O3 Mining

Marban Engineering Project

NI 43-101 Technical Report & Prefeasibility Study

Val-d'Or, Quebec, Canada

Effective Date: August 24, 2022

Prepared for:
O3 Mining Inc.
1440-155 University Ave.
Toronto, Ontario, Canada, M5H 3B7

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

List of Qualified Persons:

Renee Barrette, ing., Ausenco Engineering Canada, Inc.
James Purchase, P. Geo., G Mining Services Inc.
Carl Michaud, P. Eng., G Mining Services Inc.
Ali Hooshiar, P. Eng., Ausenco Engineering Canada, Inc.
Davood Hasanloo, P. Eng., Ausenco Engineering Canada, Inc.
Andréanne Hamel, ing., M.Sc., WSP Canada Inc.

CERTIFICATE OF QUALIFIED PERSON
Renee Barrette

I, Renee Barrette, ing., certify that I am employed as a Principal Metallurgist with Ausenco Engineering Canada Inc. ("Ausenco"), with an office address of Suite 1550 - 11 King St West, Toronto, ON M5H 4C7. This certificate applies to the technical report titled "NI 43-101 Technical Report and Prefeasibility Study for Marban Engineering, Val-d'Or, Quebec," that has an effective date of August 24, 2022 (the "Technical Report").

I graduated from Laurentian University with a Bachelor of Applied Science degree in Extractive Metallurgical Engineering, in 2001. I am a Professional Engineer registered with OIQ (No. 6019759). I have practiced my profession continuously for over 21 years with experience 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.

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.

I have not visited the Marban Engineering Project. I am responsible for Sections 1.1, 1.2, 1.3, 1.4, 1.12, 1.16, 1.17.1, 1.17.2, 1.17.3, 1.18, 1.20, 1.21, 1.22.1.3, 1.22.2.4, 1.23, 1.24.2, 2, 3, 4, 13, 17, 18.1, 18.2, 18.3, 18.4, 18.5, 19, 21.1, 21.2.1, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.2.8, 21.2.9, 21.2.10, 21.2.11, 21.2.12, 21.3.1, 21.4.1, 21.4.3, 21.4.4, 22, 24, 25.1, 25.4, 25.7, 25.10, 25.11, 25.12, 25.13, 25.14.1.3, 25.14.2.4, 25.15, 26.1, 26.6, and 27 of the Technical Report.

I am independent of O3 Mining Inc. as independence is defined in Section 1.5 of NI 43-101.

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: October 7, 2022

"Original signed and sealed"

Renee Barrette, ing.

7900 W. Taschereau Blvd, Building D, Suite 200 Brossard, Québec Canada J4X 1C2
Tel: 450.465.1950 | Fax: 450.465.6344
www.Gmining.com

CERTIFICATE OF QUALIFIED PERSON
James Purchase

I, James Purchase, P. Geo., certify that I am employed as the Vice-president of Geology and Resources for G Mining Services Inc., with an office at 7900 Taschereau Blvd, Building D, Suite 200, Brossard, Quebec, Canada, J4X 1C2. This certificate applies to the technical report titled "NI 43-101 Technical Report and Prefeasibility Study for Marban Engineering, Val-d'Or, Quebec," that has an effective date of August 24, 2022 (the "Technical Report").

I graduated from the University of Liverpool, UK with a B.Sc. in Geology in 2006. I am a Professional Geologist registered with the "Ordre des Géologues du Québec" (OGQ-Licence: #2082). I have worked as a geologist for a total of 14 years since my graduation. I have practiced my profession continuously since 2008 and have extensive experience in mineral exploration and mineral resource estimation for various commodities in Australasia, North and South America, and West Africa, with a strong focus on orogenic gold deposits. Prior relevant experience involves undertaking mineral resource estimations for the Hardrock Project and Detour Mine in Ontario, the Essakane Mine in Burkina Faso, the Tocantinzinho project in Brazil and the Touquoy Mine in Nova Scotia. I have worked in my current role with G Mining Services Inc. since February 2017.

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.

I visited the Marban Engineering Project from September 8 to September 10, 2021; for a visit duration of 3 days. I am responsible for Sections 1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.13, 1.22.1.1, 1.22.2.1, 1.22.2.2, 1.24.1, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.5, 25.14.1.1, 25.14.2.1, 25.14.2.2, 26.2, and 26.3 of the Technical Report.

I am independent of O3 Mining Inc. as independence is defined in Section 1.5 of NI 43-101. I have prior involvement with the project since March 2021 part-time as a consultant involved in the planning of infill drilling, geological modelling, and geological support.

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: October 7, 2022

"Original signed and sealed"

James Purchase, P. Geo.

7900 W. Taschereau Blvd, Building D, Suite 200 Brossard, Québec Canada J4X 1C2
Tel: 450.465.1950 | Fax: 450.465.6344
www.Gmining.com

CERTIFICATE OF QUALIFIED PERSON
Carl Michaud

To accompany the Report entitled: "NI 43-101 Technical Report and Prefeasibility Study for Marban Engineering, Val-d'Or, Quebec", prepared for O3 Mining Inc. effective as of August 24, 2022 (the "Technical Report").

I, Carl Michaud, do hereby certify that:

1) I am currently employed as Vice President of Mining Engineering with G Mining Services Inc. in an office located at 7900, W. Taschereau Blvd, Building D, Suite 200, Brossard, Quebec, J4X 1C2.
2) I have graduated from Université Laval, Canada with a B.Sc. in Mining Engineering in 1996, and from Université du Québec à Chicoutimi, Canada with an M.B.A. in 2012.
3) I am a Professional Engineer registered with the Ordre des ingénieurs du Québec, (OIQ Licence: 117090).
4) I have practiced my profession continuously in the mining industry since my graduation from university. I have been involved in mining operations, engineering and financial evaluations for 26 years. I have occupied different positions, both technical and operational, related to mining engineering, in Underground and Open pit operation. This experience includes Kiena and Sigma Gold mine (Placer Dome), Éléonore Mine (Goldcorp) and Mont Wright Mine (Arcelor Mittal).
5) I have read the definition of "qualified person" set out in the National Instrument 43-101 ("NI 43-101") and certify that by reason of my education, affiliation with a professional association and past relevant work experience, I fulfil the requirements to be a qualified person for the purposes of NI 43-101.
6) I have participated in the preparation of the Technical Report and am responsible for Sections 1.14, 1.15, 1.20, 1.22.1.2, 1.22.2.3, 1.23, 1.24.3, 15.1, 15.2, 15.3, 15.5, 15.6, 15.7, 15.8, 15.9, 15.10, 16, 21.1, 21.2.2, 21.3.2, 21.4.2, 25.6, 25.11, 25.12, 25.14.1.2, 25.14.2.3, 25.15, and 26.4 of the Technical Report.
7) I have not visited the site property that is the subject of this report.
8) 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.
9) I have read NI 43-101 and believe that the sections and sub-sections of the Technical Report listed in item 6 above have been prepared in accordance with NI 43-101.
10) I am independent of O3 Mining Inc. as "independence" is defined in Section 1.5 of NI 43-101. I have not had prior involvement with the property that is the subject of the Technical Report.

