified into four different categories (V3, S3, CONG, POR) with three distinct spatial zones (North, Central, South). The composites of the lithological categories are outlined in Table 13-13.

Table 13-13: Lithology Summary

Categorization	Gold Grade (g/t)	As/Au Ratio
V3	11	800+
S3	17	500
CONG	6	900+
POR	5	850

13.6 Deleterious Elements and Compounds

Deleterious elements or compounds are unwanted or harmful elements or compounds in the mineralized material that can negatively impact processing, the environment, or product quality. The O'Brien process contains two deleterious elements or compounds: arsenic, which is found in the feed materials, and pyrrhotite, which was observed to potentially increase cyanide consumption during historical testwork.

Head assays of the tests indicated the presence of arsenic in some samples with an average value of 0.55% and maximum value of 1.51%. Arsenic present in the mineralized material requires proper removal and treatment prior to tailings disposal to ensure regulatory compliance and prevent environmental penalties. Treatment or disposal is required by the receiving facility.

Cyanide is introduced into the process to enhance gold recovery, but direct disposal is not permitted due to environmental concerns. An existing cyanide detoxification circuit at the Westwood complex will be used to detoxify the O'Brien tailings before environmental release.

13.7 Recovery Estimates

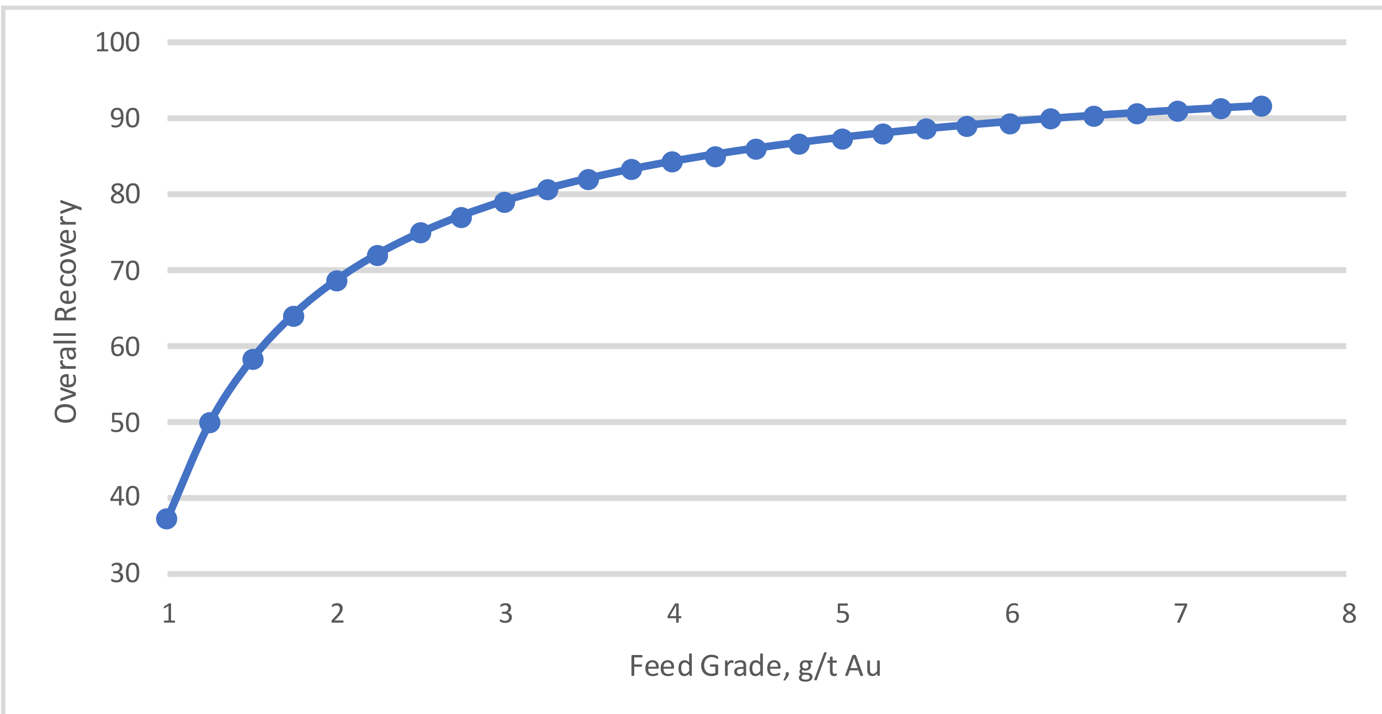
The proposed flowsheet selected from the testwork includes:

- gravity concentration
- flotation
- regrind of flotation concentrate
- leaching of flotation concentrate regrind and flotation tails.

The leachate will be processed into doré at the Westwood complex. There is no evidence from the metallurgical test results to date of any deleterious elements that would impair recovery or result in low-quality doré beyond what is discussed in Section 13.6. Recovery was estimated based on the residual gold grade of the flotation concentrate and flotation tails after leaching. The head grade of the master composite was 6.27 g/t, as used in Tests 5 and 6. With a 90% recovery for a feed grade of 6.27 g/t, supported by the average of the 2024 SGS Phase 2 Test 5 and Test 6, the residual gold grade is calculated as 0.63 g/t.

To develop a recovery curve, the residual gold grade was assumed to be constant regardless of the head grade of the gold. The recovery model based on the flowsheet of gravity concentration, flotation, regrind of flotation concentrate, and leaching of flotation concentrate and flotation tails, is shown in Figure 13-4. Due to the limited available data points at lower concentrations based on the current testwork, the same residual grade is used regardless of head grade. Additional variability testwork is recommended for future project phases to improve and validate the recovery model, noting the impact of the ratio of sulphur to gold in the feed as shown in Figure 13-3.

Figure 13-4: O'Brien Au Recovery Model



Source: Ausenco (2025).

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The mineral resource estimate for the O'Brien Gold Project was prepared by SLR using available drill hole sample data as of January 24, 2023. The mineral resource estimate is based on 1,079 drill hole collars representing 325,509 m of drilling, and 120,352 assay samples. The mineral resource estimate, with an effective date of May 6, 2025, has been prepared in accordance with CIM (2014) definitions, and is presented in Table 14-1. Indicated mineral resources are estimated to total 2.20 million tonnes (Mt) at a grade of 8.22 g/t Au, containing 582 thousand ounces (koz) Au. Inferred mineral resources are estimated to total 6.67 Mt at a grade of 4.35 g/t Au, containing 932 koz Au. Mineral resources that are not mineral reserves do not have demonstrated economic viability.

Mineralized wireframes representing vein structures were prepared in Leapfrog Geo software by Radisson and reviewed and adopted by the QP. Block model estimates were completed by SLR using Leapfrog Edge software using full-length capped composites, and a multi-pass, inverse distance cubed (ID^3) interpolation approach.

Blocks were classified using a novel automated approach which considered local drill hole spacing, composite abundance, and block-grade continuity. Indicated mineral resources were defined where there were contiguous blocks grading above 1.0 g/t Au, with the contiguous blocks containing two or more composites and defined by drill hole spacings of up to 50 m (average distance, by block, to the three nearest composites). All other estimated blocks within wireframes were classified as inferred resources. Final block classification groupings were reviewed, and manual adjustments were made to ensure cohesive classification shapes.

Wireframe and block model validation procedures, including wireframe to block volume confirmation, statistical comparisons with block model and nearest neighbour (NN) estimates, swath plots, and visual reviews in 3D, longitudinal, cross-section, and plan views, were completed.

Wireframes were defined using a nominal true thickness of 1.2 m. All blocks above the cut-off grade of 2.2 g/t Au have been included within the mineral resource estimate and existing mine workings have been excluded. Underground constraining shapes were not used to report the mineral resource but the full width compositing, minimum thickness application to wireframe building, and classification approaches taken in tandem have ensured that there is no selective reporting bias and that the criteria for reasonable prospects for eventual economic extraction (RPEEE) in an underground mining scenario has been met. In addition to SLR's internal peer and senior review processes, Radisson's technical team have reviewed the mineral resource estimate.

Drilling from January 2023 to May 2025 within the O'Brien area has validated the 2023 mineral resource and will not significantly impact the global numbers for that footprint. New drilling outside the O'Brien mineral resources, particularly at depth, appears promising for additional mineral resources, although further drilling is required before estimating mineral resources at depth.

The 2023 Technical Report (SLR, 2023) referred to the Thompson-Cadillac deposit as "New Alger." This was reasonable at the time given that the deposit was sold to Radisson as part of Renforth Resources Inc.'s New Alger property. Now that Radisson's exploration continues to drill along the Cadillac break, Radisson has chosen to rename the area back to the original name for that segment of the break, Thompson-Cadillac.

The only modifications to the mineral resource model between 2023 and 2025 were a lower cut-off grade, reflecting updated cost and metal price assumptions as outlined (Table 14-1), and a reduction in the vertical block size from 10 to 5 m to enhance flexibility in mine design, as detailed in Section 15.

The QP is not aware of any environmental, permitting, legal, title, taxation, socio-economic, marketing, political, or other relevant factors that could materially affect the mineral resource estimate.

Table 14-1: Summary of Mineral Resources, May 6, 2025

Class	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	2,204	8.22	582
Inferred	6,671	4.35	932

Notes: **1.** CIM (2014) definitions were followed for mineral resources. **2.** Mineral resources are reported above a cut-off grade of 2.2 g/t Au based on a C$172.5/t operating cost. **3.** Mineral resources are estimated using a long-term gold price of US$2,000/oz Au, a USD:CAD exchange rate of 1:1.33, and a metallurgical recovery of 90%. **4.** Wireframes were modelled at a minimum width of 1.2 m. **5.** Bulk density varies by deposit and lithology and ranges from 2.00 to 2.82 t/m^3. **6.** Mineral resources that are not mineral reserves do not have demonstrated economic viability. **7.** Numbers may not add due to rounding.

14.2 Comparison to Previous Mineral Resource Estimate

A mineral resource estimate was completed for the O'Brien Gold Project by SLR (2023) and results are compared in Table 14-2.

Table 14-2: Comparison of SLR (2023) and SLR (2025) Mineral Resource Estimates

	SLR (2023)			SLR (2025)			% Differences		
Class	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)	Tonnage (kt)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	1,517	10.26	501	2,204	8.22	582	45%	-20%	16%
Inferred	1,601	8.66	446	6,671	4.35	932	317%	-50%	109%

The principal reason for the changes to the O'Brien mineral resource estimate is the decrease in the reporting cut-off grade from 4.5 to 2.2 g/t Au (SLR) reflecting a change in the long-term gold price from US$1,600/oz Au to US$2,000/oz Au, increased milling recovery to 90% based on testwork as reported in Section 13, and engineering studies confirming the reasonableness of blocks meeting a marginal grade cut-off.

14.3 Mineral Resource Cut-off Grades

Metal prices used for mineral reserves are based on consensus, long term forecasts from banks, financial institutions, and other sources. For mineral resources, metal prices used are slightly higher than those used for mineral reserves.

A cut-off grade of 2.2 g/t Au was estimated for the O'Brien deposit based on an operating cost of C$172.5/t, which includes mining, processing, and general and administration (G&A). Mining and G&A costs were reduced to report marginal blocks. Capital costs, including sustaining capital, have been excluded. Table 14-3 lists the parameters used to calculate the cut-off grade.

Table 14-3: O'Brien Mineral Resource Cut-Off Grade Inputs

Item	Unit	O'Brien
Gold Price	US$/oz	2,000
Selling Cost	C$/oz	5
Exchange Rate	CAD:USD	1.33
Milling Recovery	%	90
Mining Cost	C$/t	110
Processing Cost	C$/t	45
G&A	C$/t	10.5
Hauling	C$/t	7
Total Operation Cost	C$/t	172.5

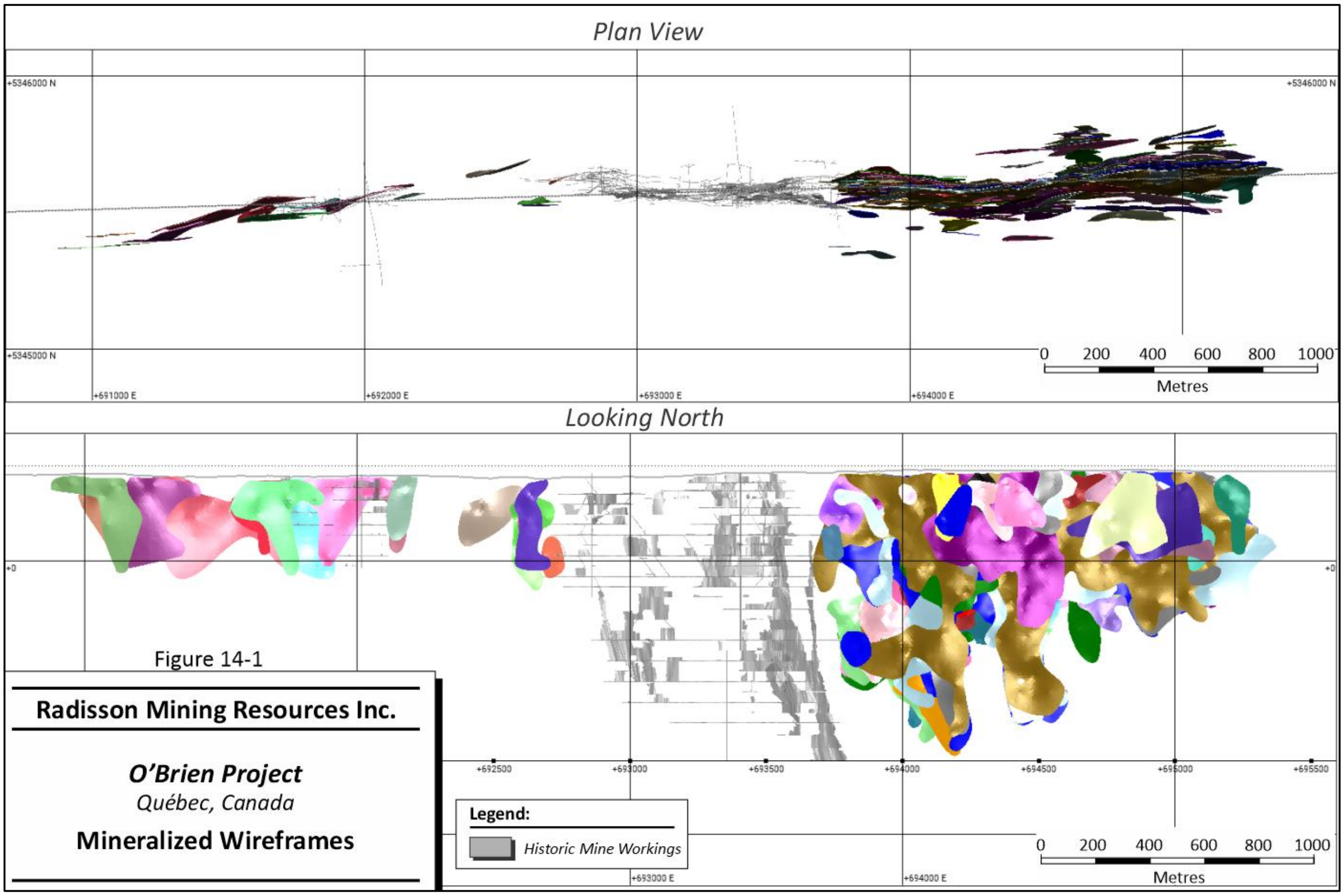
14.4 Resource Database

The drilling data is maintained in a SQL-database, Géoticlog, with drill hole location information in NAD83 projection, UTM Zone 17 using metric units. The final database for the O'Brien mineral resources consists of diamond drilling on 10 to 100 m spacing, 1,079 drill hole collars representing 325,509 m of drilling, and 120,352 assay samples completed from 1932 to 2022.

14.5 Geological Interpretation

The O'Brien mineral resource estimate is based on interpretations of vein structures and vein clusters in 112 mineralization wireframe domains. Wireframe domains were constructed by Radisson geologists using an approximate cut-off grade of 1.0 g/t Au and a nominal true minimum width of 1.2 m. Domain extensions were defined at a limit of 50% of the local drill hole spacing or distance to an excluded drill hole, or 25 m, whichever was smaller. Wireframe domains, constructed using Leapfrog Geo software, were reviewed and adopted by SLR. Vein orientations have been confirmed where possible by underground mapping and sampling, and visual comparison of related underground workings seen in the historical O'Brien mine. The final mineralization wireframe domains are presented in Figure 14-1; the colours represent individual vein wireframes.

Figure 14-1: Mineralized Wireframes



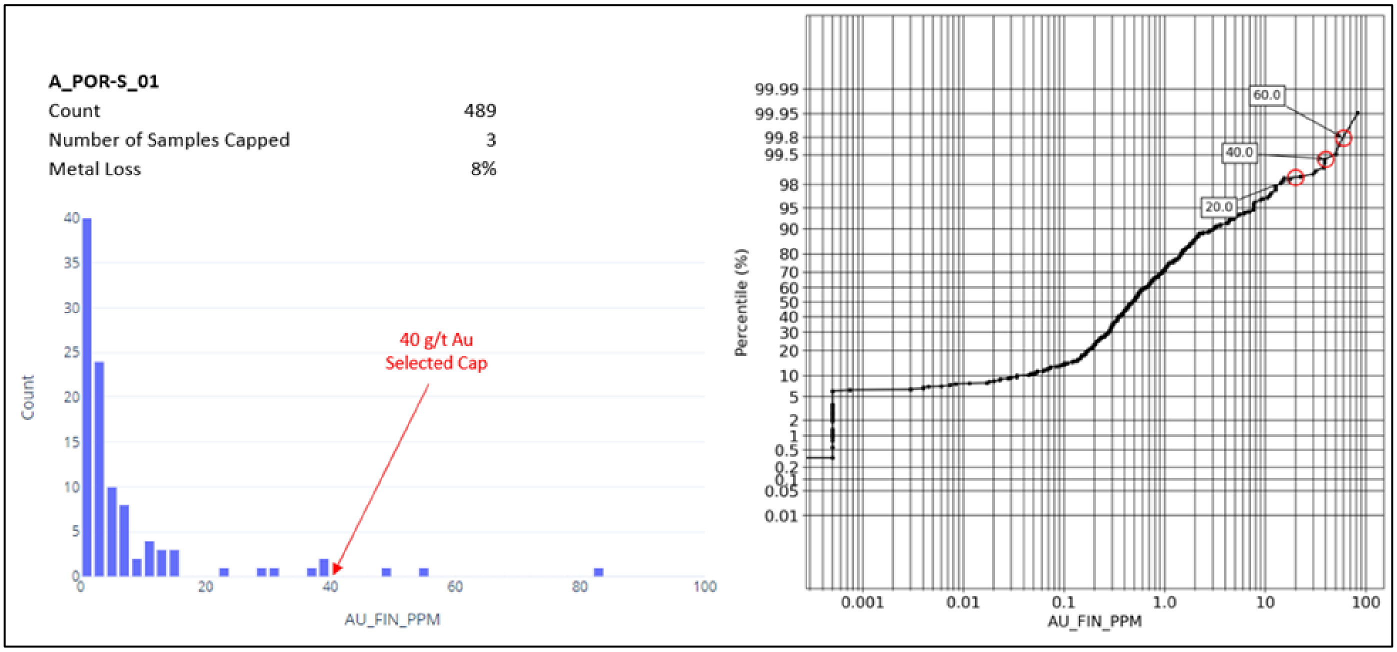
Source: SLR (2023).

The base of the overburden surface was constructed by SLR by offsetting the topographical surface downwards with respect to the logged overburden intervals in the drill holes. All resource wireframes terminate at the bottom of this surface. A lithological model supporting the modelled gold domains was also constructed by Radisson, and represents 13 lithologies, including the overburden.

14.6 Treatment of High-Grade Resource Assays

Length weighted, full length composites were grouped by individual vein, lithological domains, and northings for the purpose of trend analysis of high grades at the project. Average grade, maximum grade, coefficient of variance, and sample population size were considered. Groups were reviewed using histograms, log probability plots, basic statistics, decile analysis, and visuals to determine appropriate capping values. An example of the histograms and log probability plots used to select the capping value is provided in Figure 14-2.

Figure 14-2: Capping Analysis of Vein POR-S_01



Source: SLR (2023).

Historically at the O'Brien mine, grade restriction was accomplished through the application of a grade capping value at the assay level (i.e., the capping value was applied to the assays *prior* to compositing). In this update, to allow consideration of the grade distribution at an equal volume support, SLR composited prior to capping. By compositing all samples to the full vein thickness prior to grade restriction analysis, very short, high-grade samples were normalized to the vein thickness, facilitating the comparison with the longer samples at the deposit.

Assay statistics for ten representative veins are provided in Table 14-4.

A summary of historical capping values is provided in Table 14-5. The impact of the change in methodology change (i.e., capping assays versus compositing prior to capping) was reviewed in terms of metal loss and is summarized in Table 14-6 with respect to hosting lithological domains. While the selected gold capping value of 40 g/t is lower than the capping value used in 2019 (60 g/t) in terms of metal loss at the project, the application yields a comparable result.

Table 14-4: Vein Assay Statistics

Vein Name	Count	Length (m)	Mean (g/t Au)	CV	Lower Quartile (g/t Au)	Median (g/t Au)	Upper Quartile (g/t Au)	Maximum (g/t Au)
A_POR-S_01	1,557	1,261.74	1.86	6.85	0.07	0.34	0.92	409.67
A_V3-S_01_A	1,252	1,155.91	1.63	2.97	0.01	0.17	1.51	73.20
A_V3-S_01_B	1,184	1,285.56	1.27	5.25	0.00	0.03	0.75	331.30
A_POR-S_02	1,160	939.45	1.96	15.77	0.03	0.32	0.95	1,920.00
A_V3-N_16	748	609.27	1.26	2.65	0.02	0.27	1.31	57.94
A_V3-N_15	531	448.11	1.85	1.98	0.02	0.17	2.13	78.17
A_V3-N_14	495	401.87	2.94	8.31	0.02	0.21	2.30	822.86
A_V3-N_01	452	436.22	2.65	5.62	0.01	0.34	1.71	257.49
A_V3-C_09_A	346	299.32	1.06	1.83	0.02	0.22	1.18	13.22
A_CONG_06_A	332	256.01	5.59	5.11	0.14	0.93	3.06	455.00
A_PONT_08	330	391.91	0.93	8.13	0.00	0.02	0.11	185.11

Table 14-5: Historical Capping Levels

Firm	Year	Gold Cap Used
O'Brien Mine	1933 to 1956	2 oz/ton (62.2 g/t) on assays
RPA	1998	2 oz/ton (62.2 g/t) on assays
RPA	2007	2 oz/ton (62.2 g/t) on assays
RPA	2013	1.5 oz/ton (46.7 g/t) on assays
Ken Williamson	2019	60 g/t on assays
SLR	2023 and 2025	40 g/t on composites

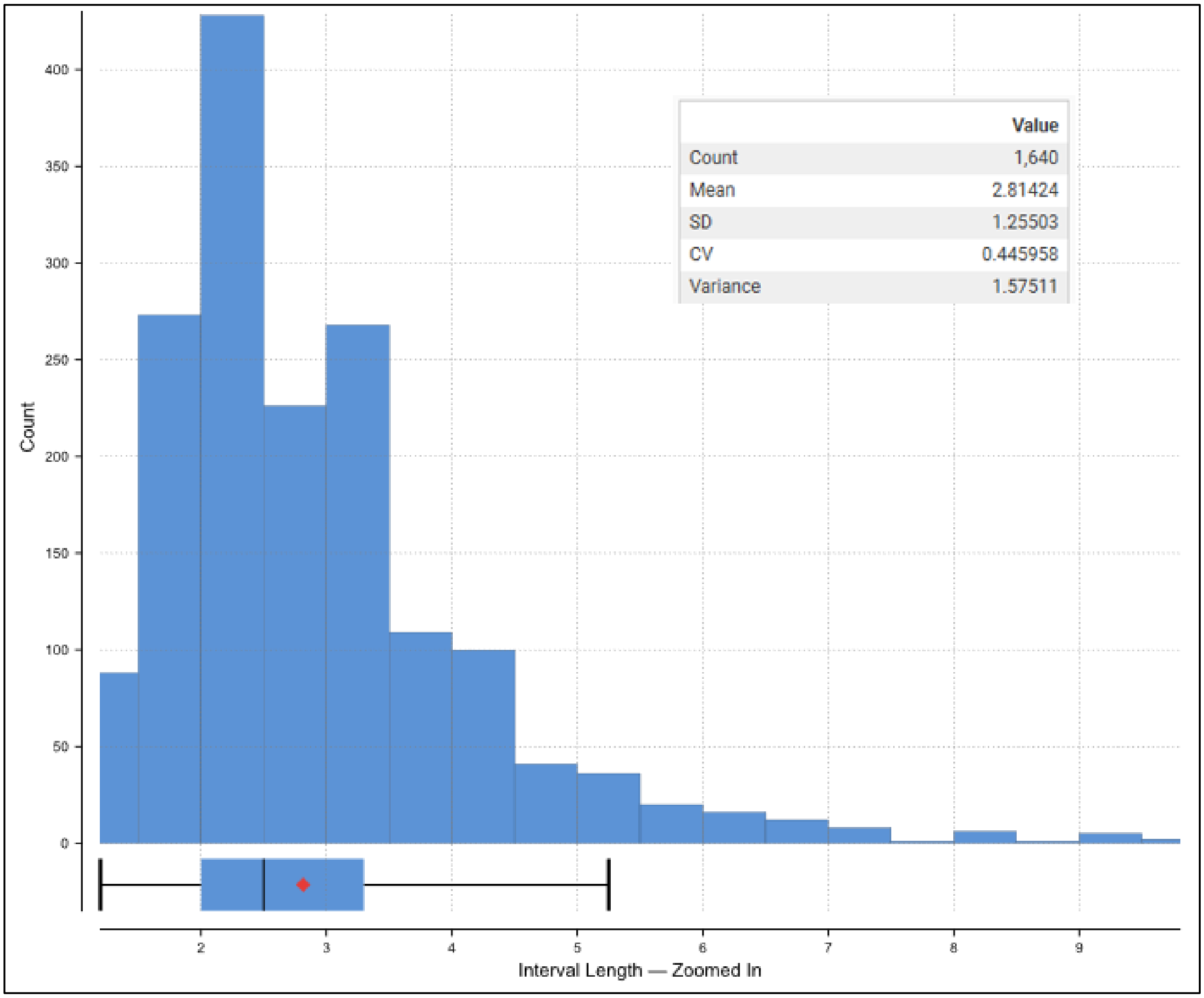
Table 14-6: Capped Assays Versus Capped Composites Statistics and Metal Loss by Domain

Domain	Count	Mean (g/t Au)	Max (g/t Au)	CV	Cap (g/t Au)	No. of Caps	Capped Mean (g/t Au)	Capped CV	Metal Loss (%)
	Capped Assays - Outliers Included								
V3-N	2,889	2.14	1019.14	6.22	60.00	14	1.81	2.73	15
V3-CEN	1,060	1.70	636.70	9.34	60.00	5	1.33	3.57	22
S1p	1,573	2.82	853.40	7.95	60.00	17	1.92	3.32	32
POR-N	1,387	3.27	616.00	7.94	60.00	10	1.92	3.08	41
POR-S	2,732	1.90	1920.00	11.71	100.00	6	1.50	4.15	21
ZFLLC	156	2.04	127.00	5.65	20.00	4	1.11	2.85	46
	Capped Assays - Outliers Removed								
V3-N	2,887	2.02	257.49	4.44	60.00	12	1.80	2.72	11
V3-CEN	1,059	1.36	120.55	4.04	60.00	4	1.30	3.50	4
S1p	1,569	2.25	199.20	4.84	60.00	13	1.86	3.29	17
POR-N	1,384	2.38	371.00	6.35	60.00	7	1.82	2.94	24
POR-S	2,729	1.59	320.23	5.74	100.00	3	1.46	4.06	8
ZFLLC	154	1.05	41.00	3.38	20.00	2	0.95	2.82	9
	Capped Full Length Composites								
V3-N	1,128	2.14	822.86	4.86	40.00	4	1.95	2.04	9
V3-CEN	490	1.70	214.97	5.43	40.00	1	1.42	2.48	17
S1p	710	2.82	157.78	4.20	40.00	9	2.11	2.49	25
POR-N	578	3.27	324.77	4.71	40.00	9	2.44	2.45	25
POR-S	1,029	1.90	132.63	3.88	40.00	5	1.67	2.78	12
ZFLLC	77	2.04	34.71	3.04	15.00	2	1.45	2.37	29

14.7 Compositing

Wireframes were modelled to a nominal 1.2 m minimum thickness, and gold assays were composited to represent the full-length intercept of each domain. Unsampled gold values were assigned a zero value. A histogram of composite lengths within mineralization domains is presented in Figure 14-3. SLR notes that some very long full-length composites are represented by a small number of drill holes which intersect barren mineralization along dip, and some composites shorter than 1.2 m occur in barren areas locally. Most drill holes intersect mineralization domains at oblique angles.

Figure 14-3: Histogram of Composite Interval Lengths within Mineralization Domains



Source: SLR (2023).

14.8 Trend Analysis

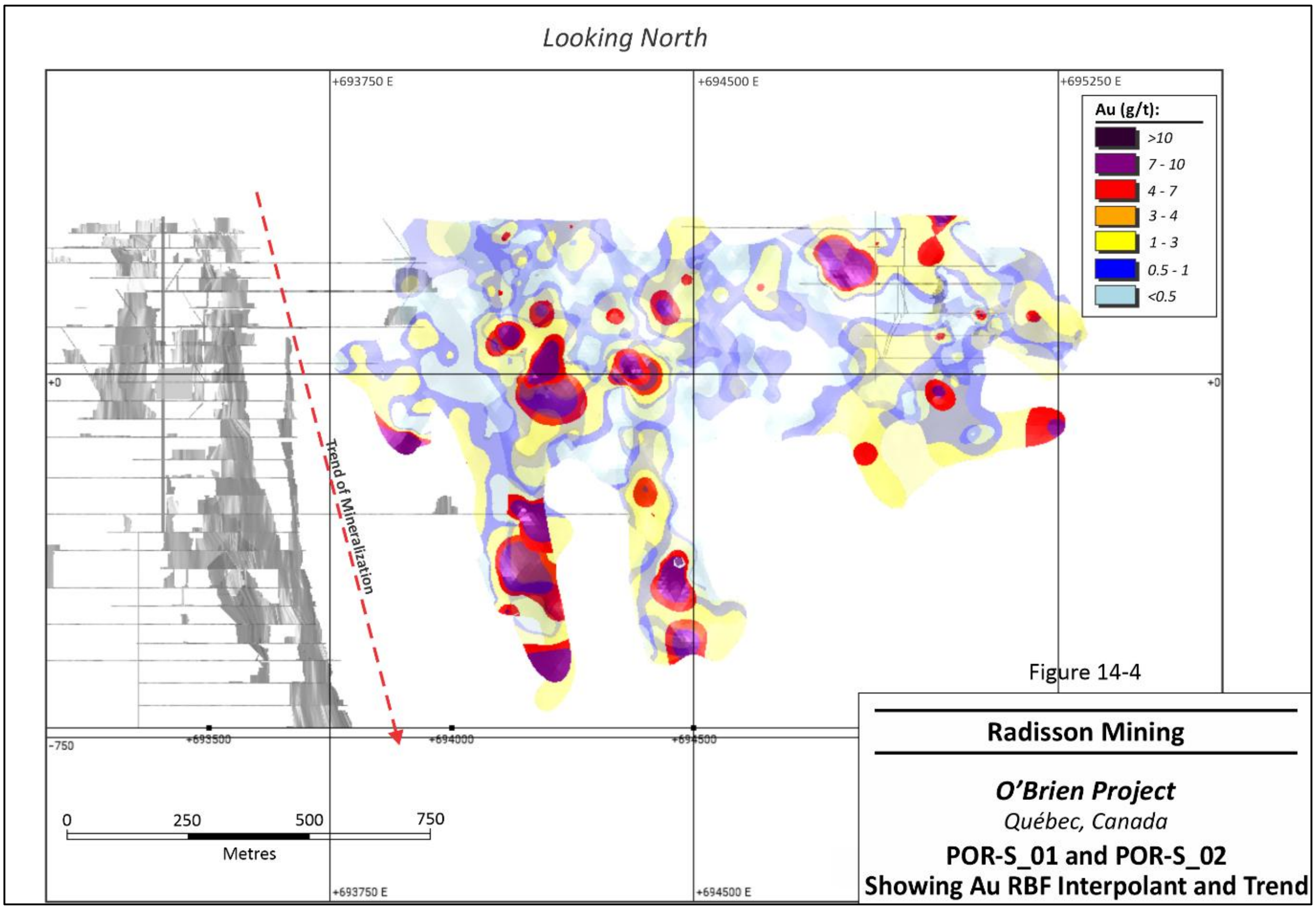
14.8.1 Grade Contouring

As an aid to understanding the distribution and continuity of the gold grades in the domain models, a study to understand overall trends was conducted. As part of this exercise, four of the larger wireframes were selected to provide information for as much of the strike length and depth of the mineralization outlined by drilling as possible.

Gold grades in these domains were contoured in three dimensions using the full-length, capped, composited assay data, using the radial basis function (RBF) interpolant feature of the Leapfrog Geo (Version 22.1.0) software package, and the results were visualized.

Historical workings in the O'Brien mine assisted with understanding gold trends, visible in the orientation of the stope meshes. Due to the presence of multiple tabular, sub-parallel mineralized gold domains, only two of these domains, POR-S_01 and POR-S_02, are shown in longitudinal section in Figure 14-4.

Figure 14-4: POR-S_01 and POR-S_02 Showing Au RBF Interpolant and Trend



Source: SLR (2023).

14.8.2 Variography

Mineralized wireframes using the modelling functions available in the Leapfrog Edge (2021.2.4) modelling software package to establish geostatistical parameters for grade interpolation into the mineral resource block model.

The variogram analysis began with the preparation of both downhole and omni-directional variograms of the gold values to provide a basis for the selection of the variogram nugget (C0). Multiple variograms were then created in the plane of the mineralization wireframes with a range of orientations to identify orientations that provided the best variogram models. A constant nugget was used for all variogram models, while the lag distances were adjusted to accommodate the data spacing characteristics along the given direction under examination. An angular tolerance of 22.5° was used in most cases; however, analysis of the impacts of alternate angular tolerances on the resulting variogram models were also examined.

The quality of the variogram models was limited by the wide-spaced nature of the informing composite samples and the resulting experimental variogram models did not fit the data well. A higher density of data points will be required to obtain a better understanding of the gold grade mineralization and improved variogram models.

14.9 Search Strategy and Grade Interpolation Parameters

Grade interpolation was performed on a parent-block basis using inverse distance cubed (ID^3) and two progressively larger interpolation passes. The first estimation pass corresponded to average drill hole spacing, and the second pass was carried out at two times the size of the first estimation pass. Search ellipses for grade interpolation were oriented using dynamic anisotropy with the longest axis (major) aligned down plunge along mineralization, and the second longest axis (semi-major) aligned along strike.

Estimation runs were carried out separately for each of the 112 mineralization domains. A summary of search parameters used for the estimation is presented in Table 14-7.

Table 14-7: Summary of Search Strategies

Search Parameters	Pass #1	Pass #2
Inverse Distance Power	3	3
Minimum Number of Full-Length Composites	3	1
Maximum Number of Full-Length Composites	5	5
Maximum Number of Samples per Hole	1	1
Length of Major Axis (m)	100	200
Length of Semi-Major Axis (m)	30	60
Length of Minor Axis (m)	30	60
Variable Orientation	Y	Y

14.10 Bulk Density

A total of 4,639 density measurements were available for analysis, 584 of which were from intercepts through mineralized wireframes, as presented in Table 14-8. The 2019 mineral resource estimate used a density of 2.82 g/cm³, a value obtained from 207 samples (Williamson, 2019). Densities within mineralization ranged from 2.63 to 4.51 g/cm³, with one outlying value of 7.74 g/cm³ being removed. In the QP's opinion, these are reasonable densities for this type of mineralization.

Based on the additional available data and new analysis, the QP has selected to continue the use of 2.82 g/cm³ for all rock material and 2.00 g/cm³ for the overburden. While there is a good coverage of density data available, the QP recommends Radisson continue to collect data, particularly within underrepresented lithologies.

Table 14-8: Density Statistics Within Mineralization by Host Lithology

Lithology	Count	Length (m)	Mean (g/cm³)	Minimum (g/cm³)	Median (g/cm³)	Maximum (g/cm³)
CAD-S1	1	0.12	2.77	2.77	2.77	2.77
CAD-S3	6	0.76	2.82	2.72	2.75	3.00
PON-S3	38	4.48	2.81	2.67	2.79	3.39
POR-N	57	7.57	2.76	2.49	2.74	3.15
POR-S	91	11.74	2.78	2.53	2.76	3.16
S1p	56	7.06	2.78	2.59	2.78	3.17
S3p	25	3.13	2.83	2.33	2.81	3.45
V3-CEN	55	6.34	2.82	2.63	2.81	3.02
V3-N	128	16.66	2.86	2.62	2.85	3.23
V3-S	123	17.77	2.85	1.81	2.83	3.31
ZFLLC	4	0.40	2.88	2.77	2.89	3.02
Total	**584**	**76.03**	**2.82**	**1.81**	**2.81**	**3.45**

Note: statistics weighting: length-weighted.

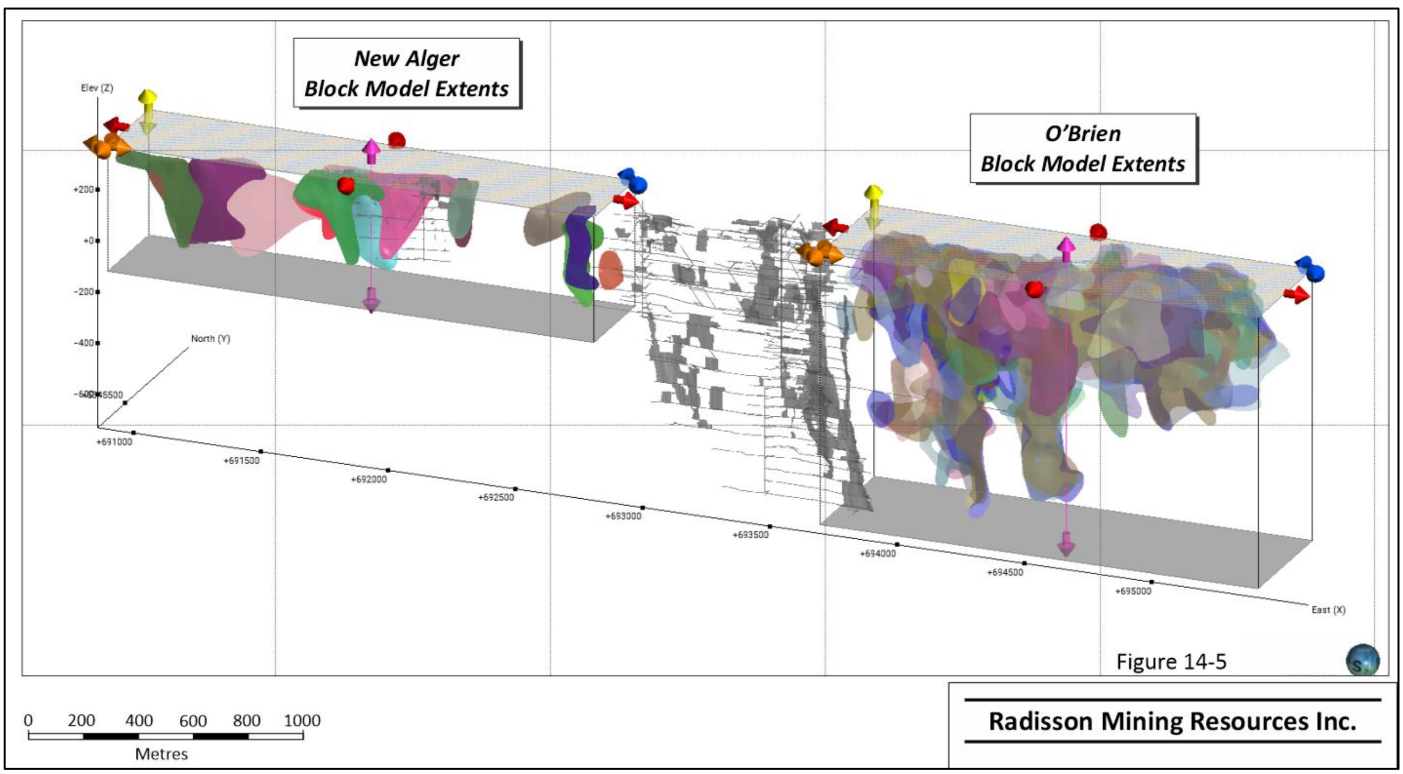
14.11 Block Models

Two block models were constructed for the O'Brien mineral resource estimate to represent the Thompson-Cadillac and O'Brien deposits. Other than location and dimension, all other block modelling and estimation parameters were identical for the two deposits. Block model construction and estimation was completed in Leapfrog Edge software. Block model dimensions for O'Brien and Thompson-Cadillac are presented in Table 14-9 and Figure 14-5. In Figure 14-5, the colours represent individual vein wireframes. The QP considers the block model sizes appropriate for the deposit geometry and proposed mining methods.

Table 14-9: Block Model Definition for the O'Brien and Thompson-Cadillac Deposits

Deposit	Parameter	Units	X	Y	Z
O'Brien	Base Point	m	693,670	5,345,300	335
	Boundary Size	m	1,720	526	1,060
	Parent Block Size	m	5	2	5
	Min. Sub-block Size	m	1.25	0.5	0.625
	Rotation	°	0	0	0
	Size in Blocks		344	263	212
Thompson-Cadillac	Base Point	m	690,860	5,345,330	335
	Boundary Size	m	1,910	396	490
	Parent Block Size	m	5	2	5
	Min. Sub-block Size	m	1.25	0.5	0.625
	Rotation	°	0	0	0
	Size in Blocks		382	198	98

Figure 14-5: O'Brien and Thompson-Cadillac Block Model Extents



Source: SLR (2023).

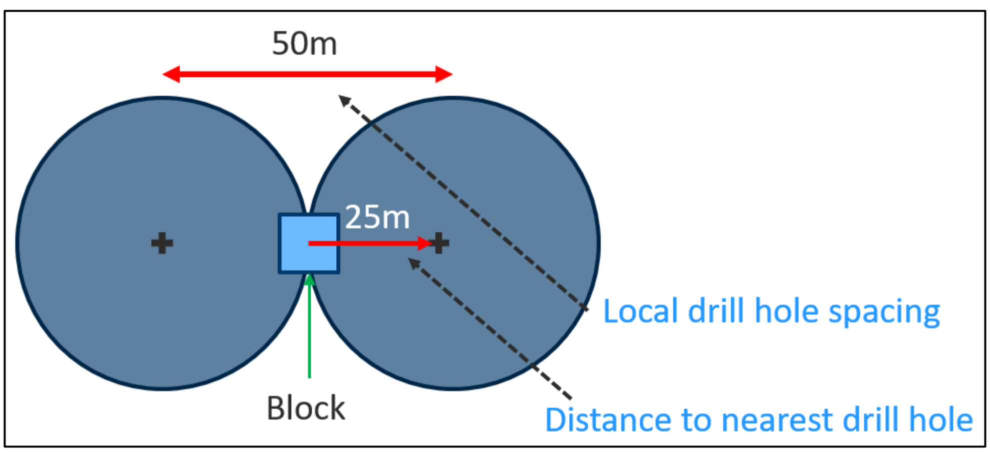
14.12 Classification

Definitions for resource categories used in this technical report are consistent with those defined by CIM (2014) and adopted by NI 43-101. In the CIM classification, a mineral resource is defined as “a concentration or occurrence of solid material of economic interest in or on the Earth’s crust in such form, grade or quality and quantity that there are reasonable prospects for eventual economic extraction.” Mineral resources are classified into measured, indicated, and inferred categories. A mineral reserve is defined as the “economically mineable part of a measured and/or indicated mineral resource” demonstrated by studies at pre-feasibility or feasibility level as appropriate. Mineral reserves are classified into proven and probable categories.

Blocks were classified using a novel automated approach which considered local drill hole spacing based on the average distance to the closest three drill holes (as opposed to the distance to the nearest drill hole (Figure 14-6), composite abundance, and block-grade continuity. Indicated mineral resources were defined where there were contiguous blocks above 1.0 g/t Au, where the contiguous blocks contained two or more composites and drill hole spacing of 50 m or less was achieved. All other estimated blocks within wireframes were classified as inferred resources. Final block classification groupings were reviewed, and manual adjustments made to ensure cohesive classification shapes. Due to the presence of multiple tabular, sub-parallel mineralized gold domains, only two of these domains are presented in longitudinal section in Figure 14-7.

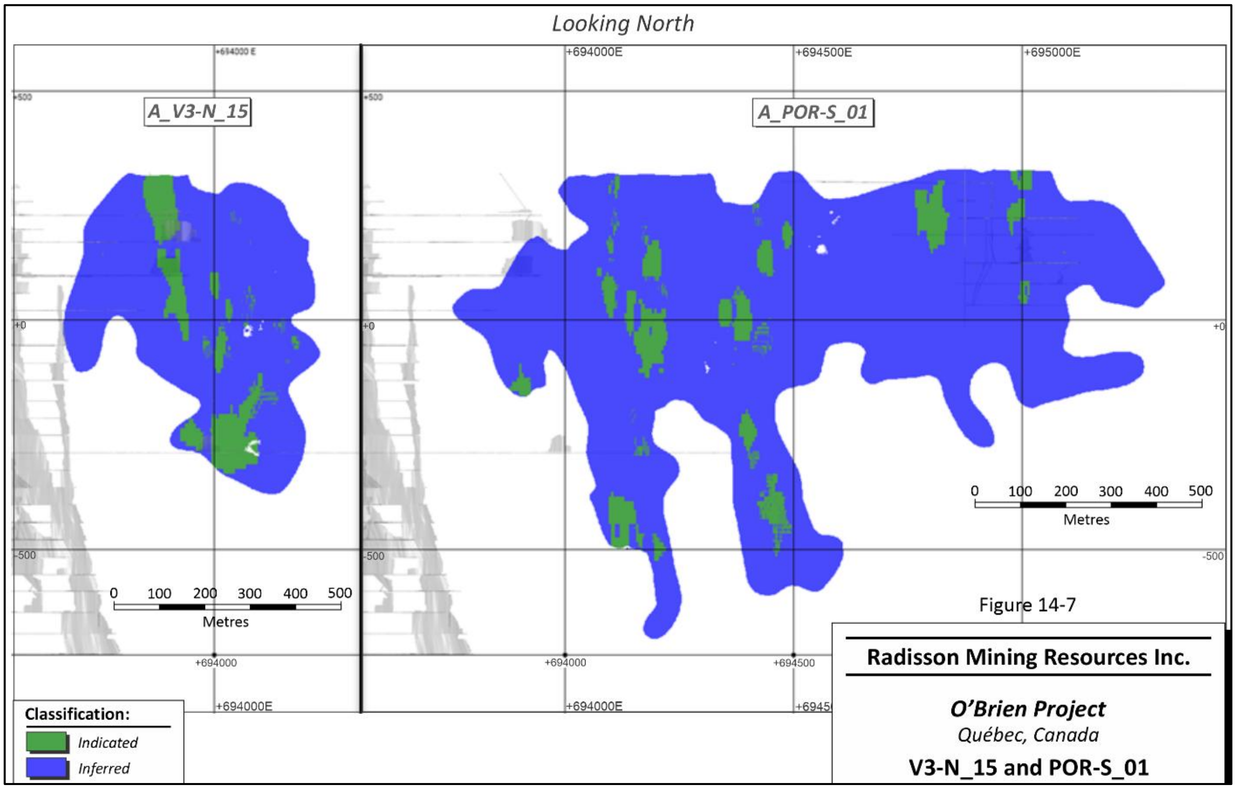
Underground constraining shapes were not used to report the mineral resource, but the full width compositing, minimum thickness application to wireframe building, and classification approaches taken in tandem have ensured that there is no selective reporting bias and that the criteria for the mineral resources meeting reasonable prospects for eventual economic extraction (RPEEE) in an underground mining scenario have been met.

Figure 14-6: Drill Hole Spacing Versus Distance to Drilling



Source: SLR (2023).

Figure 14-7: V3-N_15 and POR-S_01 Block Classification



Source: SLR (2023).

14.13 Block Model Validation

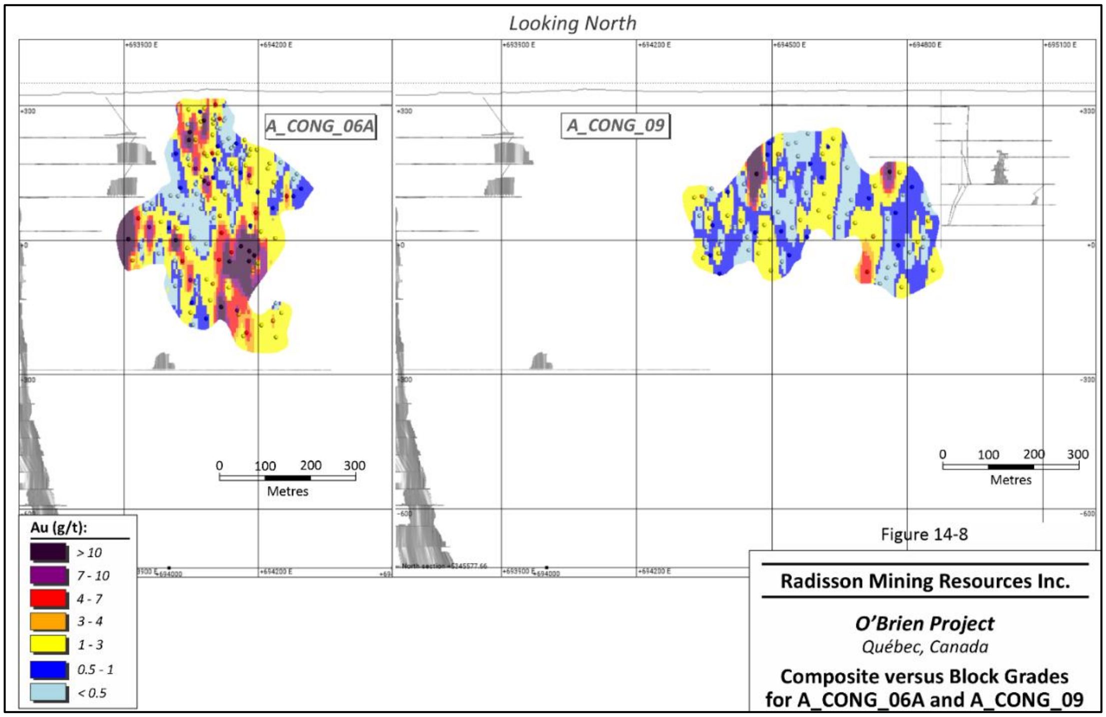
Blocks were validated using industry standard techniques, including the following:

- visual inspection of composite versus block grades (Figure 14-8)
- comparison between ID^3, nearest neighbour (NN), and composite means
- swath plots (Figure 14-9).

The QP reviewed gold grades and proportions relative to the blocks, drilled grades, composites, and modelled solids. SLR observed that the block grades exhibited general accord with drilling and sampling, and did not appear to smear significantly across sampled grades.

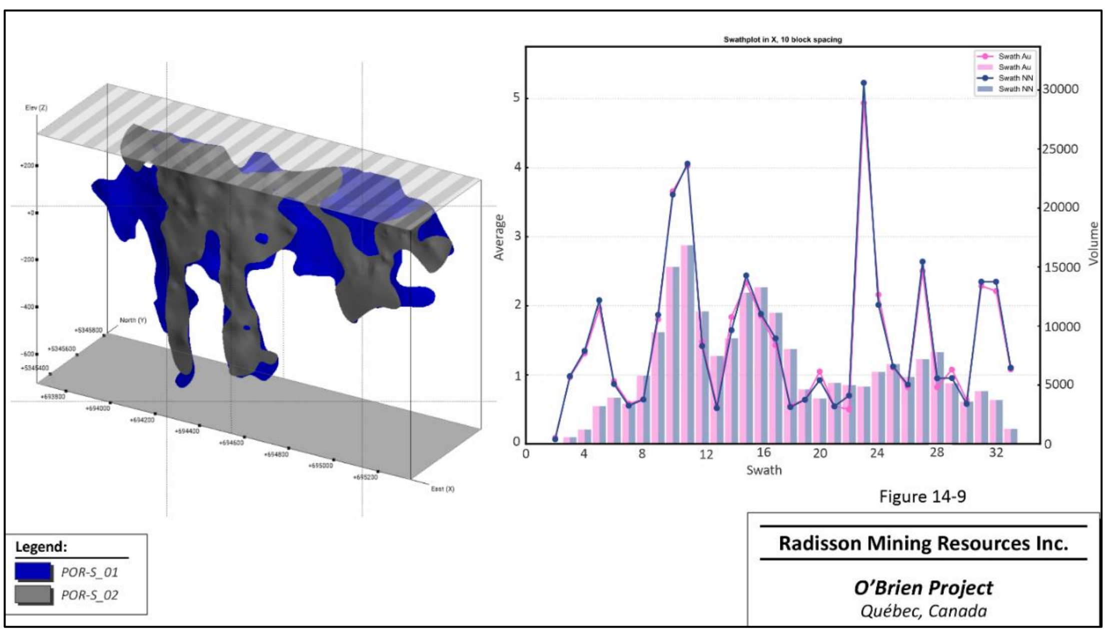
Swath plots generally demonstrated good correlation, with block grades being somewhat smoothed relative to composite grades, as expected.

Figure 14-8: Composites Versus Block Grades for A_CONG_06A and A_CONG_09



Source: SLR (2023).

Figure 14-9: Swath Plot in X-Axis for POR-S_01 and POR-S_02



Source: SLR (2023).

14.14 Mineral Resource Reporting

Mineral resources at the O'Brien Gold Project are reported as per the mineral resource estimation methodologies and classification criteria detailed in this technical report, using a cut-off grade of 2.2 g/t Au, and are summarized in Table 14-10. Existing mine workings have been depleted from the mineral resources.

Table 14-10: Summary of Mineral Resources, May 6, 2025

Class	Tonnage (000 t)	Grade (g/t Au)	Contained Metal (koz Au)
Indicated	2,204	8.22	582
Inferred	6,671	4.35	932

Notes: **1.** CIM (2014) definitions were followed for mineral resources. **2.** Mineral resources are reported above a cut-off grade of 2.2 g/t Au based on a C$172.5/t operating cost. **3.** Mineral resources are estimated using a long-term gold price of US$2,000/oz Au, a US$/C$ exchange rate of 1:1.33, and a metallurgical recovery of 90%. **4.** Wireframes were modelled at a minimum width of 1.2 m. **5.** Bulk density varies by deposit and lithology and ranges from 2.00 to 2.82 t/m^3. **6.** Mineral resources that are not mineral reserves do not have demonstrated economic viability. **7.** Numbers may not add due to rounding.

15 MINERAL RESERVE ESTIMATES

This section is not applicable to a preliminary economic assessment.

16 MINING METHODS

16.1 Introduction

This section discusses the proposed mine plan developed by Norda Stelo for the current preliminary economic assessment (PEA). The mine plan presented herein is based on the block model used to report the 2023 mineral resource estimate. The indicated and inferred resources of the 2023 mineral resource estimate were converted into economically mineable shapes using the 2023 mineral resource block model (following a reduction in the vertical block size from 10 to 5 metres to enhance flexibility in mine design), with mining parameters applied to define the economic potential of the PEA mine plan. The underground mining scenario targets subvertical mineralized veins.

The proposed O'Brien Gold Project would be an underground operation with a mining method that has been optimized to suit the deposit geometry. Longitudinal longhole retreat has been selected as the preferred mining method due to the subvertical orientation and continuity of the mineralized veins. The narrow width and subvertical dip of the veins further support the application of this method, as it allows for efficient extraction while maintaining safety and minimizing dilution. The project includes two mining zones: (1) the O'Brien East zone located, to the east of the former O'Brien mine, and (2) the Kewagama zone, located around the historical workings of the former Kewagama mine. Each zone will have its own main decline, portal, and ventilation network; however, an underground drift connecting the two zones will be driven to facilitate handling of the mineralized material. The Thompson-Cadillac area was not considered as part of the PEA.

A vertical conveyor capable of handling a daily productivity of 1,400 t/d will be used to bring the mineralized material from Level 300 to the surface. During the period when the vertical conveyor has not yet been commissioned, a fleet of 20-tonne underground trucks will transport the mineralized material to the surface dump area.

In the O'Brien East and Kewagama zones, production levels are connected by internal ramps, and each level includes all the necessary infrastructure required for mechanized longhole stoping. Major underground infrastructure, such as the maintenance bay, crusher, and main pumping station, is all located in the O'Brien East zone but will serve both zones.

Production mining will use numerous production fronts to maximize productivity and flexibility. Production centres are defined to maximized efficiency, depending on the development schedule. Mining each production centre (incorporating typically five levels) will involve ascending from the lowest to the highest level. The sill pillar level will be recovered using upper mining stopes and will not be backfilled. To ensure stability and safety, the level above the sill pillar level will be backfilled with cemented rockfill containing a higher proportion (7%) of cement. To reduce costs on lateral development, stopes located at the apex of mining areas will typically be mined using upper mining stopes. The project will be an underground mining operation using a mixed approach of both contractor and company-operated personnel.

16.2 Geotechnical Considerations

16.2.1 Past Studies

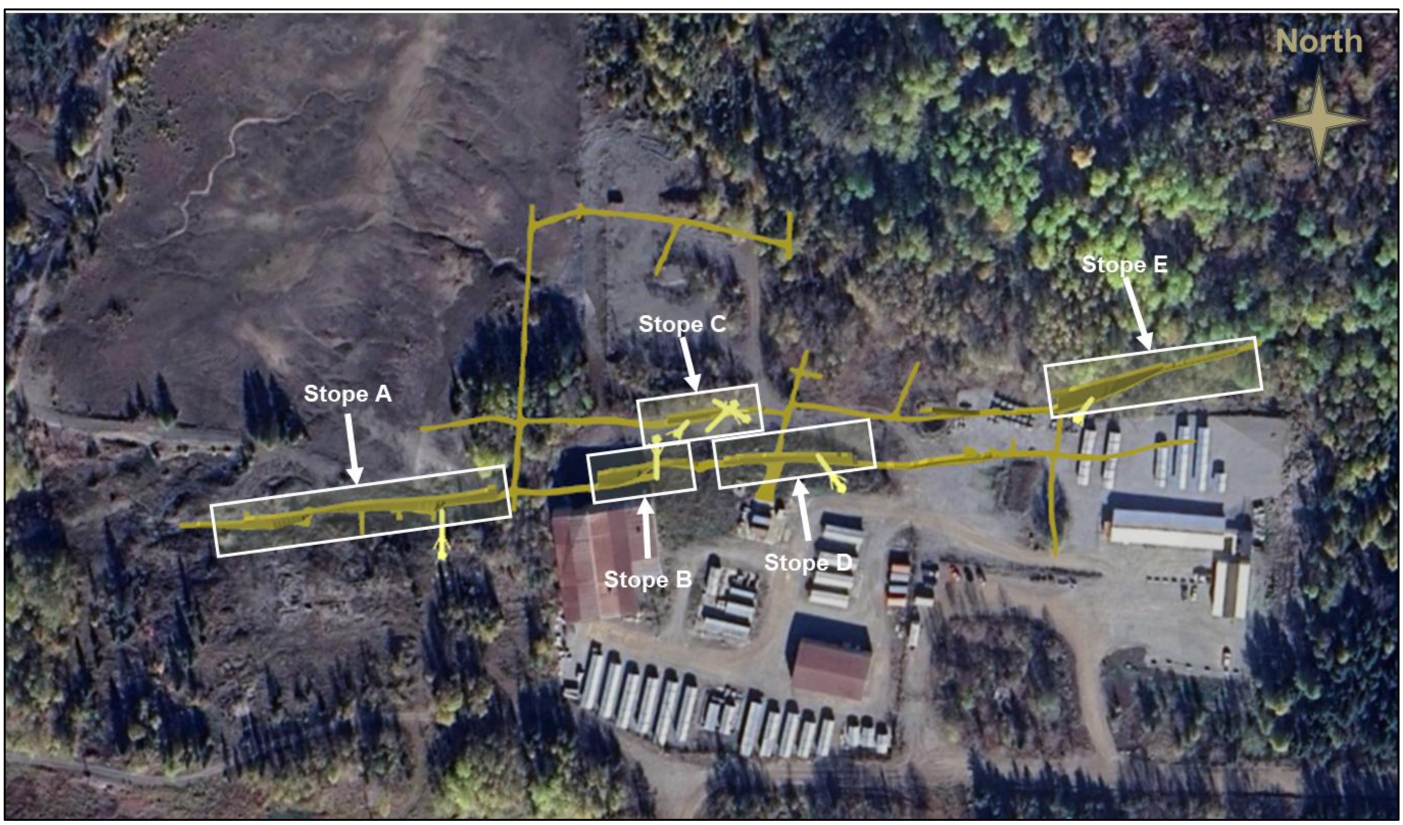
Golder Associates Ltd. (now WSP Global Inc.) conducted two geomechanical studies in the following areas:

- stability of the crown pillars for the old O'Brien mine (Golder, 2020)
- rock mass characterization at the O'Brien site (Golder, 2021).

These studies provided the basis for the geomechanical parameters used in this technical report, and were used to determine the stope design parameters and assess the stability of the surface crown pillars.

Assessing the stability of the crown pillars for the old O'Brien mine consisted of evaluating four old stopes located near the surface (Figure 16-1) in proximity to the existing surface facilities (Golder, 2020). Seven geomechanical holes for a total of 137 m were drilled to gather information on the crown pillar resting on top of each stope. A 330 m hole was achieved further east to allow bulk sampling in the O'Brien East Zone.

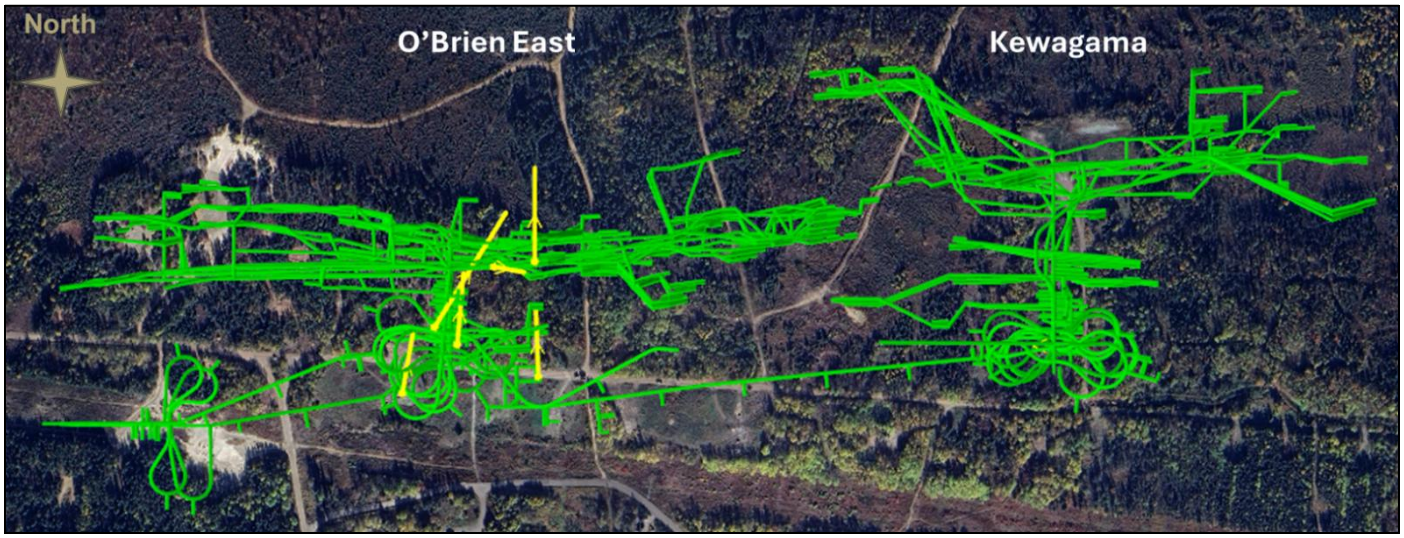
Figure 16-1: Location of the Drill Holes and the Stopes Evaluated during the Crown Pillar Study



Source: Norda Stelo (2025).

The rock mass characterization study was conducted to obtain information on the rock mass for the planned bulk sampling (Golder, 2021). The drill holes were completed in the O'Brien East Zone, which is part of the PEA study, making them a valuable basis for current geomechanical parameters. Six drill holes and the logging of 821 m were completed during the campaign. Figure 16-2 shows the locations of the drill holes.

Figure 16-2: Drill Holes Location during the Rock Mass Characterization Study (in Yellow)



Source: Norda Stelo (2025).

The two studies identified four main geomechanical domains used to simplify rock mass characterization:

- greywacke
- basalt
- porphyry
- conglomerate.

The key results obtained for each of the geomechanical units are summarized in Table 16-1.

Table 16-1: Summary of Key Parameters Obtained during Past Studies

Geomechanical Unit	UCS Used for Classification (MPa)	Q'_{35} Percentile	Q'_{50} Percentile	RMR_{76} [1]
Greywacke	103	5.1	15.8	64
Basalt	81	8.3	25.0	74
Porphyre	73	8.3	33.3	74
Conglomerate	71	8.3	33.3	74

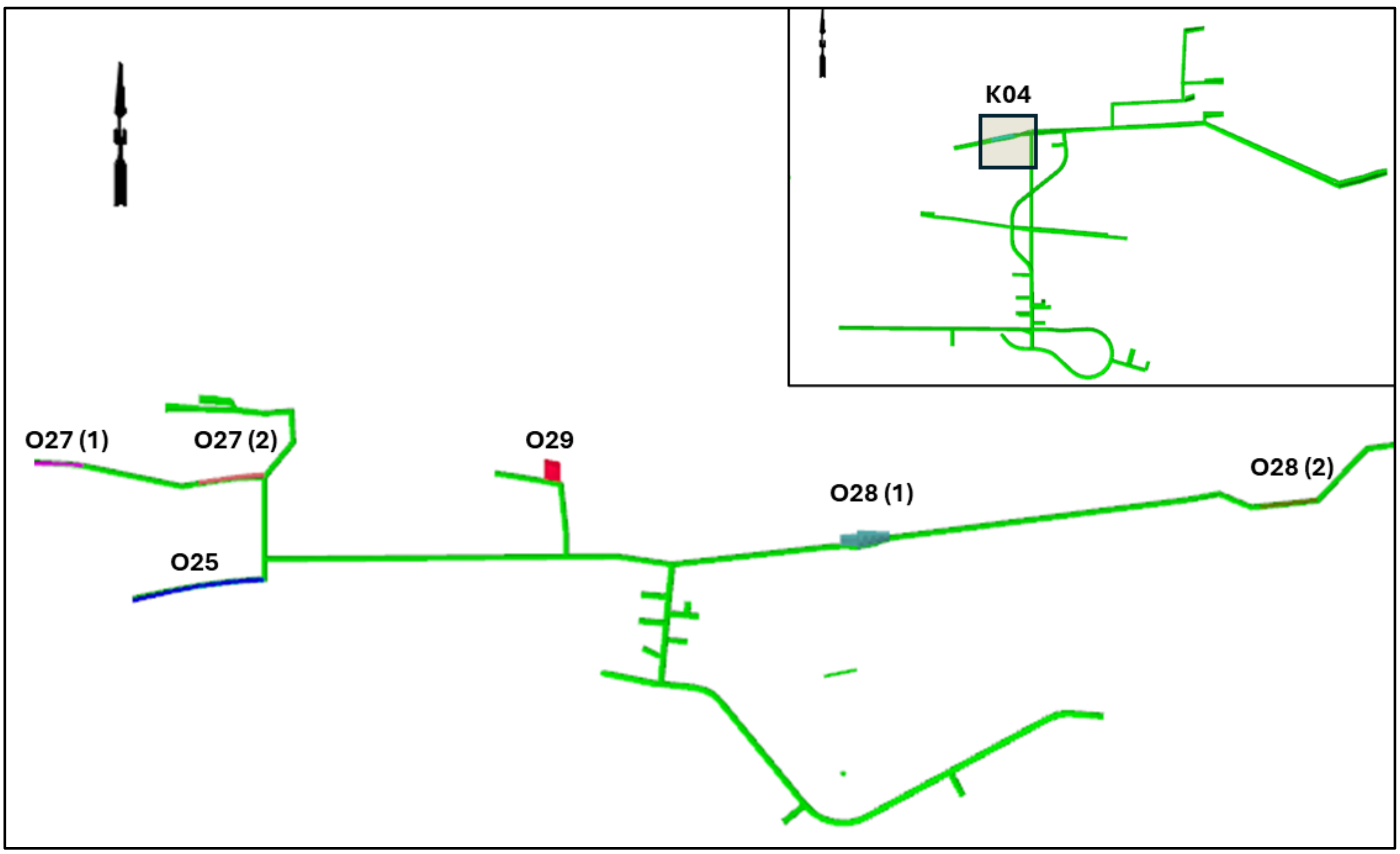
Note: Classification obtained using the 35th percentile distribution.

Based on Table 16-1, the values indicate that the rock mass at the O'Brien Gold Project is generally of good quality. These values have been used in various empirical analyses of surface crown pillar stability and stope dimensioning. It should be noted that the data presented in these studies are preliminary, and further geomechanical work is recommended to increase the level of confidence to a pre-feasibility study standard.

16.2.2 Surface Crown Pillar

The surface crown pillar stability for the stopes near the surface was analysed during the PEA. In total, seven groups of stopes were evaluated: six in the O'Brien East Zone, and one in the Kewagama Zone. The O'Brien East Zone presents more challenges due to its thicker overburden compared to the Kewagama Zone. The plan views in Figure 16-3 shows the locations in plan view of the groups evaluated. The groups were named according to their mining horizons.

Figure 16-3: Groups of Stopes Analysed for the Surface Crown Pillar



Source: Norda Stelo (2025).

During the study, limited information was available regarding geomechanical domains. As a conservative approach, Greywacke (identified in Golder's studies as having the lowest rock quality) was selected as the representative rock type to assess the stability of the surface crown pillar.

Due to the absence of data on the transition zone between overburden and competent rock, the 35th percentile Q-value was used as the basis for the stability analysis. For the Q-value calculation, a joint water reduction factor (Jw) of 1.0 and a stress reduction factor (SRF) of 2.5 were applied, resulting in an estimated Q-value of 2.0.

The scaled span method (Carter, 2014) was employed to determine the safety factor and the probability of failure for the surface crown pillar of the stope groups. This methodology considers that the overburden does not contribute to the stability of the crown pillar. Table 16-2 presents the estimated safety factors and probabilities of failure in unsupported conditions and with the overburden included in the stope depths.

Table 16-2: Scaled Span Method Results

Stopes (Mining Horizon)	Dimension (m)		Depth	Geomechanical Unit	Q	Jr	Ja	Overburden Thickness (m)	No Support	
	Width	Height							FS	Pf (%)
O27 (1)	2.2	25.0	29.6	Greywacke	2.0	0.5	1.0	9.9	5.6	0.6
O27 (2)	2.6	25.0	28.8	Greywacke	2.0	0.5	1.0	9.3	4.9	0.7
O25	2.6	25.0	32.4	Greywacke	2.0	0.5	1.0	10.6	5.0	0.7
O29	3.5	25.0	30.9	Greywacke	2.0	0.5	1.0	17.9	3.1	1.4
O28 (1)	3.2	25.0	27.8	Greywacke	2.0	0.5	1.0	14.8	3.4	1.2
O28 (2)	2.3	25.0	28.9	Greywacke	2.0	0.5	1.0	7.5	5.6	0.6
K04	2.4	25.0	25.4	Greywacke	2.0	0.5	1.0	7.6	4.9	0.7

This assessment indicates that the surface crown pillars are stable, with a low probability of failure. However, additional geomechanical data are required to more accurately define long-term stability and enhance the level of confidence.

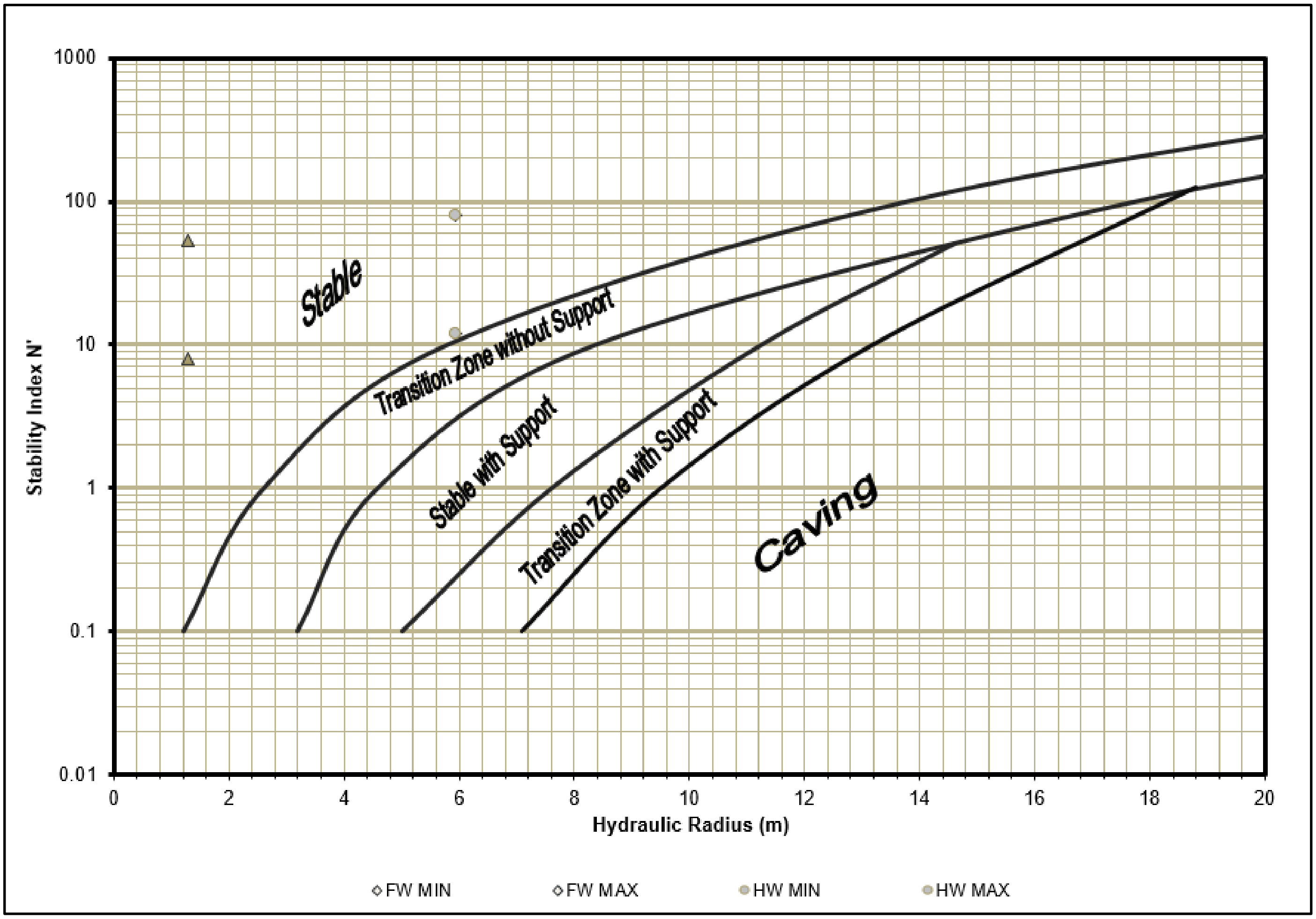
16.2.3 Stope Design

16.2.3.1 Typical Condition

The stability of open stopes depends on appropriate initial dimensioning, which can be assessed using the modified stability graph method. This method is based on the rock mass quality index Q' (where SRF/Jw = 1 in the Q index) and incorporates various geotechnical parameters, including the exposed face (hydraulic radius). The stability graph (Nickson, 1992) was used to predict stope stability based on plots of HR versus N' from case histories of unsupported stopes, cable-bolted stopes, and defined stability limits.

For the stability assessment, minimum and maximum boundaries were plotted on the stability graph using Q' values of 5.6 and 33.3, representing the range of conditions expected in both the hangingwall and footwall. An average stope thickness of 2.8 m and a total height of 29 m (25 m floor-to-floor plus a 4 m overcut drift) were considered. Figure 16-4 presents the stability graph for the project.

Figure 16-4: Stability Graph for the Stope Design of the O'Brien PEA



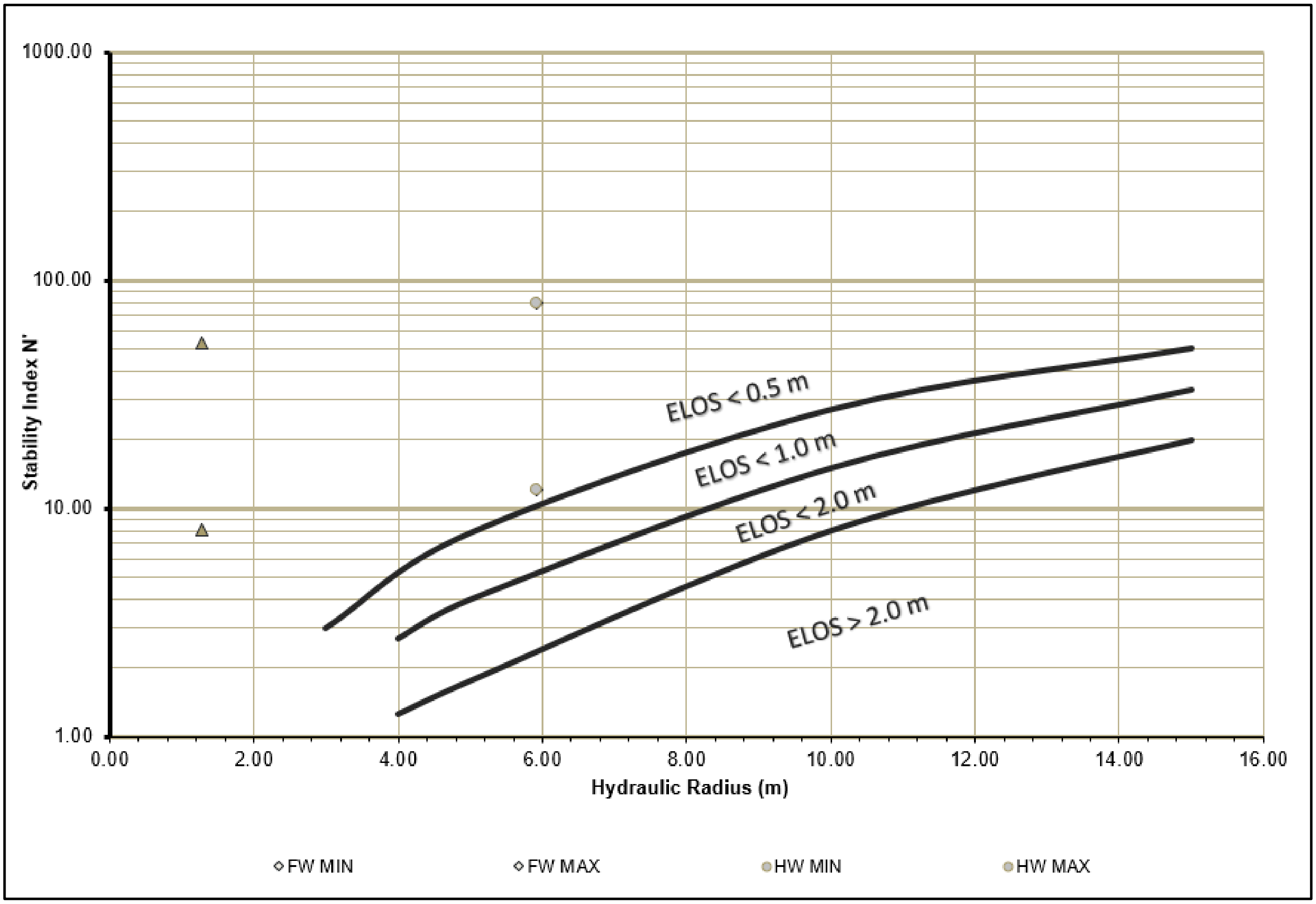
Source: After Nickson (1992).

Preliminary results indicate that 30 m long stopes could remain stable without support. However, given the narrow geometry of the veins at the O'Brien Gold Project, stope grades are sensitive to dilution. As a conservative measure, 20 m long stopes were selected to reduce dilution while maintaining practical and efficient mining geometries.

With the selected sizing parameters, the unplanned dilution was estimated based on the equivalent linear overbreak/slough (ELOS) (Clark, 1997). Figure 16-5 shows the ELOS graph of the project. Both dilutions of the hangingwall and footwall with the minimum Q' value of the Golder studies resulted in a dilution less than 0.5 m on both walls. To be conservative, and since the hangingwall yielded close to the 0.5 m line, an ELOS of 0.5 m was applied to the hangingwall and an ELOS of 0.2 m was considered on the FW accounting for the blast damage. Considering the narrow vein environment, minimal dilution is justified.

With the selected sizing parameters, unplanned dilution was estimated using the ELOS method (Clark, 1997).

Figure 16-5: ELOS Graph for the Stope Dimensions of the O'Brien Gold Project



Source: After Clark (1997).

The quantity of data and the empirical methodology of Nickson and Clark are considered adequate for a PEA-level study. However, additional geomechanical data collection and the use of more global methods should be considered to support a pre-feasibility study.

16.2.4 Ground Support

For costing requirements, preliminary ground support templates were developed based on the QP's experience in operations with a similar operating environment and common empirical support practice. Considering the size of the excavations and their anticipated influence on rock mass stability based on values commonly used in similar hard rock mines in the province of Québec, the following ground support requirements are proposed for the 4.5 m wide x 4.5 m high drifts:

- Back – 1.8 m long, 20 mm fully resin-grouted rebar on a 1.2 x 1.2 m spacing, with spherical seats and #6 welded wire mesh.
- Walls – 1.8 m long FS35 split sets on a 1.2 x 1.2 m spacing down to 1.5 m from the floor with #6 welded wire mesh.

The ground support system could be modified after the acquisition of additional data from the rock mass characterization study or depending on field conditions encountered during mining operations.

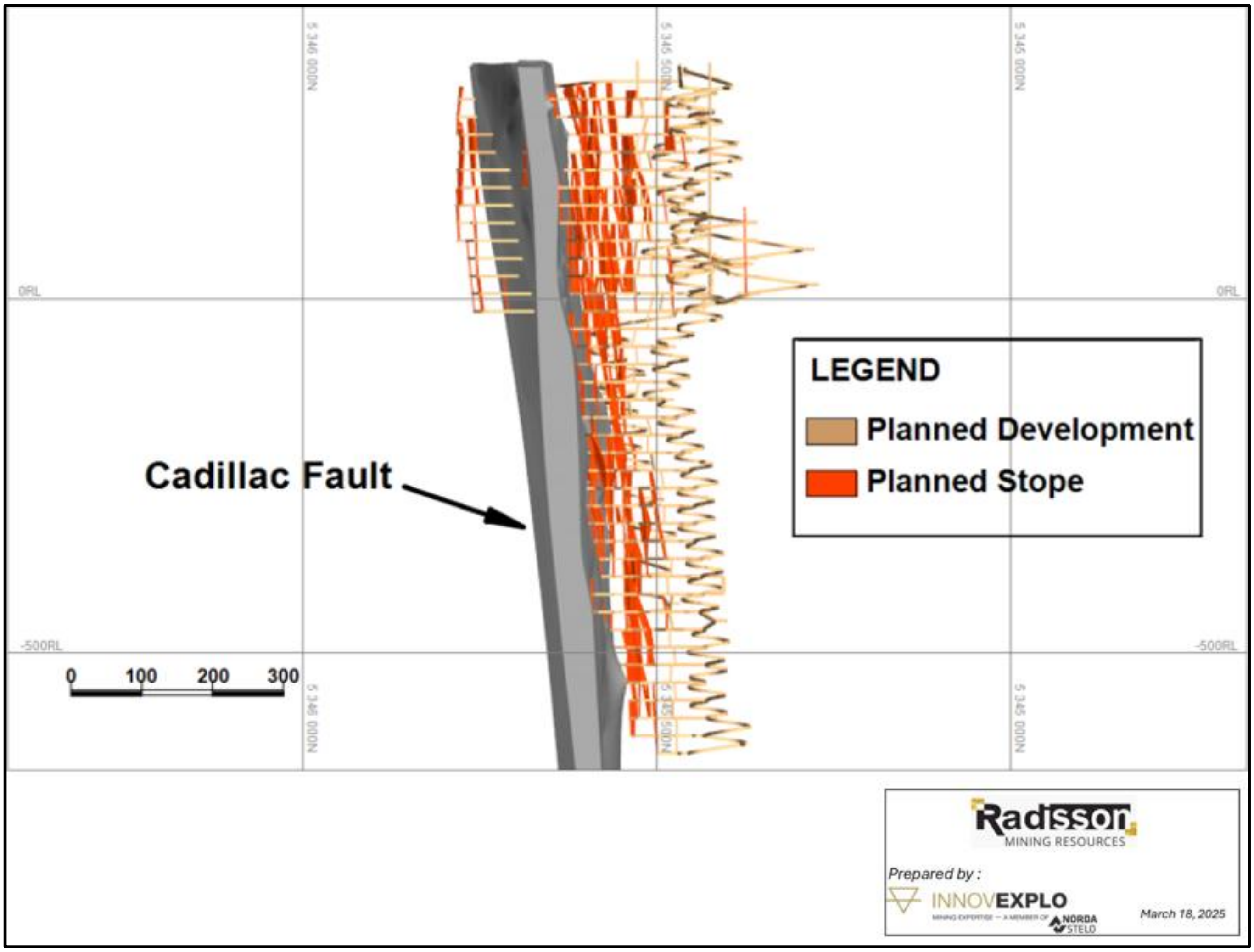
16.2.5 Cadillac Fault

The mining site is crosscut by the well-known Cadillac Fault, which in the local area, strikes east-west with an average dip of 85° to the south (Figure 16-6).

At the PEA level, a conservative approach was adopted by applying an additional dilution factor of 11% to stope and development solids located within or near the fault zone. This estimate is based on 3D mined-out solids from the historical O'Brien mine that are in contact with the fault interpretation provided by the geology team, as well as drill core photographs.

For production planning purposes, additional delays and secondary ground support were included for stopes intersecting the fault. The estimated delays are proportional to the size of the excavation. Furthermore, the fault was considered in the design of development headings, ensuring that tunnels cross the structure perpendicularly to minimize its impact on the tunnel stability. Only 4% of the mineralized material tonnage is located within the fault zone (Figure 16-6).

Figure 16-6: Mine Overview Looking East with the Cadillac Fault

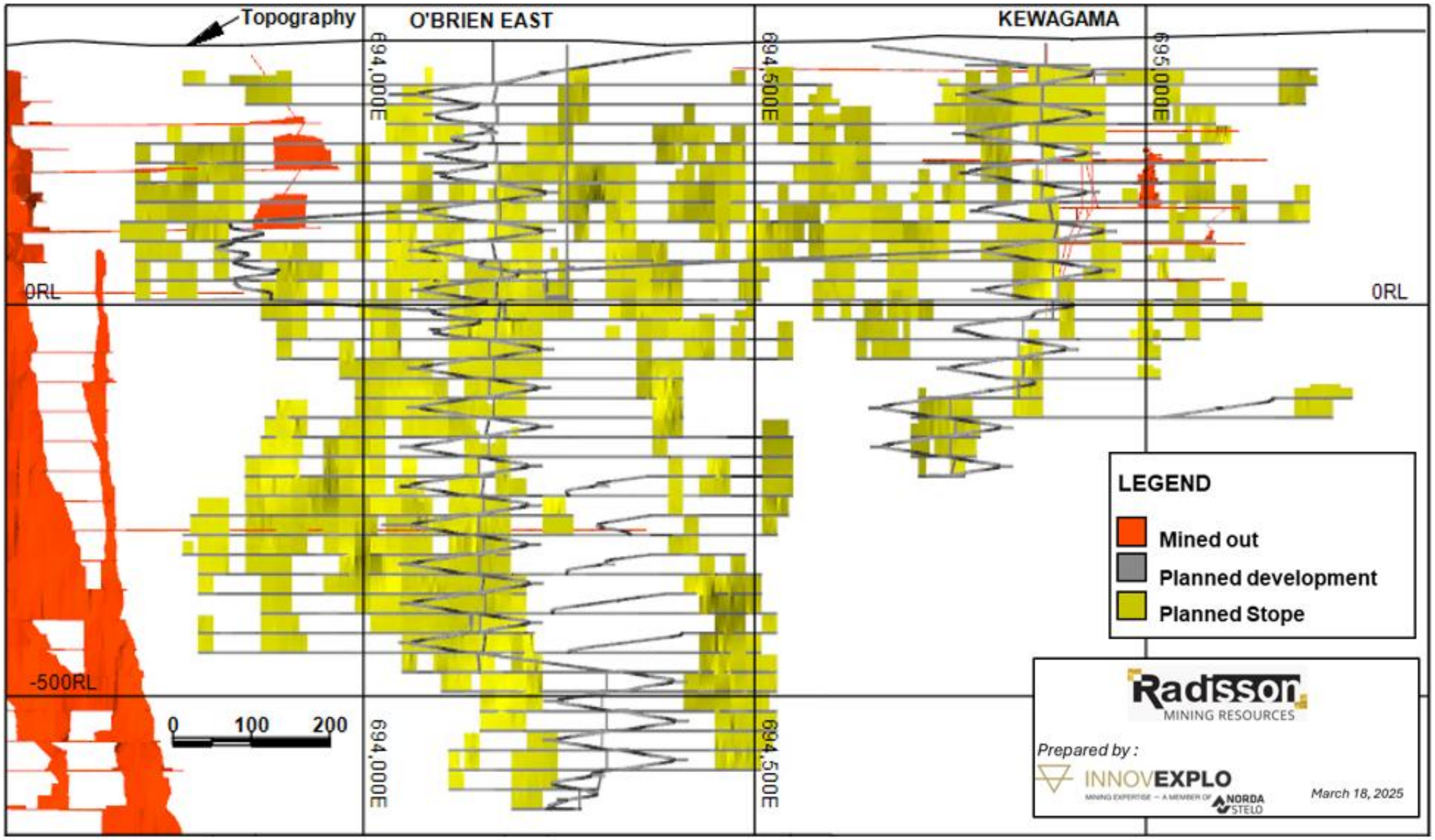


Source: Norda Stelo (2025).

16.3 Mine Design

The O'Brien Gold Project is designed as a mine with mechanized ramp access that will use a longitudinal longhole retreat method to mine the underground mineralized materials in the two mining zones (Figure 16-7). Level spacing is set at 25 m, corresponding to the best alternative between 20 m and 30 m level spacing to maximize profitability while minimizing drilling deviation and unplanned dilution.

Figure 16-7: Overview of Underground Mine Design



Source: Norda Stelo (2025).

Each mining zone is provided with its own ramp portal and main decline, starting from the surface, to serve the different levels. The development strategy on each level in the production zone is to develop the ventilation access drift and drop raise as soon as possible to extend the primary ventilation circuit to the level.

During the development phase, the extracted waste rock will be trucked to surface and temporarily dumped at the waste pile near the portal. Waste material will be returned underground to be reused as backfill material for stopes as required.

A main underground hub with a crusher and vertical conveyor will facilitate continued output of mineralized material from the mine, minimizing trucking activities and the project's surface footprint.

The dewatering system will consist of seven underground secondary pumping stations (sump pump) and one main pumping station. Turbid water will be pumped via a series of sump pumps located in the main pumping station located in the O'Brien East zone, then pumped to the surface water pond using a GEHO pump.

16.3.1 Mining Method Selection

The deposit is in the Abitibi region with a dip that varies between 70° and 90° and a thickness that ranges from 1.5 m to over 14 m and averages 2 m. Geotechnical data collected by WSP, along with historical data, indicate stable conditions for longhole mining, with adequate ground support required to maintain safety and profitability. Based on the rock mass classification, longhole mining has been identified as the most suitable mining method, particularly with level spacing of 25 meters, to minimize drilling deviations and unplanned dilution. This spacing ensures optimal recovery of mineralized material and maximizes profitability by reducing costs associated with excessive secondary drilling or overbreak.