Dated: October 7, 2022

"Original signed and sealed"

Carl Michaud, P. Eng

CERTIFICATE OF QUALIFIED PERSON
Mohammad Ali Hooshiar Fard

I, Mohammad Ali Hooshiar Fard, P. Eng., certify that I am a Professional Engineer, currently employed as Geotechnical Engineer, with Ausenco Engineering Canada Inc. (Ausenco), with an office at 1050 W Pender St, Vancouver, BC V6E 3S7. This certificate applies to the technical report titled "NI 43-101 Technical Report and Prefeasibility Study for Marban Engineering, Val-d'Or, Quebec," that has an effective date of August 24, 2022 (the "Technical Report").

I graduated from Sharif University of Technology with BSc and MSc in Materials Science and Engineering in 2003 and 2006, respectively, and the University of Alberta in 2011 with a PhD in Materials Engineering. I am a Professional Engineer registered with the Engineers and Geoscientists British Columbia (No. 40965), Engineers Yukon, and Ordre des Ingénieurs du Québec (No. 6043599). I have practiced my profession for 19 years with experience in designing tailings and waste rock storage facilities as well as managing geotechnical field investigation and lab testing programs for mining projects across the globe. A summary of the more recent portion of my professional career is as follows:

- Geotechnical Mining Engineer, Ausenco, Canada 2018–present
- Geotechnical Mining Engineer, AECOM, Canada 2013–2017
- Senior Geotechnical Consultant, SRK Consulting Inc., Canada 2011–2013

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.

I visited the Marban Project on May 27, 2022. I am responsible for Sections 1.17.6, 1.17.7, 1.22.1.4, 1.22.2.5, 1.24.4, 1.24.7, 15.4, 18.7, 18.8, 25.8, 25.14.1.4, 25.14.2.5, 26.5, 26.9, and 26.10 of the Technical Report.

I am independent of O3 Mining Inc. as independence is defined in Section 1.5 of NI 43-101.

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: September 6, 2022

"Original signed and sealed"

Mohammad Ali Hooshiar Fard, P. Eng

CERTIFICATE OF QUALIFIED PERSON
Davood Hasanloo

I, Davood Hasanloo, P. Eng., certify that I am a Professional Engineer, currently employed as Senior Water Process Engineer, with Ausenco Engineering Canada Inc., with an office at 1050 W Pender St, Vancouver, BC V6E 3S7. This certificate applies to the technical report titled "NI 43-101 Technical Report & Mineral Resource Estimate for the Marban Engineering Project," that has an effective date of August 24, 2022 (the "Technical Report").

I graduated from the Chamran University with a Bachelor of Science in Civil engineering in 2006 and University of British Columbia with a Master of Applied Science degree in Hydrotechnical Engineering. I am a Professional Engineer, registered with Ordre des Ingénieurs du Québec, member number 60275. I have practiced my profession continuously since 2009 and have been involved in hydrotechnical analysis and water resources engineering related to mining projects dealing with sitewide water management and water management design. Example of projects I've been involved with include: Wasamac Gold Mine FS, Eskay Creek Mine FS, Teck Elkview Operations Drainage and Sediment Mitigation Design, Chvaletice Mine FS, Las Chispas Mine detailed design, New Polaris Mine FS, etc.

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.

I have not visited the Marban Engineering Project site. I am responsible for Sections 1.17.4, 1.17.5, 1.22.1.5, 1.24.5, 1.24.6, 18.6, 25.14.1.5, 26.7, 26.8 of the Technical Report

I am independent of 03 Mining Inc. as independence is described by Section 1.5 of the NI 43-101. I have had no previous involvement with the Marban Engineering Project.

I have read the Instrument and that 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: October 7, 2022

"Original signed and sealed"

Davood Hasanloo, P. Eng

CERTIFICATE OF QUALIFIED PERSON
Andréanne Hamel, ing. M.Sc.

I, Andréanne Hamel, ing., M.Sc., certify that I am employed as a Project Director with WSP Canada Inc with an office address of 1135 Lebourgneuf Québec City, Qc. This certificate applies to the technical report titled "NI 43-101 Technical Report and Prefeasibility Study for Marban Engineering, Val-d'Or, Quebec," that has an effective date of August 24, 2022 (the "Technical Report").

I graduated from UQAM university of Montreal with a B.Sc. in Geology (1996) and from Laval University of Quebec City with a B.ing. (1999) and M.Sc. (2002) in geological engineering. I am a member of the Ordre des ingénieurs du Québec (OIQ No. 128249). I have practiced my profession since 2002 for a total of 20 years since my graduation. I have significant experience in environmental impact assessments (EIAs) for mining projects in different regions in the province of Quebec. For this project, I have been directly involved in the environmental assessments, baseline studies and permitting requirements.

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 for which I am responsible for preparing.

I have not visited the Marban project site. I am responsible for Sections 1.19, 1.22.1.6, 1.24.8, 20, 25.9, 25.14.1.6, and 26.11 of the Technical Report.

I am independent of O3 Mining Inc. as independence is defined in Section 1.5 of NI 43-101. I have had minor involvement with the project since March 2021 on a part-time basis. I contributed to the planning of hydrogeological and geotechnical campaigns and environmental baseline studies by WSP.

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: October 7, 2022

"Signed and sealed"

Andréanne Hamel, ing. M.Sc.

Important Notice

This report was prepared as a National Instrument 43-101 Technical Report for O3 Mining Inc. (O3 Mining) by Ausenco Engineering Canada Inc. (Ausenco), G Mining Services Inc. (GMS), WSP Canada Inc. (WSP), 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 O3 Mining Inc. subject to terms and conditions of its contracts with each of the Report Authors. Except for the purposes 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.

Table of Contents

List of Tables

List of Figures

1 SUMMARY

1.1 Introduction

Ausenco Engineering Canada Inc. (Ausenco) has prepared a prefeasibility study (PFS) and associated technical report for O3 Mining Inc. (O3 Mining) on the Marban Engineering Project located in Val-d'Or, Quebec. The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1.

The responsibilities of the engineering consultants are as follows:

- Ausenco was commissioned by O3 Mining to manage and coordinate the work related to the PFS and technical report. Ausenco has also completed the metallurgy and process design, design of major site and tailings infrastructure (except for the mining stockpile and facilities) and compiled the overall cost estimate and financial model.
- G Mining Services Inc. (GMS) was commissioned to review the infill drilling campaign and associated quality assurance/quality control (QA/QC) data, update the geological model and mineral resource, produce a mineral reserve, and complete the mining design based on the mineral reserve.
- WSP Canada Inc. (WSP) has completed the geotechnical drilling programs and environmental fieldwork for the PFS.

1.2 Terms of Reference

The report supports disclosures by O3 Mining in a news release dated 06 September 2022 entitled "O3 Mining Completes Prefeasibility Study for Marban Engineering with Post-Tax NPV of C$463 Million, Unlevered IRR of 23.2% and Annual Production of Over 160Koz Gold."

All measurement units used in this report are SI units unless otherwise noted. Currency is expressed in Canadian dollars (C$ or CAD) unless otherwise noted.

Mineral resources and mineral reserves are reported in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The Marban property contains six past-producing mines (Marban, Norlartic, Kierens, Malartic Hygrade, Orion No. 8 and Camflo (below 800 metres (m) from the surface)), which collectively produced 879,000 ounces of gold between 1959 and 1992. O3 Mining owns 100% of the Marban property.