The materialized zones within the deposit are of sufficient width and grade, ensuring that the estimated dilution does not eliminate the profitable recovery of the material. Given the decreasing availability of skilled conventional mining labour, mechanization of the mine becomes a consideration. Therefore, the selected mining method, longitudinal retreat with backfill, is considered the most appropriate, as it enables enhanced mineralized material recovery and safer, more efficient, operations using advanced mining equipment.

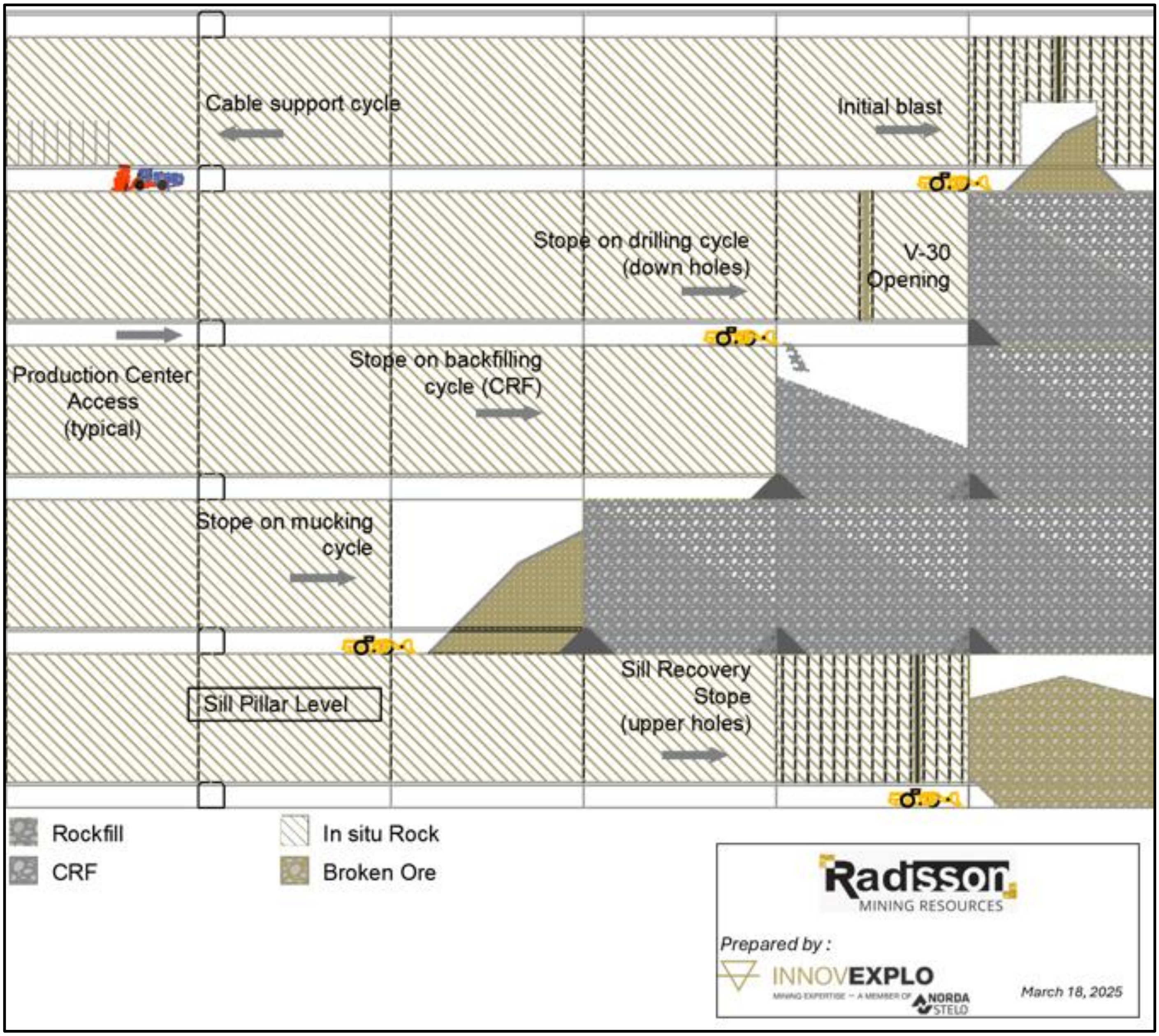
Mining will consist of an undercut level and an overcut level, both accessed from the main ramp or an access drift. Each sill will be accessed perpendicularly and developed along strike to the economic limits of the mineralization.

After sill development—and cable bolting where necessary—production holes will be drilled between the sills and blasted in two or three phases. To maximize stability, sustain production, improve profitability, and divert induced stresses, the mining area will be subdivided into production centres using the longitudinal retreat method. This method creates pyramidal shapes as mining progresses, enhancing stability by reducing stress concentrations.

Due to the distance between the mine and processing plant, and to ensure long-term stope stability, cemented rockfill is prioritized for backfilling. Rockfill will be used where conditions permit.

Figure 16-8 illustrates the mining cycle.

Figure 16-8: Mining Cycle – Longitudinal Longhole Retreat



Source: Norda Stelo (2025).

16.3.2 Mine Design Criteria

The permanent drifts dimensions are chosen to (1) accommodate the largest production equipment following regulatory requirements for health and safety in underground mines, (2) avoid ventilation bottleneck that will increase the mine static pressure, and (3) produce realistic air velocity along horizontal developments serving as airways to the primary ventilation circuit, such as ramps and level access drifts.

Ramps and level access drifts are 4.5 m wide by 4.5 m high, whereas haulage drifts for waste and production in mineralized areas are 4.0 m wide by 4.0 m high. Various development parameters are summarized in Table 16-3. The PEA design was planned with no gradient where applicable; the proposed gradient in the table describes the desired gradient in the final operation. Level developments are designed to respect the 2% minimal gradient to allow water runoff to a level sump.

Table 16-3: Mine Design Parameters

Development Heading	Width (m)	Height (m)	Gradient (%)
Ramp	4.5	4.5	13.7 (15 maximum)
Level Access	4.5	4.5	2
Production Drift	4.0	4.0	2
Remuck	4.5	4.5	2
Loading Bay	4.5	5.0	2
Mixing Bay	4.5	4.0	2
Sump	4.0	4.0	-15
Electrical Station, Ventilation Access	4.0	4.0	2

The main ramp average gradient is approximately 13.7%. A maximal gradient of 15% was applied to the ramp connecting the two levels with a minimal turning radius of 25 m where possible. To minimize maintenance and operator fatigue, ramps are designed to keep a linear portion for level access (i.e., 4 m) before and after level access. A remuck bay is also planned between every level for development efficiency.

To maintain the stability of permanent openings such as ramps, haulage drifts, underground constructions, and vertical developments, a minimum pillar is left between such openings and between permanent development and stopes. Table 16-4 summarizes the design pillars.

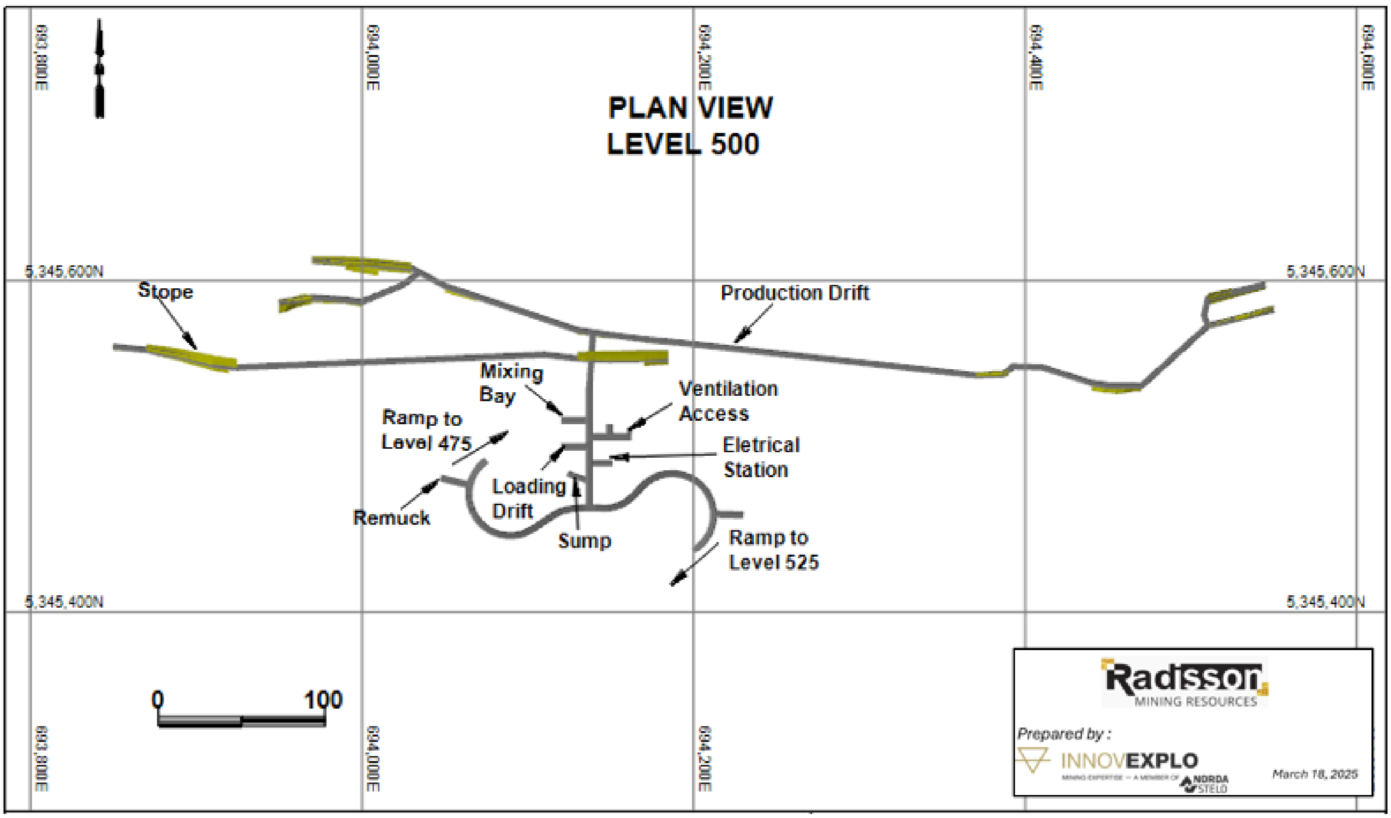
Table 16-4: Mine Design Pillars

Pillar Type	Minimum Pillars (m)
Ramp / Stope	25
Drift / Stope	10
Raise / Stope	20
Ramp / Drift or Access	12
Drift / Drift	12
Raise with Access-Way / Ramp	30
Raise / Drift	12
Raise / Fault	15
Drift / Fault	15

16.3.3 Level Design

From the main ramp, a level access drift directed north-south is driven perpendicular to mineralization. Then the production drifts, which are developed east and west from the level access drift, are aligned with the longitudinal axis of the deposit. A typical production level includes a sump, an electrical station, a ventilation access, a loading bay, a mixing bay, a pump station, and level drifts, as shown in Figure 16-9. Depending on the location, a level may also include a refuge and other relevant infrastructure.

Figure 16-9: Typical Level (Level 500 in O'Brien East Zone)

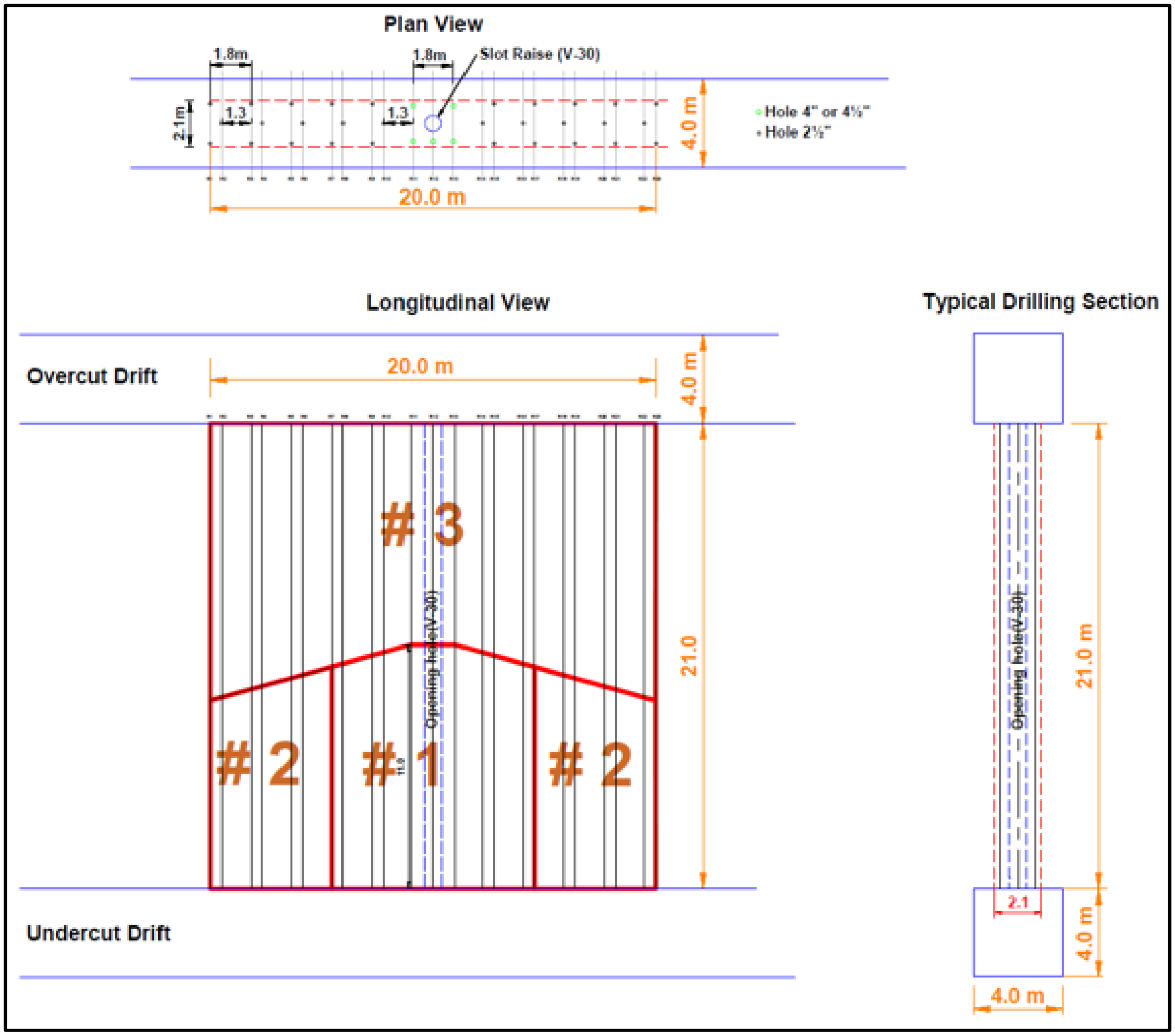


Source: Norda Stelo (2025).

16.3.4 Drill and Blast Design

Longitudinal longhole mining methods will be employed. Mining areas have individual production centres based on the sector and their dependency. 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 the loss of mineralized material and maximize the use of natural waste pillars. The production drilling generally follows a parallel drilling pattern, but some fan drillings are caried out for stopes larger than the production drift width to maximize the mining recovery. Production drill holes are drilled with a 63.5 mm (2.5") diameter using a top hammer drill. Most holes are 21 m long, but some can reach 26 m for fan drilling. The slot raise will be achieved using a V30. Figure 16-10 illustrates the drilling pattern and the blasting sequence.

Figure 16-10: Production Drilling Pattern



Source: Norda Stelo (2025).

Almost all stopes have a thickness below 8 m with an average of 2.7 m. Only seven stopes have a thickness between 8 and 14 meters. The resulting total tonnage mined by the longitudinal long-hole method is 3.3 Mt. Table 16-5 presents the number of down hole stopes and uppers holes stopes. Down hole longitudinal stoping and upper longitudinal stoping will count for 73% and 15%, respectively, of the total mineralized material and marginal metric tonnes to be mined in the project; 12% will come from development.

Table 16-5: Longitudinal Stoping Summary

Longitudinal	Number of Stopes	Tonnage	Gold (oz)
Downhole	887	2,758,127	509,446
Uphole	204	557,660	94,002

16.3.4.1 Rate and Supporting Assumptions

To maintain accurate underground mine scheduling, detailed cycle times were calculated for main underground activities. The operational parameters used for the O'Brien Gold Project are detailed in Table 16-6. Each day includes two 10-hour shifts and considers all related operational activities (e.g., shift changes, lunch breaks, refuelling, loss of time, and transportation to workplaces).

Table 16-6: Operating Parameters

Operating Parameters	Units	Quantity
Working Days per Year	Days	365
Number of Shifts per Day	Shifts	2
Effective Hours per Day	Hours	14

A summary of the main development rates is provided in Table 16-7.

Table 16-7: Main Development Rates

Heading	Single Face		Multiface Maximum Rate	
	Per Jumbo Drill (m/month)	Per Face (m/month)	Per Jumbo Drill (m/month)	Per face (m/month)
Waste	250	220	310	90
Mineralized Material	N/A	N/A	310	60
Crusher Room	N/A	N/A	310	60
Maintenance Bay	N/A	N/A	310	60
Pumping Station	N/A	N/A	310	60
Vertical Development Rate (m/month)				
Drop Raise	21	Includes support & access-way		
Cone Sump	15	Includes support		
Silo	15	Includes support		
Vertical Conveyor	50	No support & no installation		

Development planning estimates the number of jumbo drills required. This number is then used to estimate the number of other related equipment, such as bolters and load-haul-dump loader (LHD) vehicles (required for development), based on the detailed cycle time of the development path.

The same cycle time calculation process is used to estimate the vertical development rates. The rates vary based on the selected method used and the size of the excavation. To these rates, additional delays are applied to consider other activities when required, such as ground support and an access-way for construction.

For rehabilitation and slashing of excavations in old mine workings, the task rate performance is the same as standard development, ranging from 60 to 90 m per month depending on the material characteristics. However, the generated tonnes have been adjusted using a relative factor based on the ratio of the mined-out section to the final required section.

16.3.4.2 Opening Sizing

A 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 be subeconomic if specific development and mining costs were considered. Parameters used in the DSO module are presented in Table 16-8.

Table 16-8: DSO Parameters for Underground Mining

Parameters	Units	O'Brien East	Kewagama
Mineralized Material Density	t/m^3	2.82	2.82
Waste Density	t/m^3	2.82	2.82
Optimization Length	m	20	20
Minimum Mining Width	m	1.5	1.5
Minimum Stope Pillar	m	8	8
Hangingwall Dilution	m	0.5	0.5
Footwall Dilution	m	0.2	0.2
Cut-off Grade (Stopes) 0% Royalty	g/t	3.05	3.05
Cut-off Grade (Stopes) 2% Royalty	g/t	3.11	3.11
Crown Pillar	m	25	25

16.3.5 Underground Infrastructure

The main underground infrastructure serving both zones including the pumping station, maintenance bay, powder and cap magazine, crusher room, mineralized material silo feeding the crusher, grizzly room, vertical conveyor, and loading installation, primarily located on levels 275, 300, and 325 in the O'Brien East zone, are located near the decline connecting the Kewagama and O'Brien East zones.

The main surface ventilation fans and heater room are designed to be located at the surface.

16.3.5.1 Mineralized Material Handling System

The mineralized material handling system includes the crusher installation located on level 300, the grizzly room, the mineralized material silo with its fingers and the vertical conveyor raises. The full system will be excavated in Q4 Y2 of the life of mine and will be available in Q3 Y3 of the life of mine. The 6.8 m diameter circular mineralized material bin will hold about 1,800 tonnes, equivalent to around 1.5 days of production. Figure 16-11 presents an isometric view of the crusher station and surrounding infrastructure.

16.3.5.2 Other Infrastructure

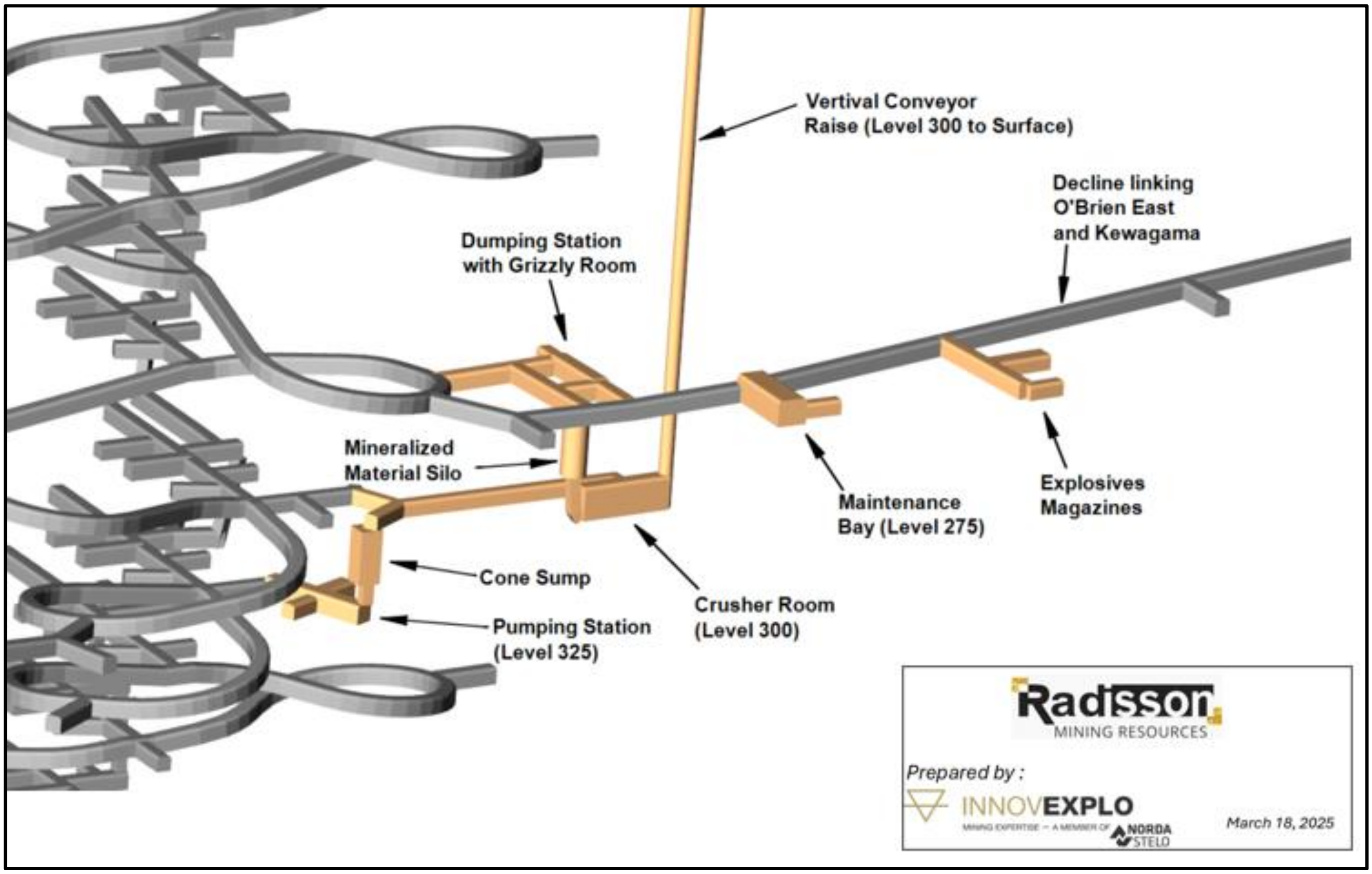
Additional infrastructure includes the fresh air raise, internal ventilation raises, refuge stations, powder, cap magazines and redundant infrastructure located on the access levels (Figure 16-11).

The Kewagama main explosive depots (powder and cap magazines) are located on Level 25. Three explosive depots are in O'Brien East, on Levels 50, 275 and 600. The one located on Level 275 in O'Brien East will also serve the lower levels of Kewagama. These can easily accommodate the explosive requirements of the project and are designed with room to spare for material manipulation and to comply with all federal and provincial requirements.

The maintenance bay was sized for small maintenance operation and for less mobile equipment. Maintenance of larger equipment such as LHDs and trucks will be carried out on the surface.

Each underground refuge station is designed and located to accommodate the necessary number of workers at any given time. The refuges are located closer than the required 1,000 m to ensure no delays in the development sequence. The ventilation system is discussed in Section 16.5.5.

Figure 16-11: Crushing Station Overview with Surrounding Infrastructure



Source: Norda Stelo (2025).

16.3.6 Dewatering

The pumping rate at the O'Brien Gold Project is estimated from the pumping rate recorded in the historical mine and then adding the following:

- water inflows that will come from the planned excavations in the two zones
- contribution of underground operating equipment.

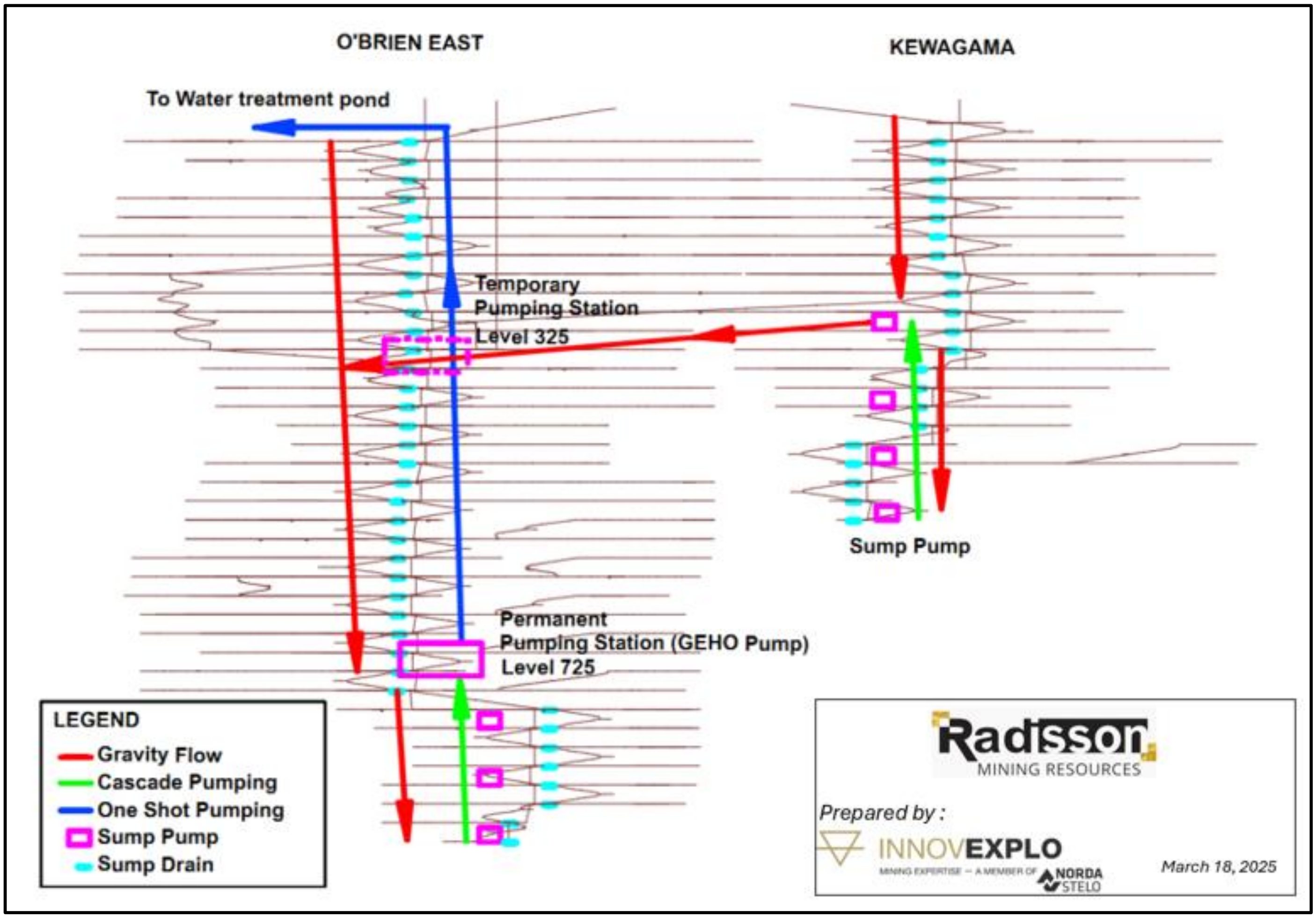
The estimated groundwater flows generated by overall planned excavations is 1,908 m^3/d.

According to mine planning, the peak water flow produced underground by all the underground equipment in operation was estimated at 437 m^3/d. Thus, the total water flow retained for pump selection is 2,345 m^3/d. The dewatering design capacity will be 2,814 m^3/d (516 gal/min), which includes 20% contingency.

Dirty water will be transported gravitationally by a system of sumps and drain holes, and then strategically placed in sumps equipped with a submersible pump to allow the water to be transported from the lower parts of the mine to the main pumping station.

A temporary pumping station is planned on Level 325 below the cone sump located around the crusher room. Following the completion of the main pumping station excavation on Level 725 (Q3 Year 4), all installation will be moved to the Level 725 permanent pumping station. A single GEHO pump with a pumping capacity of 118 m^3/h will be capable of pumping the dirty water with high solids content. This will ensure the pumping of all the turbid water from Level 725 to the water treatment pond at the surface. Figure 16-12 illustrates the dewatering strategy of the project.

Figure 16-12: Dewatering Strategy



Source: Norda Stelo (2025).

16.3.7 Dilution and Mine Losses

16.3.7.1 Typical Conditions

Considering the ground conditions currently expected, the total hangingwall and footwall planned dilution is estimated at 0.5 m for the hangingwall and 0.2 m for the footwall (for a total of 0.7 m), based on the equivalent linear overbreak/slough (ELOS) empirical method (Clark, 1998). The ELOS dilution envelope of was considered during the block model interrogation.

Based on Norda Stelo’s database, a mining recovery of 95% is applied for all stopes in the project, while recoveries of 98% and 100% are considered for development in waste and level drifts, respectively.

16.3.7.2 Cadillac Fault

At this level of the study, an unplanned dilution of 11% is considered when the stope or development is in the immediate vicinity of the Cadillac Fault. To be conservative, a grade of 0 g/t is assigned to the unplanned dilution material. Cable bolting will be used as secondary ground support system for stopes located within the fault area.

16.3.7.3 Backfill Dilution

A 5% backfill dilution at 0 g/t has been added to each stope adjacent to a backfilled stope.

16.3.7.4 Upper Recovery

Stope grades are considered to be uniformly distributed within a stope, as the process is carried out within the Deswik scheduling software rather than through DSO modelling. At a PEA level of study, this accuracy is considered appropriate.

A maximum stope height of 18 m has been considered for upper stopes located at the top of production centres, resulting in the sterilization of a 3 m mineralized pillar. When three adjacent upper stopes are present, the pillar width is set at a maximum of 3 m or 70% of the stope width, whichever is greater. However, if an opportunity arises to develop an economic bypass through an already mined upper level to enable side drilling, a reduced pillar width of 1.5 m is assumed. Upper stopes represent 12% of the total tonnes and 13% of the total ounces in the project.

16.3.8 Cut-off Criteria

Cut-off grades used in the design and scheduling process are based on preliminary revenue inputs and calculated study costs, as outlined in Table 16-9.

Table 16-9: Cut-off Grade Inputs

Factor	Unit	Assumption	
Gold Price	US$/oz	2000	
Exchange Rate	USD:CAD	1.33	
Mining Dilution	%	15	
Mining Recovery	%	95	
Mill Recovery	%	92	
Preliminary Mining Cost	C$/t	125	
Site G&A	C$/t	21	
Rd Haulage	C$/t	7	
Processing Costs	C$/t	45	
All in Mining Cost ($ CND/T)	C$/t	198	
Royalty (%)	%	0	2
Selling Cost	C$/oz	5	58.2
Total Revenue per Ounce of Gold	C$/oz	2,320	2,265
Cut-Off Grade	g/t	3.05	3.11

During the initial stope optimization process, and depending on the applicable royalty rate, break-even cut-off grades of 3.05 or 3.11 g/t Au were applied. For incremental stopes, cut-off grades of 2.66 or 2.71 g/t Au were used, and for marginal development required to access stopes, cut-off grades of 0.66 or 0.68 g/t Au were applied. These lower cut-off grades cover processing and surface road haulage, as mining and haulage costs for this material are already incurred to access stope material. The preliminary cut-off grades are summarized in Table 16-10.

Table 16-10: Preliminary Cut-off Grades

Cut-off Grades (g/t)	Royalty	
	0%	2%
Break-Even	3.05	3.11
Incremental	2.66	2.71
Marginal	0.66	0.68

Before the integration of the DSO into the schedule, the economic potential of each individual stope for inclusion in the PEA as mineralized material was evaluated using the cost and revenue assumptions summarized in Table 16-11. For each level, all operating costs were included in the analysis to validate that it generated a positive cash flow. The same process was repeated for isolated mineralized zones (pods/orphans). Additionally, an economic analysis was conducted to validate whether the recovery difference between an upper and down stope justifies the inclusion of the overcut of the stopes.

Table 16-11: Stopes Economic Parameters

Stopes Economic Inputs	Unit (CAD)	Unit Rate	
		0% Royalty	2% Royalty
Mining Cost (excluding Operating Development)	$/t	113	
Development Cost	$/m	2,500	
Capital Level Cost	$/level	1,156,500	
Revenue per Ounce of Gold	$/oz	2,582	2,653

After accounting for stope depletion by development, and applying dilution and recovery factors, the economic potential of individual DSO shapes for inclusion in the PEA was evaluated using the same established cost and revenue assumptions.

16.3.9 Backfill

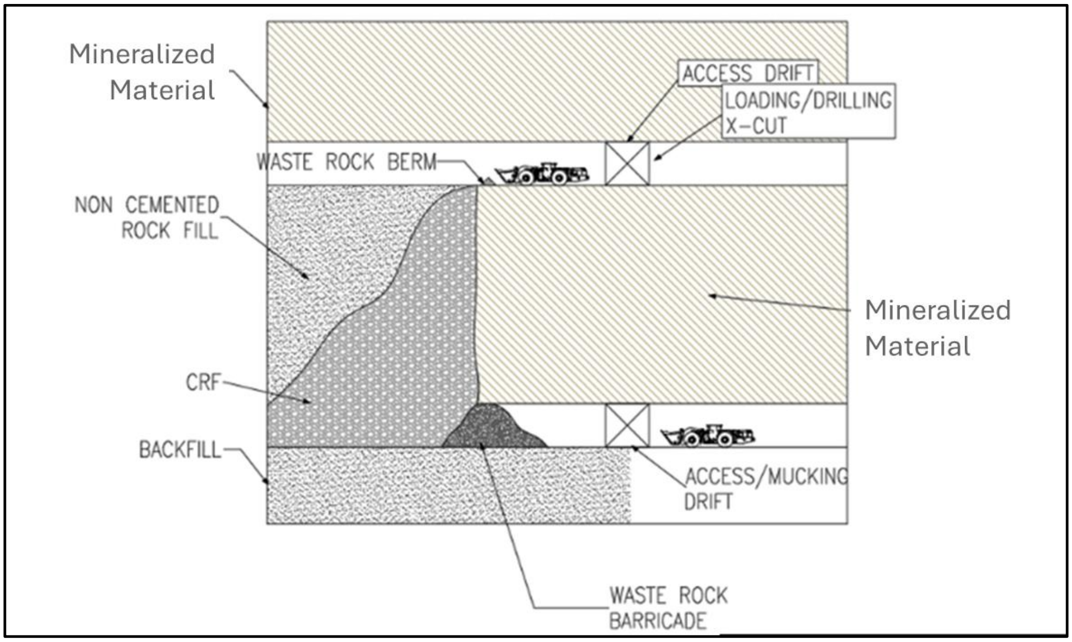
Given the distance between the processing plant and the mining camp, the use of paste backfill for stope stabilization is not feasible. As a result, cemented rockfill has been selected as the preferred backfill material.

The primary backfill will be cemented rockfill with a 4.0% cement binder, except in areas above sill pillars, where the binder content will be increased to 7.0%. This cement content may be adjusted based on the results obtained underground. The cemented rockfill will be transported through a gravity-fed distribution network of cemented slurry, with a cement plant at surface to supply the required binder. The scoop will mix the cement slurry with the development muck in the mixing bay before transporting it to the stope for deposition.

A simple rockfill method will be employed as extensively as possible, particularly in areas at the end of a longitudinal sequence or in stopes with no direct impact on adjacent excavations. Development waste rock will be used as cemented rockfill or rockfill as a priority. Excess waste rock will be hauled to the surface waste dump or stocked in unused or depleted levels, whenever possible. Some remucks may also be used to stockpiled temporarily excess waste rock for future backfill.

The required backfill strength will depend on the size of the stope and the mining sequence. To maximize safety and flexibility, the curing time used is 14 days before exposing a backfilled face. Figure 16-13 presents a typical cemented rockfill backfill operation.

Figure 16-13: Cemented Rockfill Overview



Source: Norda Stelo (2025).

16.4 Mine Services

16.4.1 Electrical Distribution

Underground power will be distributed at 25 kV. Provision was made for 31 substations (sizes varying from 200 to 1,000 kVA) to convert voltage to utilization levels (i.e., 600 V and 120/208 V). This will allow enough flexibility to address the needs for mining, dewatering, secondary ventilation, and services such as refuges and the maintenance bay.

16.4.2 Communication Network

It is assumed that a high-speed Internet link is already available close to the site. A private long-term evolution (LTE) network with basic infrastructure will be installed on surface.

An underground fibre optic network will be installed through the ramp and service holes to connect each electrical substation.

The private cellular LTE network will be deployed underground. LTE antennas will be installed every 50 m to allow complete coverage of the production levels for tele-operation. The antennas will be connected to the fibre optic network in each substation.

An underground automation programmable logic controller (PLC) network will be deployed to obtain real-time information and provide control on pumping, ventilation, and other installations.

16.4.3 Fuel Distribution Network

No underground fuel distribution network is planned for the project. Mobile equipment with fuel tanks will be used to fill equipment underground. A diesel tank on surface will be used to fill underground trucks.

16.4.4 Compressed Air and Water Supply

Limited compressed air will be required underground since most of the development and production drilling will be done by electric equipment. Compressed air will be used mainly for portable water pumps, to clean the floor prior to long hole drilling, and to service the refuges.

Compressed air will be produced at surface and will be available underground via a network of 8” diameter steel pipes installed in various underground developments.

16.4.5 Mine Ventilation

The ventilation system has been designed to allow independent operation of the ventilation networks in the O’Brien East and Kewagama zones, ensuring operational flexibility. A step-by-step ventilation network will be implemented from starting with a temporary ventilation system when the main fans are not yet operational. The final system for each zone will be in operation as soon as the fresh air raise has been completed and the surface main fans have been commissioned. The main fans will be provided with a variable frequency drive to optimize utilization. Silencers will limit the main fan noise produced at a maximum of 50 dB at the surface nearest receptor of the town of Cadillac. All the ventilation raises are provided with access-ways to provide secondary egress. The ventilation design was carried out using Ventsim, an underground mine ventilation simulation software.

16.4.5.1 Fresh Air Requirement

The ventilation requirement is based on the maximum amount of equipment operating underground at the same time. The fresh air rate used to dilute diesel equipment emissions corresponds to the certificate issued by the CANMET Mining and Mineral Sciences Laboratories (CANMET) of Natural Resources Canada The utilization rate for each piece of equipment within the fleet was based on common industry standards to meet the worst-case scenario: 100% for production equipment running full diesel, 25% for machinery that operates primarily with electricity such as drill rigs, 50% to 65% for service and support equipment. The final ventilation network of the project in full production has a capacity of 498 kcfm for the O’Brien East zone, and 273 kcfm for Kewagama zone. This includes a contingency of 15% as a provision for air leakage.

Tables 16-12 and 16-13 show the ventilation rate for each piece of underground equipment and the required total airflow for O’Brien East and Kewagama, respectively.

Table 16-12: Fresh Air Requirement for O'Brien East Zone

Equipment Type	Model	Engine Power (kW)	Quantity (No.)	CANMET Airflow Requirement per Unit (cfm)	Utilization Rate (%)	CANMET Total Airflow Required (cfm)
Development						
Jumbo	Boomer S2	90	3	7,000	25	5,250
LHD 6 t	ST3.5	173	2	10,700	100	21,400
Truck 20 t	MT2200	242	4	23,000	100	92,000
Scissor Lift	SL3	110	3	12,000	65	23,400
Bolter	DS312	74	5	9,200	25	11,500
Production						
Production Drill	Stopemaster Series HX	55	1	0	25	0
LHD 6 t	ST3.5	173	2	10,700	100	21,400
Truck 20 t	MT2200	242	3	23,000	100	69,000
ANFO Truck	Maclean	110	1	12,000	65	7,800
Rockfill						
LHD 6 t	ST3.5	173	1	10,700	100	10,700
Services						
Transmixer	UNI 40 BM	96	1	7,500	10	750
Boom Truck	UNI 40 LP	96	1	7,500	50	3,750
Personnel Carrier	PC3	96	1	7,300	50	3,650
Mechanical Truck	Land Cruiser HZJ79 - BTE 134	96	1	7,300	50	3,650
Fuel-Lube Truck	UNI 40 SV	96	1	7,500	50	3,750
Underground Grader	PG 10 HA	96	1	7,500	65	4,875
Water Truck	Maclean	150	1	14,200	50	7,100
Electric Vehicle	Land Cruiser HZJ79 - BTE 134	96	1	7,300	50	3,650
Light Vehicle	Land Cruiser HZJ79 - BTE 133	96	3	7,300	50	10,950
Tractor	M4D	55	2	3,800	50	3,800
Mine Rescue - Light Vehicle	Land Cruiser HZJ79 - BTE 134	96	0	7,300	50	0
Allowance for Conveyor			1	9,500	100	9,500
Allowance for Crusher			1	28,500	100	28,500
Subtotal						**346,375**
Contingency					**15**	**51,956**
Total						**398,331**

Table 16-13: Fresh Air Requirement for Kewagama Zone

Equipment type	Model	Engine Power (kW)	Quantity (No.)	CANMET Airflow Requirement per Unit (cfm)	Utilization Rate (%)	CANMET Total Airflow Required (cfm)
Development						
Jumbo	Boomer S2	90	2	7,000	25	3,500
LHD 6 t	ST3.5	173	2	10,700	100	21,400
Truck 20 t	MT2200	242	3	23,000	100	69,000
Scissor Lift	SL3	110	2	12,000	65	15,600
Bolter	DS312	74	3	9,200	25	6,900
Production						
Production Drill	Stopemaster Series HX	55	1	0	25	0
LHD 6 t	ST3.5	173	1	10,700	100	10,700
Truck 20 t	MT2200	242	2	23,000	100	46,000
ANFO Truck	Maclean	110	1	12,000	65	7,800
Rockfill						
LHD 6 t	ST3.5	173	1	10,700	100	10,700
Services						
Transmixer	UNI 40 BM	96	1	7,500	10	750
Boom Truck	UNI 40 LP	96	1	7,500	50	3,750
Personnel Carrier	PC3	96	1	7,300	50	3,650
Mechanical Truck	Land Cruiser HZJ79 - BTE 134	96	1	7,300	50	3,650
Fuel-Lube Truck	UNI 40 SV	96	1	7,500	50	3,750
Underground Grader	PG 10 HA	96	1	7,500	65	4,875
Water Truck	Maclean	150	1	14,200	50	7,100
Electric Vehicle	Land Cruiser HZJ79 - BTE 134	96	1	7,300	50	3,650
Light Vehicle	Land Cruiser HZJ79 - BTE 133	96	3	7,300	50	10,950
Tractor	M4D	55	2	3,800	50	3,800
Mine Rescue - Light Vehicle	Land Cruiser HZJ79 - BTE 134	96	0	7,300	50	0
Allowance for Conveyor			0	9,500	100	0
Allowance for Crusher			0	28,500	100	0
Subtotal						**237,525**
Contingency					**15**	**35,629**
Total						**273,154**

16.4.5.2 Temporary Ventilation System

In both zones, when the fresh air raise is not yet in operation, a temporary ventilation system will be put in place during main ramp development from the surface to the fresh air raise access drift. The temporary ventilation system for each zone includes one run of plastic ducting with a 48" diameter, powered by a 150 hp development fan located on the surface near the portal. Plastic ducting has a low friction factor and prevents duct maintenance.

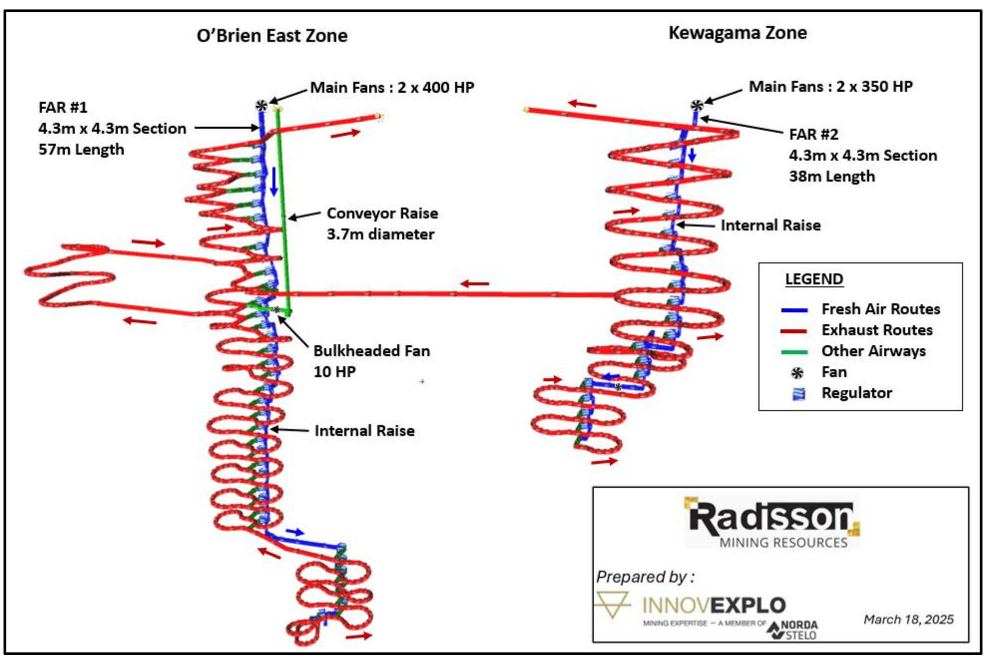
16.4.5.3 Permanent Ventilation Network

The early permanent ventilation network commences operation as soon as the fresh air raise connecting the surface to the upper ventilation access drift is completed to form the first loop of airway. The fresh air raises (sized 4.3 m x 4.3 m), the ventilation access drift, and the internal raises are fresh air routes, while the ramp is the exhaust path.