1.3 Property Description and Location

The Marban property is located in the western portion of the province of Quebec, Canada, and is comprised of 181 mining claims covering 7,701.97 hectares (ha). The property lies at the junction of Dubuisson, Fournière, Malartic, and Vassan townships, centered at latitude 48.17°N, longitude 78.05°W, on NTS sheets 32D/01 and 32C/04.

The Marban property is divided into two projects. A first one, Marban Engineering, corresponds to the southeast portion of the property and includes all the infrastructures related to this economic study. A second one, Marban Greenfield, is broader and includes all the north and west parts of the Marban property.

1.4 Accessibility, Climate, Local Resources, Infrastructure & Physiography

The project is accessible by the Gervais Road, a well-maintained, all-weather gravel road running north from Highway 117. It provides access to the historic Norlartic, Kierens, and Marban mines. The Camflo Road provides access to the western part of the project area. Winter access for snowmobiles and all-terrain vehicles is provided by trails and winter roads.

The Marban property is part of the Harricana River watershed, which is part of the James Bay hydrographic basin. The mean elevation of the property is about 300 metres above sea level (masl). The property is relatively flat, with local relief of 15 to 30 m. The surface is dominated by swamps and glacial deposits; a north-trending esker lies on the west side of the Gervais Road. The overburden is generally between 10 and 40 m thick, but locally it is very shallow and there are some outcrops (Ducharme et al., 2009).

The Canadian National Railroad (CNR) and a power line run parallel to Highway 117 immediately south of the property, and mine sites have access to power and telecommunication systems. A 120-kilovolt (kV) powerline crosses through the Marban Engineering Project property. The communities of Malartic and Val-d'Or, each within 15 km of the property, provide full services for exploration and mining, including a labour force with mining experience.

1.5 History

Exploration conducted at the Marban property dates back to at least 1940 and includes geologic mapping, sampling, compilation of geological, structural, and geochemical data, geophysical prospecting, trenching, and extensive drilling from the surface and underground. At least 14 different companies explored and/or mined parts of the property from 1940 through 1994.

Prior to O3's involvement in the project, NioGold Mining Corporation (NioGold) gained interests in the Marban property in 2006, including the work of the Aurizon-NioGold joint venture, and carried out the following activities at the Marban property:

- construction of three-dimensional computer models of the historic underground workings
- completion of high-resolution airborne magnetic surveying
- orientation induced-polarization surveying
- petrographic studies of the gold mineralization at the North-North and Marban zones
- drilling of 954 holes (representing 281,217 m of drilling)
- metallurgical testwork.

In March 2016, Niogold was acquired by Osisko Mining who drilled 26 holes (totalling 15,171 m) in 2016 and 2017 into the extensions at depth of the Marban, Norlartic, and Kierens deposits. The drillholes were widely spaced at 400 m to test the concept of a large-volume, low-grade ore body at depth. The deepest drillhole reached 1,475 vertical metres. In July 2019, O3 Mining was created, and the Marban Property was transferred from Osisko Mining into the O3 Mining project portfolio.

1.6 Geological Setting & Mineralisation

The Marban property lies within the Archean Abitibi greenstone belt of the Superior Province, Quebec, which consists of alternating east-trending metavolcanic-plutonic and sedimentary belts that are bounded by crustal-scale faults. The Abitibi belt has been divided into a Northern Volcanic Zone (NVZ) and a Southern Volcanic Zone (SVZ) (Chown et al., 1992). The Marban property is located in the southern portion of the SVZ, where the Parfouru fault separate the Blake River segment to the west from Malartic segment to the east (Daigneault et al., 2002). The western portion of the property contains the eastern end of the Blake River Group, that appear as a north-dipping panel with faulted contacts bordered with the sedimentary units of the Kewagama to the north and Cadillac to the south (De Souza et al., 2020). To the east, the Malartic segment is subdivided into the Malartic Group, plume-derived komatiitic-tholeiitic marine-plain volcanic assemblages

divided from north to south, into the La Motte-Vassan, Dubuisson, and Jacola formations and the Louvicourt Group representing an arc-type complex subdivided into the Val-d'Or Formation, a transitional to calcalkaline volcanic complex, and the Heva Formation, characterized by geochemically distinct iron tholeiites.

The metavolcanic rocks within the Marban property are cut by three major northwest- to west-northwest-striking shear zones of regional extent—the North, Norbenite, and Marbenite shears. The Marbenite shear hosts the Marban deposit, while the Norbenite shear hosts both the Kierens and Norlartic gold deposits and the North shear hosts the North zone.

The Marban deposit, one of numerous gold deposits on the greater Marban Property, is located at the Marbenite shear and extends in the hanging wall rocks for several hundreds of metres. It sits immediately to the east of a curvature in the Marbenite which is trending more east-west in this particular area. The southernmost part of the mineralisation is hosted by a strongly sheared komatiite unit within the Jacola Formation. Going north, the Mine Sequence corresponds to basaltic volcanics hosting most of the gold mineralization of the deposit. The mafic unit in the Marban deposit presents a significant thickening, compared to the correlated basalt horizon east and west, due to a multi-phased folding that shows a doubly plunging fold axis interference pattern which implies an early fold phase overprinted by subsequent east-west folding. The basaltic unit can be geochemically divided into two different basalts. A magnesium-rich basalt to the south that overlay a ferriferous basalt to the northern part of the Mine Sequence.

The Marban deposit is also characterized by minor cross-cutting shears that link early fold hinges together. The shears are plunging 40° to 70° to the north and often host gold zones. North of the basaltic units, the komatiite is injected by multiple felsic dykes, namely the Marban Dyke area. Those dykes are locally strongly altered and sheared, and contain gold mineralisation.

Gold mineralization at the Marban deposit occurs primarily within the Mine Sequence basalts mostly within the ferriferous basalt or at its folded contact with the magnesian basalt. The mineralization consists of quartz and quartz-carbonate-chlorite veins and veinlets with disseminated pyrite and pyrrhotite mostly within the carbonatized, chloritized and albitized wall rock. Veins and veinlets vary in thickness from one centimetre to few metres, but they form stockworks that can be up to 70 m thick. Mineralization is also hosted within the north-dipping shear zones and associated felsic to intermediate sills and dykes.

The Kierens and Norlartic deposits are localized along the Norbenite shear, an important northwest trending deformation zone that dips moderately to steeply northeast. Komatiites, along with mafic and intermediate dykes, are included within the shear zone. Some late, weakly deformed felsic dykes locally cut the previous units. The gold mineralization is closely associated with carbonatized, albitized and pyritized intermediate intrusions that contain a stockwork of quartz-carbonate veinlets.

The North Zone gold mineralization are emplaced as sub-parallel zones dipping 60º to the northeast recognized within the north shear and are confined to zones of quartz-carbonate veining and pyrite alteration within sheared iron-rich tholeiitic basalt (Stuart and Martin, 1988). Intermediate dykes across the deposit seem to be spatially correlated with the different zones but the gold still mostly remains in the hosting mafic volcanic.

The North-North Zone is a near-surface intrusive-hosted deposit with mineralized quartz-tourmaline stockwork. Gold mineralization is confined to a conformable quartz-albite-carbonate-pyrite alteration envelope with quartz-tourmaline-carbonate vein stockwork localized in the central portions of a 60 m wide granodiorite sill. The sill was emplaced within a sequence of deformed ultramafic and mafic volcanic rocks. The sill and alteration envelope strike northwesterly and dip 40° to 55º to the northeast.