The O'Brien East airflow requirement is supplied through the fresh air raise #1 by two 400 hp main fans mounted in parallel. A small fan will be bulkheaded in the conveyor raise access drift to force downward fresh airflow in the conveyor raise. Internal raises in the upper levels (Levels 50, 75, and 100) have the same section size as the fresh air raise. From Level 125 to the bottom, their sectional dimensions are 3.1 m x 3.1 m.

In the Kewagama zone, the required airflow is supplied through fresh air raise #2 by two 350 hp main fans mounted in parallel. The internal raises are sized 4.3 m x 4.3 m for Levels 25 and 50, and 3.1 m x 3.1 m for other levels. The main ventilation network is shown in Figure 16-14.

Figure 16-14: Main Ventilation Network



Source: Norda Stelo (2025).

16.4.5.4 Level Ventilation

Each production level should provide sufficient fresh air to accommodate two LHDs and one production truck. On each level, trucks are only expected to travel along the level access drift up to the loading bay area, where a minimum of 44.4 kcfm of fresh air will be available.

A typical level ventilation system consists of an internal air raise connected to a ventilation access drift. Within this drift a bulkhead is installed housing a 75 hp auxiliary fan that supplies fresh air through a 48-inch diameter flexible duct.

In the stope areas, where LHD loaders operate for mucking or backfilling, additional airflow is delivered via a 25 hp auxiliary fan connected to another 48-inch diameter flexible duct to meet higher ventilation demands.

16.4.5.5 Main Fans and Heating System

The main fans at full capacity have been pre-selected following the result of the ventilation simulation. The heater system will be powered with natural gas, which has been selected according to the amount of fresh air blown by the fans and the anticipated climatic conditions at the project site throughout the year. The specifications for the selected fans and gas burners were provided by the manufacturer Howden. Table 16-14 shows the selected model of fans and heater for the permanent ventilation network.

Table 16-14: Main Fans and Natural Gaz Heaters

Fan & Heater Location	Fan Model	Fan Nominal Power	Airflow	Burner Capacity
O'Brien East Fresh Air Raise #1	AFN SO 54 1200 1866	2 x 400 hp	2 x 200 kcfm	2 x 18 MMBTUH
Kewagama Fresh Air Raise #2	AFN SO 54 1200 1866	2 x 350 hp	2 x 137 kcfm	2 x 13 MMBTUH

Note: MMBTUH - Million British thermal unit hours.

16.5 Production Schedule

The production schedule was generated with the Deswik Scheduler™ module. The production strategy is to access the planned stopes in the upper levels of the O'Brien East and Kewagama zones as early as possible. Initial mineralized material development is planned for Q1 of Year 1.

Stopes mucking will start in Q4 of Year 1 with two active stopes in the Kewagama zone and four active stopes in the O'Brien East zone. During the 21 months of pre-production, mineralized material will be sourced from development headings and a limited number of stopes, for a cumulative total of 197,000 tonnes at an average grade of 3.49 g/t Au. This material will be mined and hauled to a temporary surface stockpile. It will then be transported via haulage road by truck to the Westwood complex mill (IAMGOLD) 21 km away.

Commercial production is planned for the start of Q2 of Year 2. This moment marks the time where production reaches 70% of the steady-state production average of approximately1,410 t/d. The production rate will stay above the 70% target for more than 90 days after that date. Commercial production usually is dictated by mill production; however, since the O'Brien Gold Project will deal with an independent milling site, underground mineralized material production was used.

Based on the current mineral resource estimate, the project has a mine life extending to Q1 of Year 12. The life-of-mine plan outlines a rapid production ramp-up in Year 2, reaching an average of approximately 65,000 ounces of gold per year over the commercial production. Year 11, the last complete year, is projected to deliver 264,000 tonnes at a grade of 5.95 g/t Au. All tonnages and grades presented in Item 16 refer to diluted mineralized material, accounting for mining recovery and other underground factors, but excluding mill recovery.

An average of 13.4 km (linear-equivalent metres) of horizontal development is planned annually from Year 1 to Year 3, with a peak of 14.4 km in Year 2. Development then decreases to an average of 9.2 km per year from Year 4 to Year 6, and 3.9 km per year from Year 7 to Year 11.

A summary of the underground schedule, both overall and by mining zone, is provided in Tables 16-15 and 16-16, respectively.

Table 16-15: Underground Schedule Summary

Item	Unit	1	2	3	4	5	6	7	8	9	10	11	12	Total
Development														
Horizontal Development	m	11,373	14,421	14,468	10,353	10,100	7,401	3,984	4,107	3,596	3,965	911	0	84,680
Vertical Development	m	259	326	508	234	83	84	84	62	0	43	0	0	1,683
Total Development	m	11,632	14,747	14,976	10,587	10,183	7,486	4,067	4,169	3,596	4,008	911	0	86,363
Rehabilitation & Slashing	m	197	0	0	438	800	97	0	0	0	0	0	0	1,532
Mineralized Material														
Mineralized Material Development	kt	24	79	80	45	73	48	23	32	28	29	8	0	469
Mineralized Material Production	kt	11	300	359	392	365	389	416	361	227	247	245	3	3,316
Total Mineralized Material	kt	34	379	439	438	438	438	439	393	255	276	253	3	3,785
Total Mineralized Material per Day (Average)	t/d	94	1,039	1,202	1,199	1,199	1,200	1,203	1,076	700	756	693	8	N/A
Gold Grade	g/t	5.26	5.84	5.31	5.58	5.49	5.38	5.53	5.73	6.59	6.35	6.15	3.84	5.71
Gold	koz	6	71	75	78	77	76	78	72	54	56	50	0	695
Low-Grade														
Mineralized Material Development	kt	66	129	158	93	106	77	45	34	33	37	11	0	790
Total Mineralized Material per Day	t/d	179	354	433	256	290	212	122	94	91	100	31	0	N/A
Gold Grade	g/t	1.71	1.75	1.76	1.78	1.84	1.79	1.78	1.62	1.79	1.71	1.70		1.76
Gold	koz	4	7	9	5	6	4	3	2	2	2	1	0	45
Waste & Backfill														
Waste Produced	kt	578	569	533	445	357	252	130	151	126	148	25	0	3,314
Cemented Rockfill	kt	6	130	129	181	116	142	144	108	104	85	48	0	1,194
Rockfill	kt	3	107	138	127	141	134	162	140	68	95	87	5	1,207

Table 16-16: Underground Schedule Summary per Zone

Item	Unit	1	2	3	4	5	6	7	8	9	10	11	12	Total
Kewagama														
Total Development	m	5,771	5,196	3,687	3,519	2,855	3,440	437	0	0	0	0	0	24,905
Rehabilitation & Slashing	m	197	0	0	218	800	97	0	0	0	0	0	0	1,312
Total Mineralized Material	kt	24	162	114	82	118	116	103	18	0	0	0	0	736.0
Gold Grade	g/t	5.59	6.32	4.41	4.80	5.56	5.65	4.75	4.57	0	0	0	0	5.34
Gold	koz	4	33	16	13	21	21	16	3	0	0	0	0	126
Total Low Grade	kt	33	37	29	24	25	41	9	0	0	0	0	0	197.6
Gold Grade (LG)	g/t	1.71	1.79	1.83	2.04	2.01	1.83	2.15	0.00	0	0	0	0	1.86
Gold (LG)	koz	2	2	2	2	2	2	1	0	0	0	0	0	12
O'Brien East														
Total Development	m	5,861	9,551	11,289	7,068	7,328	4,045	3,631	4,169	3,596	4,008	911	0	61,458
Rehabilitation & Slashing	m	0	0	0	221	0	0	0	0	0	0	0	0	221
Total Mineralized Material	kt	10	217	325	356	320	322	336	375	255	276	253	3	3,048
Gold Grade	g/t	4.50	5.48	5.63	5.75	5.46	5.28	5.77	5.79	6.59	6.35	6.15	3.84	5.80
Gold	koz	2	38	59	66	56	55	62	70	54	56	50	0	568
Total Low Grade	kt	32	92	129	70	81	36	36	34	33	37	11	0	592.1
Gold Grade (LG)	g/t	1.72	1.74	1.75	1.69	1.78	1.75	1.69	1.62	1.79	1.71	1.70	0	1.73
Gold (LG)	koz	2	5	7	4	5	2	2	2	2	2	1	0	33

16.6 Mining Sequence

To maximize the stability of the mining area by diverting induced stresses, the two zones will be subdivided into production centres. This strategy will also increase the number of working areas to sustain the production level. The longitudinal retreat method is used to create pyramidal shapes as mining progresses in a production centre.

A typical mining cycle includes secondary ground support, where required. V-30 slot-drilling will be done in advance of the production drill mobilization, followed by the complete production drilling of the stope. Longitudinal stopes will be blasted in three phases: a primary blast for the void, and then secondary and tertiary blasts 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 and the stope is backfilled with cemented rockfill (rockfill is used when possible). Table 16-17 summarizes the operating rates and corresponding delays assigned to each mining activity.

Table 16-17: Main Production Activity Rates

Activity	Units	Rate	Comments
Production Cables	m/d	75	Only for stopes in contact with the fault
Slot	day	3	V30
Production Drilling	m/d	150	Drilling factor of 3,38 t/m[1]
Blast and Pull Void	day	4	
Uncertainty	day	6.2	30% of the average production cycle
Mucking	t/d	700	Average remuck distance: 275 m
Cemented Rockfill	t/d	485	One day of delay for barricades
Rockfill	t/d	700	

Notes: [1]The drilling factor was evaluated based on the size of an average stope (2.1 m) without internal dilution (ELOS method).

A 14-day curing period for the backfill was assumed before blasting adjacent stopes to minimize backfill dilution. A seven-day delay was also applied before blasting an upper stope located above a backfilled stope to prevent the scoop from digging.

A 20-day delay was applied between the last development activity and the first stope in the sequence to allow additional rounds or slash to be excavated for drilling purposes and to complete the geological interpretation of the mining horizon.

An allowance for uncertainty has been built into the schedule to reflect potential issues that may arise during stope excavation, such as a block detaching from the wall and hindering the excavation process, equipment breakdown, blocked drill holes, misfires, and other operational challenges. The goal was to create a realistic sequence that allows room for performance improvement.

16.7 Grade Control

Grade control includes definition drilling at a 12.5 m spacing, as well as carrying out operating development test holes, muck samples and channel samples, as well as stope muck samples.

Development within the mineralized material zone will require collaboration between the operations and geology departments, which is why development in the mineralized material is slowed down. The interpretation of geological data and the results of sampling can delay the advance cycle of a stope. For this reason, the advance fronts have been limited to one round every two days. The cost per tonne has been estimated based on Norda Stelo's database.

16.8 Mining Equipment Fleet

The operational quantities required for all major and critical equipment (i.e., jumbo drill, cable bolter, production drills, LHDs, trucks, etc.) were estimated during the planning process. Yearly operation hours have been estimated for secondary services equipment based on typical operation and current mine scheduling requirements.

It is assumed all the equipment listed for the project will be acquired by the Owner between Years -1 and 9. The mobile equipment fleet required for underground operation is presented by year in Table 16-18.

Table 16-18: Mobile Equipment Fleet for Underground Operation

Equipment Type	Brand	Model	Max.	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Development															
Jumbo	Epiroc	Boomer S2	5	2	5	5	5	3	3	2	2	1	1	1	0
LHD 6 t	Epiroc	ST3.5	4	2	4	4	4	2	2	1	1	1	1	1	0
Truck 20 t	Epiroc	MT2200	7	4	7	7	7	5	5	3	3	3	3	1	0
Scissor Lift	XCMG	SL3	5	2	5	5	5	3	3	2	2	1	1	1	0
Bolter	Sandvik	DS312	8	4	8	8	8	5	5	3	3	2	2	2	0
Production															
Production Drill	RESEMIN	Stopemaster Series HX	3	1	2	1	3	2	2	1	1	1	1	1	1
LHD 6 t	Epiroc	ST3.5	3	2	3	4	3	3	3	2	2	1	1	1	1
Truck 20 t	Epiroc	MT2200	5	4	5	5	5	5	5	3	3	2	2	2	2
ANFO Truck	Paus	Maclean	2	1	1	1	2	1	1	1	1	1	1	1	1
Services															
Transmixer	Paus	UNI 40 BM	2	0	2	2	2	2	2	1	1	1	1	1	1
Boom Truck	Paus	UNI 40 LP	2	2	2	2	2	2	2	1	1	1	1	1	1
Personnel Carrier	Toyota	PC3	2	2	1	2	2	2	2	1	1	1	1	1	1
Mechanical Truck	Toyota	Land Cruiser HZJ79 - BTE 134	2	2	1	2	2	2	2	1	1	1	1	1	1
Fuel-Lube Truck	Paus	UNI 40 SV	2	2	1	2	2	2	2	1	1	1	1	1	1
Underground Grader	Paus	PG 10 HA	2	0	2	2	2	2	2	1	1	1	1	1	1
Water Truck	WS3	Maclean	2	2	2	2	2	2	2	1	1	1	1	1	1
Electric Vehicle	Toyota	Land Cruiser HZJ79 - BTE 134	2	2	2	2	2	2	2	1	1	1	1	1	1
Light Vehicle	Toyota	Land Cruiser HZJ79 - BTE 133	6	5	6	6	6	6	6	3	3	3	3	3	3
Tractor	Kubota	M4D	4	4	4	4	4	4	4	2	2	2	2	2	2
Mine Rescue - Light Vehicle	Toyota	Land Cruiser HZJ79 - BTE 134	1	1	1	1	1	1	1	1	1	1	1	1	1
Total Equipment			**69**	**44**	**64**	**68**	**69**	**56**	**56**	**32**	**32**	**27**	**27**	**25**	**19**

16.9 Mine Personnel

Mine personnel are allocated across three primary operational areas: (1) underground services (including supervision, construction, development, and production), (2) underground maintenance (mechanical and electrical), and (3) technical services.

Electrical and mechanical supervision will alternate between day and night shifts as required, ensuring that a supervisor or senior employee is always present to oversee operations. Additional supervisory staff, technicians, and specialized workers will be scheduled from Monday to Friday (5-2), on day shifts only.

16.9.1 Mine Production

“Production personnel” refer to the operations personnel and maintenance staff. Operations personnel include underground supervisory (e.g., mine superintendent, captain, and trainer) personnel, as well as blasters and individuals required to operate major and secondary equipment. Maintenance personnel include the maintenance management staff, mechanics, electricians, and helpers.

The projected staffing requirements for production over the life of mine are detailed in Table 16-19.

16.9.2 Technical Services

Most technical services personnel will be scheduled to work from Monday to Friday (5-2) on dayshift only. Certain roles may be assigned to a seven-days-on/seven-days-off rotation to ensure continuous operational support.

A list of technical services personnel required over the life of mine is presented in Table 16-20.

Table 16-19: Production Personnel

Production	Max.	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Operations													
Mine Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Captain	1	1	1	1	1	1	1	1	1	1	1	1	1
Shifter-Development	4	2	4	4	4	4	4	4	4	4	4	4	2
Shifter-Production	4	2	4	4	4	4	4	4	4	4	4	4	2
Jumbo Operator	8	4	8	8	8	8	8	8	8	4	4	4	2
Bolter Operator	16	4	16	16	16	16	16	16	16	8	8	8	4
Long Hole Driller	8	4	8	8	8	8	8	8	8	4	4	4	2
Blaster	4	2	4	4	4	4	4	4	4	2	2	2	2
Scoop Operator	20	10	20	20	20	20	20	20	20	20	20	20	10
Truck Operator	24	12	24	24	24	24	24	24	24	24	24	24	12
Grader Operator	4	2	4	4	4	4	4	4	4	2	2	2	2
Service Miner	8	8	8	8	8	8	8	8	8	8	8	8	8
Trainer	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance													
Maintenance Manager	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Supervisor	1	1	1	1	1	1	1	1	1	1	1	1	1
Maintenance Planner	1	1	1	1	1	1	1	1	1	1	1	1	1
Electricians	10	8	10	10	10	10	10	10	10	10	10	10	8
Mechanics	12	8	12	12	12	12	12	12	12	12	12	12	8
Helpers	4	4	4	4	4	4	4	4	4	4	4	4	2
Total	**132**	**76**	**132**	**132**	**132**	**132**	**132**	**132**	**132**	**112**	**112**	**112**	**70**

Table 16-20: Technical Service Personnel

Technical Services	Max.	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8	Year 9	Year 10	Year 11	Year 12
Geology													
Chief Geologist	1	1	1	1	1	1	1	1	1	1	1	1	1
Database Technician	1	0	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist, Resources	1	0	1	1	1	1	1	1	1	1	1	1	1
Senior Geologist, Production	1	1	1	1	1	1	1	1	1	1	1	1	1
Intermediate Geologist	1	0	1	1	1	1	1	1	1	1	1	1	1
Junior Geologist	1	0	1	1	1	1	1	1	1	1	1	1	1
Senior Geology Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Geology Technician	2	1	2	2	2	2	2	2	2	2	2	2	1
Journeyman Core Shack	2	1	2	2	2	2	2	2	2	2	2	2	1
Engineering													
Technical Superintendent	1	1	1	1	1	1	1	1	1	1	1	1	1
Chief Engineer	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mining engineer (Planning)	1	1	1	1	1	1	1	1	1	1	1	1	1
Senior Mining engineer (Rock Mechanics)	1	0	1	1	1	1	1	1	1	1	1	1	1
Intermediate Mining Engineer (Planning + Development)	1	0	1	1	1	1	1	1	1	1	1	1	1
Intermediate Mining Engineer (Ventilation)	1	0	1	1	1	1	1	1	1	1	1	1	1
Intermediate Mining Engineer (Stoping)	1	0	1	1	1	1	1	1	1	1	1	1	1
Intermediate Mining Engineer (Construction + Cost)	1	0	1	1	1	1	1	1	1	1	1	1	1
Junior Mining Engineer	1	0	1	1	1	1	1	1	1	1	1	1	1
Senior Mine Technician	1	1	1	1	1	1	1	1	1	1	1	1	1
Mining Technician (Survey)	4	2	4	4	4	4	4	4	4	4	4	4	4
Mining Technician (Rock Mechanics)	4	2	4	4	4	4	4	4	4	4	4	4	4
Mining Technician (Stoping)	4	0	4	4	4	4	4	4	4	4	4	4	4
Mining Technician (Construction)	4	0	4	4	4	4	4	4	4	4	4	4	4
Junior Mining Technician	4	0	4	4	4	4	4	4	4	4	4	4	4
Total	**41**	**13**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**41**	**39**

17 RECOVERY METHODS

17.1 Overview

A testwork program was conducted for the preliminary economic assessment. The program was designed to determine the optimal recovery method for the processing of O'Brien mineralized material at the Westwood complex. Radisson entered into a memorandum of understanding titled "Memorandum of Understanding Between Radisson Mining Resources Inc. and IAMGOLD Corporation," (September 6, 2024) with IAMGOLD Corporation to assess the design criteria for processing mineralized material from the O'Brien deposit at the Westwood complex.

An economic assessment was conducted on the testwork results, with the results indicating that the separation of gold through gravity separation, flotation, flotation concentrate regrind, carbon in leach (CIL) and carbon in pulp (CIP) circuit would provide the highest NPV and be compatible with the existing process plant at the Westwood complex. The recovery method is therefore based on the separation method described above, utilizing the existing crushing and primary grinding equipment as well as Westwood complex infrastructure as needed.

The process design has been based on a toll milling campaign basis, where the Westwood complex would alternate between processing mineralized material from the IAMGOLD Westwood mine, and then from the O'Brien mine, without processing the two materials concurrently. A design throughput of 3,000 t/d was selected based on existing equipment sizing and material competency and hardness. Overall mill capacity utilization depends on the production rate from the O'Brien project mine site, and ranges from 1,160 to 2,000 t/d.

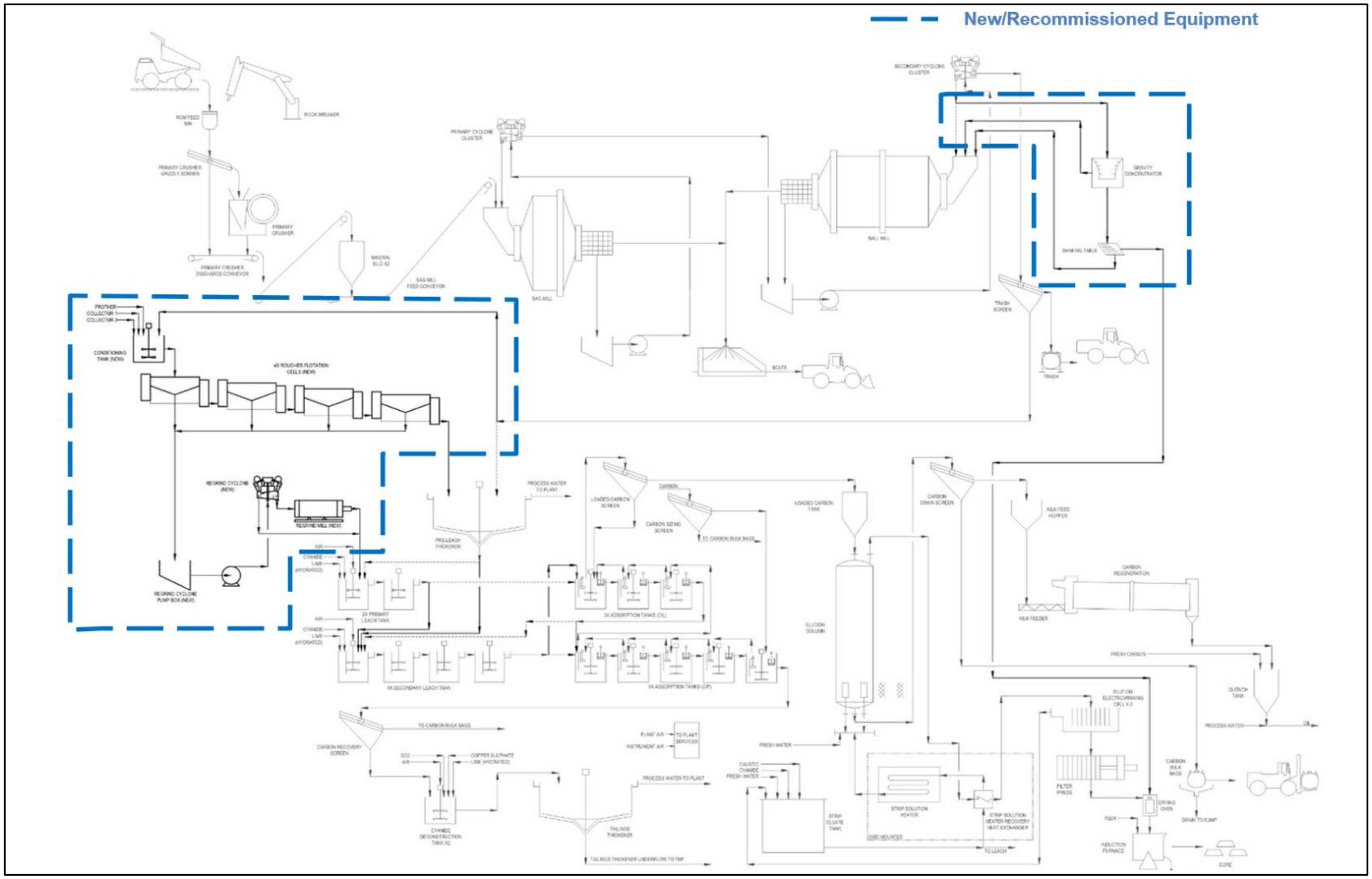
17.2 Process Flowsheet

The process flowsheet includes the circuits listed in Table 17-1. The overall process flow diagram is presented in Figure 17-1. The Westwood complex is also equipped with a paste plant, but use of this facility has not been considered necessary for processing the O'Brien material at this time.

Table 17-1: Process Flowsheet Description

Equipment/Circuit Description	New or Existing Equipment
Primary Crushing	Existing
Feed Material Storage	Existing
SAG Mill and Ball Mill	Existing
Gravity Concentration and Shaking Table	Existing
Rougher Flotation Circuit with Regrind Mill	New
Pre-Leach Thickener, Leach and Carbon Adsorption	Existing
Pressure Zadra-Type Elution followed by Electrowinning and Smelting	Existing
Carbon Regeneration by Rotary Kiln	Existing
Cyanide Destruction of Tailings using the SO_2/Air Process	Existing
Tailings Thickening	Existing

Figure 17-1: Process Flow Diagram (New/Recommissioned Equipment in Blue)



Source: Ausenco (2025).

17.3 Process Design Criteria

Key process design criteria for the O'Brien project are listed in Table 17-2.

Availability of the circuit has been considered as 100%, assuming that when toll milling is being carried out at the Westwood complex on O'Brien feed, the plant will be available 100% of the time. Any maintenance, either planned or unplanned, is not assumed to occur during toll milling production hours.

Existing equipment specifications were used to determine the capacity for processing O'Brien material. A primary grind size of 75 µm (P_{80}) was selected based on the available capacity of the grinding circuit achieving at 3,000 t/d.

Table 17-2: Process Design Criteria and Existing Equipment

Design Parameter	Units	O'Brien
Plant Throughput	t/d	3,000
Gold Grade – Design Mill Head	g/t	4.9 to 6.9
Crushing Plant Availability	%	100% (de facto)
Mill Availability	%	100% (de facto)
Bond Crusher Work Index (CWi), 75th Percentile	kWh/t	18.7
Bond Rod Mill Work Index (RWi), 75th Percentile	kWh/t	18.2
Bond Ball Mill Work Index (BWi), 75th Percentile	kWh/t	12.8
SMC Axb, 25th Percentile	-	35
Bond Abrasion Index (Ai)	g	0.39
Material Specific Gravity	-	1.58
Primary Crusher	-	Jaw, 1.07 m x 1.22 m
SAG Mill	-	5.4 m diameter x 6.3 m EGL
SAG Mill Installed Power	kW	2,013 (2,700 hp)
Ball Mill	-	3.5 m diameter x 4.9 m EGL
Ball Mill Installed Power	kW	745.7 (1,000 hp)
Primary Grind Size (P_{80})	µm	75
Fraction of Feed to Gravity Gold	% of new feed	100
Gravity Gold Recovery (Design)	% Au	42
Total Gold Recovery (Life of Mine)	% Au	87.5
Rougher Residence Time	min	4
Rougher Mass Pull	%	10
Concentrate Pulp Density	% w/w	25
Pre-Leach Thickener Diameter	m	27
Pre-Leach Thickener Underflow Density	% w/w	53
Primary Leach Capacity	m^3	772
Primary Leach Residence Time	h	10
Primary Leach Extraction, Average	% Au	80.8
Secondary Leach Capacity	m^3	5,545
Secondary Leach Residence Time	h	26

Design Parameter	Units	O'Brien
Secondary Leach Extraction, Average	% Au	80.6
Carbon Adsorption, Number of Stages	Quantity	3 CIL + 5 CIP
Carbon Adsorption Capacity	m^3	1,112 (CIL), 1,860 (CIP)
Adsorption Tail Solution Concentration, Target	ppm Au	< 0.1
Primary Leach Sodium Cyanide Addition	kg/t leach feed, concentrate	15.6
Secondary Leach Sodium Cyanide Addition	kg/t leach feed, combined	1.2
Primary Leach Hydrated Lime Addition	kg $Ca(OH)_2$/t leach feed, concentrate	5.9
Secondary Leach Hydrated Lime Addition	kg $Ca(OH)_2$/t leach feed, combined	0.5
Elution Method	-	Pressure Zadra
Carbon Batch Size	t	4
Detoxification Residence Time (per Tank)	min	45
Detoxification Tanks	Quantity	2
Detoxification SMBS Addition	g SO_2/g CN_{WAD}	5.0
Detoxification Copper Sulphate	g Cu/m^3	21
Detoxification Lime Addition	g $Ca(OH)_2$/g SO_2	1.0
Detoxification Discharge CN_{WAD}, Design	mg/L CN_{WAD}	1
Tailings Thickener Diameter	m	20
Tailings Thickener Underflow Density	% w/w	53

17.4 Plant Design

The following subsections include a description of the IAMGOLD Westwood Complex process plant as well as descriptions of new equipment, the flotation and regrind circuit.

17.4.1 Mineralized Material Handling and Crushing (Existing)

O'Brien mineralized material will be delivered by truck to the Westwood complex and be dumped onto the run-of-mine pad adjacent to the existing crusher feed location and stored alongside Westwood mineralized material. The run-of-mine pad can store up to 60,000 tonnes of feed material. The material will be crushed via a 1.07 m by 1.22 m jaw crusher. The crushed mineralized material will be conveyed to the milling feed bins using the existing conveyor systems. The mineralized material from the milling feed bins will be then conveyed to the SAG mill feed conveyor.

Key equipment for the materials handling and crushing circuit includes the following:

- milling feed bins
- 1.07 m x 1.22 m jaw crusher
- various conveyor systems
- material handling equipment.

17.4.2 Grinding Circuit (Existing)

The existing grinding circuit consists of a 2,000 kW SAG mill in closed circuit followed by a 745 kW ball mill in closed circuit. The SAG mill discharge is sent to the primary cyclone feed pump box where the underflow of the primary cyclones recycles back to the SAG mill and the overflow of the primary cyclones is sent to the secondary cyclone feed pump box. The secondary cyclones overflow feeds directly to the trash screen before entering the flotation circuit while the underflow of the secondary cyclones is the feed to the ball mill. The ball mill discharges back into the secondary cyclone feed box completing the closed circuit. It is assumed that the availability of the grinding circuit during active toll milling is a de facto 100%.

Key equipment for the grinding circuit includes the following:

- 5.4 m diameter x 6.3 m effective grinding length (EGL) 2,000 kW SAG mill
- 3.5 m diameter x 4.9 m effective grinding length (EGL) 745 kW ball mill
- primary cyclone feed and secondary cyclone feed pump box
- primary and secondary cyclone feed pumps
- primary and secondary cyclones
- trash screen.

17.4.3 Gravity Recovery Circuit (Recommissioned)

The existing gravity circuit comprises of one scalping screen and a centrifugal batch Knelson concentrator. Feed to the circuit is directed from the secondary cyclone underflow directed by gravity flow to the scalping screen. Gravity scalping screen oversize reports back to the secondary cyclone pump box mill via gravity flow.

Scalping screen undersize is fed to the centrifugal concentrator. Operation of the gravity concentrator is processed in a semi-batch mode where the gravity concentrate is collected in the concentrate storage cones and subsequently discharged to the vibrating table. The tailings from the gravity concentrators also report back to the ball mill via gravity flow.

The vibrating table having received the gravity concentrate will oscillate with water addition so that the lighter material will be washed away, further concentrating the gravity concentrate. The vibrating table concentrate will then be sent directly to the gold drying ovens and into the induction furnace to produce gold doré.

Key equipment for the gravity circuit includes the following:

- gravity feed scalping screen
- gravity concentrator
- vibrating table.

17.4.4 Flotation and Regrind Circuit (New)

Existing decommissioned flotation cells will be removed, and four new flotation cells and a rougher conditioning tank will be installed for use as a rougher flotation circuit. The rougher conditioning tank will be fed by the secondary cyclone overflow with the density adjusted to 25% w/w solids prior to flotation.

Two collectors and a frother reagent will be added to the rougher conditioning tank with the option to stage add reagents along the flotation bank. The flotation concentrate mass pull is targeted to be 10% based on observations from current testwork.

The flotation concentrate will be collected in launders and flow via gravity to a new open-circuit regrind circuit. The regrind mill is fed by the regrind cyclone pump box where the cyclone overflow will go to regrind discharge pump box while the regrind cyclone underflow will go to an IsaMill for regrind. The ground IsaMill product will be discharged into the regrind discharge pump box and will combine with regrind cyclone overflow to be pumped to the primary leach tanks at 15 µm (P_{80}).

The flotation tailings will flow to the flotation tailings pump box and will be pumped directly to the secondary leach tanks.

17.4.5 Leach and Adsorption Circuit (Existing)

The leach-adsorption circuit consists of two primary leach tanks, four secondary leach tanks, three carbon-in-leach (CIL) tanks, and five carbon-in-pulp (CIP) tanks. The primary leach tanks provide 10 hours of leach residence time while the secondary leach, CIL, and CIP tanks combined for a residence time of 35.5 hours.

The primary leach tanks are fed by the flotation reground concentrate at a density of 25% w/w solids. Air is sparged into alternating tanks to maintain adequate dissolved oxygen levels. Hydrated lime is added to the achieve the operating pH to the desired set point of 10.5 and a heavy dose of cyanide solution is added to the first leach tank for intense leaching.

The secondary leach tanks are fed by both the leached concentrate from the primary leach tanks as well as the flotation tails. Hydrated lime is added to the achieve the operating pH to the desired set point of 10.5 and a dose of cyanide solution is added to the first of the secondary leach tanks.

Fresh/regenerated carbon from the carbon regeneration circuit is returned to the last tank of the CIP circuit and is advanced counter-currently to the slurry flow to the CIL circuit by pumping slurry and carbon. Slurry from the last CIL tank flows to the cyanide detoxification tanks. The intertank screen in each CIL/CIP tank retains the carbon while allowing the slurry to flow by gravity to the downstream tank. This counter-current process is repeated until the loaded carbon reaches the first CIL tank. Recessed impeller pumps are used to transfer slurry between the CIL/CIP tanks and from the lead tank to the loaded carbon screen in the elution circuit.

The leach and carbon adsorption circuit key equipment includes:

- pre-leach thickener
- leach (primary/secondary)/CIL/CIP tanks and agitators
- loaded carbon screen
- intertank carbon screens
- carbon sizing screen.

17.4.6 Cyanide Destruction (Existing)

Tailings are treated with an SO_2/air cyanide destruction process. Leach residues are split between two parallel reactors. During current operations under IAMGOLD, when processing Westwood feed material, a part of the process discharge residues is used to feed the paste backfill plant while the remainder is pumped to the TMF. A buffer tank (14 hours of backfill plant operation capacity) is maintained at maximum level with detoxified slurry to ensure continuous feed to the backfill plant in case of a temporary process plant shutdown. Usage of the paste plant has not been considered for the processing of O'Brien material.

17.4.7 Electrowinning & Gold Room (Existing)

Gold is recovered from elution pregnant solution by electrowinning and smelted to produce doré bars. The pregnant solution is pumped through one of two electrowinning cells fitted with stainless steel mesh cathodes. An electrical current is applied across the cells, causing gold to deposit on the surface of the cathodes. The remaining barren solution transferred to the leach circuit. The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and gravitates to the sludge hopper. The sludge is filtered, dried, mixed with fluxes, and smelted in an electric induction furnace to produce gold doré. The electrowinning and smelting process takes place within a secure and supervised gold room.

17.4.8 Paste Plant (Existing)

The Westwood complex is equipped with a paste plant; however, tailings from the O'Brien material are not planned to report to the paste plant. They will instead report to tailings management.

17.4.9 Tailing Thickening and Management (Existing)

The tailings will be pumped to the existing Westwood tailings management facility. The facilities have not been reviewed for compatibility or investigate for required modifications for this report. Discussions with IAMGOLD indicated that milling scenarios should currently assume the use of existing tailings facilities for the disposal of O'Brien tailings, but this is not based on an assessment of the geochemical properties of the O'Brien tailings.

17.5 Energy, Water, and Process Materials Requirements

17.5.1 Process Materials

Reagents and consumables required for the flotation cells and regrind mill to process O'Brien mineralized material are listed in Table 17-3 below in blue. Also included in the table are the estimated annual consumption rates of reagents and consumables for the other areas of the Westwood complex, in black text. These estimates were developed independently of IAMGOLD based on a review of the existing equipment and current technical reports. The estimates were not based on information provided by IAMGOLD.

Table 17-3: Reagents and Consumables

Description	Unit	Consumption
Consumables		
Jaw Crusher Cheek and Swing Jaw Set	set/a	1
Crusher Screen Panel	set/a	4
Ball Mill Grinding Media	t/a	1,095
SAG Mill Grinding Media	t/a	177
SAG Mill Liner	set/a	1
Ball Mill Liner	set/a	1
Regrind Mill Media	t/a	41
Regrind Mill Liner	set/a	1
Crucibles	#/a	12
Reagents		
Sodium Hydroxide	t/a	71.2
Sodium Cyanide	t/a	3,023
PAX	t/a	137
MIBC	t/a	11.0
R208	t/a	82.1
Lime	t/a	2,142
Activated Carbon	t/a	66
Hydrochloric Acid	t/a	1,718
Sulphamic Acid	t/a	0.62
Borax	t/a	4
Silica	t/a	2
Sodium Nitrate	t/a	0.34
Sodium Carbonate	t/a	0.34
SMBS	t/a	1,473
Copper Sulphate	t/a	28
Antiscalant	t/a	14.2
Dust Suppression	t/a	32.2
Effluent Treatment Reagents		
Ferric Sulphate	t/a	158
Hydrogen Peroxide	t/a	6.9
Flocculant	t/a	1.1

17.5.2 Water Requirements

Water requirements for the process plant will be met using existing water treatment and management processes and systems.

Fresh water is sourced from Bousquet River. A water balance for the Westwood complex was not completed for this report.

Water usage during operations is not expected to be significantly different when processing O'Brien mineralized material compared to current operations. Future project phases should review the overall Westwood complex water balance when toll milling.

17.5.3 Power Requirements

The process plant with the new flotation cells and regrind mill will have an estimated nominal demand of 5.1 MW and an estimated annual power consumption of 43 MWh/a.

The additional installed power requirements of the regrind and flotation circuits are estimated to be approximately 1.8 MW. Discussions with IAMGOLD regarding capacity indicated that there is sufficient additional capacity available with the existing electrical infrastructure and with Hydro-Quebec. Future studies should consider reviewing the electrical infrastructure of the Westwood complex in more detail to confirm sufficient capacity.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

The mine site infrastructure required at O'Brien includes civil infrastructure, site facilities/buildings, water management infrastructure, waste rock and process feed material storage facilities, and electrical infrastructure.

Buildings located at the mine site will include the following:

- mine dry and administration office
- cemented rockfill building
- water treatment building
- mineralized material temporary stockpile building.

Siting of the facilities has been based on locating the infrastructure around the portal locations. The overall site layout is shown in Figure 18-1.

18.2 Site Access

The project site can be reached from the town of Rouyn-Noranda by travelling approximately 50 km east along Highway 117. The former mine site is approximately 750 m north of Highway 117 as it crosses through the town of Cadillac. From Highway 117, the project site can be accessed by Rue du Petit-Canada. The town of Val D'or is located approximately 55 km west of the project on Highway 117.

Rouyn-Noranda and Val D'or both have regional airports with regularly scheduled flights to and from Montreal, which acts as a hub for flights to the north. The project site is a 6.5-hour drive northwest from Montreal, and there is daily bus service between Montreal and the other cities in the Abitibi region.

A 120 kV powerline is located in the right-of-way directly south of the property.

Supplies, labour, and service providers are readily available in the general area (in Rouyn-Noranda and Val-d'Or). Local resources include commercial laboratories, federal government underground mining research office, construction contractors, drilling companies, exploration service companies, engineering and various other consultants, and equipment vendors and suppliers.

Figure 18-1: Overall Mine Site Layout

Source: Ausenco (2025).

18.3 Built Infrastructure

18.3.1 Introduction

The existing roads at site will be improved to support heavy vehicles, including haul trucks.

Areas where waste rock, buildings and other infrastructure are to be located will be cleared of overburden. The typical method of clearing, removing topsoil, and excavating will be employed. Infrastructure to be installed will include drainage ditches, collection ponds, stockpile pads, and roads.

Site civil work includes design for the following infrastructure:

- light vehicle and heavy equipment roads
- laydown areas
- mine portals
- building support
- water management ponds and contact and non-contact water channels
- waste rock storage facilities
- temporary mineralized material storage, prior to hauling.

18.3.2 Accommodation

Due to the project’s proximity to the towns of Rouyn-Noranda and Val-d’Or, mine employee accommodations will not be required on site, as these towns currently provide housing for similar local mining operations.

18.3.3 Buildings

Buildings to be constructed at the mine site are listed in Table 18-1.

Table 18-1: List of Mine Site Buildings

Building Description	Length (m)	Width (m)	Height (m)
Mine Dry/Administration Office	12.2	9.6	3
Crusher Rockfill Building	20	11	8
Water Treatment Building	21	36.5	N/A
Temporary Mineralized Material Stockpile Building	25 (m) diameter		~12.5

There are some existing buildings approximately 500 m west of the mine site that are owned, leased, and used by Radisson. These buildings include a core shack and offices, and an approximately 15 m by 25 m warehouse.

18.4 Waste Rock and Mineralized Material Storage Facilities

Underground development will generate approximately 3.3 Mt tonnes of waste rock. Of this, 2.4 Mt will be required for backfilling mined-out stopes. About 975,000 tonnes of waste rock material will be hauled to surface and will be stockpiled at a single location identified as the "waste rock storage facility." Over the life of mine, about 4.6 Mt of mineralized material will be transiting through the surface and loaded on trucks for transportation to the mineral process plant. The mineralized material will need to be temporarily stored near the vertical conveyor outlet on a laydown platform named the "mineralized material storage facility." The general arrangement plan presented in Figure 18-1 shows the location of the waste rock and mineralized material storage facilities.

The preliminary concepts of the waste rock and mineralized material storage facilities were developed by taking into consideration the site requirements with the intention to integrate, at an early-stage, applicable regulations and governmental recommendations embedded in Québec provincial guidelines: Directive 019 for the mining industry (MELCC, 2025) and the guidelines for preparing mine closure and restauration plans (MRNF, 2024). Further site investigation and analysis would be required to better characterize the ground foundation material properties, waste rock, and mineralized materials geochemical characteristics. Following completion of the laboratory testing and field works, the local and regional hydrogeological model will need to be refined and the long-term geotechnical stability of both storage facilities will need to be properly assessed.

18.4.1 Waste Rock Storage Facility

The waste rock storage facility is designed to contain waste rock production that will not be salvaged for underground backfilling operation over the entire life of mine. The storage facility will receive waste rock material trucked from the underground operation to the surface and will accommodate a total of approximately 553,000 m^3 of waste rock.

A geochemical characterization of the waste rock material was conducted at an early engineering level. Preliminary results indicate that waste rock material has a generally high net neutralization potential and a high NP/AP ratio (Genivar, 2012). Based on these results, the waste rock could be considered as non-potentially acid generating (non-PAG) and therefore, no specific ground protection beneath the facility is expected.

The bulk density of the placed waste rock material at the waste rock storage facility was conservatively assumed to be in the order of 1.77 t/m^3 with no significant moisture content.

No site-specific investigation and soil characterization beneath the footprint of the proposed facility have been conducted. However, some relevant information could be excerpted from the interactive map of SIGEOM website (MRNF) for the main overburden layers (quaternary geology deposits) observed in the area. These various soils layers were also identified in the vicinity of the storage facility during the hydrogeological site investigation conducted by Richelieu Hydrogéologie (2020). The foundation soils beneath the proposed waste rock storage facility would consist of a layer of organic matter (approximately 0.20 m thick), glacial sediments (i.e., till with a maximum thickness of approximately 3.50 m) and glaciolacustrine sediments (fine- to medium-grained sand with gravel and pebbles with a maximum thickness of 6 m).

The waste rock storage facility is located within the project site limits and positioned near the mine portals to minimize hauling distance from the underground operation and ensure proper water management. The footprint of the facility is also located more than 60 m away from the shoreline of any lake or regular or intermittent watercourses. The construction of the facility will include a waste rock base layer nominally compacted, a perimeter access road, two peripheral water collection ditches, and a water collection pond on the north side, namely, the sedimentation pond.

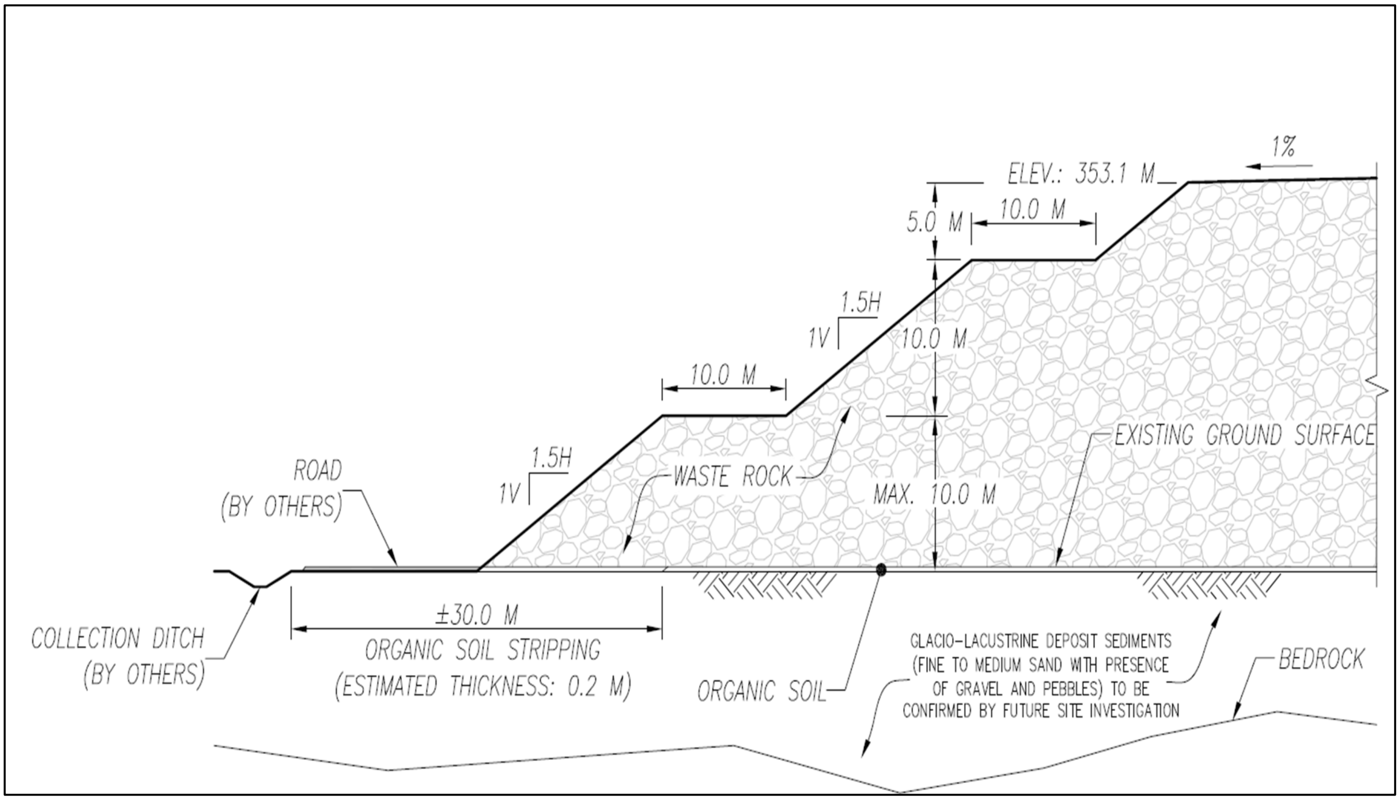
Runoff water from the waste rock storage facility will be collected by the peripheral ditches and conveyed by gravity to the sedimentation pond. The collected water will then be pumped towards the sedimentation basin, where it will be settled, monitored and then discharged to the environment. The sedimentation pond will have associated emergency spillways and water pumping infrastructure. Staged construction of the storage facility could be considered to accommodate mine waste production and water management infrastructure implementation.