1.7 Deposit Types

The various mineralization styles present on the Marban property can be characterized as sub-types of the orogenic class of gold deposits, most commonly shear-zone hosted gold mineralization within greenstone terrains.

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 terrains 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). They are also known as mesothermal, orogenic. 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.

1.8 Exploration

Since the creation of O3 Mining, exploration work on the Marban property has focused mainly on exploration drilling. Some other exploration works have been carried out by O3.

In May 2021, an air-borne drone magnetometric survey was completed covering the Camflo, Orion, Malartic Hygrade, and Malartic H areas. Vision 4K inc. performed the survey using the AIM-LOW™ system. A second drone survey was completed in March 2022 and covered all the western portion of the property with flight lines spaced at 50 m; in total, 712 linear km were flown.

In November 2021, Abitibi Geophysique Inc. undertook a Orevision® induced polarization (IP) survey on nine lines covering the Marban, Norlartic, Kierens, Gold Hawk, and Orion deposits. The purpose of the survey was to validate whether the known mineralization present at those deposits had distinguishable IP responses. Lines were between 400 and 1,000 m apart and 1 to 3 km long.

In July 2021, a soil and spruce bark survey was carried out over mineralized and barren areas. The survey covered Marban, Norlartic, Kierens, Gold Hawk, and Orion deposits. Eighty-five soil samples and 74 bark samples were collected along five lines with a sample spacing of 25 m along the lines.

In July 2021, the Orion No. 8 outcrop was sampled by 12 channel that totalled 136 samples covering 131 m. The surface map of Blanchette (2016) has been updated.

From June 2022 to September 2022 a soil survey and prospection were carried out on the western portion of the property. In total, 4,135 soil samples, 284 channel samples and 50 grab samples were taken. All assays are pending for that campaign.

1.9 Drilling

Drilling at Marban is mainly oriented 180° to 220° with dip generally between 45° and 65°. This orientation is more or less perpendicular to the main trend of mineralisation. Conversely, the historic underground drillholes have highly variable azimuths and dips, as they were drilled in series of fans from individual drill stations, which creates a multitude of angles between drillholes and mineralization. Some of them may then be considered parallel to the mineralization and be not representative of true mineralized width. However, this impact is mitigated by the modelling technique that constrains all intercepts within interpreted domains that respect the true mineralized widths.

There are three main generations of drilling at Marban; Historical (Pre 2006), Niogold/Osisko-Era (2006–2017) and O3 Mining era (2020 to present). Most recent drilling has been conducted using either NQ or HQ core sizes, with occasional telescoping to BQ when multiple stopes were traversed.

The drill-hole data were reviewed in the context of the geology of the Marban, Kierens, and Norlartic mineralization, and the sample collection methods are appropriate for the style of mineralization in each target area.

1.10 Sample Preparation, Analyses & Security

The sample preparation, analysis, and security procedures used by NioGold and Osisko are considered to be adequate, although blanks and certified reference material (CRM) data during the Niogold-era show a high failure rate. Documentation of the procedures employed in the pre-NioGold drilling programs is lacking. The drillhole data subsequent to the 1960's drilling that is lacking QA/QC has been validated through analysis and comparison of re-assays as discussed in Section 11. Sampling by O3 Mining since 2020 conforms to best industry practice and has undergone rigorous QA/QC verifications for inclusion into the Mineral Resource Estimate (MRE).

1.11 Data Verification

Mr. James Purchase, P.Geo and QP, visited the site in September 2021, and has subsequently checked certificate checks, undertaken numerous statistical analyses, particularly to validate the pre-NioGold assays (pre-2006). In the opinion of the QP:

- The drillhole database appears to be complete and no errors were discovered upon comparison to the certificates.
- The Marban historic data from the 1980s and earlier has not been included in the data set for resource estimation due to detection limit, precision issues, and a demonstrated bias at grades between 0–1 gram per tonne (g/t).
- The Kierens-Norlartic historic data earlier than 1986 has shown to be biased (for the same reasons as Marban) and unreliable and has not been included in resource estimation.
- The Marban data since the 1980s and the Kierens-Norlartic data from 1986 forward is considered of acceptable quality for resource estimation. In addition, any re-assayed drillholes from the excluded period have been included in the resource estimate.
- All assay data from the 2020–2022 drilling campaign by O3 has been included in the resource estimate.

1.12 Mineral Processing and Metallurgical Testing

Metallurgical testwork programs were conducted on samples from Osisko between 2015 and 2017, as described in Section 13.2. The testwork programs were performed on the Marban and Norlartic deposits.

The following historical sources of technical and project information were referenced in developing the process plant design for the prefeasibility study, along with the current PFS metallurgical testwork program:

- SGS, The Grinding Circuit Design Based on Ten Bench-Scale Grindability Test results from the Marban Engineering Project report 149647-003, 2016.

The metallurgical program for the prefeasibility study was conducted in March 2022 at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as project BL886, and was performed on composites from Marban, Norlartic, Kierens, North Zone, and Gold Hawk Zone deposits.

The testwork program included two Marban and one Norlartic major composite samples for head analysis, bulk mineralogy, gravity recoverable gold evaluation and leach testing development. Thirteen variability samples were included for evaluation of optimized gravity and leach conditions. A Marban/Norlartic master composite was used for bulk leach testing, cyanide detoxification, solid-liquid separation, and pressure filtration testing. Problematic elements such as copper and arsenic are at low concentrations and will not pose any metallurgical issues.

The comminution testing program included four Marban and two Norlartic samples for Bond ball mill work index and Bond abrasion index tests. An additional three variability samples from smaller zones were included for Bond ball mill work index tests. The comminution test results placed the Marban and Norlartic samples into the medium hard category with low to moderate abrasiveness. The North Zone composite would be considered soft.

Extended gravity recoverable gold (E-GRG) tests were completed on the two Marban and Norlartic primary composite samples. The test results showed high levels of gravity recoverable gold, which is indicative of plant scale gravity gold amenability. The E-GRG test results demonstrate that samples are amenable to gravity concentration in the grinding circuit to remove coarse free gold prior to leaching. Results indicated highly variable gravity recoveries with range spanning from 9% to 41%, with an average gold recovery of 26%.

The whole ore leach tests on the two composites displayed gold extractions, that ranged from 97.6% to 98.9% for the two Marban composites, and one result of 90.4% for the Norlartic composite. The leach variability samples were all tested using the optimized conditions from the primary composites. In general, leach residue grades increased with increasing head grades. The Norlartic samples result in higher leach residue grades than the Marban samples for similar head grades.

The gravity leach test results were analyzed to provide a recovery model for use with the mine production schedule to provide gold recovery and production data. Recoveries for material from the Marban and Kierens pits are estimated as 94.9% based on the recovery model produced from the gravity leach test results, whereas the equation below was derived to predict plant gold recovery for the material from the Norlartic pit:

$$Norlartic\ Recovery = \frac{Head\ grade\ (g/t) - 0.09}{Head\ Grade\ (g/t)} * 100 + 1.5\% - 0.5\%$$

$$Marban\ and\ Kierens\ Recovery = 94.9\%$$

1.13 Mineral Resource Estimate

The mineral resource estimate for O3 Mining's Marban and Kierens-Norlartic deposits have been updated since the previous estimate in 2020 due to a significant infill drilling campaign, an updated geologic model, and updated economic assumptions. The mineral resource with an effective date of 27 February 2022 is listed in Table 1-1. CIM Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the updated CIM Guidelines (CIM, 2019) were followed for the mineral resource estimate.