The waste rock storage facility has a footprint of approximately 4.7 ha, excluding the peripheral ditches and access roads, and reaches heights varying between 17.4 m and 25.0 m for a maximum elevation of 353.1 m. The embankment will have an overall slope of 2.5H:1V to provide sufficient slope stability, and will be built with benches having a width of 10 m, a maximum inter-bench height of 10 m and a bench slope angle of 1.5H:1V. Waste rock material will be stockpiled with upstream lifts to provide sufficient drainage to avoid water accumulation within the facility. A typical cross-section of the embankment slopes is shown in.

During construction of the waste rock storage facility, peat and organic topsoil will be stripped along a 30 m side band beneath the slope of the first bench and downstream of the toe to improve global embankment slope stability. Stripping requirements will be developed at the next level of study and will depend on the observed foundation conditions and the results of the slope stability analysis. The topsoil material will be temporarily stored downstream of the facility footprint and used immediately to progressively reclaim the completed lower slopes or will be stockpiled at a dedicated overburden storage facility.

A cross-section of the waste rock storage facility is shown in Figure 18-2.

Figure 18-2: Waste Rock Storage Facility – Typical Cross-Section



Source: Norda Stelo (2025).

18.4.2 Mineralized Material Storage Facility

Throughout the life of mine, the underground operation will produce approximately 4.6 Mt of mineralized material. A storage facility for mineralized material will be constructed adjacent to the vertical conveyor discharge point. This facility will include a 4,000 m³ laydown platform, on which a 25 m diameter dome will be installed, providing sufficient temporary storage capacity for the mineralized material delivered to surface. This overflow storage will be available in the event material cannot be loaded and transported to the mineral processing plant. During normal operations, the mineralized material will be continuously reclaimed using a front-end loader to feed the haul trucks travelling to the processing plant.

A geochemical characterization of the mineralized material was conducted at an early engineering level. Preliminary results show that mineralized material has a high potential for acid generation (Genivar, 2012). Samples of mineralized material have also been tested for their metal leaching potential. Laboratory results from testing, such as the toxicity characteristic leaching procedure (TCLP), indicate that the mineralized material is leachable for some metals (Genivar, 2012). The management of the storage facility, including the infiltration and runoff water collection system, was designed accordingly.

The bulk density of the stockpiled mineralized material at the storage facility was conservatively assumed to be in the order of 1.86 t/m^3 with no significant moisture content.

No site-specific investigation and soil characterization beneath the footprint of the proposed facility have been conducted. However, some relevant information could be excerpted from the interactive map of SIGEOM website (MRNF) for the main overburden layers (quaternary geology deposits) observed in the area. These various soils layers were also identified during the hydrogeological site investigation conducted by Richelieu Hydrogéologie (2020) in the vicinity of the mineralized material storage facility. The foundation soils beneath the proposed facility are expected to consist of a layer of unsaturated organic matter (approximately 0.15 m thick) underlaid by a fluvio-glacial deposit (i.e., an esker) or a glacio-lacustrine deposit which could contain fine- to medium-grained sand with gravel and pebbles with a maximum thickness of 6 m.

The mineralized material storage facility is located between the vertical conveyor outlet and the western mine portal on a high-topographical point overlapping two local catchment areas. The site was selected to facilitate the surface disposal and loading operation of the mineralized material, to allow safe truck circulation and maintain operations close to the mine entrance, and to minimize transport distances to the mineral process plant. The footprint of the storage facility is located more than 60 m away from the shoreline of any lake or any regular or intermittent flowing watercourses. The facility has a footprint of approximately 0.5 ha, excluding the peripheral ditches, and will reach a height of about 2.5 m above ground level. In general, the mineralized material storage facility will present a side-slope, having an overall inclination angle of 1.5H:1V.

Construction of the mineralized material storage facility will include a multi-layer environmental protection system, two peripheral water collection ditches, and one collection ditch running from the facility towards the western mine portal. The multi-layer environmental protection includes the following features, from top to bottom:

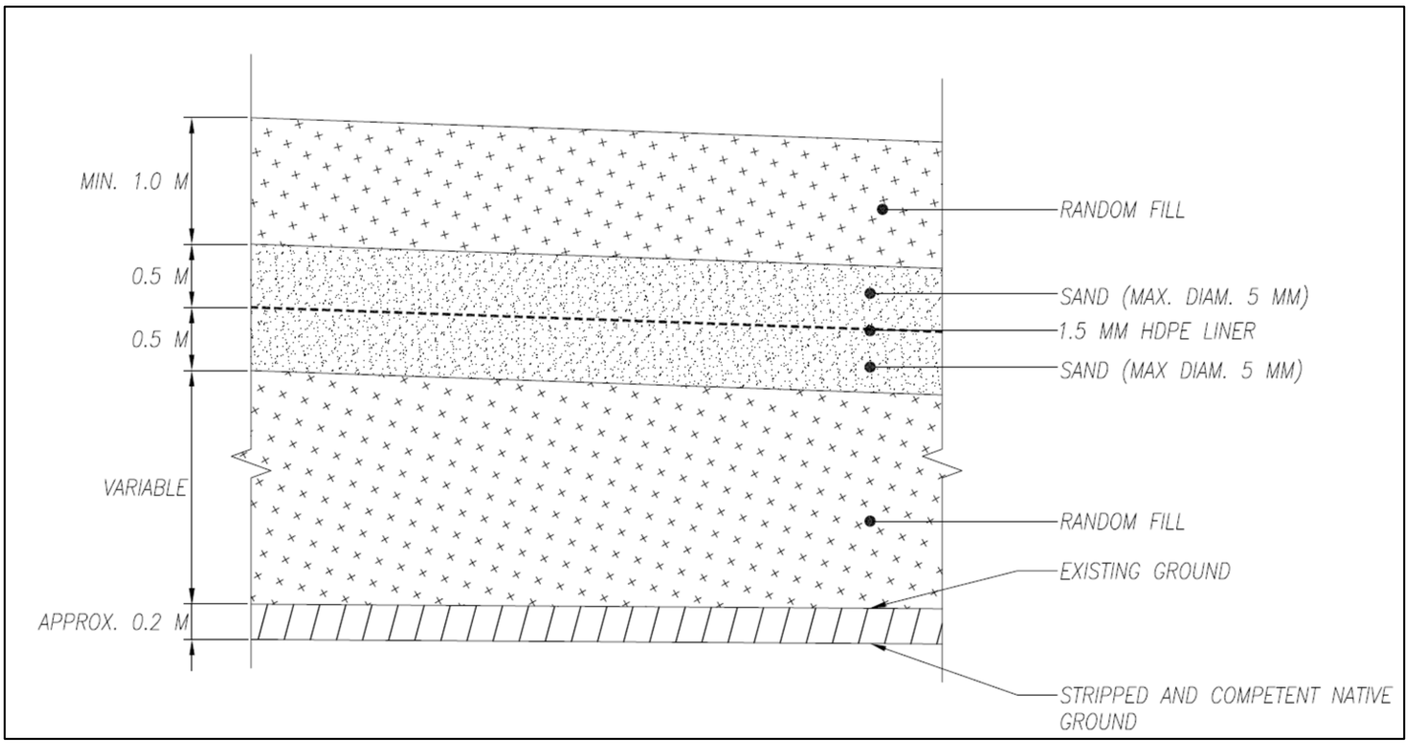
1. random fill layer (minimum thickness of 1.0 m), providing a uniform surface for material disposal and for trucking
2. upper sand layer (minimum thickness of 0.5 m), providing a protective cushion for the underlying geomembrane liner and a material transition with the overlying random fill layer
3. high-density polyethylene (HDPE) geomembrane liner, providing an impervious layer to collect and to direct the infiltration water
4. lower sand layer (minimum thickness of 0.5 m), providing a protective cushion for the overlying geomembrane liner and a material transition with the underlying random fill
5. random fill layer (approximately 0.5 m), providing competent foundation and allowing initial grading of the structure for water collection.

Details of the muti-layer environmental protection are provided in Figure 18-3.

Surface grading of the mineralized material storage facility will be designed to ensure a maintainable surface that is not subject to flooding or erosion. Runoff water will be collected either by the peripheral lined ditches or will flow at the surface until it reaches the connection point of the peripheral ditches and the ditch conveying water to the western mine portal. A minimum 1% downslope gradient toward the connection point will be implemented at the surface of the facility. Infiltration water collected by the impervious HDPE geomembrane liner will be directed towards the peripheral ditches and conveyed by gravity to a third collection ditch that will discharge the collected water in the underground mine. A safety berm around the top of the facility will be required to prevent vehicles from falling off the surface.

During construction of the mineralized material storage facility, peat and organic topsoil will be stripped over the footprint to ensure adequate surface preparation before placement of the multi-layer environmental protection. The topsoil material will be temporarily stored next to the footprint. It might be used later for site reclamation or will be stockpiled at a dedicated overburden storage facility.

Figure 18-3: Muti-Layer Environmental Protection Details of the Mineralized Material Storage Facility



Source: Norda Stelo (2025).

18.5 Hauling

The haul route between the O'Brien mine site and Westwood complex will be along Highway 117 for approximately 21 km. For the purposes of this PEA, it is assumed that access to Highway 117 will be through Rue du Petit-Canada. Further study, including community consultation, is recommended to assess the optimal haul road configuration including, if required, alternate access options to Highway 117.

18.6 Power and Electrical

Primary power to the O'Brien site will be provided by Hydro-Québec via a 120 kV high-voltage overhead transmission line. This existing 120 kV overhead line is located approximately 100 m south of the portal location. The line will be branched off from and travel perpendicular to the existing 120 kV line. The new 120 kV branch will travel north and terminate at the outdoor high-voltage substation located near O'Brien's underground mining portal.

The voltage will be stepped down from 120 kV to 4.16 kV at the substation, which will have utility metering. The substation will comprise one 7.5/10 MVA oil-filled, forced-air-cooled-type substation transformer to carry the maximum power required by the site. This includes minor future (5%) growth in demand.

Power factor correction equipment will be installed to improve the power factor to 0.9 or better at the point of interconnection with the utility.

The underground power distribution will operate at 4,160 kV, with provisions to step down the voltage to levels suitable for mining operations, dewatering, secondary ventilation, and other services. This setup ensures sufficient flexibility to meet the various electrical demands of the mine.

Backup power will be provided by a 4 MW, 4.16 kV diesel generator.

18.7 Fuel

To support operations, a refueling station will be installed at the surface level to accommodate haul trucks and fuel delivery vehicles. Over the life of the mine, total fuel consumption is estimated at 33.8 million litres (ML), corresponding to an average annual usage of approximately 2.8 ML. Peak consumption is expected in Year 7, reaching 3.9 ML.

Propane storage and distribution will also be required on site, primarily for heating the underground air intakes. Two rented tanks are planned at the mine site to supply the air heating system, with propane delivered by a local supplier. Over the life of the mine, total propane consumption is estimated at 32.5 ML, representing an average seasonal usage of approximately 2.7 ML per winter. The highest seasonal demand is expected in Year 4, with a total of 3.2 ML.

18.8 Process Plant, Tailings and Storage Facilities

Infrastructure related to the process plant is located at the Westwood complex. No additional infrastructure will be required.

Tailings will be stored at the Westwood complex tailings management facility (TMF) as part of a potential toll milling agreement. An assessment of the TMF was not conducted for this PEA. Under a toll milling agreement, tailings management would be the responsibility of the process plant owner.

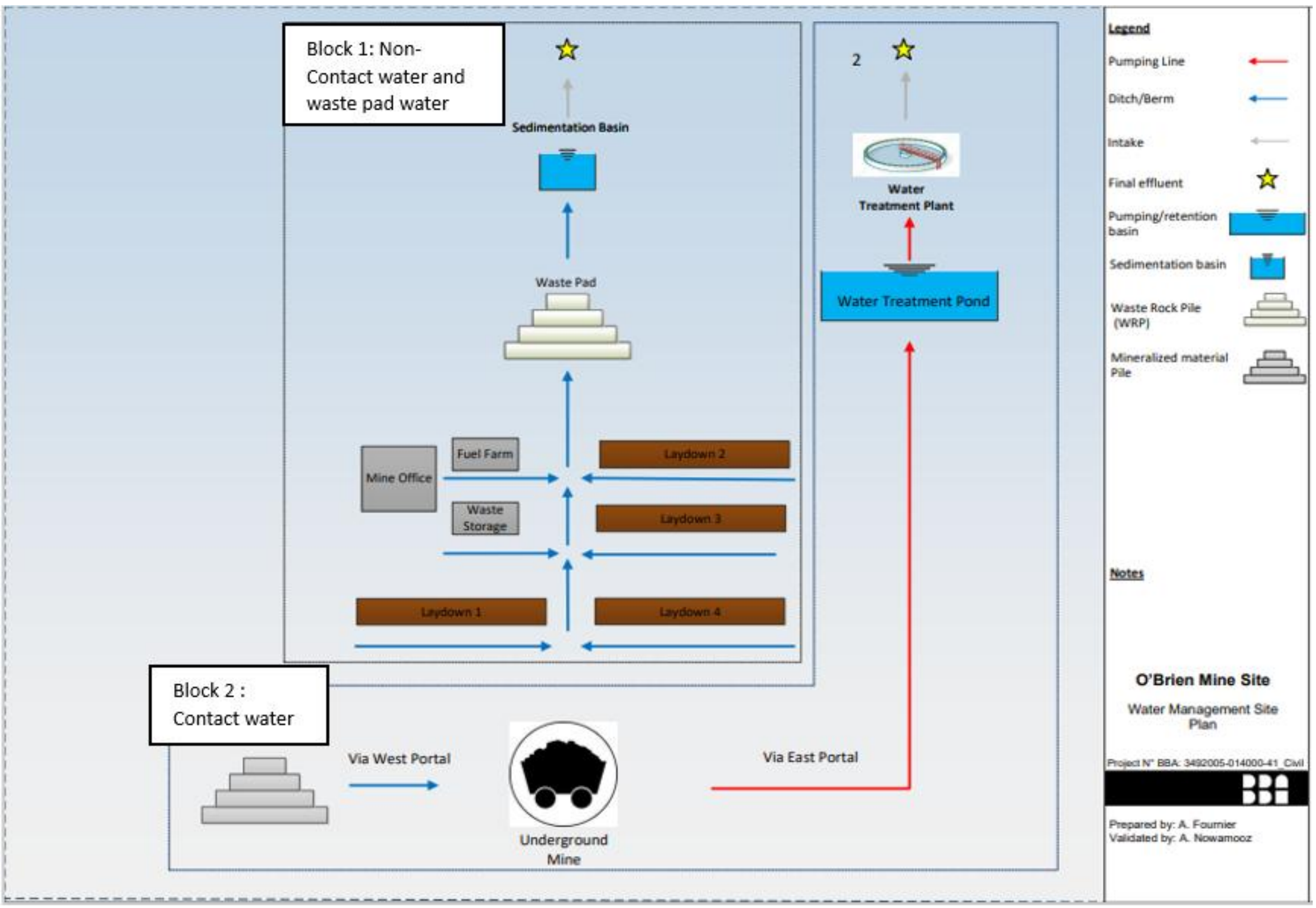
18.9 Water Supply and Management

This section outlines the water supply sources and water management strategy for the O'Brien mine site over the life of mine. It includes a description of water sources, estimated water requirements, associated infrastructure, and management strategies for both contact and non-contact water. The design complies with Directive 019 sur l'industrie minière and incorporates climate change considerations, as required.

18.9.1 Water Management Plan

The water management plan for the O'Brien mine site has been developed for the life of mine. A comprehensive water management plan is essential for both acid-generating and non-acid-generating structures to ensure the responsible treatment and discharge of water, in accordance with the Loi sur la qualité de l'environnement (LQE, c. Q-2). Figure 18-4 shows the surface water flow diagram for the O'Brien site.

Figure 18-4: Flux Diagram of the O'Brien Mine for the Life of Mine



The water management strategy is divided into two categories: contact water and non-contact water:

- Block 1 – Contact Water: The water from the acid-generating pile (mineralized material pad) will be drained into the underground mine from the West portal via a drainage ditch. The underground water will be pumped outside of the mine from the East portal to a retention or water treatment basin. This water will then be pumped to the treatment unit to receive proper treatment and neutralizing acidity. The treated water will be quality tested before being safely discharged into the environment, in full compliance with applicable environmental regulations.

- Block 2 – Non-Contact Water and Waste Pile: Non-contact water includes surface runoff from civil infrastructure (laydown areas, mine office, waste storage areas, and fuel storage), the waste pad, and the natural watershed. A network of drainage ditches will convey this runoff to a sedimentation basin located north of the site. After proper settling and quality verification, the water will be safely discharged into the environment, in full compliance with applicable environmental regulations.

18.9.2 Water Management Infrastructure

The O'Brien mine site will include one pad for mineralized material, one pad for waste, civil infrastructure (fuel farm, waste storage, laydown areas, and mine office), two portals (east and west) to access the underground mine, and an access road. Ditches will also be constructed around the infrastructure to drain surface water. The infrastructure required is summarized in Table 18-2. The proposed site arrangement is shown on the following page in Figure 18-5.

Table 18-2: Required infrastructure

Related Pads and Access Road	Area Managed	Basin Associated	Required Storage Capacity (m^3)	Associated Ditches
Waste Pad	Waste pad area	Sedimentation basin	9,018	Waste pad ditches Ditch 4 – 726 m Ditch 5– 112 m
Mineralized Material Pad	Mineralized material pad area	-	-	Mineralized material pad ditches Ditch 1 – 94 m Ditch 2 – 177 m Ditch 3 – 96 m
Access Road	Access road and civil infrastructure area	-	-	Access road ditches

The surface water management infrastructure was designed in accordance with the criteria from the Directive 019 sur l'industrie minière, as summarized in Table 18-3, and is based on regional climatological data (precipitation and snow water equivalent) from the Val-d'Or meteorological station. A 14% increase in precipitation was included to account for the projected impacts of climate change.

Figure 18-5: Surface Layout for the O'Brien Mine Site

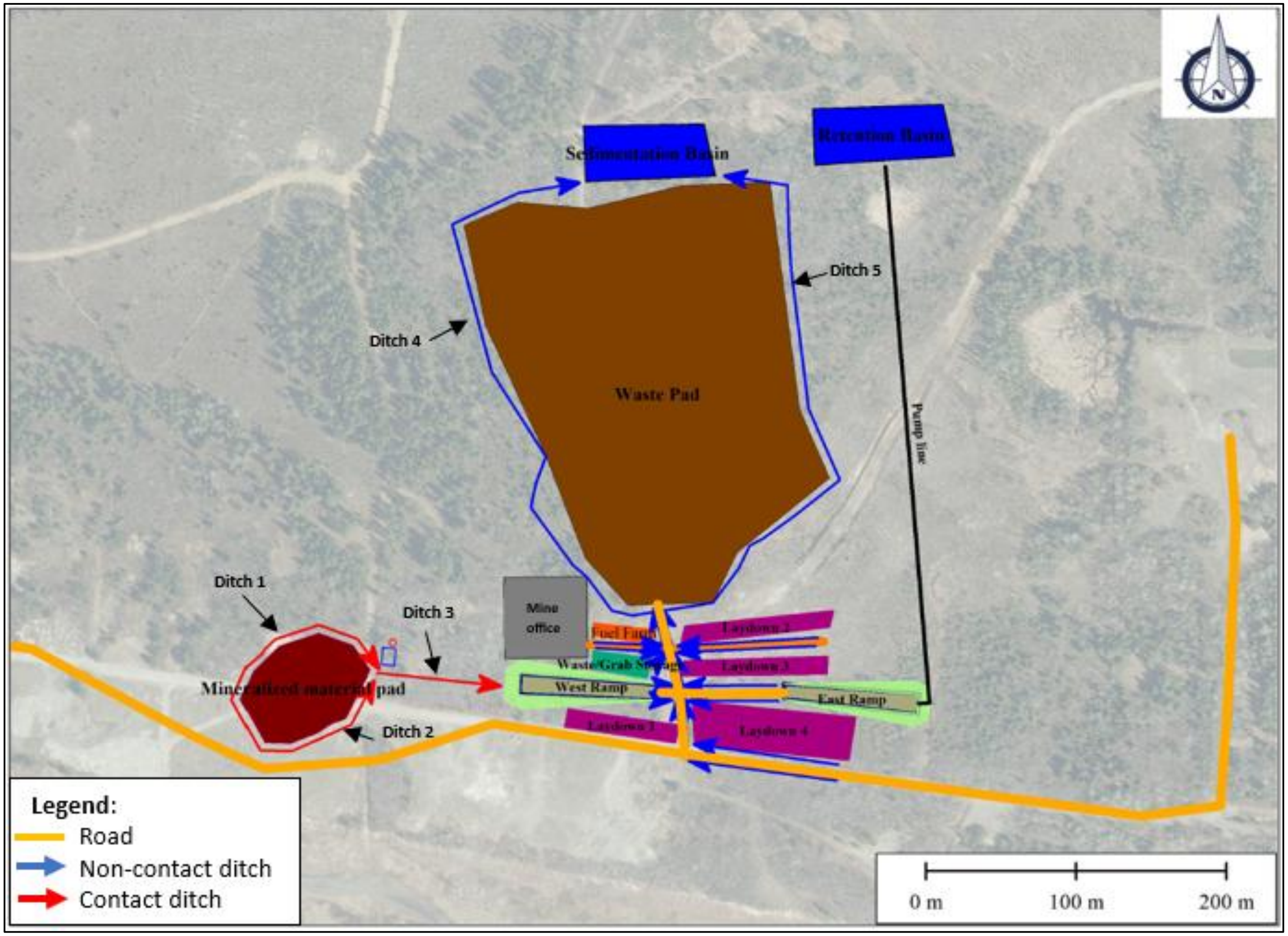


Table 18-3: Design Criteria for the Hydraulic Infrastructure

Infrastructure	Parameter	Criteria	Reference/Comment
Sedimentation Basin	Maximum capacity	24-hours rainfall with a 1000-year return period and 100-year snowmelt over 30 days	Directive 019 sur l'industrie minière (MDDEP, 2025)
	Dead volume for TSS accumulation, minimum slide from the bottom	0.5 m	Dredging could be done regularly
	Emergency spillway capacity	Probable maximum flood	Directive 019 sur l'industrie minière (MDDEP, 2025)
	Minimum freeboard	1 m	For safety and calculation uncertainties
Drainage Ditches	Hydraulic capacity: frequent flow (return period)	100-year rainfall and 100-year snowmelt over 30 days	Directive 019 sur l'industrie minière (MDDEP, 2025)
	Cross-section	Trapezoidal	-
	Minimum velocity	0.5 m/s	To avoid sedimentation and/or ice during winter
	Minimum base width	1 m	For maintenance
	Minimum depth	1 m	To take account of snow
	Maximum side slopes	2H:1V	In detailed design, flared sections with low slopes should be favoured where possible.
Retention Basin	Maximum storage capacity	Rainfall of 2,000 years/24-hour and 100-year snowmelt over 30 days	Directive 019 sur l'industrie minière (MDDEP, 2025) The mineralized material pad is considered leachable and potentially acid-generating (PAG)
	Freeboard	2.0 m	Sensitive component
Pumping System	Type of pipe	HDPE	N/A
	Maximum velocity in pressurized pipe	2.5 m/s	N/A

The water treatment unit and retention basin were also designed to accommodate a maximum dewatering rate from the underground mine of 118 m^3/h (approximately 520 USGPM).

The mineralized material pad is considered as leachable and potentially acid-generating (PAG). Therefore, any water originating from the mineralized material pad or underground mine is considered contact water and requires treatment. The waste pad is considered non-leachable and non-acid-generating. Any water coming from the waste pad, along with the rest of the mine infrastructure, is considered non-contact water and does not require treatment other than sedimentation.

18.9.3 Sedimentation Basin

The surface water runoff from the civil infrastructure (laydowns, waste storage, fuel farm, and mine storage) and waste pad will be drained to a sedimentation basin via drainage ditches.

Based on the non-acid-generating potential of the water going to the sedimentation basin, the maximum storage capacity includes a 100-year, 30-day snowmelt and a 1,000-year, 24-hour rainfall. The volume of water to be managed by the sedimentation basin are presented in Table 18-4. The maximum storage capacity of the sedimentation basin is estimated at 9,018 m^3, representing a surface area of 0.3 ha and a 3.0 m depth without considering the freeboard (1.0 m) and the dead volume for the sediment (0.5 m). Dredging should be done at least once per year to ensure available dead volume in the basin and maintain the storage volume as designed.

Based on a minimum diameter of ±10 µm for spherical suspended solids particles in the basin, the sedimentation rate is estimated at 5.5 x 10^{-5} m/s using Stokes law, resulting in a sedimentation time of 24 hours.

Table 18-4: Volume of Water to be Managed by the Sedimentation Basin

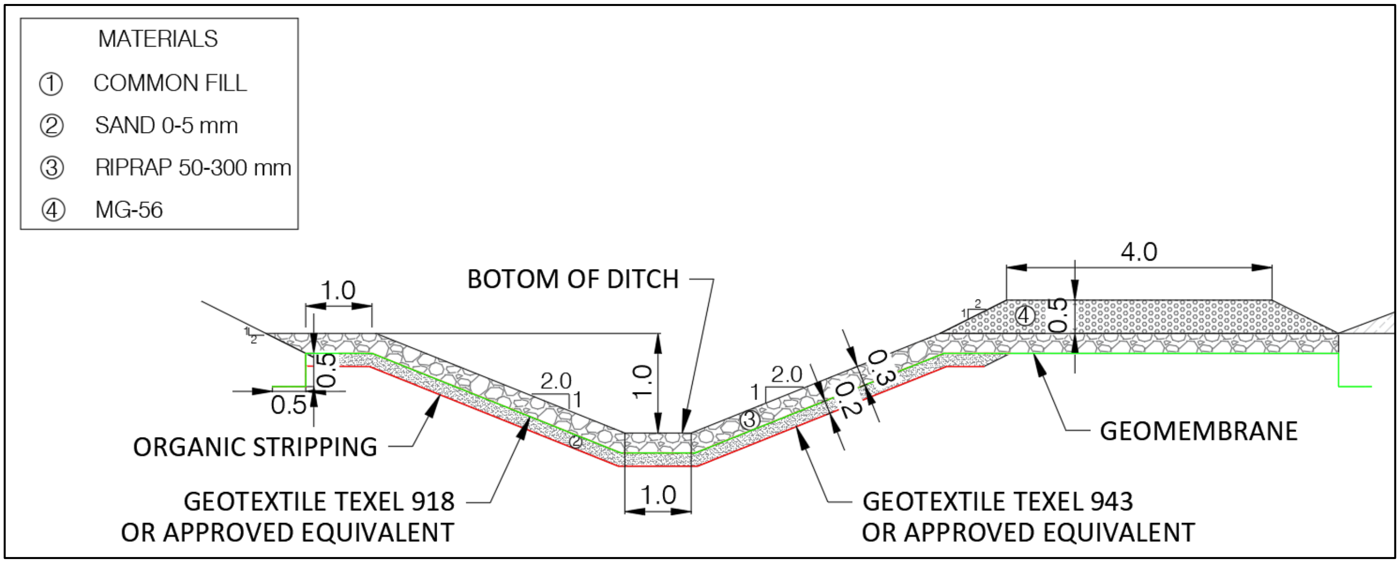
Infrastructure	Area (ha)	Runoff Coefficient	Snow & Rain (100-Year / 30-d) * (mm)	Precipitation (24-h/1000-Year) * (mm)	Volume of Snow (m^3)	Volume of Rain (m^3)
Substation	0.1	0.9	489.4	106.4	470	102
Mine Dry Office	0.3	0.9	489.4	106.4	1,332	290
Laydown 1	0.2	0.9	489.4	106.4	677	147
Laydown 2	0.2	0.9	489.4	106.4	677	147
Laydown 3	0.3	0.9	489.4	106.4	1,357	295
Laydown 4	0.1	0.9	489.4	106.4	339	74
Waste storage	0.1	0.9	489.4	106.4	237	52
Waste Pad	4.7	0.9	489.4	106.4	20,816	4,526
Fuel Farm	0.1	0.9	489.4	106.4	478	104
Natural Watershed	5.6	0.3	489.4	106.4	8,222	1,788
Sedimentation Basin	0.3	1.0	489.4	106.4	1,468	319
Total	**11.9**	**-**	**-**	**-**	**36,074**	**7,844**

Notes: *Including a 14% increase due to climate change.

18.9.4 Drainage Ditches

All ditches will be constructed following the typical cross-section presented in Figure 18-6.

Figure 18-6: Typical Ditch Cross-Section



Source: BBA (2024).

A geomembrane will be placed at the bottom of the drainage ditches of the mineralized material pad to prevent contamination through infiltration, while the rest of the ditches will not have the geomembrane.

The details of the ditches 1, 2, 3 (mineralized material pad) and 3 and 4 (waste pad), presented in Table 18-5, were designed based on a 100-year rainfall and 100-year snowmelt over 30 days including a 14% increase for the impact of climate change. The other ditches along the access road are discussed in earlier sections.

Table 18-5: Parameters of the Drainage Ditches

Pad Drained	Ditch ID	PK (m)		J	n	b	h	z	Q_p	D	V	R	Caliber	Riprap Thickness
		Start	End	(%)	(-)	(m)	(m)	(xH:1V)	(m^3/s)	(m	(m/s)	(m)	(mm)	(mm)
Mineralized Material Pad	1	0.0	94.0	3.3	0.035	1.0	1.0	2.0	0.04	0.05	0.69	0.95	0-200	300
	2	0.0	177.0	1.0	0.035	1.0	1.0	2.0	0.09	0.12	0.62	0.88	0-200	300
	3	0.0	96.0	2.3	0.035	1.0	1.0	2.0	0.12	0.11	0.90	0.89	0-200	300
Waste Pad	4	0.0	726.0	5.0	0.035	1.0	1.0	2.0	2.08	0.45	2.64	0.55	200-300	500
	5	0.0	112.0	1.0	0.035	1.0	1.0	2.0	0.27	0.23	0.87	0.77	0-200	300

Notes: J: Slope of the ditch. n: Manning's coefficient. b: Base width of the ditch. h: Depth of the ditch. z: Side slopes of the ditch. Qp: Peak flowrate in the ditch. D: Depth of water in the ditch. V: Average velocity in the ditch. R: Freeboard.

18.9.5 Retention or Water Treatment Basin

Water collected from the acid-generating and leachable pile (mineralized material pad) will be transferred to the underground mine via the east portal. Water from the underground mine will then be pumped outside to the treatment basin for proper treatment, before being released into the environment.

The volume to be treated in the water treatment unit includes both the 24-hour dewatering rate (118 m^3/h) and the 100-year, 30-day snowmelt over the retention basin surface, treated at an average rate of 2.0 m^3/h. The details of the treatment plan are discussed in Section 18.10.6. The capacity of the retention basin is calculated based on a 24-hour holding period for the dewatering water from the underground mine, as well as the excess water that is generated during peak snowmelt. The volume of water to be managed by the retention basin is presented in Table 18-6. The total storage volume of the basin is 3,200 m^3, corresponding to a depth of 1.1 m without freeboard (1.0 m) over a surface area of 0.3 ha.

Table 18-6: Volume of Water to be Managed by the Retention Basin

Infrastructure	Area (ha)	Runoff Coefficient (-)	Snow and Rain (100 years / 30 d) * mm)	Precipitation (24 h / 2000 year) * (mm)	Volume of Snow (m^3)	Volume of Rain (m^3)	Dewatering Rate (m^3/d)
Retention basin	0.3	1.0	489.4	112.6	338	1 468	2,832

Notes: *Including a 14% increase due to climate change.

18.9.6 Water Treatment Plant

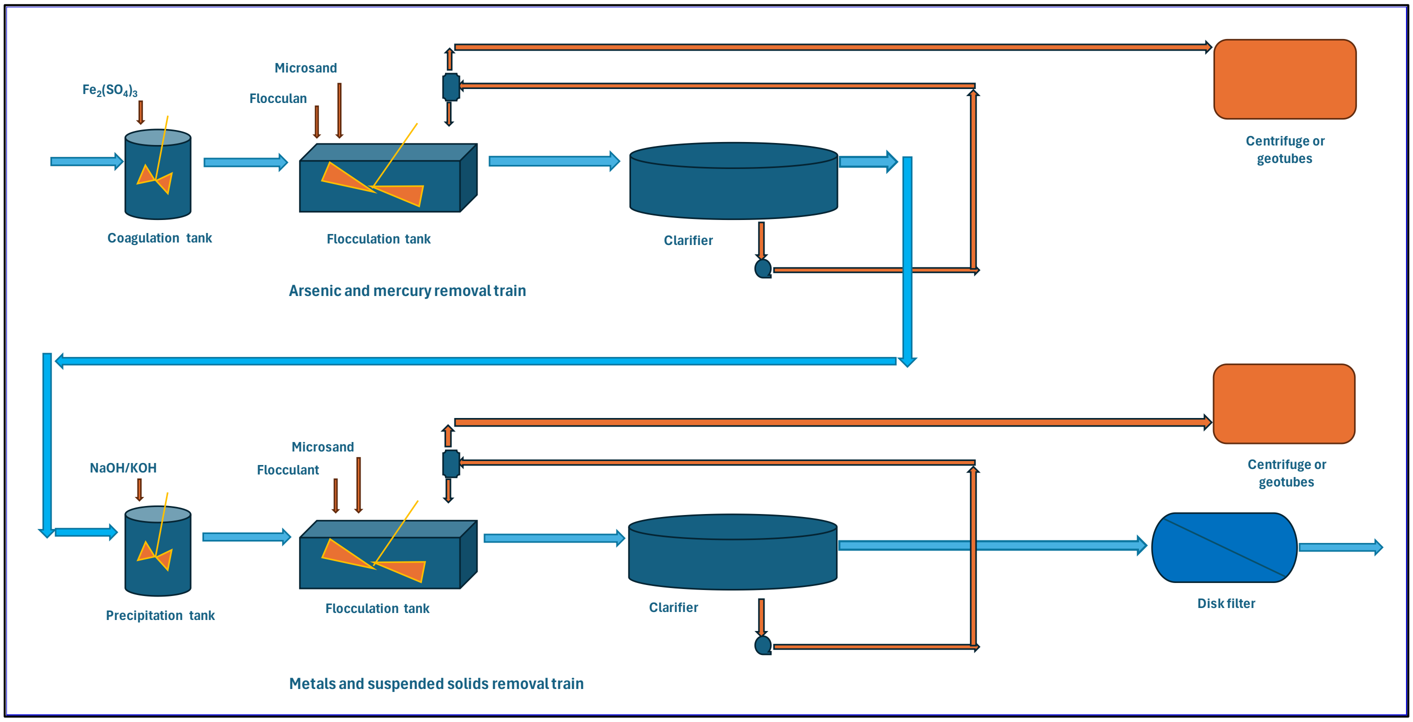
Based on the review of the groundwater analyses, dissolved metals such as arsenic, copper, cadmium, mercury, and zinc were the parameters of concern. The water treatment system proposed by ASDR for the bulk sample was specifically aimed at arsenic removal, but also addressed total suspended solids (TSS) and pH adjustment.

Based on these observations, two water treatment technologies are recommended to be used in series. The first consists of arsenic and mercury removal, while the second is for the removal of dissolved metals and TSS, as well as pH adjustment. The water treatment plant was designed based on the 120 m^3/h treatment rate presented above.

The schematics of the treatment flow recommended are presented in Figure 18-7. The treatment technology proposed for the two water treatment plants is a ballasted sand flocculation process (e.g., RAPISAND® or ACTIFLO®) similar to the technology proposed in ASDR proposal. The main equipment for this technology is as follows:

- a reactor tank where a coagulant is added
- a second reactor tank where polymer and sand are added
- a clarifier tank where flocculated particles are settled out and parts are recycled back to polymer reactor tank.

Figure 18-7: Water Treatment Plant Schematics



Source: BBA (2025).

The first treatment technology will use ferric sulphate as a coagulant for arsenic and mercury removal. The wasted sludge is dewatered using a centrifuge or geotubes. The dewatered sludge is disposed of in an authorized site. The treated water from the first treatment technology becomes the inlet of the second treatment technology.

The second treatment technology will use sodium hydroxide and/or potassium hydroxide to raise the pH and precipitate metals. After the flocculated particles are settled, the clarified water is sent to a disk filter to polish the water before discharge to the environment; waste sludge is sent again to a centrifuge or geotubes for dewatering. The dewatered sludge is disposed of in an authorized site.

A constant treatment flow of 120 m^3/h was estimated based on the dewatering rate of the underground mine and the treatment of snow during spring melt (1 in 100-year recurrence and a melt of over 30 days). Assuming the treatment occurs at a rate of 120 m^3/h, 24 hours per day, 365 days each year, the total volume of water to be treated will be approximately 1,051,200 m^3/a. The estimated operating costs will range from $473,040 to $525,600 per year.

The total capital cost estimate (direct and indirect costs) for the water treatment plant is $17,027,940.

18.10 Comments on Project Infrastructure

The mine site location does not have existing infrastructure, but is located approximately 500 m east of previous mining infrastructure. Warehousing, offices, the core shack, and storage areas are expected to be reused to support the mine.

As it is anticipated a process plant will not be constructed for the project, infrastructure requirements are limited to mine-related items, which reduces overall construction requirements compared to a mine complex with mining, process plant, and tailings infrastructure.

19 MARKET STUDIES AND CONTRACTS

This PEA establishes criteria for the development of O'Brien based on processing and tailings management at an existing off-site facility, namely IAMGOLD's Westwood complex, under a toll milling arrangement. A milling assessment for the processing of O'Brien mined material at the Westwood complex was conducted under the auspices of the September 2024 MOU between Radisson and IAMGOLD. The MOU is non-binding and non-exclusive and contains no specific terms around potential commercial arrangements between the parties. IAMGOLD has not independently confirmed the processing assumptions, metallurgical results and/or cost assumptions assumed in this study.

It was assumed in this PEA that the O'Brien Gold Project will produce gold in the form of doré bars. The market for doré is well-established and accessible to new producers. The doré bars will be refined in a certified North American refinery—of which there are many in the eastern United States and Canada—and the gold will be sold on the spot market.

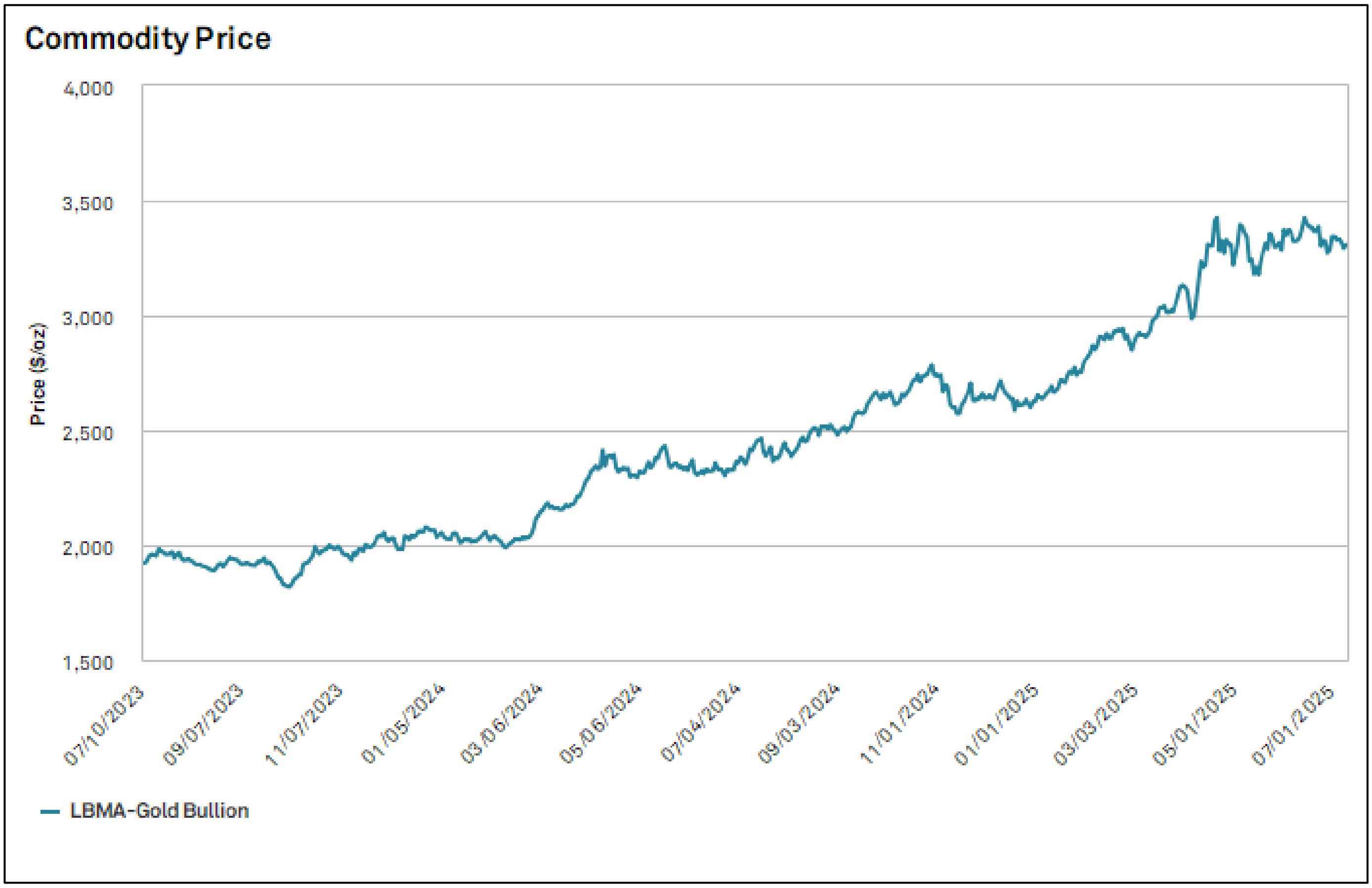
19.1 Market Studies

No market studies have been conducted by Radisson Mining or its consultants on the gold doré that will be produced. Gold is a freely traded commodity on the world market for which there is a steady demand from numerous buyers. Gold production is expected to be sold on the spot market. Terms and conditions included as part of the sales contracts are expected to be typical for this commodity. Gold is bought and sold on many markets in the world, and it is not difficult to obtain a market price at any time. The gold market is very liquid with many buyers and sellers active at any given time.

19.2 Commodities Price and Refining Assumptions

As of July 10, 2025, the trailing two-year gold price was US$2,466/oz, and the trailing three-year gold price was US$2,253/oz (Figure 19-1). For this report, a gold price of US$2,550/oz was assumed. The exchange rate used in the study is 1.00:0.73 (CAD:USD). The assumptions used for the estimation of the US$2,550/oz are considered reasonable based on current spot prices and historical trending.

Figure 19-1: Two-Year Gold Price Chart



Source: S&P Cap IQ, July 10th, 2025

20 ENVIRONMENTAL STUDIES, PERMITTING, & SOCIAL OR COMMUNITY IMPACT

20.1 Environmental Considerations

This section describes the main components of the current physical, biological, and human environments in the project area. Data in this section is based on existing inventories and public databases. Radisson commissioned environmental baseline studies using the services of SNC Lavalin in 2019.

The content of this chapter considers that processing the mineralized material for the O'Brien Gold Project will occur off site. Consequently, no tailings or processing-related infrastructure will be present at the project site.

Further studies will be undertaken to provide the level of information required for the environmental and social impact assessment.

20.1.1 Baseline and Supporting Studies

20.1.1.1 Air Quality and Greenhouse Gas Emissions

The presence of the LaRonde Mine complex, about 7 km west of the property, is likely to affect the air quality in the area.

The air quality monitoring station located in Rouyn-Noranda is part of the Réseau de surveillance de la qualité de l'air du Québec, which monitors the ambient air quality across the province. Parameters monitored at the Rouyn-Noranda station include sulphur dioxide (SO_2), ozone (O_3), and particulate matter ($PM_{2.5}$).

Between 2006 and 2024, the following results were observed at the Rouyn-Noranda - Parc Tremblay station:

- Annual mean for SO_2 varied between 2.6 and 1.2 ppb.
- Annual mean for O_3 varied between 25.9 and 29.0 ppb.
- Annual mean for $PM_{2.5}$ varied between 6.0 and 11.1 µg/m_3.

The concentrations of air pollutants in Rouyn-Noranda may vary from those at the Radisson property.

An ambient air quality monitoring program is ongoing to collect data on background concentrations of certain contaminants that will be generated by the future activities of the mine. This data will also support the air quality modelling study required for the environmental and social impact assessment.

Greenhouse gas (GHG) emissions have been evaluated for the mining operation phase of the project based on data inputs used in the operating cost financial model. The objective of this evaluation is to provide a high-level estimate of the project's GHG emission footprint. Data inputs used included fossil fuel consumption (diesel and propane) by mobile equipment and vehicles as well as fixed heating equipment for the ventilation system (natural gas). The Quebec Guide de quantification des émissions de gaz à effet de serre was used to identify emission factors for diesel/propane driven light and heavy vehicles as well as for natural gas fired heating equipment. Annual volumes of fossil fuels consumed were determined based on typical usage frequency of mobile/fixed equipment for a mining operation such as the O'Brien Gold Project. Annual GHG emissions were estimated to be 16,300 tonnes of CO_2 equivalent per year. Radisson is engaged in limiting its carbon footprint by implementing industry-recognized environmental management practices.

20.1.1.2 Ambient Noise Level

The nearest residence is located approximately 400 m from site in an area designated as rural, where permitted uses include mobile or single-unit homes, as well as natural resource activities, including land and subsurface exploitation.

For the environmental and social impact assessment, a noise modelling survey will be carried out since the project is in an area with numerous human activities.

20.1.1.3 Geology

The bedrock in the project area lies at the boundary between the Pontiac and Abitibi subprovinces in the Superior geological province. The project straddles the Larder Lake-Cadillac Fault Zone (or Cadillac Fault). The property straddles the basalt volcanics of the Piché Group, which separate the metasedimentary rocks (wacke, mudrock and shale) of the Pontiac Group to the south from the metasedimentary rocks (volcaniclastic sandstone) of the Cadillac Group to the north.

20.1.1.4 Hydrology

The mine site is located in the Blake River watershed, which covers an area of 102.2 km^2. The river flows in a northeast direction, with its ultimate outlet being Baie à Cormier in Preissac Lake. The natural drainage on site flows south to north, with the road delineating the sub-watershed.

20.1.1.5 Protected Areas

There are no protected areas within the O'Brien property. The closest to site are two heronries (wildlife habitats) located on two islands in Preissac Lake. There are also two biological refuges approximately 9 km north and southwest of the project area.

20.1.1.6 Vegetation

The project is located south of the boreal zone, which extends between the 48th and 58th northern latitudes in Quebec, and is dominated by boreal coniferous species. It is part of the balsam fir – white birch bioclimatic domain, Western subdomain. In the southernmost area of the boreal zone, several temperate species reach their northern limit of distribution, such as yellow birch (*Betula alleghaniensis*), red maple (Acer rubrum), eastern white pine (*Pinus strobus*), and red pine (*Pinus resinosa*).

The most representative forests of the area, the balsam fir – paper birch stand, are composed of balsam fir (*Abies balsamea*), paper birch (*Betula papyrifera*), and white spruce (*Picea glauca*). However, a large part of the area's landscapes is made of young forests resulting from logging or fires, including many stands dominated by paper birch or trembling aspen (*Populus tremuloides*), as well as jack pine (*Pinus banksiana*) forests. Black spruce (*Picea mariana*) forests, often with balsam fir, are also common throughout the area.

20.1.1.6.1 Wetlands

Twenty-nine wetlands in the project area were characterized during the field surveys, covering a total area of 90.3 ha. Four main classes of wetlands were observed, namely the treed peatland, treed swamp, shrub swamp, and marsh. Shrub swamps, mainly dominated by speckled alder, were the more abundant and represented 63% (57.1 ha) of all the wetlands. Treed swamps occupied an area of 13.7 ha and were often dominated by black spruce or speckled alder (*Alnus incana* subsp. *rugosa*). Treed peatland covered an area of 7.3 ha and were mainly dominated by black spruce and tamarack (*Larix laricina*). Marshes covered an area of 12.2 ha, mainly dominated by bluejoint reedgrass (*Calamagrostis canadensis*), bulrushes (*Scirpus sp.*), and sedges (*Carex sp.*).

Overall, most wetlands had an ecological value rated at medium due to their low specific richness, small area, or disturbances present (partial cutting, line right-of-way, beaver activity, etc.). Several small, isolated wetlands were also judged to have a low ecological value due to their small area, the absence of surface hydrological connectivity, and the (sometimes significant) disturbances observed. No wetlands in the project area presented a high ecological value.

20.1.1.6.2 Species at Risk

According to the *Centre de données sur le patrimoine naturel du Québec* (CDPNQ) interactive map, there are no reported occurrences of species at risk within a radius of 15 km from the project area. In addition, no potential habitat was identified and no species at risk was observed in the wetlands characterized in 2019.

20.1.1.7 Fauna

20.1.1.7.1 Large Mammals

The project area is in hunting zone 13, which corresponds to the entire administrative region of Abitibi-Témiscamingue. The large mammal species found in this area are the white-tailed deer, which is relatively unabundant because it is at the northern limit of its range, as well as the moose and the American black bear, which are very common and abundant. According to hunting and trapping harvest data compiled by the MELCCFP, 11 white-tailed deer were harvested in hunting zone 13 in 2023, compared to 2,593 moose and 572 American black bears.

During the field surveys carried out, indications of the presence of American black bear and moose were noted.

20.1.1.7.2 Small Mammals and Micromammals

Indications of the presence (observations of individuals, tracks, or faeces) of snowshoe hare, woodchuck, red squirrel, and eastern chipmunk were noted during the field surveys.

A specific inventory for micromammals was carried out in August 2019 in the project area to check for the presence of rock vole and southern bog lemming. Fifty-nine micromammals of at least nine species were captured. By far the main species captured was the meadow vole, with 32 captures. The other species captured were, in increasing order of abundance, meadow jumping mouse, northern short-tailed shrew, smoky shrew, common shrew, star-nosed mole, deer mouse, American pygmy shrew, and southern red-backed vole. No species with special status was captured. The presence of the southern bog lemming in the project area, especially since its preferential habitats are present, cannot be excluded, but the presence of the rock vole is much less likely due to the absence of preferential habitats in the project area.

20.1.1.7.3 Bats

According to the geographic distributions of bat species in Quebec and the results of bat inventories of 2019, the project area is potentially frequented by six of the eight species of bats present in Quebec. Bats can migrate thousands of kilometres to their hibernation site, typically caves or abandoned mines, where the environment remains at above-freezing temperatures and humid. They hibernate approximately from October to April.

The results of the acoustic inventory carried out in the project area confirmed that five of the six bat species were potentially present.

Only the presence of the northern myotis has not been confirmed acoustically. Undetermined records of bats in the genus *Myotis* were noted, that could potentially be the northern myotis or the little brown myotis. However, the few signals collected during this inventory as well as the overlap of echolocation signals between these two species did not make it possible to validate the presence of northern long-eared bats beyond reasonable doubt.

20.1.1.7.4 Amphibians and Reptiles

The American toad and the wood frog were observed during field inventories, although no specific surveys targeting amphibians were conducted.

Three reptile species have a special status either only in Quebec, or in both Quebec and Canada: the northern ring-necked snake, the wood tortoise, and the smooth green snake. The snapping turtle is a species of special concern in Canada but is very common and abundant in the province of Quebec.

A turtle inventory and a snake inventory were carried out in the project area to verify the presence of species at risk. During these surveys, only one snapping turtle was observed on the banks along the Blake River, and three specimens of common garter snakes were seen.

20.1.1.7.5 Birds

Surveys for songbirds and nightjars were carried out in June 2019 in the project area. In total, 85 species of birds have been recorded and the nesting was qualified as possible, probable, or confirmed for 83 of them, two species at risk: the common nighthawk and the Canada warbler.

20.1.1.7.6 Fish and Fish Habitat

The biophysical characterization of the watercourses (potential habitat for fish) and the inventory of the fish fauna show that generally, the watercourses in the project area have low habitat potential for fish, particularly due to their biophysical characteristics. However, the numerous beaver ponds along the watercourses provide good quality feeding and nursery habitats for fish. Five species were captured during fish inventories in the project area: the northern pearl dace (*Margariscus margarita*), the mottled sculpin, the brook stickleback, the white sucker, and the northern redbelly dace (*Chrosomus eos*). None of these species have protected status.

The restriction period for work in fish habitat would generally extend from April 15 to June 15 regarding the species present in the watercourses of the project area. To minimize the impact of mining operations on these aquatic environments, mitigation measures and an appropriate ecological monitoring will be implemented to preserve sensitive habitats.

20.1.1.7.7 Species at Risk

The Government of Quebec is committed to protecting genetic biodiversity as reflected in the *Act Respecting Threatened or Vulnerable Species* (chapter E-12.01). The regulation respecting threatened or vulnerable wildlife species and their habitats (chapter E-12.01, r. 2) identifies wildlife species that are legally protected. Species at risk also include species that receive legal protection under Schedule 1 of Canada's *Species at Risk Act*.

In the project area, eight mammal wildlife species, three reptile species and 12 bird species at risk have been observed or are likely to frequent the territory (Table 20-1).

According to the CDPNQ Interactive map, there are three known occurrences of bank swallow and one occurrence of smooth greensnake within a 15 km radius of the study area.

The occurrence of the woodland caribou—woodland ecotype (population No. 14 – Val-d'Or)—is also included inside the 15 km radius, south of the study area. This small population of a few individuals is subject to monitoring and protection measures by the Government of Quebec, so caribou are very unlikely to be present in the study area.

Table 20-1: Fauna Species at Risk Susceptible to be Found within the Study Area

Latin Name	Common Name	Status in Quebec	Status in Canada (Schedule 1)
Mammals			
Mustela nivalis	Least weasel	Likely to be designated	None
Microtus chrotorrhinus	Rock vole	Likely to be designated	None
Synaptomys cooperi	Southern bog lemming	Likely to be designated	None
Aeorestes cinereus	Hoary bat	Likely to be designated	Endangered (COSEWIC), not on Schedule 1
Lasionycteris noctivagans	Silver-haired bat	Likely to be designated	Endangered (COSEWIC), not on Schedule 1
Lasiurus borealis	Eastern red bat	Vulnerable	Endangered (COSEWIC), not on Schedule 1
Myotis lucifugus	Little brown myotis	Threatened	Endangered
Myotis septentrionalis	Northern myotis	Threatened	Endangered
Reptiles			
Diadophis punctatus edwardsii	Northern ring-necked snake	Likely to be designated	None
Glyptemys insculpta	Wood tortoise	Vulnerable	Threatened
Opheodrys vernalis	Smooth greensnake	Likely to be designated	None
Birds			
Antrostomus vociferus	Eastern Whip-poor-will	Vulnerable	Threatened
Asio flammeus	Short-eared Owl	Likely to be designated	Threatened
Cardellina canadensis	Canada Warbler	Likely to be designated	Threatened
Chaetura pelagica	Chimney Swift	Threatened	Threatened
Chordeiles minor	Common Nighthawk	Likely to be designated	Special concern
Contopus cooperi	Olive-sided Flycatcher	Vulnerable	Threatened
Coturnicops noveboracensis	Yellow Rail	Threatened	Special concern
Dolichonyx oryzivorus	Bobolink	Vulnerable	Special concern (COSEWIC), not on Schedule 1
Euphagus carolinus	Rusty Blackbird	Likely to be designated	Special concern
Falco peregrinus anatum	Peregrine Falcon	Vulnerable	Not at risk
Haliaeetus leucocephalus	Bald Eagle	Vulnerable	None
Riparia	Bank Swallow	Candidate[1]	Threatened

[1] Candidate species : species monitored by the CDPNQ whose situation seems worrying, but which is not designated as threatened or vulnerable and which does not appear in the list of species likely to be designated as threatened or vulnerable published in the *Gazette officielle du Québec*. Candidate species include most species designated federally under the *Species at Risk Act* which do not have official status in Quebec.

20.2 Permitting Considerations

20.2.1 Provincial Procedure

The project is subject to Quebec's *Environment Quality Act* ("EQA," c. Q-2), which states that all mining developments, including the additions to, and alterations or modifications of, existing mining developments are automatically subject to assessment and review. The O'Brien Gold Project will therefore be subject to the same provincial process.

20.2.2 Federal Procedure

From the information BBA has collected on the project, it should not be subject to the federal impact assessment examination procedure (*Impact Assessment Act* (S.C. 2019, c. 28, s. 1), Physical Activities Regulations (SOR/2019-285). However, the project will be subject to the Metal and Diamond Mining Effluent Regulations (*Fisheries Act*).

20.2.3 Environmental Permits

Table 20-2 presents the most significant acts, regulations, directives, and guidelines that apply to the project. This list is not exhaustive and is based on information known to date. The applicability of the permits listed will have to be reviewed as the project becomes further defined.

Table 20-2: Environmental Permits – Acts and Regulations

Level of Govern-ment	Acts and Regulations
Provincial	***Environment Quality Act* (c. Q-2)**
	Regulation Respecting Environmental Impact Assessment and Review (Q-2, r. 23).
	Regulation respecting the regulatory scheme applying to activities based on their environmental impact (Q-2, r. 17.1)
	Design code of a storm water management system eligible for a declaration of compliance (Q-2, r. 9.01)
	Clean Air Regulation (Q-2, r. 4.1)
	Regulation respecting the operation of industrial establishments (Q-2, r. 26.1)
	Snow, road salt and abrasives management regulation (Q-2, r. 28.2)
	Regulation respecting used tire storage (Q-2, r. 20)
	Regulation respecting the declaration of water withdrawals (Q-2, r. 14)
	Regulation respecting mandatory reporting of certain emissions of contaminants into the atmosphere (Q-2, r. 15)
	Regulation respecting halocarbons (Q-2, r. 29)

Level of Govern-ment	Acts and Regulations
	Regulation respecting hazardous materials (Q-2, r. 32)
	Regulation respecting the reclamation of residual materials (Q-2, r. 49)
	Regulation respecting activities in wetlands, bodies of water and sensitive areas (Q-2, r. 0.1)
	Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1)
	Protection policy for lakeshores, riverbanks, littoral zones, and floodplains (Q-2, r. 35)
	Water withdrawal and protection regulation (Q-2, r. 35.2)
	Land protection and rehabilitation regulation (Q-2, r. 37)
	Regulation respecting the quality of the atmosphere (Q-2, r. 38)
	Regulation respecting the charges payable for the use of water (Q-2, r. 42.1)
	Directive 019 sur l'industrie minière (2025)
	Protection and rehabilitation of contaminated sites policy (1998)
	***Mining Act* (c. M-13.1)**
	Mining regulation (M-13.1, r. 2)
	***Threatened or Vulnerable Species Act* (c. E-12.01)**
	Regulation respecting threatened or vulnerable wildlife species and their habitats (E-12.01, r. 2)
	Regulation respecting threatened or vulnerable plant species and their habitats (E-12.01, r. 3)
	Compensation Measures for the Carrying out of Projects Affecting Wetlands or Bodies of Water Act (M-11.4)
	***Act respecting the conservation of wetlands and bodies of water* (2017, chapter 14; Bill 132)**
	***Watercourses Act* (c. R-13)**
	Regulation respecting the water property in the domain of the State (R-13, r. 1)
	***Conservation and Development of Wildlife Act* (c. C-61.1)**
	Regulation respecting wildlife habitats (C-61.1, r. 18)
	***Act respecting the lands in the domain of the state* (chapter T-8.1)**
	Regulation respecting the sale, lease and granting of immovable rights on lands in the domain of the State (chapter T-8.1, r. 7)
	***Sustainable Forest Development Act* (chapter A-18.1)**
	Regulation respecting the sustainable development of forests in the domain of the State (chapter A-18.1, r. 0.01)
	Regulation respecting forestry permits (chapter A-18.1, r. 8.)
	***Building Act* (c. B-1.1)**

Level of Govern-ment	Acts and Regulations
	Safety Code (B-1.1, r. 3)
	Construction Code (B-1.1, r. 2)
	Cultural Heritage Act **(c. P-9.002)**
	Occupational Health and Safety Act **(c. S-2.1)**
	Regulation respecting occupational health and safety in mines (S-2.1, r. 14)
	Explosives Act **(c. E-22)**
	Regulation under the *Act respecting explosives* (E-22, r. 1)
	Highway Safety Code (c. C-24.2)
	Transportation of Dangerous Substances Regulation (C-24.2, r. 43)
Federal	***Fisheries Act (R.S.C., 1985, c. F-14)***
	Authorizations Concerning Fish and Fish Habitat Protection Regulations (SOR/2019-286);
	Metal Mining Effluent Regulations (SOR/2002-222)
	Canadian Environmental Protection Act (S.C. 1999, c. 33)
	PCB Regulations (SOR/2008-273)
	Environmental Emergency Regulations, 2019 (SOR/2019-51)
	Federal Halocarbon Regulations (SOR/2003-289)
	National Pollutant Release Inventory
	Canadian Wildlife Act (R.S.C., 1985, c. W-9)
	Wildlife Area Regulations (C.R.C., c. 1609)
	Migratory Birds Convention Act, 1994 (S.C. 1994, c. 22)
	Migratory Birds Regulations (C.R.C., c. 1035)
	Species at Risk Act (S.C. 2002, c. 29)
	Nuclear Safety and Control Act (S.C., 1997, c. 9)
	General Nuclear Safety and Control Regulations (SOR/2000-202)
	Nuclear Substances and Radiation Devices Regulations (SOR/2000-207)
	Hazardous Products Act (R.S.C., 1985, c. H-3)
	Explosives Act (R.S.C., 1985, c. E-17)
	Transportation of Dangerous Goods Act (1992)
	Transportation of Dangerous Goods Regulations (SOR/2001-286)

20.2.4 Mining Permits

Table 20-3 presents a non-exhaustive list of required approvals, authorizations, permits, or licenses based on the known components of the project and typical activities related to mining projects.

Table 20-3: Authorities Related to Mining Activities

Activities	Type of Request	Authority
Closure plan	Approval	MRNF[1]
Mining operations	Lease	MRNF
Mine waste rock dump location	Approval	MRNF
Infrastructure implantation on public land	Lease	MRNF
Construction and operation of an industrial establishment, the use of an industrial process, and an increase in the production of property or services	Authorization	MELCCFP[2]
Withdrawal of water, including related work and works	Authorization	MELCCFP
Establishment of potable, wastewater and mine water management and treatment facilities	Authorization	MELCCFP
Work, structures or other interventions carried out in wetlands and bodies of water	Authorization	MELCCFP
Installation and operation of any other apparatus or equipment designed to treat water to prevent, abate, or stop the release of contaminants into the environment	Authorization	MELCCFP
Installation and operation of an apparatus or equipment designed to prevent, abate, or stop the release of contaminants into the atmosphere	Authorization	MELCCFP
Carry out an activity likely to modify a wildlife habitat	Authorization	MELCCFP
Harvest wood on public land where a mining right is exercised	Authorization	MRNF
Build or improve a multi-use road	Authorization	MRNF
Use of high-risk petroleum equipment	Permits	RBQ[3]
Construction	Permits	RCM[4]
Explosives possession, magazine, and transportation	Permit	SQ[5]
Explosives transportation	Permit	NRCan[6]
Notice and Environmental Emergency Plan	-	ECCC[7]

Notes: [1]Ministère des Ressources naturelles et des Forêts. [2]Ministère de l'Environnement, de la lutte contre les changements climatiques, de la Faune et des Parcs. [3]Régie du Bâtiment du Québec. [4]Regional County Municipality. [5]Sureté du Québec. [6]Natural Resources Canada. [7]Environment and Climate Change Canada.

20.3 Social Considerations

20.3.1 Administrative Framework

The project is located in the northwest area of the Abitibi-Témiscamingue Administrative Region.

20.3.1.1 Rouyn-Noranda

The City of Rouyn-Noranda is the only municipality in the region that functions both as a municipality and as a Regional County Municipality (MRC), ranking 28th among municipalities and 44th among MRCs in terms of population size in Quebec.

Covering an area of 6,484 km^2, Rouyn-Noranda is in the centre of the Abitibi-Témiscamingue region and is composed of a central urban hub, two urban-rural districts, and twelve rural districts.

The central hub has a high population density, while moving away from it, more rural characteristics appear.

20.3.1.2 Cadillac District

The mining village of Cadillac was founded in 1938 and serves as the secondary hub to the City of Rouyn-Noranda. It was established following the first well drilling by O'Brien Mines Limited and Thompson Cadillac in 1924. In 2002, the Town of Cadillac was merged with Rouyn-Noranda and became one of its neighbourhoods. It is the farthest district from the central hub. This neighbourhood serves as the eastern gateway to the City of Rouyn-Noranda and is equidistant from the major activity centres of Abitibi-Témiscamingue, including the central hub of Rouyn-Noranda, Val-d'Or, and Amos.

The project is located a little more than 1 km from the urban perimeter of the Cadillac District.

20.3.2 Aboriginal Groups and Territory

20.3.2.1 Algonquin Community of Abitibiwinni

The project site overlaps with the traditional territory of the Algonquin community of Abitibiwinni (Pikogan). The Algonquin community of Abitibiwinni Pikogan is located ~45 km northeast of the project site and approximately 3 km north of the Town of Amos. The community, known by its traditional name Abitibi8innik in reference to Abitibi Lake, was kept informed and regularly consulted by previous owners of the property claims.

Founded on a land base of 72 ha, Pikogan's territory now spans over 277 ha, accommodating around 600 community members on site, with an additional 300 members living outside the community. Young people make up a significant portion of the population, with over 40% of Abitibiwinni Nation members under the age of 18. The primary language spoken is French, followed by English and Algonquin. The Algonquin language, still spoken by elders and some community members, is actively promoted through educational and awareness initiatives aimed at younger generations to keep the language alive.

The community is governed by the Tribal Council of the Algonquin Anishinabeg Nation. Elections are conducted in accordance with traditional practices, though influenced by Canadian electoral laws. Council members serve three-year terms, while the Chief and Deputy Chief serve four-year terms.

Under the Entente Pikogan, an agreement on consultation and accommodation between the Abitibiwinni First Nation Council and the Government of Quebec, any holder of a claim within the applicable territory is encouraged to contact the Abitibiwinni First Nation's Natural Resources Secretariat. This engagement helps ensure the community is informed about planned exploration activities, allows for open communication, and offers an opportunity to address any concerns the Abitibiwinni First Nation may have.

20.3.2.2 Long Point First Nation

Long Point First Nation is an Anishinabeg community situated on the unceded territory of Anishnabe Aki. This Algonquin community comprises approximately 800 members, with about half residing in Winneway, located in the Abitibi-Témiscamingue region of Western Quebec and ~75 km south of the project site.

The Long Point First Nation Council asserts ancestral land claims in the Preissac area and has requested negotiations with Agnico Eagle regarding the LaRonde mine. These claims may also extend to the project property, which is near Agnico Eagle's operations.

20.3.3 Land Tenure

The mining claims included for O'Brien are located on public land. There is no forest shelter leases or vacation leases within the O'Brien property.

Radisson currently has surface rights to one area via annually renewable leases, which are in good standing.

20.3.4 Resource Use

20.3.4.1 Mining Activities

The Abitibi-Témiscamingue region was built around the mining industry, with mining projects dating back to 1926 when the first mining city was established in Rouyn-Noranda.

Glencore's Horne smelter, a major complex on the territory, specializes in metal processing and sustainable recycling. The city also hosts an operational gold mine (Elder, owned by Abcourt Mines Inc.), as well as numerous junior mining companies, exploration, drilling firms, and other services. Several mining projects are currently under development in the area, such as Horne 5 (Falco Resources), Wasamac (Agnico Eagle Mines), and Granada (Granada Gold Mine).

Located 50 km from the city of Rouyn-Noranda and 55 km from the city of Val-d'Or, the O'Brien Gold Project benefits from the strong mining-based economies of both cities.

According to the SIGEOM platform (Quebec Geomining Information System), adjacent to the O'Brien property, there are numerous active mineral claims held by various exploration companies. The closest active mine is the LaRonde mine complex, operated by Agnico-Eagle and located approximately 7 km west of the O'Brien property.

20.3.4.2 Forestry Activities

Under Quebec mining legislation, the owner of the mining rights can make use of the timber on the leased property by paying a nominal fee if such timber is deemed to be of commercial value.

Forestry plays a major role in the province at the economic level. The forest management unit (FMU) is the reference for territorial units that manage forests on public lands. There are 59 FMUs in Quebec and all are subject to commercial timber harvesting. The O'Brien Gold Project, which overlaps with FMU 082, involves logging and other forest management activities as defined in section 4 of the *Sustainable Forest Development Act* (c. A-18.1).

20.3.5 Recreational Activities

Preissac, situated northeast of the O'Brien property, is recognized for the economic benefits generated by its recreational tourism sector. Preissac Lake, which is 73 km^2 and with its shores just 3.5 km from the O'Brien property, occupies a significant portion of Preissac Township. This lake is central to the area with numerous outfitters, cottages, and nautical clubs.

Preissac Lake is a highly popular destination for both vacationers and tourists, and the regional development plan emphasizes recreational development across an extensive area near the lake. Additionally, Camping du Lac Normand, a major campsite with 350 sites located approximately 7.5 km from the project site, confirm the region's reputation as a vacationing area.

Currently, a network of 85.4 km of snowmobile trails, 17.9 km of four-season ATV trails, and 8.6 km of winter ATV trails have been established in the Rouyn-Noranda territory. The Trans-Québec snowmobile provincial trails and the Route Verte are key recreational tourism routes that connect the region to other RCMs and to Ontario.

The Trans-Québec snowmobile Trail number 83 runs through the Radisson property from Highway 117 to Lake Preissac.

20.3.6 Traditional Land and Resource Use

In the Abitibi-Témiscamingue region, the members of the Algonquin Anishinabeg traditionally use the territory for hunting, trapping, and fishing. Before the colonization of Abitibi-Témiscamingue and the exploitation of agricultural, forestry, and mining resources, the First Nations were harvesting wildlife resources primarily for subsistence purposes (food, shelter, clothing, and utilitarian objects). The close link between wildlife and the region's history is illustrated by memories of the Algonquin presence and the fur trade.

The beaver reserve, which occupies the northern portion of Abitibiwinni Aki (from the 49th parallel), grants the Abitibiwinni First Nation exclusive trapping rights. This section of ancestral territory comprises 34 family traplines totalling 11,400 km^2. Passed down from generation to generation, these family traplines are favoured sites for cultural and subsistence activities.

According to our research, there is no traditional trapline within the project property.

20.4 Consultations

Mining activities within the project's territory are covered by the Pikogan Agreement and require consultation with the local Indigenous community. Radisson will initiate future consultations.

As a first step, the company has identified key stakeholders to be consulted as part of the project. This preliminary list includes Indigenous communities, local governments, land users, local businesses, and economic development organizations. The objectives of the consultation process will be to:

- ensure that relevant information is communicated in a transparent and accessible manner (inform stakeholders of the project scope, potential impacts, and timeline, etc.)
- Identify and understand concerns, interests, and expectations related to the proposed mining project, particularly from stakeholders most directly affected
- support the development of mitigation measures and opportunities that respond to feedback received, and foster long-term, collaborative relationships with stakeholders.

20.5 Mineralized Material, Waste Rock, and Tailings Management

20.5.1 Mineralized Material Management

Mineralized material temporarily stored on site will be placed on a lined pad designed to contain and control any potential leachate. The mineralized material stockpile will remain relatively small in volume, as the material will be transported off site for processing. During storage, the mineralized material will be subject to appropriate environmental monitoring to ensure that any potential impacts, including seepage or dust emissions, are identified and managed.

Upon completion of mining activities, the mineralized material storage area will be fully decommissioned and restored in accordance with applicable regulatory requirements and the site closure plan.

20.5.2 Waste Rock Management

A waste pile will be established near the mineralized material stockpile; however, it is not expected to pose environmental concerns.

20.5.3 Tailings Management

As previously noted, the mineralized material will be transported off site for processing; therefore, no tailings will be generated or managed on site.

20.6 Water Management

A water management plan was developed by BBA in 2025 and includes preliminary hydrotechnical analyses and water balance calculations. The plan was established for the full life of mine. A retention basin and a treatment facility will provide appropriate treatment to neutralize acidity from the mineralized material pile and reduce environmental impacts. A system of ditches will collect and drain surface water runoff. Non-contact water will be diverted to the environment and runoff from the waste pile will be sent to a sedimentation basin. The surface water will then be safely discharged to the environment in compliance with applicable environmental standards.

A water quality monitoring program will be implemented to verify the effectiveness of the treatment system and ensure compliance with provincial discharge criteria. Monitoring results will be reviewed regularly and reported to the relevant regulatory authorities, as required. The site-wide water management plan is presented in Section 18.

20.7 Baseline Hydrogeology

Hydrogeological information is primarily derived from existing data and mapping, as well as from a hydrogeological study conducted by Richelieu in 2020. That study aimed to characterize the hydrogeological conditions of the bedrock surrounding the project area and included installing observation wells.

The hydraulic conductivity of the hydro stratigraphic units was determined using samples collected for grain size analysis, along with falling-head permeability tests conducted in observation wells. Overall, combining all methods, results indicate that the average hydraulic conductivity of the upper portion of the bedrock is 5.73×10^{-7} m/s (n=228), with values ranging from 4.2×10^{-13} to 2.76×10^{-6} m/s. Profile tracer tests (PTTs) conducted in exploration boreholes further indicate a decrease in hydraulic conductivity to average values on the order of 10^{-9} m/s throughout the profile, except in localized fractured zones.

Water level measurements indicate that the piezometric surface of the surficial aquifer generally flows toward the south and west. Overall, groundwater flow appears to follow the topographic gradient of the area. Most boreholes intersect the piezometric surface at approximately 1.5 m below ground surface.

Groundwater recharge occurs primarily in topographically elevated areas, where fine-grained glaciolacustrine sediments and glacial till tend to be thin or absent. Conversely, groundwater discharge generally takes place in lower-lying areas associated with the surface water drainage network. The preliminary estimate of average annual recharge per unit area ranges from 170 to 470 mm/a, depending on soil texture and slope. Based on the DRASTIC vulnerability index, the regional aquifer is considered to have low to moderate vulnerability, with index values calculated for the property ranging from 85 to 185.

20.7.1 Groundwater Quality

Groundwater sampled from the observation wells is generally of the calcium-bicarbonate type, with a neutral to alkaline pH and low mineral content (average electrical conductivity of 267 µS/cm). The geochemical profile of groundwater circulating in the overburden deposits is similar to that of groundwater flowing through the fracture network in the bedrock. Between 2016 and 2024, groundwater quality data indicate occasional exceedances of drinking water guidelines, particularly for aluminium, arsenic, manganese, mercury, and nickel. Exceedances of surface water discharge criteria were also noted for arsenic, silver, copper, mercury, and zinc.

The proximity of residential wells along Petit-Canada Road necessitates careful consideration of potential groundwater contamination risks. Groundwater quality will be monitored systematically to evaluate the potential for arsenic remobilization associated with future mining activities. A hydrogeological model will be developed and periodically updated to assess the risk of contamination of groundwater, whether related to current operations or to the remobilization of legacy contaminants from the former mine.

20.7.2 Geochemistry

Static testing was carried out by Golder in 2020 on 60 samples, including 20 mineralized samples, 28 waste rock samples, and 12 tailings composites. A waste rock and mineralized material sampling campaign were carried out in spring 2019 by a Golder geologist. Table 20-4 shows the samples submitted for geochemical characterization.

Table 20-4: Samples Submitted for Geochemical Characterization

Sample Type	Lithology	Number of Samples Analysed
Mineralized Material	Basalt	9
	Porphyritic andesite	5
	Conglomerates	5
	Greywacke	1
Tailings	Cyanide tailings	8
	Flotation tailings	3
	Cyanide tailings from flotation concentrate	1
Waste Rock	Greywacke	13
	Basalt	11
	Porphyritic andesite	4

Source: Golder (2020).

The analytical program was carried out at SGS laboratory (Lakefield, Ontario), a laboratory accredited by the *Centre d'expertise en analyse environnementale du Québec* (CEAEQ) for selected analyses. Static tests were carried out to classify the mining material samples according to the criteria of MELCCFP guidelines for geochemical characterization of mineralized material and mining wastes.

Results showed acid rock drainage (ARD) potential for 11 mineralized samples (55%) and 1 sample of tailings (8%) due to their low carbonate content and neutralization potential, as well as their variable sulphur content. No waste rock lithologies showed ARD potential. Table 20-5 presents a summary of ARD and leaching potential.

Table 20-5: Classification of the Material Analysed in the Geochemical Characterization of the Mining Material

Type	Lithology	No. of Samples	Potential ARD	Leaching Potential	High Risk	Classification
Ore	Basalt	9	4 (44%)	As(9), Ba(1), Cu(4), Mn(4), Zn(1)	As(4)	High risk (4/9), Leachable (9/9 ; 100%)
	Porphyritic Andesite	5	4 (80%)	Ag(1), As(5), Cu(2), Mn(1), PB(1)	As(1)	High risk (1/5), Leachable (5/5 ; 100%)
	Conglomerates	5	2 (40%)	As(3), Cr(1), Cu(3), Mn(1)	-	Leachable (4/5 ; 40%)
	Greywacke	1	0 (0%)	Cu(1), Mn(1)	-	Leachable (1/1 ; 100%)
Tailings	Cyanide Tailings	8	1 (13%)	As(3), Mn(1)	-	Leachable (4/8 ; 50%)
	Flotation Tailings	3	0 (0%)	Mn(1)	-	Leachable (1/3 ; 33%)
	Cyanide Tailings from Flotation Concentrate	1	1 (100%)	As, Zn	As	High risk, Leachable
Waste Rock	Greywacke	13	0 (0%)	-	-	Low risk
	Basalt	11	0 (0%)	Cu(3), Mn(5)	-	Leachable (7/11 ; 64%)
	Porphyritic Andesite	4	0 (0%)	As(2), Ba(3), Cr(2)	-	Leachable (4/4 ; 100%)

Source: Golder (2020).

Results for leaching tests carried out with SPLP, CTEU-9, and TCLP procedures showed most samples are classified as leachable (Ag, As, Ba, Cu, Mn, Pb, Zn). Three samples of mineralized material from the porphyritic andesite are classified as high risk for arsenic. Arsenic and aluminium are also released in concentrations exceeding the groundwater quality criteria (RES) for arsenic and aluminium.

Samples of cyanide tailings are classified as leachable for arsenic and manganese. Flotation tailings and concentrate samples were also analysed, and all three flotation tailings samples are leachable for manganese. The concentrate sample is leachable for arsenic.

For waste rock, all porphyritic andesite and most basalts are classified as leachable (As, Ba, Cu, Cr, Mn) and all greywacke samples are classified as low risk based on the results of the static geochemical testing. Complementary kinetic geochemical testing should confirm that most of the waste rock does is not metal leaching, so mitigation measures to protect groundwater (e.g., the use of a geomembrane) should not be required.

Additional static tests on waste rock samples are recommended to more accurately assess sulphur, arsenic, and manganese content, as well as other parameters. Kinetic tests are also suggested to better define the acidification potential and leaching risk of the mining materials.

20.8 Closure and Reclamation Planning

20.8.1 Concepts

Closure and rehabilitation planning will take place in collaboration with stakeholders. The main goals of closure and rehabilitation activities will be as follows:

- eliminate unacceptable health hazards and ensure public safety
- limit the production and spread of contaminants that could damage the receiving environment
- eliminate long-term maintenance and monitor requirements
- return the site to a visually acceptable condition
- return infrastructure areas to a state compatible with future use.

The main measures for restoring the mining site will include the following:

- plugging the two ramps, ventilation raises, and vertical conveyor
- revegetating the project footprint in accordance with regulations
- revegetating the waste rock pile and temporary mineralized material stockpile area
- demolishing and removing all buildings and other surface infrastructure
- managing the materials generated during dismantling of the facilities by applying the principles of reduction, reuse, recycling, and reclamation (and, if necessary, disposing of materials at authorized sites, according to the level of contamination)
- conducting a land characterization study to identify the presence of contaminants with concentrations in excess of regulatory values and taking the necessary measures, in compliance with the provisions of the *Environment Quality Act* and the Land Protection and Rehabilitation Regulations
- scarifying the roads built as part of the mining activities and restoring the natural drainage patterns
- dismantling the industrial wastewater treatment installations when deemed no longer necessary and dismantling all related installations (e.g., removal of pumps, pipelines, etc.)
- creating a breach in the water management ponds, levelling dams, covering the surface with topsoil before revegetation
- restoring the hydrological drainage to passive flows when appropriate.

Implementing an environmental monitoring program will demonstrate that reclamation works have achieved their goals.

20.8.2 Closure Cost Estimates

The total cost of reclamation (and the guarantee) is estimated at $5.3 million. This cost includes the direct and indirect costs of site rehabilitation as well as post-closure monitoring, engineering costs (30%), and a mandatory 15% contingency. A detailed cost estimate will be developed during the closure planning process in accordance with the Guidelines for Preparing Mine Closure Plans in Québec (MRNF, October 2024).

A financial bond corresponding to the total anticipated cost of completing all the work set forth in the rehabilitation and reclamation plan will be provided to the Minister of Finance of Québec as required by regulations.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The capital and operating cost estimates presented in this PEA provide substantiated costs that can be used to assess the preliminary economics of the O'Brien Gold Project. The estimates are based on the following:

- underground mining operations and surface infrastructure at the O'Brien site
- transportation of mineralized material to the Westwood complex
- new equipment to be installed at the Westwood process plant
- processing costs of the mineralized material at the Westwood process plant
- Owner's costs and provisions
- toll milling fees.

21.2 Capital Costs

21.2.1 Introduction

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2025 Canadian dollars based on quotations, Ausenco's in-house database of projects and studies, and experience from similar operations.

21.2.2 Overview

The capital cost estimate includes underground mine development, surface infrastructure at the O'Brien site, modifications at the Westwood process plant, project indirect costs, project delivery, Owner's costs, and contingency.

The capital cost estimate excludes both pre-production mine operating costs and revenue, which are reflected in the life-of-mine operating cost and revenue estimates, and excludes development costs incurred prior to the commencement of early works.

The capital cost summary is presented in Table 21-1. The total initial capital cost for the O'Brien Gold Project is C$174.8 million; and life-of-mine sustaining costs are C$172.7 million. O'Brien mine closure costs are estimated at C$5.3 million, and salvage credits are C$3.1 million. (Note: closure costs and salvage credits are not included in Table 21-1.)

Table 21-1: Summary of Capital Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
1000	Mining	93.2	168.7	261.9
3000	Process Plant	20.5	-	20.5
5000	On-Site Infrastructure	15.7	-	15.7
6000	Off-Site Infrastructure	8.2	-	8.2
	Total Directs	**137.7**	**168.7**	**306.7**
7000	Project Indirects and Delivery	13.7	-	13.7
8000	Owner's Cost	3.9	-	3.9
	Total Indirects	**17.6**	-	**17.6**
9000	Contingency	19.6	4.0	23.6
	Contingency	**19.6**	**4.0**	**23.6**
	Project Totals	**174.8**	**172.7**	**347.5**

Note: Excludes closure costs and salvage credits.

21.2.3 Basis of Estimate

21.2.3.1 Exclusions

The following are not included in the estimate:

- allowance for exchange rate fluctuations
- escalation.

21.2.3.2 Sources of Information

Data for the estimates have been obtained from numerous sources, including the following:

- mine schedules
- conceptual engineering design by Ausenco, Norda Stelo, and BBA
- mechanical equipment costs for flotation cells and the regrind mill from budgetary quotes received in 2025
- electrical equipment costs from pricing using Ausenco's database of recent Canadian studies and projects
- factoring of material take-offs for concrete, steel, instrumentation, in-plant piping by benchmarking against similar projects with equivalent technologies and unit operations
- engineering design to a preliminary economic assessment level

- budgetary equipment quotes from suppliers based in the USA and Canada
- data from similar recently completed studies and projects.

Major cost categories (permanent equipment, material purchase, installation, subcontracts, indirect costs, and Owner's costs) were identified and examined.

Costs were developed based on Ausenco's in-house database of costs and labour rates. The estimate is prepared in the base currency of Canadian dollars (currency: CAD; symbol: C$). Pricing has been converted to Canadian dollars from United States dollars (currency: USD; symbol: US$) at an exchange rate of 0.73 USD/CAD.

21.2.4 Mine Capital Costs (WBS 1100, WBS 1200 & WBS 1400)

The mine capital costs were sourced from third-party equipment manufacturers, contractors, and vendors and Norda Stelo's internal capital database. The capital estimation was completed with an accuracy of +40%/-30%.

The mine capital cost estimate does not include:

- costs for pre-feasibility and feasibility studies
- provision for changes in exchange rates
- GST/QST
- project financing and interest charges
- price/cost escalation during construction
- import duties and custom fees
- pilot plant and other testwork
- sunk cost
- exploration activities
- severance cost for employees at the cessation of operations
- any additional costs (but can partly be absorbed in contingency allowance).

Mine initial and sustaining capital costs are estimated at C$261.9 million, representing approximately 75% of the overall capital cost estimate. Mining-related capital costs are summarized by category in Table 21-2.

The mine rehabilitation and dewatering category includes all costs related to rehabilitating the Kewagama mine.

Table 21-2: Capital Cost Estimate for Mining

Description	Initial Capital Cost ($M)	Sustaining Cost ($M)	Total Cost ($M)
Contractor Mobilization-Demobilization	1.9	0.0	1.9
Mine Rehabilitation and Dewatering	2.2	4.0	6.2
Mine Development (Waste)	24.2	92.9	117.1
Underground Mobile Equipment	25.7	45.8	71.5
Underground Infrastructure	21.4	13.7	35.1
Surface Infrastructure and Equipment	17.5	12.2	29.7
Total	**93.2**	**168.7**	**261.9**

The mobile equipment fleet consists of underground production and development units to support the underground mine. All equipment will be a capital purchase agreement signed with vendors. Equipment capital costs represent a 15% down payment spread over one year and monthly payments over four-year terms (equipment costs plus financial costs). The sustaining capital cost includes a major overhaul for every 15,000 hours of operation. The major overhauls were estimated between 15% to 55% of the initial cost of the equipment without financing, depending on the type of equipment.

Underground infrastructure represents the equipment required to support secondary ventilation, electrical and communication distribution, and an underground pumping system. As for the surface infrastructure, it includes the portal, mineralized material and waste piles, surface material handling components, and ventilation system.

21.2.5 Mine Dry and Office Building (WBS 1300)

The mine dry and office building cost was developed based on a recent quotation for a similar office and mine dry facility in Northern Ontario. It includes the supply and installation of the building, foundation, and indirect costs. The building cost is listed in Table 21-3.

Table 21-3: Mine Dry and Office Building Costs

WBS	Building Description	Cost (C$M)
1300	Mine Dry, Offices, Lunchroom, Washrooms	0.4

21.2.6 Process Capital Costs (WBS 3000)

Process plant capital costs are summarized in Table 21-4. Process equipment requirements are based on conceptual process flowsheets and process design criteria as defined in Section 17. The process flowsheets and process design criteria are based on installing new process equipment to be integrated with the existing Westwood complex process facility.

Major equipment to be installed at the process plant was sized based the mechanical equipment list and process design criteria. The major mechanical equipment costs for the flotation cells and regrind mill were based on 2025 budgetary quotes. Tanks for reagent storage were based on Ausenco's database of recent Canadian studies and projects.

No major electrical equipment was included for the process plant integration. Discussions with the Westwood complex team indicated there is sufficient available MCC room and high-voltage transformer capacity, so this existing capacity was carried forward in this report. Electrical equipment costs were based on recent and historical budget quotes from similar projects, adjusted to reflect the size of the project.

To support the major mechanical and electrical equipment packages, the process plant and infrastructure engineering designs were completed to a PEA-level of study, allowing for bulk material costs (i.e., steel, concrete, earthworks) to be derived for major commodities based on factoring.

The total direct costs for materials and equipment for other disciplines were developed by applying factors (percentages) to the total direct cost (supply and installation) of the mechanical equipment. The factors are based on Ausenco's historical data for similar types of work and are specific to both discipline and area.

Table 21-4: Summary of Process Capital Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
3100	Flotation	4.5	0	4.5
3200	Regrind	14.0	0	14.0
3300	Reagents	2.0	0	2.0
	Process Plant Total	**20.5**	**0**	**20.5**

21.2.7 On-Site Infrastructure (WBS 5000)

On-site infrastructure costs are summarized in Table 21-5. The costs include the following:

- earthworks for roads, waste rock storage and mineralized material stockpile pad
- site water treatment
- site water management and handling
- cemented rockfill plant
- cemented rockfill plant building.

Table 21-5: On-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
5100	Earthworks and Water Management	11.3	0	11.3
5200	Cemented Rockfill Plant and Building	4.5	0	4.5
	On-Site Infrastructure Total	**15.7**	**0**	**15.7**

21.2.7.1 Earthworks and Water Management (WBS 5100)

21.2.7.1.1 Earthworks

Earthwork costs were based on material take-offs developed for the ditches, basins and roads and other site earthworks. Total earthwork costs are estimated as $2.45 M in Direct Costs.

21.2.7.1.2 Site Water Treatment, Management, and Handling

The direct expenses related to water treatment and supply are estimated at $8.8 million. These costs cover equipment, piping, the treatment building, and installation. The water treatment building is assumed to be a cornercast structure.

21.2.7.2 Cemented Rockfill (WBS 5200)

21.2.7.2.1 Cemented Rockfill Plant

The cement plant will be located on surface near the vertical conveyor The plant will primarily be used to produce cement slurry for underground backfilling with cemented rockfill. The facility will be fully automated and housed within an insulated, heated building designed for northern climate conditions to ensure consistent mixture quality and equipment reliability.

21.2.7.2.2 Cemented Rockfill Plant Building

The cemented rockfill plant building cost was developed based on 2023 and 2024 reference quotes escalated for inflation. The cost includes the building shell, overhead doors, HVAC, lighting, overhead crane, and indirects. The building cost is $0.8 million.

21.2.8 Off-Site Infrastructure (WBS 6000)

The costs were developed based on Ausenco's in-house database of costs and labour rates and includes the cost for a high-voltage transformer to power the underground mine and surface infrastructure. Off-site infrastructure costs are summarized in Table 21-6.

Table 21-6: Off-Site Infrastructure Capital Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
6200	Off-Site Infrastructure	8.2	0	8.2
	On-Site Infrastructure Total	**8.2**	**0**	**8.2**

21.2.9 Indirect Costs (WBS 7000, WBS 8000)

Indirect costs are summarized in Table 21-7 and described in the following subsections.

Table 21-7: Indirect Costs

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
7000	Project Indirects and Project Delivery	13.7	0	13.7
8000	Owner's Project Management	3.9	0	3.9
	Indirects Total	**17.6**	**0**	**17.6**

21.2.9.1 Project Indirects and Project Delivery (WBS 7000)

Project indirects are required during project delivery to enable and support construction activities. WBS 7000 also includes project delivery costs. These costs include the following:

- temporary construction facilities and services
- on-site materials transportation and storage
- commissioning representatives and assistance
- spares (commissioning, initial, and insurance)
- first fills and initial charges
- project delivery.

The project indirect and project delivery costs have been based on Ausenco's historical project costs of similar nature and is composed of the components, as shown in Table 21-8.