The base case cut-off grade is 0.30 g/t Au based on a recovery of 93.7%, processing + G&A costs of C$18.2/tonne, mining cost of $2.40/tonne and a US$1900/oz Au price, with smelter terms as detailed in the notes below. The underground resource has been filtered to only include contiguous blocks above the underground cut-off at sufficient widths to demonstrate reasonable prospects of eventual economic extraction (RPEEE). At these cut-offs, the total indicated mineral resource is estimated at 67,692 kt at a grade of 1.09 g/t Au for a total of 2,374 koz, and the inferred mineral resource is estimated at 3,149 kt at a grade of 2.21 g/t Au for a total of 223 koz.

These mineral resources are not mineral reserves as they have not demonstrated economic viability. The quantity and grade of reported inferred mineral resources in this news release are uncertain in nature and there has been insufficient exploration to define these resources as indicated or measured; however, it is reasonably expected that the majority of inferred mineral resources could be upgraded to indicated mineral resources with continued exploration.

Mr. James Purchase, P.Geo., is not aware of any factors or issues that materially affect the mineral resource estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors and additional risk factors regarding indicated and inferred resources.

Table 1-1: Mineral Resource Estimate (effective date: February 27, 2022)

Deposit		Indicated			Inferred		
		Tonnes (kt)	Gold (g/t)	Ounces (koz)	Tonnes (kt)	Gold (g/t)	Ounces (koz)
Open Pit (OP)	Marban	52,437	1.03	1,736	1,038	0.97	32
	Kierens - Norlartic	14,795	1.22	582	1,068	1.42	49
	Total	67,232	1.07	2,318	2,106	1.20	81
Underground (UG)	Marban	162	4.47	23	860	4.43	123
	Kierens - Norlartic	297	3.36	32	182	3.36	20
	Total	460	3.75	55	1,043	4.25	142
Combined Mineral Resources - OP and UG		67,692	1.09	2,374	3,149	2.21	223

Notes: **1.** The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019). **2.** The QP for this mineral resource estimate is James Purchase, P. Geo of G Mining Services Inc. Mr. Purchase is a member of L'Ordre des Géologues du Québec (#2082). **3.** The effective date of the mineral resource estimate is 27 February 2022. **4.** The lower cut-off used to report open-pit mineral resources is 0.30 g/t Au. Underground mineral resources have been reported using a 3.0 g/t lower cut-off at Marban, and a 2.5 g/t lower cut-off at Kierens-Norlartic. **5.** The Marban and Kierens-Norlartic deposits have been classified as indicated and inferred mineral resources according to drilling spacing and estimation pass. No measured resource has been estimated. Underground mineral resources have been categorized manually to remove isolated areas and have been reported using a 3 m minimum thickness. **6.** Known underground works have been incorporated into the block model, and zero density has been assigned to the blocks located within the voids. **7.** The density has been applied based on measurements taken on drill core and assigned in the block model by lithology. **8.** In general, a minimum thickness of 3 m was used when interpreting the mineralized bodies. **9.** The MRE is based on subblock models with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 2.5 m x 2.5 m. **10.** Tonnage have been expressed in the metric system, and gold metal content has been expressed in troy ounces. **11.** The tonnages have been rounded to the nearest 1,000 tonne and the metal content has been rounded to the nearest 1,000 ounce.

1.14 Mineral Reserve Estimate

The proven and probable ore reserve for the Marban Engineering Project is estimated at 56.4 Mt at an average grade of 0.91 g/t Au for 1,647 koz of contained fold. There is no reserve within the proven category. This is summarized in Table 1-2.

Table 1-2: Marban Engineering Project Ore Reserve Estimate (August 17, 2022)

	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)
Proven	-	-	-
Probable	56,437	0.91	1,647
Proven and Probable	56,437	0.91	1,647

Notes: **1.** The mineral reserve is estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources & Mineral Reserves, May 19th, 2014. **2.** The Qualified Person for the estimate is Mr. Carl Michaud, Eng. M.B.A., Vice President of Mining Engineering for GMS. Effective date of the estimate is August 17, 2022. **3.** Mineral reserves are estimated for a long-term gold price of US$ 1,600/oz. **4.** Mineral reserve cut-off grade is 0.3 g/t Au for all materials. **5.** A dilution skin width of 1 m was considered resulting in an average mining dilution of 5.4%. **6.** The average strip ratio is 5.07:1. **7.** Numbers may not add due to rounding.

The open pit mine design and ore reserve estimate have been prepared by GMS to a level appropriate for a prefeasibility study. The mineral reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the mineral reserves are based solely on measured and indicated mineral resources with applicable modifying factors and therefore exclude any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as waste for reporting purposes.

The parameters used for optimization were updated from previous work done on the Marban Engineering Project as well as benchmarking on similar projects. Optimization parameters were updated from previous work done on the Marban Engineering Project as well as benchmarking on similar projects. A long-term metal price assumption of US$ 1,600/oz was used. The mining reference cost (i.e., for a block near surface) is US$ 2.40/t with an incremental cost of C$ 0.003/t per 1 m added to account for the additional haulage cycle time. The total ore-based cost is C$16.55/t. The ore-based cost is based on a nominal throughput of 6.0 Mt/a. A cut-off grade (CoG) of 0.3 g/t was set for the project. A prefeasibility level pit slope design study was carried out by Ausenco.

A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic CoG. The block contacts are then used to estimate a 1 m dilution skin around ore blocks to estimate an expected dilution during mining. To account for dilution of the historic underground voids, a 3 m skin was applied to the hanging wall material that will fill the voids post blast. This material is expected to be lost in dilution as is treated as such. One million tonnes of ore are considered lost due to ore loss at a grade of 0.39 g/t; in addition, 1,230 Mt of ore at 1.08 g/t is lost to the voids. Added to the ore is 8.6 Mt of dilution at an average grade of 15 g/t.

1.15 Mining Methods

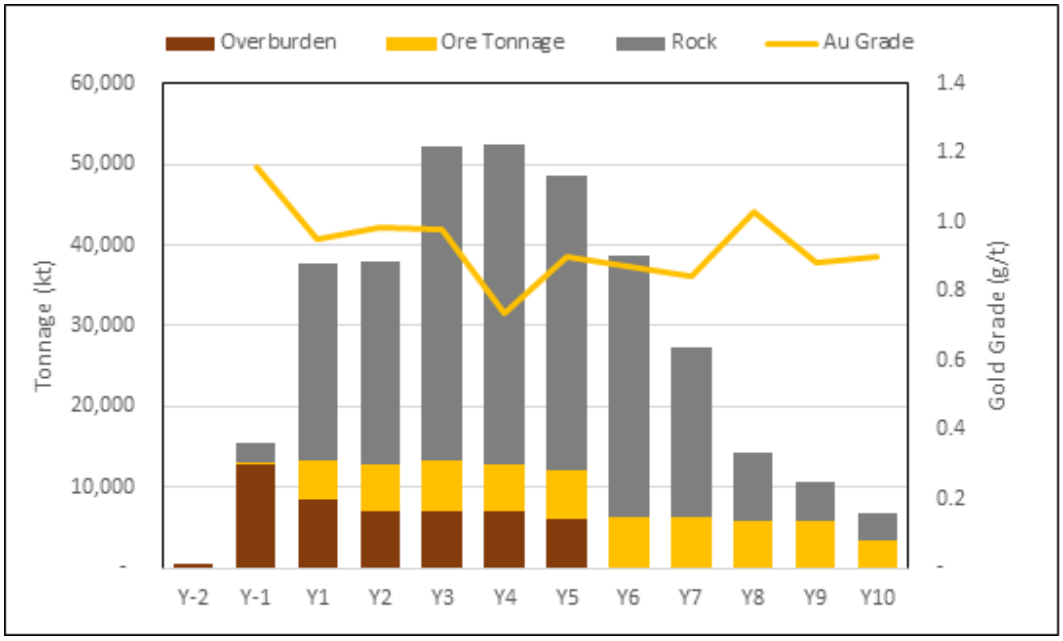
Mining of the Marban project will be carried out using conventional open pit techniques using 10 m benches. An owner-mined open pit operation is planned with hydraulic shovels and mining trucks, including the outsourcing of certain support activities such as blasting.

Production drilling of the 10 m benches will be carried out by 6.5-inch (165.1 mm) production drills with the capabilities of rotary drilling and down the hole (DTH) drilling. Blast holes are loaded with high energy bulk emulsion. The majority of the loading of the pit will be carried out by three 16 m^3-electric hydraulic shovels, one 12 m^3 diesel electric shovel, and a 10.7 m^3 diesel front end loader. The loading fleet with be augmented with 60 t shovels that will handle overburden material and scaling. The primary loading fleet with service a fleet of 150 t trucks and the secondary overburden loading units wills service a fleet of 100 t trucks.

The mining of the Marban project will occur in five separate pits. The Marban pit is the largest and contains three nested phases. The Norlartic pit is the second largest and is composed of two nested pits. The four remaining pits are small, single-phased pits. Waste rock will be sent to the primary waste dump which is phased into two sections to reduce initial footprint and hauling from pits. Overburden material will go to one of two overburden stockpiles located adjacent to each of the pit groups. The pits generate 237.3 Mt of waste rock and 48.9 Mt of overburden at a total stripping ratio of 5.07.

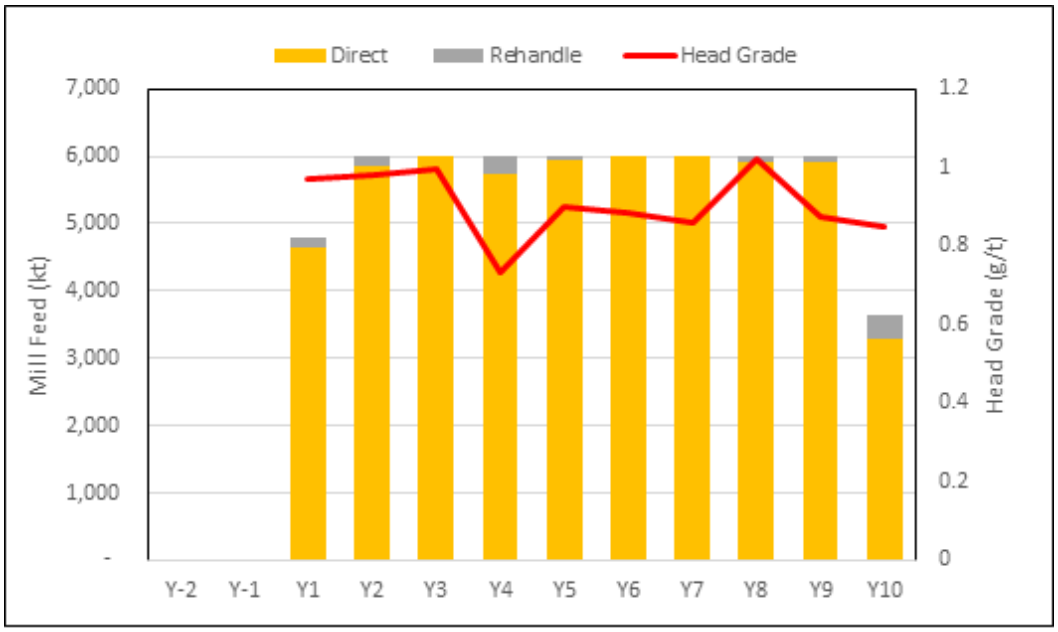
Mining activities include two years of pre-production and construction and a total mine life of 12 years. The initial year of commercial production includes a milling ramp up at 80% of the name plate capacity until it reaches the peak milling rate of 6.0 Mt/a which is sustained for 8.5 years. Peak mining rate is 52.3 Mt/a. Figure 1-1 depicts the mining schedule by material type and ore grade. Figure 1-2 depicts the mill feed by designation and the associated head grade. Minimal stockpiling and blending are used. The peak stockpile inventory is 0.5 Mt of low-grade material which is recovered at the end of mine life.

Figure 1-1: Mine Production by Material Type



Source: GMS, 2022.

Figure 1-2: Mill Feed



Source: GMS, 2022.

1.16 Recovery Methods

The testwork provided was analysed and several options for process routes were reviewed in the initial stages of the prefeasibility study. Based on the analysis, a conventional leach and carbon-in-pulp process route was chosen as the most suitable for the deposit and project economics.

The process plant was designed using conventional processing unit operations to treat up to 6.0 Mt/a (16,438 t/d) based on an availability of 8,059 hours per annum or 92%. The crushing plant section design is set at 70% availability and the gold room availability is set at 52 weeks per year. The plant will operate two shifts per day, 365 days per year, and will produce doré bars.

Ore is hauled from the mine to the primary crushing facility equipped with an apron feeder, grizzly feeder, and jaw crusher. The material will be conveyed to the secondary scalping screen, from where undersize material will bypass the secondary cone crusher while oversized material will be crushed. The two streams will combine and be conveyed to the covered stockpile.