Table 21-8: Project Indirects and Project Delivery

WBS	WBS Description	Initial Capital Cost (C$M)	Sustaining Capital Cost (C$M)	Total Capital Cost (C$M)
7100	Field Indirects	3.2	0	3.2
7200	Project Delivery (EPCM)	9.7	0	9.7
7300	Commissioning/Operations Readiness	0.1	0	0.1
7400	Vendor Representatives	0.1	0	0.1
7500	Spares	0.4	0	0.4
7600	First Fills	0.2	0	0.2
	Indirects Total	**13.7**	**0**	**13.7**

The Level 2 WBS costs have been calculated as shown in Table 21-9. The costs have been calculated by applying the percentage listed under the "Factor Percentage" column to the costs listed under the "Basis of Factor" column, excluding any WBS numbers shown under the "Excluded WBS Numbers" column. For example, field indirects are calculated based on 5% of the total direct cost of $64.5 million, which is calculated from the total direct costs for the project of $137 million, less WBS 1100 and 1200 ($25.7 million and $47.4 million, respectively).

Table 21-9: Project Indirects and Project Delivery Factors

WBS	WBS Description	Basis of Factor	Excluded WBS Numbers	Factor Percentage (%)
7100	Field Indirects	Total Direct Costs	1100, 1200	5
7200	Project Delivery (EPCM)	Total Direct Costs	1100, 1200	15
7300	Commissioning/Operations Readiness	Total Equipment Supply	1100, 1200	1
7400	Vendor Representatives	Total Equipment Supply	1100, 1200	2
7500	Spares	Total Equipment Supply	1100, 1200	5
7600	First Fills	Total Equipment Supply	1100, 1200	2.5

21.2.9.2 Owner Capital Costs (WBS 8000)

Owner’s costs were factored as 5% of total directs excluding WBS nos. 1100 and 1200 ($63.6 million) and including the following:

- project staffing and miscellaneous expenses
- pre-production labour
- home office project management
- home office finance, legal, and insurance.

21.2.10 Contingency (WBS 9000)

Contingency accounts for the difference in costs from the estimated and actual costs of materials and equipment. The level of contingency varies depending on the nature of the contract and the client's requirements. Due to uncertainties at the time the capital cost estimate was developed, it is essential that the estimate includes a provision to cover the risk from these uncertainties.

The estimate contingency does not allow for the following:

- abnormal weather conditions
- changes to market conditions affecting the cost of labour or materials
- changes of scope within the general production and operating parameters
- effects of industrial disputations
- financial modelling
- technical engineering refinement
- estimate inaccuracy.

The total estimated contingency in initial capital costs for the O'Brien Gold Project is $19.6 million. The estimated contingency percentages are listed in the following subsections.

21.2.10.1 Underground Mining Contingency

Contingencies of up to 15% were applied variously to underground infrastructure line items and development costs, developed within the cost, productivity or material estimates. Non-underground mine items and surface facilities were assigned a cash contingency of 25%.

21.2.10.2 Other Contingency Costs

A contingency of 25% were assigned to total direct and indirect costs, except those associated with underground mining (refer to Section 21.2.9.1).

21.2.11 Sustaining Capital

21.2.11.1 Mining (WBS 1000)

Life-of-mine sustaining capital costs are estimated at $173 million (including contingencies). This estimate includes all capital expenditures incurred after the first quarter of Year 2, such as underground development in waste rock and underground infrastructure, but excludes mine closure and salvage costs.

Mobile mining equipment purchases are split between initial and sustaining capital; any payments or deposits made during the construction period are classified as initial capital, while those made during operations are included under sustaining capital.

Sustaining capital for mobile equipment includes major overhauls, which are scheduled approximately every 15,000 operating hours and range from 15% to 55% of the equipment's purchase cost, depending on the type of equipment.

Other sustaining capital items include surface and underground construction costs incurred after the start of production.

21.3 Operating Costs

21.3.1 Overview

The operating cost estimate is presented in Q2 2025 Canadian dollars (currency: CAD; symbol: C$). The estimate was developed to have an accuracy of ±50%. The estimate includes mining, water treatment at the O'Brien site, processing at the Westwood complex, general and administration (G&A) costs, and toll milling fees.

Toll milling fees have been calculated based on a percentage of estimated processing costs to reflect a profit component that may be charged above and beyond the estimated operating costs for processing O'Brien mineralized material at the Westwood complex. The selected toll milling fees for this study were 30% of the estimated processing and related general and administrative costs, which equates to $18.94/t.

Processing operating costs, including both variable and fixed costs, have been developed based on per tonne milled basis. For processing operating costs that are variable in nature (such as reagents, power), costs have been developed on a per tonne basis and are independent of the fraction of utilization of the Westwood complex by Radisson.

Processing operating costs that are fixed in nature were estimated on an annual basis and then divided by a nominal average annual throughput of 1.095 Mt/a (3,000 t/d) to determine a cost per tonne of milled material. The cost per tonne of milled material was then multiplied by the yearly throughput in the mine plan to allow for the consideration of the percent utilization of the Westwood complex by the O'Brien Gold Project. For example, if the O'Brien Gold Project provided 0.438 Mt of mineralized material for processing in a year at the Westwood complex, then 40% (1.095 Mt /0.438 Mt = 40%) of these fixed costs have been attributed to the project for that year. This method applies to fixed costs such as labour, maintenance, and vehicle costs and allows for an equitable attribution of costs between Westwood and O'Brien during the use of the Westwood complex with multiple feeds.

The overall life-of-mine operating cost, excluding toll milling fees, is $661 million over 11 years, or an average of $144.43/t milled. The overall life-of-mine operating costs, including toll milling fees is $747 million over 11 years, or an average of $163.38/t milled. Table 21-10 provides a summary of the project operating costs.

Table 21-10: Operating Cost Summary

Cost Area	Life-of-Mine Total ($M)	Milled Cost ($/t)	% of Operating Cost Subtotal	% of Total Costs, with Toll
Mining	346	75.66	52	46
Process	173	37.71	26	23
G&A	142	31.06	22	19
Operating Cost Subtotal	661	144.44	100	N/A
Toll Milling Fees	87	18.94	N/A	12
Total with Toll Fees	**747**	**163.38**	**N/A**	**100**
Off-Site Costs, Refining & Transport	6	-	-	-
Royalties	10	-	-	-
Total Cash Costs	**763**	**-**	**-**	**-**

21.3.2 Mine Operating Costs

The total operating cost for mining is $346.1 million, which represents approximately 46% of the overall operating cost. Table 21-11 shows the distribution for mining operating costs.

Table 21-11: Capital Cost Estimate for Mining

Description	Operating Cost (C$M)	Cost per Tonne (C$/t)	% of Mining Operating Cost
Contractor Indirect Operating Cost	12.7	2.77	3.7
Definition Drilling	14.6	3.20	4.2
Stope Development	81.5	17.81	23.5
Stope Production	76.7	16.76	22.1
Underground Services	133.8	29.25	38.7
Water Management	5.7	1.24	1.6
Mine Site to Process Plant Haulage	21.1	4.62	6.1
Total	**346.1**	**75.66**	**100**

21.3.3 Process Operating Costs

The processing operating cost estimate includes costs related to the following:

- reagent and consumable consumption
- process plant maintenance
- power use
- laboratory
- process plant labour
- processing mobile equipment.

The life-of-mine process operating cost is $173 million over 11 years. The process operating costs are estimated as an average of $34.70/t of milled. Process operating costs are summarized in Table 21-12.

Table 21-12: Processing Operating Cost Summary

Cost Area	Life-of-Mine Average Cost (C$M/a)	Milled Cost ($/t)	% of Total
Reagents	6.8	17.68	47
Consumables	1.5	4.00	11
Maintenance	0.4	1.15	3
Power	0.8	2.08	6
Laboratory	0.1	0.18	0
Labour (O&M)	3.5	9.19	24
Mobile Equipment (O&M)	0.2	0.43	1
Tailings Operating Cost	1.1	3.01	8
Total	**14.4**	**37.71**	**100**

21.3.3.1 Reagents

The reagent profile was developed according to the testwork results outlined in Section 13. The testwork enabled the addition and consumption rates of reagents to be estimated. Where testwork was not available, benchmarking against proven unit technologies was used. Costs for each reagent were based on 2023 and 2024 email quotes escalated to 2025 costs. The details are presented in Table 21-13. The reagent costs are approximately $17.68/t, or 47% of the total process operating cost.

Table 21-13: Reagent Cost Summary (Life-of-Mine Average)

Reagent	Life-of-Mine Average (C$M/a)	C$/t of Mill Feed
Activated Carbon	0.1	0.24
Antiscalant	0.0	0.07
Borax	0.0	0.01
Copper Sulphate	0.0	0.11
Dust Suppression	0.0	0.10
Ferric Sulphate	0.0	0.08
Flocculant	0.0	0.01
Hydrochloric Acid	0.4	1.06
Hydrogen Peroxide	0.0	0.01
Lime	0.5	1.11
MIBC	0.0	0.04
Sodium Cyanide	5.3	12.83
PAX	0.2	0.45
R208	0.1	0.30
SMBS	0.5	1.19
Project Total	**7.4**	**17.68**

21.3.3.2 Consumables

Consumables were identified as non-reagent requirements/replacements that are related to the crushing circuit and griding circuit. The following items are considered consumables:

- jaw crusher liners
- crusher screen panel
- SAG mill liner
- SAG mill media
- ball mill liner
- ball mill media
- regrind media
- regrind mill liner.

The usage was estimated from benchmarking databases of similar mineralization. The unit costs were based on a regression of internal data obtained from vendor quotations for similar projects. The details are shown on Table 21-14. The consumable costs are approximately $4.00/t, or 11% of the total process operating cost.

Table 21-14: Consumable Cost Summary (Life-of-Mine Average)

Consumable	Life-of-Mine Average (C$M/a)	Unit Cost (C$/t Milled)
Jaw Crusher Liners	0.0	0.01
Crusher Screen Deck	0.1	0.14
SAG Mill Liners	0.2	0.51
Ball Mill Liners	0.2	0.44
SAG Mill Media	0.1	0.32
Ball Mill Media	0.8	1.98
Regrind Mill Media	0.1	0.13
Regrind Mill Liner	0.1	0.17
Crucibles	0.0	0.02
Propane	0.0	0.02
Diesel	0.1	0.27
Subtotal	**1.6**	**4.00**

21.3.3.3 Labour

Staffing was estimated by developing a site-specific estimate of the mill operations and maintenance staff. The estimate includes staff for the following areas, as well as an allowance for contract labour:

- mill staff
- administration
- mill operations
- laboratory
- mill maintenance.

The total operational labour is an estimated 99 employees. Organizational staffing plans outlining the labour requirement are shown in Table 21-15. The labour costs are approximately $9.19/t, or 24% of the total process operating cost.

Table 21-15: Processing Plant Labour Summary

Labour Description	Number of Personnel
Mill Management	**7**
Mill Manager	1
Operations Superintendent	1
Metallurgy Superintendent	1
Maintenance Superintendent	1
Maintenance Planner	1
Senior Metallurgist	1
TMF Engineer-in-Training	1
Mill Operations	**32**
Shift Supervisor	4
Assistant Super	4
Crushing Operator	4
Grinding Operator	4
Leach/Elution Operator	4
Equipment Operator	4
Control Room	4
TMF Labourer	4
Milll Maintenance	**24**
Mechanical Supervisor	2
Millwright	10
Apprentice	2
Electrical Supervisor	2
Electrician	4
Instrumentation Technician	2
Process Control	2
Metallurgy & Laboratory	**15**
Senior Metallurgist	1
Metallurgist in Training	1
Metallurgical Technical	2
Goldroom Security	1
Main Gate	0
Senior Assayer	1
Sample Preparation	4
Assayer	4
Goldroom Refiner	1
O'Brien Surface Operations	**5**
Operators	4
Millwright	1
Total	**83**

Source: Ausenco (2025).

21.3.3.4 Power

The power cost is calculated from the current Westwood process plant power usage of 35 to 36 kWh/t as described in the January 2025 IAMGOLD Westwood Technical Report, plus additional power usage requirements such as for flotation and regrind equipment. The calculated additional consumption for flotation, regrind, and other equipment is 4.7 kWh/t, for a combined total of 39.5 kWh/t.

The cost for power was based on the 39.5 kWh/t consumption combined with an average all-in delivered power cost of $0.052/kWh, which includes both variable and fixed electricity costs. The power costs are approximately $2.08/t, or 6.0% of the total process operating cost.

21.3.3.5 Laboratory

Operating costs associated with laboratory and assay activities were estimated according to the anticipated number of assays per day and per year, estimated by Ausenco. Assay costs include samples taken from various samplers throughout the plant, solution samples, tests on the loaded, barren, and regenerated carbon, bullion bar testing, cyanide detoxification sampling, and environmental sampling and assaying. The laboratory and assays comprise approximately $0.18/t, or 0.5% of the total process operating cost. An estimated 13,000 internal assays are required per year.

21.3.3.6 Mobile Equipment

The process plant mobile equipment operating costs are based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares, and tires, as well as annual registration and insurance fees. Mobile equipment costs have been based on a total estimated cost. The mobile equipment costs comprise approximately $0.43/t, or 1.2% of the total process operating cost.

21.3.3.7 Equipment Maintenance

Annual maintenance costs were estimated based on total installed capital costs from two reference facilities in 2021 and one in 2023 that have similar process flowsheets and throughputs, multiplied by a 5% maintenance factor. The estimated total maintenance operating cost is $1.15/t milled, or 3.3% of the total process operating cost.

21.3.4 Tailings Operating Costs

Tailings-related operating costs for processing the O’Brien mineralized material at the Westwood complex were included under total operating costs. An operating cost of $3.01/t was added to the total operating costs as an allowance for additional tailings-related costs that may be incurred by the combined operations. Total tailings operating costs over the life of mine are $14 million.

21.3.5 General and Administrative Operating Costs

General and administrative (G&A) costs are expenses not directly related to the production of doré and include expenses not captured under mining, processing, external refining, and/or transportation costs. These costs were developed based on the January 2025 IAMGOLD Westwood Technical Report G&A on a cost-per-tonne basis, which represents the total G&A costs for the IAMGOLD process plant and mines. This G&A cost has been included to account for Westwood process costs. Additional G&A costs to account for personnel and insurance at the O'Brien site were included and were fixed per annum. The G&A cost listed in the 2025 IAMGOLD Westwood Technical report for 2025 and 2026 averaged $25.42/t. Additional G&A positions are listed in Table 21-16 for an estimated cost of $1.45 million per year.

Table 21-16: O'Brien Site Related G&A Staff Positions

Labour Description	Number of Personnel
Administration Staff	**2**
Administration Assistance	1
Finance Manager	1
Finance, Procurement and Accounts Payable	**2**
Payroll	1
Buyers	1
Security	**4**
Main Gate – O'Brien Site	4
Safety	**3**
Safety Officers	2
Safety Trainer	1
Environment	**3**
Supervisor	1
Technicians	1
Engineer	1
Human Resources	**2**
Human Resources Generalist	2
Total	**16**

Source: Ausenco (2025).

Insurance costs for the O'Brien site were estimated at $0.89 million per year. The fixed annual cost for G&A was estimated as a total of $2.3 million per year. The fixed and variable costs are summarized in Table 21-17.

Table 21-17: G&A Cost Summary

Item	Cost	Cost Basis
Westwood Process Plant G&A Costs	$25.42	Variable, per O'Brien tonne processed
Additional G&A Staff Positions	$1.45 million	Fixed, per annum
O'Brien Site Insurance	$0.89 million	Fixed, per annum

Source: Ausenco (2025).

21.3.6 Toll Milling Fees

Toll milling costs have been calculated based on a percentage of processing costs to reflect a profit component that can be charged above the estimated operating costs for processing O'Brien material at the Westwood complex. The selected toll milling costs for this study were 30% of the estimated processing and related general and administrative costs, which equates to $18.94/t or approximately $7.9 million per year.

This cost must be supported in future negotiations with IAMGOLD with a formal proposal and contract for toll milling. Alternative toll milling facilities may also be investigated as the processing flowsheet is typical of a gold-bearing sulphide feed.

22 ECONOMIC ANALYSIS

22.1 Forward-Looking Information Cautionary Statements

The results of the economic analyses discussed in this chapter represent forward-looking information as defined under Canadian securities law. The results depend on inputs that are subject to known and unknown risks, uncertainties, and other factors that may cause actual results to differ materially from those presented here.

Information that is forward-looking includes the following:

- mineral resource estimates
- assumed commodity prices and exchange rates
- proposed mine production plan
- projected mining and process recovery rates
- assumptions regarding mining dilution and estimated future production
- sustaining costs and proposed operating costs
- assumptions regarding closure costs and closure requirements
- assumptions regarding environmental, permitting, and social risks.

Additional risks to the forward-looking information include the following:

- changes to costs of production from what is assumed
- unrecognized environmental risks
- unanticipated reclamation expenses
- unexpected variations in quantity of mineralized material, grade, or recovery rates
- accidents, labour disputes, and other risks of the mining industry
- geotechnical or hydrogeological considerations during mining being different from what was assumed
- failure of mining methods to operate as anticipated
- failure of plant, equipment, or processes to operate as anticipated
- changes to assumptions as to the availability of electrical power, and the power rates used in the operating cost estimates and financial analysis
- changes to site access, use of water for mining purposes, and to time to obtain environment and other regulatory permits
- ability to maintain the social licence to operate
- changes to interest rates
- changes to tax rates.

Readers are cautioned that a preliminary economic analysis is preliminary in nature. It includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

Mineral resources are not mineral reserves and do not have demonstrated economic viability.

22.2 Methodologies Used

The project was evaluated using a discounted cash flow analysis based on a 5% discount rate. Cash inflows consisted of annual revenue projections. Cash outflows consisted of capital expenditures, including pre-production costs; operating costs; refining and transport costs; taxes; and royalties. These were subtracted from the inflows to arrive at the annual cash flow projections.

Cash flows were taken to occur at the end of each period. It must be noted that tax calculations involve complex variables that can only be accurately determined during operations and as such, actual post-tax results may differ from estimates. A sensitivity analysis was performed to assess the impact of variations in gold price, exchange rate, initial capital costs, operating costs, and mill head grades.

The capital and operating cost estimates are presented in Section 21 in Q2 2025 Canadian dollars. The economic analysis was run based on a constant dollar value with no inflation.

22.3 Financial Model Parameters

The economic analysis was performed assuming a gold price of US$2,550/oz, which was based on long-term consensus analyst estimates. The forecasts are meant to reflect the average metals price expectation over the life of the project. No price inflation or escalation factors were considered. Commodity prices can be volatile, and there is the potential for deviation from the forecast.

The economic analysis also used the following assumptions:

- a pre-production period of 21 months
- a mine life of 11 years
- 100% ownership
- an exchange rate of 0.73 CAD/USD based on a review of current long-term consensus
- capital cost funded with 100% equity (no financing cost assumed)
- cash flows that are discounted to the start of construction using an end-period discounting convention
- metal products that are sold in the same year they are produced
- project revenue derived from the sale of gold doré
- no contractual arrangements for refining exist.

22.3.1 Taxes

The project has been evaluated on an after-tax basis to provide an approximate value of the potential economics.

The calculations are based on the tax regime in place as of the date of the preliminary economic analysis. At the effective date of the cashflow, the project was assumed to be subject to the following tax regime: the effective tax rate over the life of mine is 29.9%, consisting of a 30% corporate income tax.

At the base case gold price assumption, total tax payments are estimated to be $343 million over the life of mine.

22.3.2 Working Capital

An estimate of working capital has been incorporated into the economic analysis based on the following assumptions:

- accounts receivable – 0 days
- inventories – 30 days
- accounts payable – 30 days.

22.3.3 Closure Costs and Salvage Value

Closure costs are applied in the first 30 months of the project based on Quebec requirements, and the salvage value is applied at the end of the life of mine. Closure costs were estimated to be $5 million, and salvage value was estimated to be $3 million.

22.3.4 Royalties

A 2% net smelter royalty (NSR) is applied to gold production on certain claims on the easternmost portion of the property in favour of Globex Mining Enterprises Inc., which covers approximately 22% of the scheduled gold production and totals $10 million.

22.4 Financial Analysis

The pre-tax NPV discounted at 5% is $782.5 million; the IRR is 65.1%; and payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is $532.4 million; the IRR is 47.6%; and payback period is 2.0 years. The project is cash positive post-tax at gold prices above US$1,260/oz.

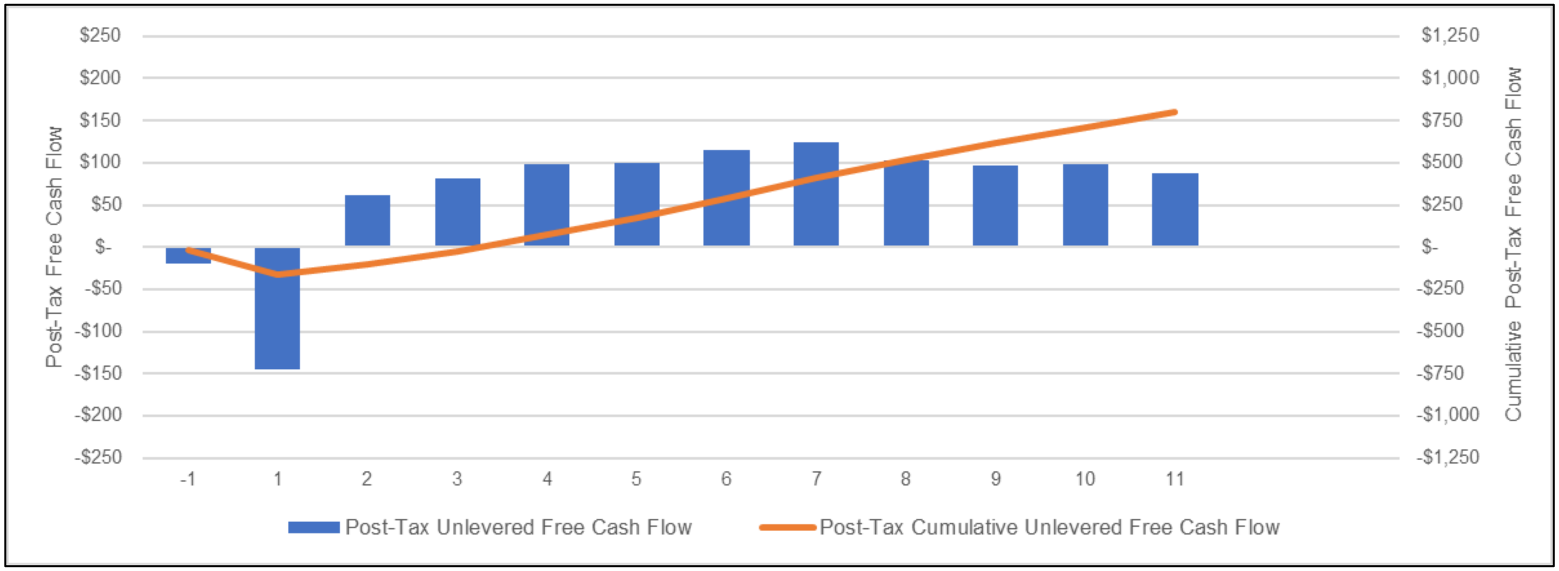
A summary of project economics is shown in Table 22-1 and illustrated in Figure 22-1. The analysis was done on an annual cashflow basis; the cashflow output is presented in Table 22-2.

Table 22-1: Economic Analysis Summary

Description	Unit	Life-of-Mine Total / Average
General Assumptions		
Discount Rate	%	5.0
Gold Price	US$/oz	2,550
Exchange Rate	CAD/USD	0.73
Mine Life	years	11.0
Total Overburden and Waste Tonnes Mined	kt	3,314
Total Mill Feed Tonnes	kt	4,575
Production		
Mill Head Grade (Au)	g/t	5.03
Mill Recovery Rate	%	87.5
Total Mill Ounces Recovered	koz	647
Total Average Annual Production	koz	59
Transport, Refining, Royalties		
Gold Payable	%	99.95
Refining & Transport Cost	US$/oz Au	6.25
NSR Royalty	%	2.0
Operating Costs		
Mining Cost	C$/t mined	75.66
Processing Cost	C$/t milled	37.71
G&A Cost	C$/t milled	31.06
Toll Milling	C$/t milled	18.94
Total Operating Cost	C$/t milled	163.38
Cash Costs and All-In Sustaining Costs		
Cash Costs*	US$/oz Au	861.06
All-In Sustaining Cost (AISC)**	US$/oz Au	1,058.54
Capital Expenditures		
Initial Capital Cost	C$M	175
Sustaining Capital Cost	C$M	173
Closure Capital Cost	C$M	5
Salvage Value	C$M	(3)
Economics		
Pre-Tax NPV @ 5%	C$M	782.5
Pre-Tax IRR	%	65.1
Pre-Tax Payback	years	1.4
Post-Tax NPV @ 5%	C$M	532.4
Post-Tax IRR	%	47.6
Post-Tax Payback	years	2.0

Notes: * Cash costs consist of mining costs, processing costs, mine-level G&A, refining and transport charges and royalties. ** AISC includes cash costs plus sustaining capital, closure costs, and salvage value.

Figure 22-1: Projected Life-of-Mine Post-Tax Unlevered Free Cash Flow



Source: Ausenco (2025).

Table 22-2: Cash Flow Forecast on an Annual Basis

Dollar Figures in Real 2025 C$M Unless Otherwise Noted Macro Assumptions	Units	Total / Avg.	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Gold Price - Flat	**US$/oz**	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550	$2,550
FX	**CAD:USD**	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73	$0.73
Free Cash Flow Valuation															
Revenue	**C$M**	**$2,258**	**--**	**$26**	**$238**	**$251**	**$255**	**$253**	**$244**	**$247**	**$229**	**$175**	**$182**	**$158**	**$1**
Operating Cost	C$M	$747	--	$31	$88	$104	$93	$83	$75	$71	$64	$48	$50	$38	$1
Refining & Transport Charges	C$M	$6	--	$0	$1	$1	$1	$1	$1	$1	$1	$0	$0	$0	$0
Royalties	C$M	$10	--	$0	$2	$2	$1	$2	$1	$1	$0	--	--	--	--
EBITDA	**C$M**	**$1,496**	**--**	**($6)**	**$147**	**$144**	**$160**	**$168**	**$166**	**$174**	**$164**	**$127**	**$131**	**$119**	**($0)**
Initial Capex	C$M	($175)	($17)	($138)	($19)	--	--	--	--	--	--	--	--	--	--
Sustaining Capex	C$M	($173)	--	--	($35)	($27)	($21)	($26)	($10)	($8)	($23)	($4)	($6)	($12)	--
Closure Capex	C$M	($5)	($3)	($1)	($1)	--	--	--	--	--	--	--	--	--	--
Salvage Value	C$M	$3	--	--	--	--	--	--	--	--	--	--	--	--	$3
Change in Working Capital	C$M	--	--	--	--	--	--	--	--	--	--	--	--	--	--
Pre-Tax Unlevered Free Cash Flow	**C$M**	**($0)**	**($20)**	**($145)**	**$92**	**$118**	**$139**	**$142**	**$156**	**$166**	**$141**	**$123**	**$125**	**$107**	**$3**
Pre-Tax Cumulative Unlevered Free Cash Flow	C$M		*($20)*	*($165)*	*($73)*	*$44*	*$183*	*$325*	*$481*	*$647*	*$788*	*$911*	*$1,036*	*$1,143*	*$1,146*
Tax Payable	C$M	($343)	--	--	($30)	($37)	($41)	($42)	($41)	($43)	($38)	($26)	($27)	($20)	--
Post-Tax Unlevered Free Cash Flow	**C$M**	**($0)**	**($20)**	**($145)**	**$61**	**$81**	**$98**	**$100**	**$115**	**$124**	**$103**	**$97**	**$98**	**$88**	**$3**
Post-Tax Cumulative Unlevered Free Cash Flow	C$M		*($20)*	*($165)*	*($104)*	*($23)*	*$75*	*$176*	*$291*	*$415*	*$517*	*$614*	*$712*	*$800*	*$803*
Production															
Production Summary															
Total Resource Mined	kt	4,575	--	100	508	597	531	544	515	484	427	289	313	264	3
Total Waste Mined	kt	3,314	--	578	569	533	445	357	252	130	151	126	148	25	--
Total Material Mined	kt	7,889	--	678	1,077	1,130	976	901	768	613	578	414	461	289	3
Mill Feed	kt	4,575	--	100	508	597	531	544	515	484	427	289	313	264	3
Mill Head Grade (Au)	g/t	5.03	--	2.94	4.80	4.37	4.91	4.78	4.84	5.19	5.40	6.04	5.81	5.95	3.84
Contained (Au)	koz	740	--	9	78	84	84	84	80	81	74	56	58	51	0
Mill Recovery (Au)	%	87.5%	--	78.5%	86.9%	85.6%	87.2%	86.8%	87.0%	87.8%	88.3%	89.6%	89.2%	89.4%	83.6%
Gold Production	koz	647	--	7	68	72	73	72	70	71	66	50	52	45	0
Gold % Payable	%	99.95%	--	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%	99.95%
Payable Gold	**koz**	**647**	**--**	**7**	**68**	**72**	**73**	**72**	**70**	**71**	**66**	**50**	**52**	**45**	**0**
Revenue	**C$M**	**$2,258**	**--**	**$26**	**$238**	**$251**	**$255**	**$253**	**$244**	**$247**	**$229**	**$175**	**$182**	**$158**	**$1**
Operating Costs															
Total Operating Costs	**C$M**	**$747**	**--**	**$31**	**$88**	**$104**	**$93**	**$83**	**$75**	**$71**	**$64**	**$48**	**$50**	**$38**	**$1**
Mine Operating Costs	C$M	$346	--	$20	$44	$53	$47	$36	$31	$29	$27	$22	$22	$14	$1
Processing Costs	C$M	$173	--	$4	$19	$23	$20	$20	$19	$18	$16	$11	$12	$10	$0
G&A Costs	C$M	$142	--	$5	$15	$18	$16	$16	$15	$15	$13	$10	$10	$9	$0
Toll Milling Fees	C$M	$87	--	$2	$10	$11	$10	$10	$10	$9	$8	$5	$6	$5	$0

Dollar Figures in Real 2025 C$M Unless Otherwise Noted Macro Assumptions	Units	Total / Avg.	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	Y11	Y12
Operating Costs per Tonne Milled	C$/t Milled	*$163*	--	*$310*	*$174*	*$174*	*$175*	*$152*	*$146*	*$147*	*$150*	*$165*	*$161*	*$146*	*$456*
Refining, Transport & Royalties															
Refining & Transport Charges	**C$M**	**$5.5**	**--**	**$0.1**	**$0.6**	**$0.6**	**$0.6**	**$0.6**	**$0.6**	**$0.6**	**$0.6**	**$0.4**	**$0.4**	**$0.4**	**$0.0**
Royalties	**C$M**	**$9.7**	**--**	**$0.3**	**$2.2**	**$1.6**	**$1.2**	**$1.5**	**$1.5**	**$1.1**	**$0.3**	**--**	**--**	**--**	**--**
Cash Costs															
Cash Cost *	US$/oz Au	$861	--	$3,093	$976	$1,081	$949	$856	$809	$752	$722	$701	$713	$628	$3,233
All-in Sustaining Cost (AISC) **	US$/oz Au	$1,059	--	$3,222	$1,361	$1,354	$1,164	$1,121	$915	$836	$981	$762	$796	$818	($4,702)
Capital Expenditures															
Initial Capital	**C$M**	**$175**	**$17**	**$138**	**$19**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Mine	C$M	$93	$0	$74	$18	--	--	--	--	--	--	--	--	--	--
Process Plant	C$M	$21	--	$21	--	--	--	--	--	--	--	--	--	--	--
On-Site Infrastructure	C$M	$16	$2	$13	--	--	--	--	--	--	--	--	--	--	--
Off-site Infrastructure	C$M	$8	$8	--	--	--	--	--	--	--	--	--	--	--	--
Total Indirect Costs	C$M	$14	$2	$11	$0	--	--	--	--	--	--	--	--	--	--
Owners Costs	C$M	$4	$1	$3	$0	--	--	--	--	--	--	--	--	--	--
Contingency	C$M	$20	$3	$16	$0	--	--	--	--	--	--	--	--	--	--
Total Sustaining Capital	**C$M**	**$173**	**--**	**--**	**$35**	**$27**	**$21**	**$26**	**$10**	**$8**	**$23**	**$4**	**$6**	**$12**	**--**
Mining	C$M	$169	--	--	$32	$26	$21	$26	$10	$8	$23	$4	$6	$12	--
Mining Contingency (Non-Underground)	C$M	$4	--	--	$2	$0	$1	$1	$0	--	--	--	--	--	--
Closure Cost	**C$M**	**$5**	**$3**	**$1**	**$1**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**
Salvage Value	**C$M**	**($3)**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**--**	**($3)**
Total Capital Expenditures Including Salvage Value	**C$M**	**$350**	**$20**	**$140**	**$55**	**$27**	**$21**	**$26**	**$10**	**$8**	**$23**	**$4**	**$6**	**$12**	**($3)**

Notes: * Cash costs consist of mining costs, processing costs, mine-level G&A, refining and transport charges and royalties. ** AISC includes cash costs plus sustaining capital, closure costs, and salvage value.

22.5 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and post-tax NPV and IRR of the project using the following variables: gold price, exchange rate, initial capital costs, operating costs, and mill head grades. Table 22-3 shows the pre-tax sensitivity analysis results; post-tax sensitivity results are shown in Table 22-4.

Table 22-3: Pre-Tax Sensitivity Analysis

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.0%	$485	$837	$1,146	$1,543	$1,808	$2,425
3.0%	$363	$654	$909	$1,237	$1,455	$1,965
5.0%	$298	$557	$783	$1,073	$1,267	$1,718
8.0%	$222	$439	$629	$874	$1,037	$1,417
10.0%	$180	$375	$546	$765	$912	$1,253

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.0%	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
3.0%	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
5.0%	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
8.0%	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
10.0%	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%

Pre-Tax NPV Sensitivity To FX

FX	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.65	$441	$731	$985	$1,311	$1,528	$2,036
0.70	$348	$617	$853	$1,156	$1,358	$1,829
0.73	$298	$557	$783	$1,073	$1,267	$1,718
0.80	$197	$433	$639	$904	$1,081	$1,493
0.85	$135	$357	$551	$800	$966	$1,354

Pre-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.65	39.4%	61.2%	80.5%	105.6%	122.5%	162.3%
0.70	32.4%	52.6%	70.4%	93.6%	109.2%	146.0%
0.73	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
0.80	20.9%	38.8%	54.3%	74.3%	87.8%	119.7%
0.85	16.1%	33.1%	47.6%	66.4%	79.1%	109.0%

Pre-Tax NPV Sensitivity To Opex

Opex	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$411	$669	$895	$1,185	$1,379	$1,831
(10.0%)	$355	$613	$839	$1,129	$1,323	$1,775
--	$298	$557	$783	$1,073	$1,267	$1,718
10.0%	$242	$500	$726	$1,017	$1,210	$1,662
20.0%	$186	$444	$670	$961	$1,154	$1,606

Pre-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	38.4%	58.4%	76.0%	98.8%	114.1%	150.1%
(10.0%)	33.5%	53.2%	70.5%	92.9%	108.1%	143.7%
--	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
10.0%	24.0%	43.1%	59.9%	81.7%	96.4%	131.1%
20.0%	19.4%	38.4%	54.8%	76.3%	90.8%	125.1%

Pre-Tax NPV Sensitivity To Initial Capex

Initial Capex	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$331	$589	$815	$1,105	$1,299	$1,751
(10.0%)	$315	$573	$799	$1,089	$1,283	$1,735
--	$298	$557	$783	$1,073	$1,267	$1,718
10.0%	$282	$540	$766	$1,057	$1,250	$1,702
20.0%	$266	$524	$750	$1,040	$1,234	$1,686

Pre-Tax IRR Sensitivity To Initial Capex

Initial Capex	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	35.2%	58.2%	78.6%	105.5%	123.6%	166.4%
(10.0%)	31.7%	52.7%	71.2%	95.5%	111.9%	150.5%
--	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
10.0%	26.1%	44.2%	59.9%	80.3%	94.0%	126.3%
20.0%	23.8%	40.8%	55.4%	74.3%	87.0%	116.9%

Pre-Tax NPV Sensitivity To Mill Head Grade

Mill Head Grade	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$33	$232	$406	$630	$780	$1,128
(10.0%)	$166	$394	$594	$852	$1,023	$1,423
--	$298	$557	$783	$1,073	$1,267	$1,718
10.0%	$431	$719	$971	$1,294	$1,510	$2,014
20.0%	$564	$881	$1,159	$1,516	$1,754	$2,309

Pre-Tax IRR Sensitivity To Mill Head Grade

Mill Head Grade	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	7.8%	23.5%	36.6%	53.2%	64.4%	90.7%
(10.0%)	18.4%	35.8%	50.8%	70.1%	83.1%	113.9%
--	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
10.0%	38.7%	60.4%	79.6%	104.6%	121.5%	161.1%
20.0%	48.8%	72.9%	94.3%	122.2%	140.9%	185.0%

Pre-Tax NPV Sensitivity To Recovery

Recovery	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$67	$273	$454	$687	$841	$1,203
(10.0%)	$183	$415	$618	$880	$1,054	$1,461
--	$298	$557	$783	$1,073	$1,267	$1,718
10.0%	$414	$698	$947	$1,266	$1,479	$1,976
20.0%	$466	$761	$1,020	$1,352	$1,573	$2,090

Pre-Tax IRR Sensitivity To Recovery

Recovery	Gold Price (US$/oz)					
	$1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	10.6%	26.8%	40.4%	57.8%	69.6%	97.2%
(10.0%)	19.8%	37.5%	52.7%	72.5%	85.8%	117.2%
--	28.7%	48.1%	65.1%	87.2%	102.2%	137.3%
10.0%	37.4%	58.7%	77.6%	102.1%	118.7%	157.6%
20.0%	42.1%	64.8%	84.9%	111.0%	128.6%	170.1%

Table 22-4: Post-Tax Sensitivity Analysis

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.0%	$340	$587	$803	$1,081	$1,266	$1,698
3.0%	$244	$448	$626	$856	$1,009	$1,366
5.0%	$193	$374	$532	$736	$871	$1,188
8.0%	$134	$286	$419	$591	$705	$971
10.0%	$102	$239	$358	$512	$614	$853

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.0%	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
3.0%	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
5.0%	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
8.0%	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
10.0%	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%

Post-Tax NPV Sensitivity To FX

FX	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.65	$293	$496	$674	$902	$1,055	$1,410
0.70	$228	$417	$582	$794	$935	$1,265
0.73	$193	$374	$532	$736	$871	$1,188
0.80	$122	$287	$432	$617	$741	$1,030
0.85	$79	$234	$370	$545	$661	$933

Post-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
0.65	28.9%	44.8%	58.7%	76.9%	89.2%	118.5%
0.70	23.8%	38.6%	51.5%	68.2%	79.5%	106.4%
0.73	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
0.80	15.3%	28.5%	39.8%	54.3%	64.0%	87.2%
0.85	11.7%	24.3%	34.9%	48.6%	57.7%	79.4%

Post-Tax NPV Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$272	$453	$611	$815	$950	$1,267
(10.0%)	$233	$414	$572	$775	$911	$1,227
--	$193	$374	$532	$736	$871	$1,188
10.0%	$154	$335	$493	$696	$832	$1,148
20.0%	$115	$295	$454	$657	$793	$1,109

Post-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	28.0%	42.5%	55.2%	71.7%	82.8%	109.2%
(10.0%)	24.5%	38.8%	51.4%	67.6%	78.6%	104.6%
--	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
10.0%	17.7%	31.8%	44.0%	59.7%	70.4%	95.7%
20.0%	14.4%	28.4%	40.4%	55.9%	66.4%	91.3%

Post-Tax NPV Sensitivity To Initial Capex

Initial Capex	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$218	$399	$557	$761	$896	$1,213
(10.0%)	$206	$387	$545	$748	$884	$1,200
--	$193	$374	$532	$736	$871	$1,188
10.0%	$181	$362	$520	$723	$859	$1,175
20.0%	$169	$349	$508	$711	$847	$1,163

Post-Tax IRR Sensitivity To Initial Capex

Initial Capex	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	25.8%	42.5%	57.2%	76.6%	89.8%	121.3%
(10.0%)	23.2%	38.6%	52.0%	69.5%	81.4%	109.6%
--	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
10.0%	19.1%	32.4%	43.9%	58.7%	68.6%	92.1%
20.0%	17.4%	30.0%	40.7%	54.4%	63.6%	85.3%

Post-Tax NPV Sensitivity To Mill Head Grade

Mill Head Grade	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$6	$147	$269	$426	$530	$774
(10.0%)	$101	$261	$401	$581	$701	$981
--	$193	$374	$532	$736	$871	$1,188
10.0%	$286	$488	$664	$891	$1,042	$1,395
20.0%	$379	$601	$796	$1,046	$1,212	$1,601

Post-Tax IRR Sensitivity To Mill Head Grade

Mill Head Grade	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	5.5%	17.2%	26.9%	39.0%	47.1%	66.2%
(10.0%)	13.5%	26.3%	37.2%	51.2%	60.7%	82.9%
--	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
10.0%	28.4%	44.2%	58.1%	76.2%	88.4%	117.6%
20.0%	35.8%	53.2%	68.7%	88.9%	102.7%	135.4%

Post-Tax NPV Sensitivity To Recovery

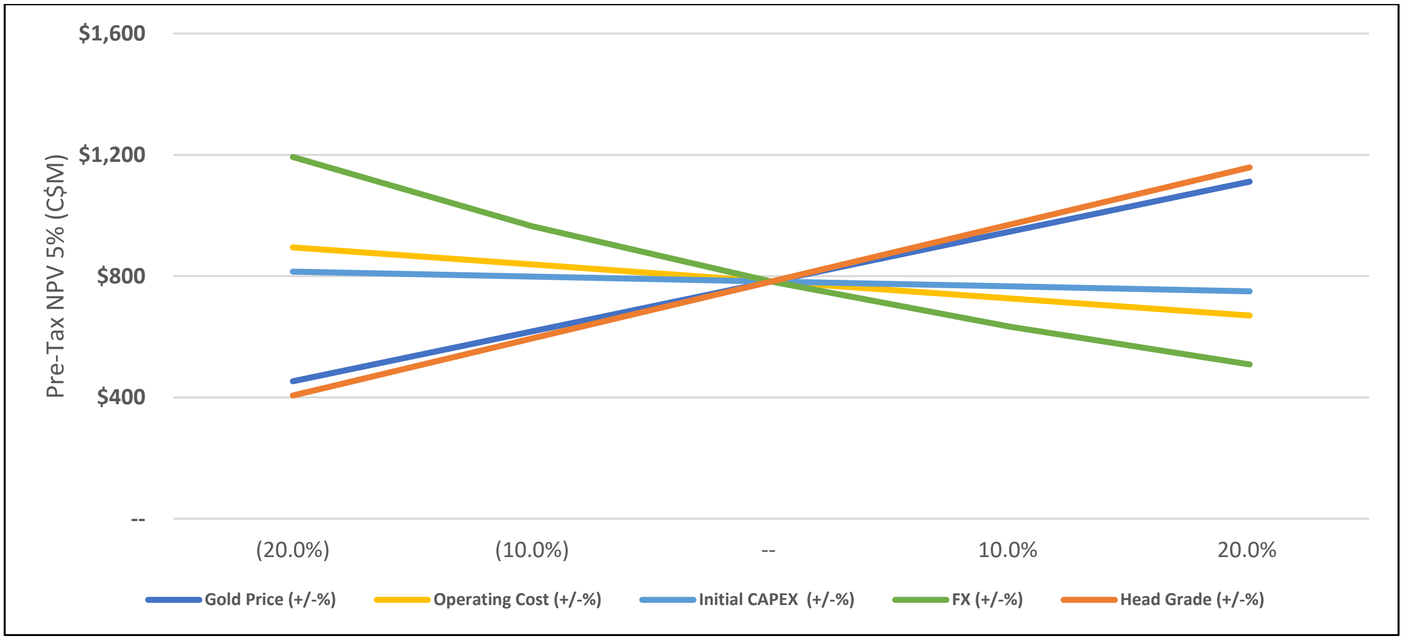
Recovery	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	$31	$176	$302	$465	$574	$827
(10.0%)	$112	$275	$417	$601	$723	$1,007
--	$193	$374	$532	$736	$871	$1,188
10.0%	$275	$473	$647	$871	$1,020	$1,368
20.0%	$310	$517	$698	$931	$1,086	$1,448

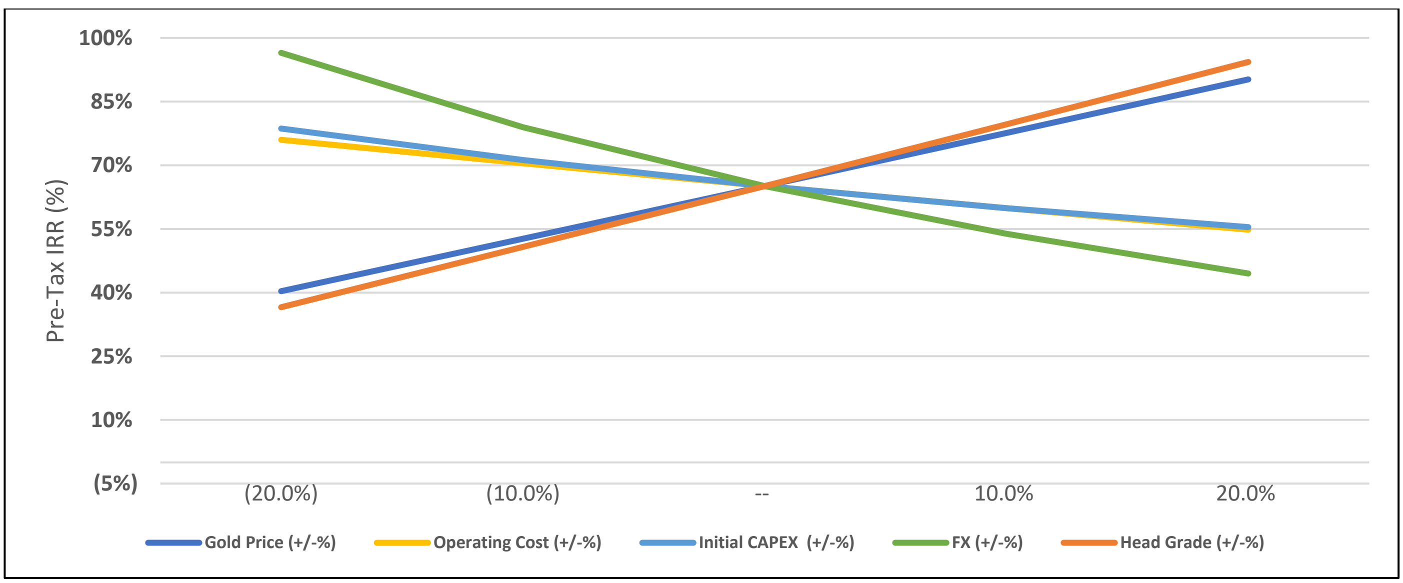
Post-Tax IRR Sensitivity To Recovery

Recovery	Gold Price (US$/oz) $1,800	$2,200	$2,550	$3,000	$3,300	$4,000
(20.0%)	7.6%	19.6%	29.6%	42.4%	50.8%	70.8%
(10.0%)	14.5%	27.5%	38.6%	52.9%	62.6%	85.4%
--	21.0%	35.3%	47.6%	63.6%	74.4%	100.1%
10.0%	27.5%	43.0%	56.6%	74.4%	86.4%	115.0%
20.0%	30.8%	47.2%	61.8%	80.7%	93.6%	124.2%

As presented in Figures 22-2 and 22-3, the sensitivity analysis showed that the project is most sensitive to changes in head grade, followed by gold price, and foreign exchange.