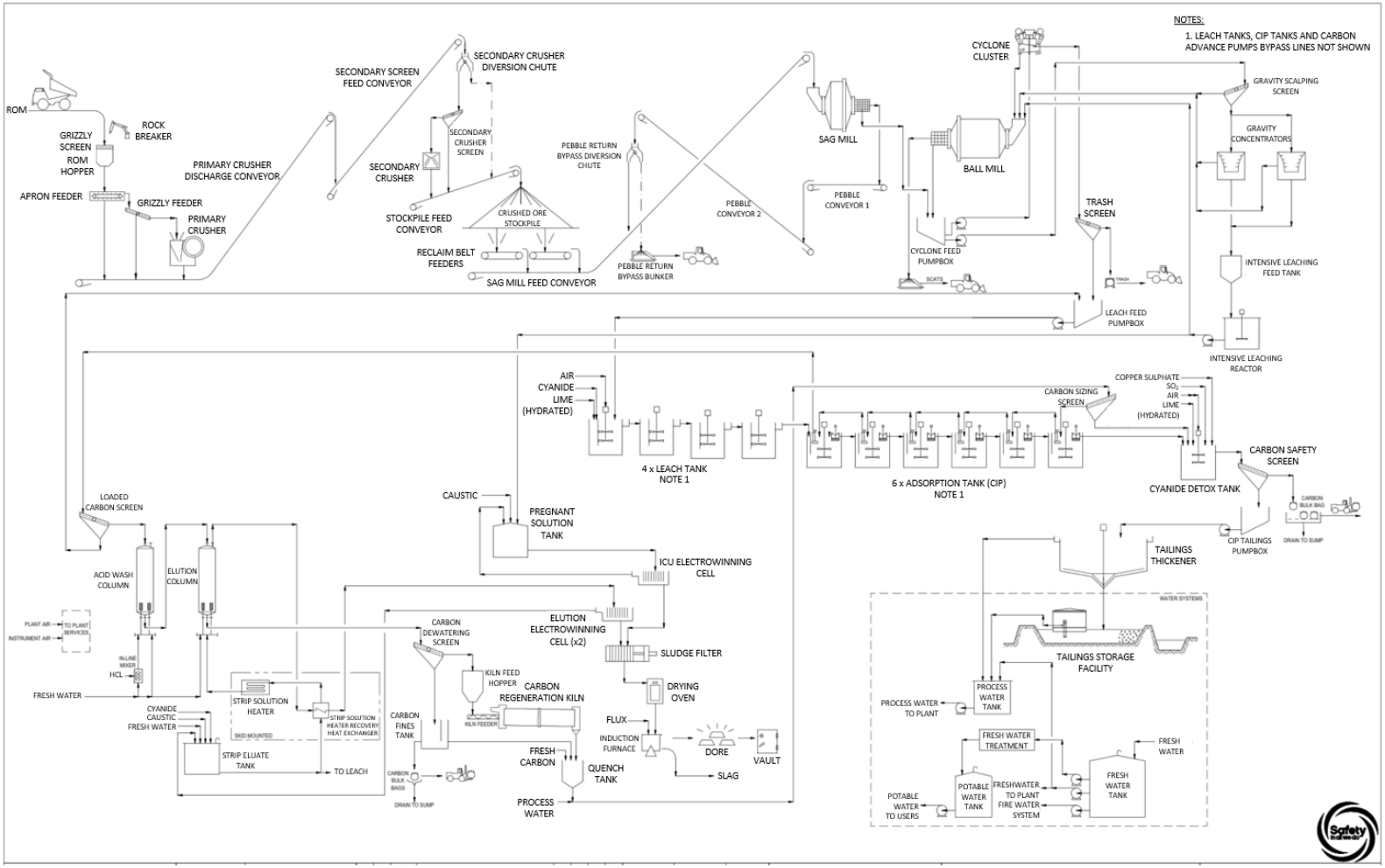
The crushed ore will be ground by a SAG mill followed by a closed-circuit ball mill with hydrocyclones classification. The cyclone feed pump will feed either the cluster of hydrocyclones or a gravity circuit equipped with an intensive cyanidation reactor. The gravity circuit will be comprised of one scalping screen and two centrifugal batch concentrators in parallel. The scalping screen undersize will feed to the centrifugal concentrator, and the concentrate will be collected and subsequently leached by the intensive cyanidation reactor circuit.

The cyclone overflow will flow to the conventional leach and carbon-in-pulp (CIP) circuit with a final grind size of 80% passing 85 µm. The cyclone underflow will report back to the ball mill.

Gold and silver are leached in the leach tanks and the dissolved gold and silver adsorbed in the CIP circuit onto activated carbon. The loaded carbon is separated and eluted using a pressure Zadra elution circuit. Gold and silver will then be recovered by electrowinning in the gold room. The gold-silver precipitate will be dried in an oven and mixed with fluxes and smelted in a furnace to pour gold doré bars.

Carbon will be reactivated in a carbon regeneration kiln before being returned to the CIP circuit. CIP tails slurry will be treated in cyanide destruction using an $SO_2/_{air}$ process with liquid SO_2 before reporting to a final tailings thickener. Figure 1-4 demonstrates the overall flow diagram for the process plant.

Figure 1-3: Overall Process Flow Diagram



Source: Ausenco, 2022.

1.17 Project Infrastructure

The overall site plan in Figure 1-3 outlines the mining pits and storage facilities, water management facilities (including the creek diversion), the process plant, tailings storage facility (TSF), the existing Hydro-Quebec high-voltage line, existing roads and railways, and the new access roads for the site. Access to the process plant and truck shop area will be from a new 2.5 km gravel road, from a new turnoff at the existing Highway 117.

1.17.1 Access

An existing public secondary road (Chemin Gervais) currently crosses over the footprint of the Marban pit. A new alignment of this road, approximately 4 km long, will be constructed east of the project site, to maintain access to the existing bridge over Keriens Creek and the properties north of the creek.

The Canadian National Railway runs through the Marban property, parallel to Highway 117. A new railway crossing will be necessary on the new main access road to the process plant.

1.17.2 Utility Power Supply

An existing Hydro-Québec 120 kV transmission line traverses the Marban property and runs adjacent to the new process pant location. A new brand 120 kV overhead transmission line will be constructed between the existing line and the new 120 kV outdoor substation, located next to the process plant.

1.17.3 Buildings

The buildings for the Marban Engineering Project consist of either pre-engineered, fabric, or modular type. Pre-engineered buildings will be supported by reinforced concrete footings with concrete slabs and pedestals, and fully enclosed with metal cladding. The process plant consists of two main pre-engineered buildings, the mill building (grinding/elution) and the gold room building. The plant is located north of the domed ore stockpile cover, and the gold room building is located immediately west of the process plant.

Fabric buildings will be fully enclosed by a fabric cover. The secondary crushing building, cyanide storage and mixing shed, reagents building, reagents storage building, plant warehouse and maintenance building, truck wash building, truck shop building, and truck shop warehouse will all be fabric buildings.

Modular buildings in the plant site area include the assay and metallurgy laboratory, the security gatehouse, the mine dry and mining office, and the mill office.

1.17.4 Water Management

A site-wide water balance analysis was completed to account for various water inflows, containments, losses, reclaims, make-up water and discharges from and into the project site. Water management components consist of collection ponds, berms, diversion ditches, and pumps to collect and contain surface water runoff from waste rock, overburden stockpiles, the process plant stockpile, and pits.

Contact water across the mine site was designed to be collected using a collection system. Additionally, clean runoff is designed to be diverted by diversion ditches. Runoff from the majority of mine facilities is considered contact water and will be collected by a collection system and contained in ponds before discharge to the environment. The runoff is conveyed throughout the site using collection ditches.

Figure 1-4: Overall Site Layout

Source: Ausenco, 2022.

1.17.5 Keriens Creek Diversion

The project requires relocation of a section of Keriens Creek, upstream of the outlet to De Montigny Lake, in order to provide access to the Kierens and Norlartic Pits, located underneath the existing creek alignment.

Based on the combination of LiDAR elevation datasets (MFFP, 2020) and bathymetric data (Niogold, 2013), grade and alignment of the diversion channel were optimized to minimize excavation volumes. Designs and quantity estimates were completed for the 100-year, 24-hour storm event (1:100 year), including setting up a hydrologic model for this weather scenario to determine peak flow rates and designing the optimum channel dimensions to safely convey the storm runoff.

The diversion channel to safely convey the 100-year, 24-hour event is 25 m wide, and 4 m deep (assuming a 2H:1V side slope). Inlet and outlet invert elevations were set to +294.5 m and +294.0 m respectively, to provide a longitudinal slope similar to that of the natural channel.