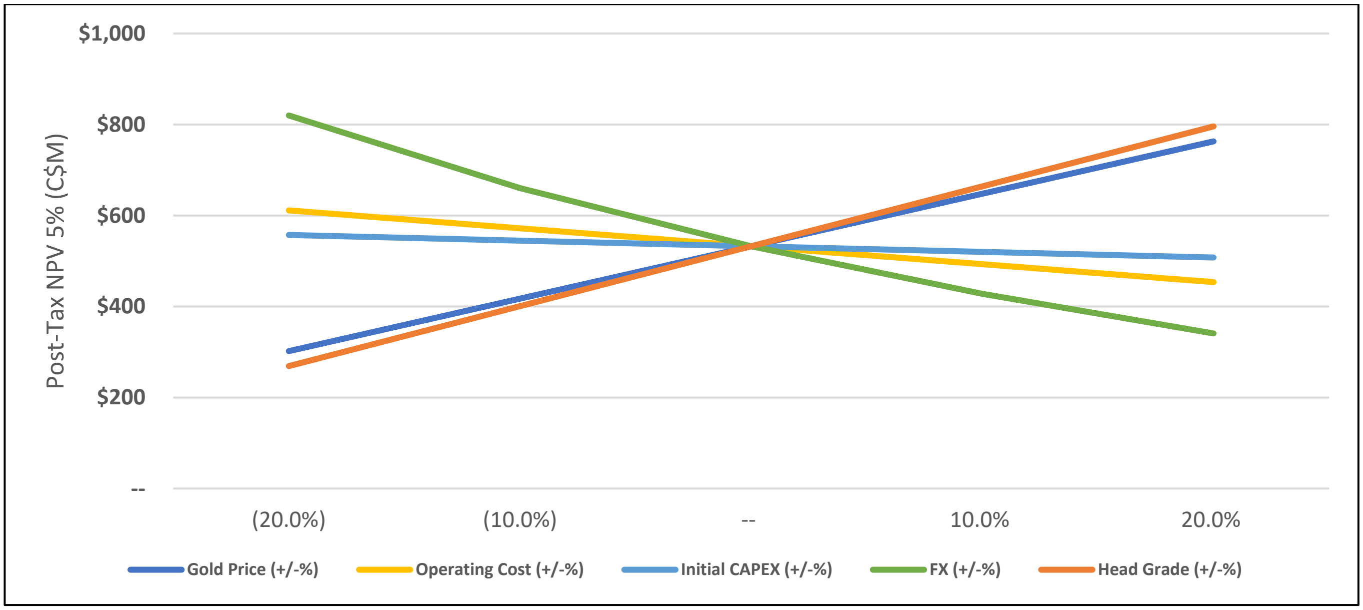
Figure 22-2: Pre-Tax NPV, IRR Sensitivity Results

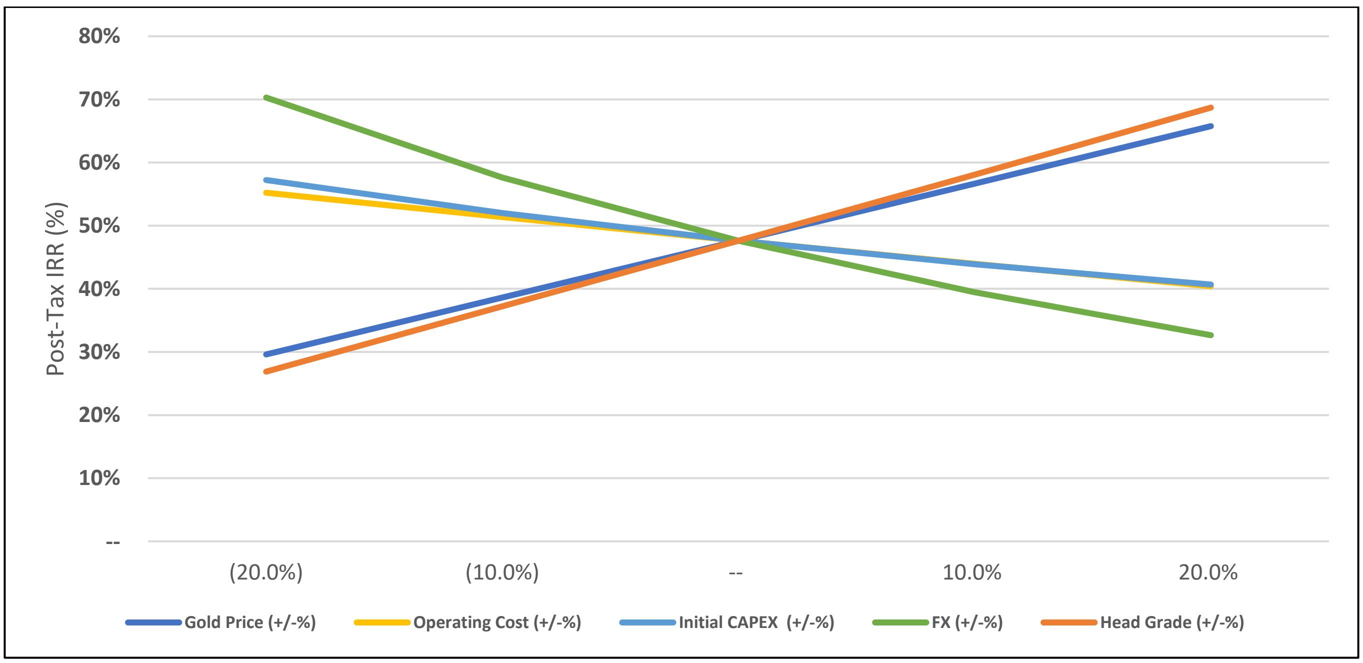




Source: Ausenco (2025).

Figure 22-3: Post-Tax NPV, IRR Sensitivity Results





Source: Ausenco (2025).

23 ADJACENT PROPERTIES

This section is modified from Section 23, Adjacent Properties, in the technical report by Williamson (2019). The QP has reviewed and compared Williamson's description in their report to other accounts in publicly available documents and considers it accurate to the best of its knowledge.

The region surrounding the O'Brien Gold Project has seen exploration and mining activities, some of which are ongoing. Several producers and mineral occurrences are found within a few kilometres of the project, as illustrated in Figure 23-1.

The QP has been unable to verify the information presented below for properties adjacent to the project site. The presence of significant mineralization within these properties is not necessarily indicative of similar mineralization at O'Brien. The QP did not review the technical and economic parameters used to produce the mineral resource estimates for these properties.

23.1 Agnico Eagle

Two major deposits, Bousquet-1 and Bousquet-2, are found on the properties held by Agnico Eagle Mines Limited (Agnico Eagle) along the northern boundary of the O'Brien Gold Project. The Bousquet deposits are located approximately 7 km west-northwest of the resource area presented in this technical report. They were mined by Lac Minerals Ltd. between 1979 and 1996. By 1996, production totalled 10.8 Mt at 5.96 g/t Au (Beaudoin et al, 2014).

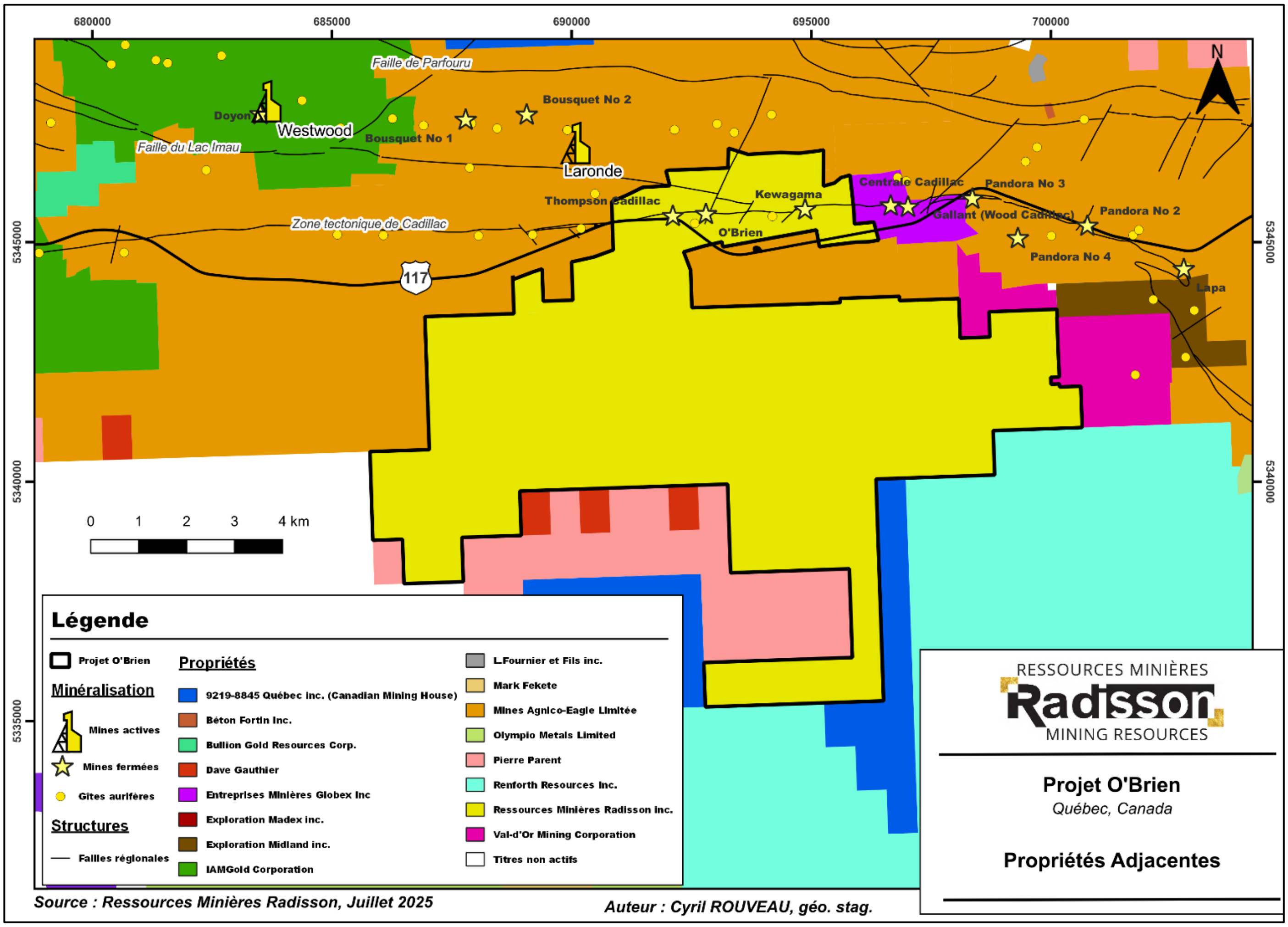
Along the same stratigraphical horizon as the Bousquet deposits, and less than 2.0 km to the east, the LaRonde mine has been in operation since 1988, and has produced more than 5.0 Moz of gold as well as valuable byproducts (silver, zinc, copper, and lead). The mine still has 3.2 Moz of gold in proven and probable reserves (22.7 Mt grading 4.42 g/t Au). The deep extension of the LaRonde mine achieved commercial production in November 2011 and is the focus of ongoing mining activities, with an estimated mine life that will last until 2032 with the LaRonde Zone 5 (Agnico Eagle, 2023).

The stratigraphical horizon related to the Bousquet and LaRonde-Dumagami deposits is located within the bimodal volcanics of the Blake River Group.

These deposits are described as gold-rich volcanogenic massive sulphide (VMS) deposits and cannot be compared or associated with the deposits found at the O'Brien Gold Project. They occur along a different stratigraphic horizon, approximately 2 km north of the resource estimate area presented in this technical report.

In April 2015, Agnico Eagle acquired the property adjacent to the southern boundary of the O'Brien Gold Project. In 2015, the property was registered to 9265-9911 Québec Inc.

Figure 23-1: Adjacent Properties



Source: Radisson (2025).

23.2 Westwood Mine Complex

IAMGOLD's Westwood mine complex is approximately 21 km west of O'Brien. Westwood began initial production in 2013 and commercial production in July 2014, producing 70,000 ounces in the first two quarters at an average diluted grade of 7.98 g/t Au. Ore from the Westwood underground mine is hauled to the Westwood complex mill for processing by ball and SAG mill, and to the carbon-in-pulp mill complex.

The Westwood area is underlain by meta-volcanics of the Blake River Group and partly by meta-sediments of the Cadillac and Kewagama Groups. Mineralization is mainly associated with units nos. 4.2, 4.3, 4.4, 5.1, and 5.2 of the Bousquet Formation. These units host gold-rich, VMS-type mineralization at the Bousquet and LaRonde, deposits and gold-sulphide vein-type mineralization in Zones 1 and 2 at Westwood. These deposits are described as gold-rich volcanogenic massive sulphide (VMS) deposits and cannot be compared or associated with the deposits found at the O'Brien Gold Project.

In September 2024, Radisson entered into an MOU with IAMGOLD to assess the design criteria for processing mined material from O'Brien. Testwork results were found to be favourable and are discussed in Section 13 of this technical report.

23.3 Pandora Wood

The Pandora Wood property, held 100% by Globex Mining Enterprises Inc. (Globex), hosts two former gold producers: the Central Cadillac mine and the Wood-Cadillac mine (Pressacco, 2008). The Central Cadillac mine was discovered in 1933 and is approximately 3 km east of the resource area presented in this report. From 1939 to 1943, production from the Central Cadillac mine was 185,541 tonnes at 5.14 g/t Au for a total of 954 kg of gold and 115 kg of silver. From June 1947 to August 1949, production was reported as 233,329 tonnes at 4.33 g/t Au for a total of 1,010 kg of gold and 130 kg of silver; however, it is thought that all or most of the production was from the Wood-Cadillac mine as the contribution from the Central Cadillac mine was not specified. The combined production for these two periods amounts to 418,870 tonnes at 4.69 g/t Au for a total of 1,964 kg of gold.

Mineralization in these deposits is also orogenic, closely related to the CLLFZ. Most of the mineralization comes from horizontal quartz-tourmaline veins found in a 15 m interval between the CLLFZ and iron formations. The veins and their strongly tourmalinized wallrocks are slightly mineralized with pyrite, arsenopyrite, and free gold. The veins also contain chalcopyrite and massive scheelite. Late quartz veinlets containing gold crosscut the older mineralized veins as well as silicified greywackes. Gold mineralization associated with arsenopyrite and pyrite was also found in talc-chlorite schists of the CLLFZ.

Globex reported high-grade drill results in December 2024, including hole SIW-24-04, with 23.22 m true width averaging 23.82 g/t Au, and hole SIW-24-05, with 6.65 m true width at an average grade of 5.69 g/t Au (Globex, 2024). The May 2025 NI 43-101 report states an indicated mineral resource of 234,800 tonnes at a grade of 14.38 g/t Au for 108,530 oz gold and inferred resources of 37,100 tonnes at 7.22 g/t Au for 8,610 oz gold, effective April 15, 2025 (SLR 2025).

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data and information to share at this stage of project development.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

This section provides interpretations and conclusions from the QPs in their areas of expertise, based on their review of available data for this report.

25.2 Mineral Tenure, Surface Rights, Water Rights, Royalties and Agreements

The O'Brien Gold Project consists of a contiguous block of 146 exploration claims that cover an area of 7,137.74 ha and one mining concession that covers an area of 288.19 ha. Radisson has a 100% interest in the project, which is comprised of three properties named O'Brien, Kewagama, and Thompson-Cadillac (previously New Alger).

25.3 Geology, Exploration, Drilling, and Mineral Resources

The QP offers the following conclusions:

- Indicated mineral resources are estimated to total 2.20 million tonnes (Mt) at a grade of 8.22 g/t Au, containing 582 thousand ounces (koz) of gold. Inferred mineral resources are estimated to total 6.67 Mt at a grade of 4.35 g/t Au, containing 932 koz Au.
- The principal reason for the changes to the O'Brien mineral resource estimate is the decrease in the reporting cut-off grade from 4.5 g/t Au to 2.2 g/t Au, reflecting a change in the long-term gold price from US$1,600/oz Au to US$2,000/oz Au, increased milling recovery to 90% based on testwork as reported in Section 13, and engineering studies confirming the reasonableness of blocks meeting a marginal grade cut-off.
- Drilling from January 2023 to April 2025 within the O'Brien area has validated the 2023 mineral resource and will not significantly impact the global numbers for that footprint.
- New drilling outside the O'Brien mineral resources, particularly at depth, appears promising for additional mineral resources, although further drilling is required before estimating mineral resources at depth. There is excellent potential to increase the mineral resources at O'Brien, particularly at depth, and additional exploration and technical studies are warranted.
- There is a good understanding of the geology and nature of the gold mineralization at the property; however, risks to the mineral resource estimate are associated with the nuggety nature of the gold mineralization which could impact assumptions about the continuity of the mineralization.
- The sample collection, preparation, analytical, and security procedures and the quality assurance/quality control (QA/QC) program, as designed and implemented by Radisson, are adequate, and the assay results within the database are suitable for use in mineral resource estimation.

- The QA/QC program indicates generally good precision, negligible sample contamination, and a relatively low bias at the primary laboratory. Some rates of failure that are higher than average for the certified reference material samples in 2020 are explained by the sample preparation issues described by Radisson geologists. While further work is warranted to resolve some outstanding issues with these results, they are sufficient to support the use of the underlying data for mineral resource estimation at this level of study.

25.4 Exploration

Since acquisition, Radisson has continued exploration work on the property with drilling, trenching, and geophysical programs, and by advancing the resource with multiple mineral resource estimates.

In 2022, Radisson embarked on an extensive exploration campaign on the southern New Alger area of the O’Brien Mine property, focusing on the Pontiac Group sedimentary units. The work was separated into three distinct phases and was carried out between January and October 2022. The work consisted of a compilation/planning phase, a prospecting and sampling phase, and a trenching phase.

In 2024, Radisson began drilling deep holes beneath the O’Brien mine and has succeeded in intercepting mineralization. Additional drilling is warranted.

In the QP’s opinion, the property has excellent exploration potential for gold mineralization.

25.5 Drilling

From the 1930s to 2025, approximately 1,207 diamond drill holes totalling 374,461.8 m have been drilled.

25.6 Analytical Data Collection in Support of Mineral Resource Estimation

Before 1995, the O’Brien mine used its internal laboratory for assaying. Between 1995 and 2025, a number of laboratories were used such as Chimitec Ltd. (Chimitec) in Val d’Or, Québec; XRAL Laboratories (XRAL) in Rouyn-Noranda, Québec; Techni-Lab Inc. (Techni-Lab) in Ste-Germaine, Québec; Laboratoire Expert Inc. (Laboratoire Expert) in Rouyn-Noranda, Québec; Swastika Laboratories Ltd. (Swastika) in Swastika, Ontario; ALS Minerals (ALS) in Val d’Or, Québec; and SGS Canada Inc. (SGS) in Val d'Or, Québec.

Except for the internal O’Brien mine laboratory, all the laboratories are independent of Radisson. Commercial laboratories Swastika, ALS, and SGS are accredited to the International Organization for Standardization/International Electrotechnical Commission (ISO/IEC) 9001:2008 standards for quality management and to ISO/IEC 17025:2005 for all relevant procedures. Accreditation of the other laboratories is unknown.

Sample preparation and analysis procedures have remained consistent over time, despite changes to the primary laboratory employed. Laboratories have generally employed a standard approach whereby samples are crushed and pulverized prior to gold analysis by fire assay (with AAS or gravimetric finish), with or without follow-up of metallic screen gold analysis on selected high-grade samples.

In the QP's opinion, the sample preparation, analytical protocols, QA/QC, and security procedures are acceptable for the purposes of mineral resource estimation.

25.7 Metallurgical Testwork

The 2024 metallurgical testwork was conducted in two phases. The objective was to confirm the most suitable flowsheet to maximize the economic outcome for the O'Brien Gold Project and to determine a recovery model for the economic assessment.

Phase 1 and Phase 2 testwork consisted of the tests as shown in Table 25-1 and Table 25-2.

Table 25-1: 2024 SGS Phase 1 Tests Summary

Flowsheet Description	Test Set 1 (Whole Ore Leach)	Test Set 2 (Gravity, Flotation, Leaching)
Whole Ore Leaching	X	-
Gravity Concentration	-	X
Leaching of Gravity Tails	-	-
Flotation of Gravity Tails	-	X
Flotation without Gravity	-	-
Leaching of Flotation Concentrate	-	X
Leaching of Flotation Tails	-	-
Master Composite Used	-	-
Number of Samples Processed	13	11

Table 25-2: 2024 SGS Phase 2 Tests Summary

Description	Test Set 3 (Gravity & Gravity Tails Leaching)	Test Set 4 (Gravity, Gravity Tails Flotation)	Test 5 (Gravity, Flotation, Regrind, Separate Leaching of Flotation Concentrate & Tails)	Test 6 (Gravity, Flotation, Regrind, Combined Leaching of Flotation Concentrate & Tails)
Whole Ore Leaching	-	-	-	-
Gravity Concentration	X	X	X	X
Leaching of Gravity Tails	X	-	-	-
Flotation of Gravity Tails	-	X	X	X
Flotation without Gravity	-	-	-	-
Regrind of Flotation Concentrate	-	-	X	X
Separate Leaching of Flotation Concentrate	-	-	X	-
Separate Leaching of Flotation Tails	-	-	X	-
Combined Leaching			-	X
Master Composite Used	-	X	X	X
Total Number of Samples Processed	12	3	1	1

Based on the completed testwork, the following interpretations were made:

- The optimum process was determined to be gravity concentration, flotation of gravity tails, regrind of flotation concentrate, and cyanidation of flotation concentrate and flotation tails to extract gold. With an 85 µm primary grind size (P_{80}) through this process, a recovery of 90% was achieved.
- Pre-aeration decreased cyanide consumption and may be considered during processing to reduce operating costs.

25.8 Mining Methods

The project is planned as an underground mining operation with off-site processing to minimize its surface footprint. The selected mining method is a modern, mechanized approach using longitudinal longhole stoping. Most of the waste rock will be returned underground as stope backfill, and the remainder will be stored on surface.

The project is near the historical O'Brien and Kewagama underground mines, which requires careful consideration of mine dewatering, waste management, and pillar assessment.

Each zone will have its own main decline, portal, and ventilation network. An underground drift will connect the two zones to facilitate mineralized material handling and allow the mines to share permanent infrastructure.

Six-tonne load-haul-dump (LHD) loaders will load trucks underground. For the upper part of the mine, mineralized material from the stopes will be hauled to surface using 20-tonne trucks. For the lower part of the mine, the mineralized material, including the low-grade material, will be transported by 20-tonne trucks to a vertical conveyor. Waste material will be hauled to a surface stockpile using 20-tonne trucks via the decline.

Underground mine development is scheduled to start in Q1 of Year -1, with initial stope production beginning in Q4 of Year -1. Commercial production is expected to start in the third month of Year 1. The mine is planned to operate until Q1 of Year 11, with potential for extension through mineral resource conversion and further exploration.

The life-of-mine plan outlines a rapid production ramp-up with production increasing to an average of 76,000 ounces recovered per year from Q2 of Year 1 through Year 7. The pre-production period includes a total of 197,000 tonnes of mineralized material grading 3.49 g/t Au.

Mine development is expected to average 8,500 m of horizontal development per year.

25.8.1 Modern Underground Operation

The project is designed as a mechanized underground mining operation employing optimized mining methods and sequences to reduce worker exposure and minimize the surface environmental footprint.

25.8.2 Vertical Conveyor

The vertical conveyor system reduces the size of the required truck fleet and facilitates access to deeper parts of the deposit, enhancing economic extraction at depth.

25.8.3 Internal Decline

The internal decline between the two mines reduces traffic congestion, provides exploration access, and enables shared use of permanent infrastructure such as the maintenance bay, pump stations, and explosives magazine.

25.8.4 Gradual Production Ramp-Up

The project follows a structured production schedule with a gradual ramp-up over the first two years. This approach supports infrastructure development and enables a smoother transition to full, steady-state production.

25.8.5 Mine Life Extension Potential

There is potential to extend the project beyond its current mine life through ongoing exploration and mineral resource conversion.

25.8.6 Environmental Measures

The project includes measures to reduce surface waste and tailings disposal. The use of pastefill and backfilling of stopes contributes to minimizing the environmental impact of the operation.

25.8.7 Operational Flexibility and Reduced Risk

The shallow depth of the deposit, the redundancy provided by two independent declines, the historical production record of the deposit, and the presence of nearby independent processing facilities capable of toll milling contribute to a lower initial capital investment requirement.

In summary, the project is presented as a well-structured underground mining operation with efficient material handling and an emphasis on environmental management.

25.9 Recovery Methods

The Westwood complex process plant is expected to process material at a rate of 3,000 t/d using toll milling for the O'Brien mineralized material. The process plant modifications at Westwood were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and impacts to existing operations during installation of the equipment.

The selected comminution and recovery processes are widely in the industry used with no significant elements of technological innovation.

25.10 Infrastructure

25.10.1 Project Infrastructure

The main site infrastructure consists of underground mining, electrical infrastructure, access roads, waste rock and temporary mineralized material storage, water management, and mine site support buildings.

As no process plant or tailings management facility are planned for the mine site, limited project infrastructure is required compared to a similar sized, standalone project. This reduces the overall complexity of mine site operations.

25.10.2 Tailings Facility

The necessary process plant infrastructure at the Westwood complex already exists. No additional facilities are required to serve the new equipment that will be installed at Westwood to process the O'Brien mineralized material.

Tailings will be stored at the Westwood complex tailings management facility as part of the toll milling agreement. An assessment of the tailings management facility was not conducted for this PEA. Under a toll milling agreement, tailings would be expected to be the responsibility of the process plant owner.

25.11 Water Management

The water management plan for the O'Brien mine site has been established based on a hydrological analysis. It includes the optimal strategy for managing contact water, sizing collection ditches, and estimating basin volumes, as discussed in Chapter 18.

For the water management plan, the best practices and standards applicable to the mining industry and the design of civil engineering structures were adopted. The design considers climate change risks and their impact on water management based on projections by Environment Canada. The retention basin was designed for runoff generated by 24-hour rainfall with a 2,000-year recurrence since the mineralized material pad and underground water are considered acid-generating. To comply with Directive 019, runoff from melting snow cover with a 100-year recurrence must also be managed, and this will be done at the water treatment plant. The maximum treatment capacity for the water treatment unit has been predetermined for a 100-year snowmelt event with the dewatering rate.

The detailed design of the storage basins, drainage ditches, and emergency spillways will be prepared in subsequent phases of the project.

25.12 Environmental, Permitting and Social Considerations

To date, the inventories carried out on the project property did not identify any environmental issues that pose a risk. Environmental characterization field studies must be continued to obtain the environmental baseline data required for the environmental and social impact assessment that will be required. However, the historical O'Brien Mine hosts 8,938 barrels of arsenic trioxide stored underground in a Class 1 dangerous waste site. The current project resources are located away from this area. Should the project encroach on this area, further studies will be required to assess the potential impact of the historical waste.

The project benefits from strong existing infrastructure and proximity to mining hubs such as Rouyn-Noranda and Val-d'Or, which support efficient access, labour, and services.

The off-site processing strategy significantly reduces the on-site environmental footprint and complexity of permitting, presenting a favourable profile from both an environmental and social perspective.

25.13 Capital Cost Estimate

The capital cost estimate conforms to Class 5 guidelines for a PEA-level estimate with a ±50% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The capital cost estimate was developed in Q2 2025 Canadian dollars based on Ausenco's in-house database of projects and studies as well as experience from similar operations.

The estimates are based on the following:

- underground mining operations and surface infrastructure at the O'Brien site
- transportation of mineralized material to the Westwood processing plant
- new equipment to be installed at the Westwood process plant
- processing costs at the Westwood process plant
- Owner's costs and provisions
- toll milling fees.

The capital cost estimate includes costs for underground mining, surface infrastructure at the O'Brien site, modifications to the Westwood complex process plant, project indirect costs, project delivery, Owner's costs, and contingency. The total initial capital cost for the O'Brien Gold Project is C$174.8 million, and life-of-mine sustaining costs are C$172.7 million. Closure costs for the O'Brien mine are estimated at C$5.3 million, and salvage credits at C$3.1 million.

25.14 Operating Cost Estimate

The overall life-of-mine operating cost, excluding toll milling fees, is C$660 million over 11 years for an average of C$144.35/t milled. The overall life-of-mine operating cost, including toll milling fees, is C$747 million over 11 years for an average of C$163.32/t milled. Table 25-3 provides a summary of the project operating costs.

Table 25-3: Operating Cost Summary

Cost Area	Life-of-Mine Total ($M)	Milled Cost ($/t)	% of Operating Cost Subtotal	% of Total Costs, with Toll
Mining	346	75.66	52	46
Process	173	37.71	26	23
G&A	142	31.06	22	19
Operating Cost Subtotal	661	144.44	100	N/A
Toll Milling Fees	87	18.94	N/A	12
Total with Toll Fees	**747**	**163.38**	**N/A**	**100**
Off-Site Costs, Refining & Transport	6	-	-	-
Royalties	10	-	-	-
Total Cash Costs	**763**	**-**	**-**	**-**

25.15 Economic Analysis

An engineering economic model was developed to estimate the project's annual pre-tax and post-tax flows and sensitivities based on an 5% discount rate. The economic analysis also used the following assumptions:

- a pre-production period of 21 months
- a mine life of 11 years
- 100% ownership
- capital cost funded with 100% equity (no financing cost assumed)
- cash flows that are discounted to the start of construction using an end-period discounting convention
- metal products that are sold in the same year they are produced
- project revenue derived from the sale of gold doré
- no existing contractual arrangements for refining.

The pre-tax NPV discounted at 5% is $782.5 million; the IRR is 65.1%; and payback period is 1.4 years. On a post-tax basis, the NPV discounted at 5% is $532.4 million; the IRR is 47.6%; and payback period is 2.0 years. The project is cash positive after-tax at gold prices above US$1,260/oz.

A sensitivity analysis was conducted on the base case post-tax NPV and IRR of the project using metal prices, foreign exchange rates, operating cost, and initial capital cost as the variables. The sensitivity analysis showed that the project is most sensitive to changes in gold price, head grade and foreign exchange

The preliminary economic assessment is preliminary in nature, that it includes inferred mineral resources that are considered too speculative geologically to have the economic considerations applied to them that would enable them to be categorized as mineral reserves, and there is no certainty that the preliminary economic assessment will be realized.

25.16 Risks and Opportunities

25.16.1 Risks

25.16.1.1 Mining Methods

Below are risks associated with mining:

- Stope Thickness – The minimum stope thickness of 1.5 m undiluted (or 2.2 m diluted) considered in the mine plan may appear narrower than what is commonly seen in other Abitibi deposits. However, similar stope dimensions have been successfully mined in Ontario, demonstrating the feasibility of this approach under appropriate conditions. It is important to note that the average stope thickness across the deposit is 2.0 m undiluted, or 2.7 m when incorporating ELOS, which aligns more closely with industry standards observed in the Abitibi region.
- Vertical Conveyor – Vertical conveyor systems, while successfully implemented in various mining operations worldwide, remain uncommon in the Abitibi region. This limited regional precedent introduces certain risks related to operational integration, workforce familiarity, and supplier support.
- Dilution Estimation – In narrow vein deposits, it is challenging to estimate dilution using empirical methods because the results are highly dependent on the proper execution of drilling and blasting plans. The historical mine was operated using conventional methods, and it would be appropriate to conduct a bulk sample to validate whether modern techniques are suitable for controlling dilution.
- Deposit Extent – Given the extent of the deposit and the significant development required to access the stopes, scheduling and overall performance will present a notable challenge in maintaining continuous production.
- Dewatering – Safe extraction of certain stopes near former excavations requires dewatering of historical mine areas. Should mine water quality or treatment capacity pose constraints, the resulting impact is expected to affect less than 2% of the estimate.
- Maintenance Bay Dimensions – The dimensions of the maintenance bay are limited relative to the equipment fleet, and its location could become problematic if the depth of the resource increases.
- Geomechanical Data – Additional geomechanical data is required to fully understand the behaviour of the rock mass at depth. Furthermore, since no development currently crosses the fault, the available data suggests limited issues. Until such development is carried out, it remains difficult to accurately assess.

25.16.1.2 Metallurgical Testing – Recovery

Metallurgical testwork conducted using the process of leaching, without gravity or flotation resulted in overall gold recovery results ranging from 53% to 99%. Final testing using the gravity/flotation/regrind/leach process selected for processing the O'Brien mineralized material resulted in an overall gold recovery of 90%, based on Tests 5 and 6 as described in Section 13. Continued testwork including variability testing should be conducted to reduce operational risk. Additional testwork should also include variability testing on sulphur/arsenic levels in the test feed to investigate the potential relationship between sulphur/arsenic levels and overall gold recovery.

The leaching of copper and arsenic into solution under the selected leaching conditions should also be tested to confirm that both the leachate is suitable for adsorption onto typical activated carbon as well as suitable for SO_2/air cyanide detoxification and/or arsenic precipitation if required.

The recovery model derived in this program was developed from the available body of testwork and should be confirmed for all lithologies and compared to variability testing as it is completed.

25.16.1.3 Recovery Methods

The condition of the Westwood facility was not assessed in detail as part of this design, and the proposed retrofit to the existing circuit should be examined in detail for ease of installation and operability. Other alternative configurations may be suitable for the project and minimize disruption to the operation of the Westwood facility during construction.

The mineralized material contains arsenic at concentrations ranging from 0.0% to 1.5%. Processing the master composite material resulted in tailings that contain arsenic at approximately 0.5%. The storage of arsenic-containing tailings in the existing Westwood tailings ponds has not been evaluated during this study; therefore, the storage of the tailings may require upgrades (or may not be permitted by the regulatory agency), and a newly constructed tailings pond may be required. Discussions with IAMGOLD indicated that milling scenarios should assume the use of the existing tailings facilities for the disposal of O'Brien tailings, but this is not based on an assessment of the geochemical properties of the tailings.

25.16.1.4 Power Availability

The process plant with the new flotation cells and regrind mill will have an estimated nominal demand of 5.1 MW. The estimated annual power consumption is 43 MWh/a. The additional installed power requirements of the regrind and flotation circuits are estimated to be approximately 1.8 MW. Discussions with IAMGOLD indicated that there is sufficient additional capacity available with the existing electrical infrastructure and with Hydro Quebec. Future studies should consider reviewing the electrical infrastructure of the Westwood complex in further detail to ensure sufficient capacity.

25.16.1.5 Water Management

The detailed design of the storage basins, drainage ditches, and emergency spillways will be prepared in subsequent phases of the project. In addition, the following risks have been identified and should also be addressed:

- The impact of extreme storm events on the groundwater ingress rate into the underground drifts needs to be considered.
- The impact of sedimentation on the storage capacity needs to be evaluated.
- Contact water in the underground mine could infiltrate fractured rock or unsealed areas.
- Geochemical testing is required to evaluate the contact and groundwater quality.

25.16.2 Opportunities

25.16.2.1 Mining

25.16.2.1.1 Alternative Mining Methods

For simplification, a single mining method was considered in this study: longitudinal retreat stoping. However, certain areas of the deposit exhibit more elongated and vertical geometries which could be well-suited to a semi-mechanized method such as Alimak mining.

While operating costs associated with Alimak mining are generally higher, further analysis would be required to determine whether these could be offset by a reduction in the amount of horizontal waste development. Recent examples in the Abitibi region have demonstrated the potential success of such methods in similar geological contexts. A comparative mining scenario analysis could help further optimize the overall mine plan.

25.16.2.1.2 Materials Handling

Given the current ramp dimensions, it would be feasible to operate with 30-tonne trucks instead of 20-tonne units. This increase in truck capacity could improve overall mine productivity, partially compensating for the removal of the vertical conveyor system.

Switching to a larger truck fleet would result in higher operating costs but could lead to a savings in capital costs by eliminating the need for a vertical conveyor. The development distance saved by removing the vertical conveyor could be reallocated to building a ramp that connects the bottom of the Kewagama sector to the core of the O’Brien East deposit.

This alternative would help alleviate potential traffic congestion in the main ramp, provide a well-positioned exploration drift for the geology, and contribute to simplifying the Kewagama dewatering network.

25.16.2.1.3 Definition Drilling

Further definition drilling should convert some of the existing inferred mineral resources to indicated or measured category. This will be a benefit for future technical studies.

25.16.2.1.4 Geomechanical Study

Further geomechanical work would increase the knowledge on the rock mass behaviour. This, in turn, would better define which ground control tactics to employ for improved dilution control, and provide guidance in selecting appropriate ground support supplies.

25.16.2.2 Metallurgical Testing

The recovery model is based on the average recovery observed in Tests 5 and 6 as described in section 13. A recovery curve was calculated based on a residual grade of 0.63 g/t regardless of the head grade of the feed material. Additional testwork should include variability testing to determine recovery at lower and higher grades to refine the recovery model. A refined recovery model may increase the recovery applied to lower gold grades in the economic analysis.

25.16.2.3 Recovery Methods

The proposed flowsheet at the Westwood complex for processing the O'Brien material was based on using the existing SAG and ball mills. An alternative flowsheet may be possible utilizing existing equipment and layout at the Westwood Complex.

There is an existing decommissioned 261 kW ball mill which may allow for simultaneous grinding of O'Brien material separately from the currently used SAG mill and ball mill. This scenario may provide improved economics if the grinding capacity of the mills can be fully utilized, and the two separate feed materials can be simultaneously processed through the existing leach circuit. This alternate scenario was not investigated in this technical report, but is recommended for review in future studies in concert with the mine plans for all IAMGOLD and Radisson deposits that may provide potential feed to the mill.

The flowsheet selected is typical of gold-bearing sulphide processing, and similar modifications can be replicated at alternative toll milling locations equipped with cyanide leaching circuits in the region such as Agnico Eagle's LaRonde property.

25.16.2.4 Primary Grind Size

Tests 5 and 6 of the 2024 SGS metallurgical testwork utilized a primary grind of 82 µm. The existing mill circuit at the Westwood complex can maintain a 3,000 t/d throughput at 75 µm, based on current comminution testwork. Future studies should include additional comminution testwork and variability testing of different primary grind sizes to optimize the throughput and recovery, based on different primary grind sizes.

25.16.2.5 Process Tie-ins and Downtime

The proposed site for the new equipment is where a decommissioned ball mill and flotation cells are currently located. Downtime associated with installing the new equipment is expected to be limited primarily to tie-ins. Future studies should consider how to optimize the new equipment tie-ins to minimize downtime to existing operations.

26 RECOMMENDATIONS

26.1 Introduction

The work carried out to date has justified the continued exploration and development of the project. The work described in this section is recommended to advance the project to a prefeasibility study level of engineering, with additional drilling and metallurgical testwork and an improved mineral resource estimate.

The estimated cost of the recommended program is $16.6 million (Table 26-1).

Table 26-1: Phase 1 Recommended Work Program and Budget

Program Component	Estimated Total Cost ($k)
Drilling	
Resource Expansion at Depth (35,000 m)	7,000
Phase 1 Mineral Resource Infill (25,000 m)	5,000
Drilling Contingency (10%)	1,200
Subtotal	13,200
Mining	
Perform a Prefeasibility-Level Hydrogeological Study	350
Perform a Prefeasibility-Level Geomechanical Study on the O'Brien and Kewagama Zones, as well as the Surface Crown Pillar	500
Perform a Groundwater Flow Study	100
Subtotal	950
Metallurgy	
Sample Preparation, Storage, Head Assays	42
Hardness Determination	81
Open Circuit Flotation Testwork	276
Locked Cycle Tests	180
Mineralogy / Liberation Analysis	100
Test Management / Supervision / Reporting	150
Metallurgy Contingency	124
Subtotal	954
Pre-Feasibility Study	**1,000**
Environmental Studies and Community Engagement	**500**
Grand Total	**16,604**

26.2 Geology, Exploration, Drilling, and Mineral Resources

Radisson has proposed an initial program with a total budget of $13.2 million, as presented in Table 26-2, to advance the project. A secondary phase would include additional drilling and continued engineering studies, infill drilling, additional exploration drilling, and is dependent upon results from the initial phase. The secondary phase budget will total approximately $30 million. The SLR QP concurs with the proposed geology and exploration program to advance the project.

Table 26-2: Geology and Exploration – Recommended Work Program & Budget

Work Description	Estimated Total Cost (C$k)
Drilling – Resource Expansion at Depth (35,000 m)	7,000
Drilling – Phase 1 Mineral Resource Infill (25,000 m)	5,000
Contingency (10%)	1,200
Grand Total	**13,200**

The QP recommends Radisson continue to develop and improve standard operating procedures (SOPs), including clear failure criteria and follow-up actions for QA/QC.

26.3 Mining Methods

Norda Stelo recommends that the work below be carried out to further develop the O'Brien Gold Project. The estimated total cost of the work is listed in Table 26-3.

- Perform a prefeasibility-level hydrogeological study to support next stage of the O'Brien Project dewatering design.
- Perform a prefeasibility-level geomechanical study, including the O'Brien East and Kewagama zones, to support the next stage of project design.
- Perform a groundwater field investigation, instrumentation, and preliminary analysis to confirm the comparable site assumptions used in this study. This will serve as a basis for the future feasibility study modelling.

Table 26-3: Mining – Recommended Work Program & Budget

Work Description	Estimated Total Cost (C$k)
Prefeasibility-Level Hydrogeological Study	*350
Prefeasibility-Level Geomechanical Study	*500
Groundwater Flow Study	100
Total	**950**

Note: *Excludes drilling cost.

26.4 Metallurgical Testing and Recovery Methods

Work should be conducted to further variability testwork and to potentially improve project economics as part of a pre-feasibility study. A pre-feasibility level metallurgical testwork plan is costed in Table 26-4 and recommended to include the following:

- samples reflecting the different styles and geological settings of mineralization to test recoveries near cut-off grade, as well as higher-grade areas of the resources; samples need to also reflect spatial distribution of each deposit and potential underground mineable mineralization
- variability testwork with different arsenic and sulphur content by lithology, based on the selected flowsheet
- comprehensive comminution testing including Bond rod and ball mill work indices, SMC testing and abrasion index tests
- leach tests with primary grind size and regrind size optimization
- regrind energy tests on the concentrates at the optimum grind sizes to confirm the capabilities of the regrind mills.

Table 26-4: Metallurgical Testwork – Recommended Work Program & Budget

Activity	Purpose	Units	Quantity	Estimated Cost (C$k)
Sample Preparation, Storage, Head Assays	Sizing for subsequent test phases, metal contents	Samples	15	42
Hardness Determination	For SMC, rod and ball mill work indices; signature plot for regrinding	Samples	15	81
Open Circuit Flotation Testwork	Optimization of reagent regime, selection of regrind size, retention times	Tests	45	276
Locked Cycle Tests	Confirmation of open circuit indication, effect of circulating loads on circuit performance	Tests	10	180
Mineralogy / Liberation Analysis	Confirm minerals, grain size, precious metals deportment basis	Samples	15 heads 30 tails	100
Test Management / Supervision / Reporting	-	%	22	150
Contingency	-	%	15	124
Total		-	-	**954**

26.4.1 Recovery Model Grade Variability Testwork

Additional metallurgical testwork is recommended to refine the recovery model, particularly regarding feed grade. The current recovery model is based on a residual gold grade of 0.63 g/t in the tailings. Variability testing may allow for an optimized recovery model which may indicate a lower gold residual grade when feeding lower grade material. The lower residual would correspond with a higher recovery at lower feed grade than is currently observed in the recovery model.

26.4.2 Recovery Model Primary Grind Variability Testwork

It is recommended that additional variability testwork of different primary grind sizes be carried out to optimize throughput and recovery for the existing Westwood complex process plant. This testwork would be enhanced with additional comminution testwork to refine the modelling of the mill throughput.

26.4.3 Regrind Particle Size

It is recommended that additional leach tests be conducted on variable particle sizes of flotation concentrate regrind to determine the effect on gold recovery. The regrind size should be economically optimized to balance capital and operating costs of the regrind mill against additional revenue from gold recovery.

26.4.4 Arsenic Content

Arsenic is identified as potentially being associated with lower recovery. It is also a potential concern for using the existing tailings facilities. Mining low-arsenic mineralized material may provide the opportunity to increase average recovery and reduce the risk associated with tailings. It is recommended that arsenic concentrations in future assays and mining models be considered explore this opportunity.

Samples are assumed to be readily available from exploration and infill drilling works.

26.5 Infrastructure

Decommissioned flotation cells and a decommissioned ball mill currently stand on the site of the proposed location for the new process equipment. It is recommended that future studies review the work required to install the new equipment and equipment tie-ins, and to optimize construction scheduling to minimize downtime to existing operations.

26.6 Environmental, Permitting, Social and Community Responsibility

Continued environmental work is recommended to support the project as it advances through economic and development studies.

A consultation plan should also be developed to support project development.

Complementary kinetic geochemical testing should be conducted to confirm that most of the waste rock is not metal leaching. Additional static tests on waste rock samples are recommended to more accurately assess sulphur, arsenic, and manganese content, as well as other parameters.

Kinetic tests are also suggested to better define the acidification potential and leaching risk of all mining materials.

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