Flooding from the diversion would not reach Lac Vassan (northeast of the diversion channel), but high water levels of the diversion could cause backflow of runoff from the lake and cause localized flooding adjacent to the lake.

1.17.6 Site Geotechnical Conditions

The project site consists of overburden overlying bedrock. Overburden material consists of backfill material and natural soil. The 2021 field investigation indicated the overburden thickness varies from 0.7 m to 32.6 m across the project site. Native soil across the site generally contains five stratigraphic units deposited in the sequence of the oldest to the youngest from the bottom to the top, including organic soil near the ground surface underlain by oxidized glaciolacustrine sediments. Beneath the oxidized glaciolacustrine sediments is glaciolacustrine sediments overlying glaciofluvial sediments that underlain by till sediments.

1.17.7 Tailings Storage Facilities

Ore processing will produce 56.4 Mt of tailings, according to the mine plan, which will be stored within two tailings facilities:

- 19.3 Mt in a conventional tailings storage facility (TSF 1), between Years 1 and 4.5; and
- 37.1 Mt in in-pit storage (TSF 2), after the Norlartic pit is fully mined out after Year 4.5.

Seepage through the TSF 1 embankment will primarily be controlled by the geomembrane, low permeability soil, filter zone on the upstream face of the embankment, and the seepage cut-off structures along the upstream toe of TSF 1. The tailings facility water management includes diversion ditches and sediment ponds.

TSF 2 will be placed into operation in Year 4.5, which is in-pit tailings disposal into the Norlartic open pit that has a capacity for 37.1 Mt. A tailings pipeline and water reclaim pipeline will be constructed at the beginning of Year 4 so operations can commence approximately in June of Year 4. The tailings pipeline will be constructed around the pit with multiple spigots to evenly discharge tailings into the facility. There are no berms or diversion channel required since all surface runoff structure were constructed for development of the open pit.

A start-up water pond volume of approximately 1.6 Mm^3 is expected to be sufficient for the thickened tailings operation based on Ausenco's design criteria of a minimum 180 days of make-up water. As tailings are deposited into the facility, water will be released from the tailings stream during deposition and subsequent consolidation and will eventually report as supernatant water to the main tailings operations pond. A portion of the supernatant water will be lost due to evaporation, interstitial voids, and from seepage into the foundation. The remaining water will be available as recycle to the plant site. The TSF 1 water management consists of diverting non-contact surface water from the surrounding area around the ring dike in trapezoidal diversion channels lined with riprap to existing drainages. The channels are design to convey the 1:100-year storm event. TSF 1 is designed to contain and then pass the Probable Maximum Precipitation (PMP) of 355 mm. The TSF 2 water management consists of diversion channel and dikes to convey surface runoff around the Norlartic

Pit. These structures are designed to convey the 1:100-year storm event around the pit. TSF 2 is designed to contain and then pass the PMP. The water from large storm events can be used for makeup water or be pumped downstream into Keriens Creek.

1.18 Market Studies and Contracts

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 of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with a large number of buyers and sellers active at any given time.

O3 Mining plans to contract out the transportation, security, insurance, and refining of doré gold bars. O3 Mining may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in Canada and in countries throughout the world. For the PFS, a cost of C$2.5/oz Au was assumed for transportation, and refining.

1.19 Environmental Studies, Permitting & Social or Community Impact

The Marban Engineering Project is subject to the provincial and federal environmental impact assessment procedure as forecasted daily production is over the thresholds outlined in the applicable regulation.

1.19.1 Environmental Baseline Studies

Several environmental baseline studies have been completed in 2021 and 2022 to supplement those of 2016. As the project continues to be defined, and in regard of the Environmental Impact Assessment (EIA) procedure requirements, additional baseline data collection and assessment will be necessary.

The studied environmental components include the following:

- Hydrological conditions;
- Hydrogeology study and groundwater quality (including hydrogeological modelling);
- Surface water and sediment quality;
- Soil natural background assessment;
- Soil environmental assessment of three old mines sites;
- Biological components (vegetation and wetlands, small mammals, chiropterans, avifauna, herpetofauna, species at risk and aquatic fauna);
- Ambient air quality;
- Ambient noise and vibration;
- Waste rock and ore geochemical assessment (static and kinetic); and
- Tailings geochemical assessment (ongoing).

Environmental site monitoring will be implemented. The objective of the environmental monitoring program is to detect and document any changes in the environment in relation to the baseline (whether or not related to the project), to verify the impact assessment and to evaluate the effectiveness of mitigation or compensation measures proposed in the impact assessment.

The main components of the environmental site monitoring program are as follows:

- Effluents Quality Monitoring (Directive 019 and Metal and Diamond Mining Effluent Regulations (MDMER));
- Groundwater Quality and Piezometric Level (Directive 019);
- Water Quality Monitoring Studies (MDMER);
- Biological Monitoring Studies (MDMER); and
- Mitigation Measures Monitoring (air quality, noise, vibration, runoff, etc.).

1.19.2 Social and Community Considerations

Early information and consultation meetings have been held with local communities, First Nations Communities, local, provincial, and federal governmental authorities to initiate collaborative work to obtain social acceptability of the project. O3 Mining is advocating for open dialogue with concerned parties to enable the inclusion of comments and suggestions in the development of the Marban Engineering Project. O3 Mining's commitments include keeping stakeholders informed on project advancement, transparency and respect for the voicing of opinions; and listening and being receptive to questions and concerns from interested parties.

Land tenure is a mix of public, private, and municipal properties. No federal land is located within the project area. No federal land will be used for to carry out the project. Since the project will require lands on which permanent residences, businesses, and public roads are located within the proposed layout, agreements will have to be settled with respective owners. O3 Mining has initiated discussion with some residents and business owners on the footprint of the project, but no agreements have been signed.

The project site is located on the ancestral territory of the Algonquin Anishinabeg Nation (Anicinabek). No land in a reserve is located within the proposed layout. The project area is, however, located on land that is subject to a comprehensive land claims agreement or a self-government agreement.

1.20 Capital and Operating Costs

1.20.1 Capital Costs

The capital cost estimate was developed in Q3 2022 to a level of accuracy of ±30% (Class 4) in accordance with the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes mining, processing, onsite infrastructure, offsite infrastructure, project indirect costs, project delivery, owners' costs, and provisions. The total initial capital cost for the Marban Engineering Project is C$435M and Life-of-Mine (LOM) sustaining costs are C$283M. Closure costs are estimated at C$48.9M. The capital cost summary is presented in Table 1-3.

Table 1-3: Capital Cost Estimate Summary

WBS2	Cost Centre	Initial Capital (C$M)	LOM Sustaining (C$M)	Total Capital (C$M)
1100	Mining General and Administration	7.77	0.0	7.77
1200	Drill and Blast	4.74	0.0	4.74
1300	Material Movement	19.8	0.0	19.8
1400	Mining Civil Infrastructure	11.7	0.0	11.7
1500	Mining Infrastructure & Services	2.71	0.0	2.71
1600	Mine Major Equipment	17.0	211.8	228.8
1700	Mine Support Equipment	8.09	0.00	8.09
Mining Total		71.8	211.8	283.7
2100	Crushing	17.9	0.0	17.9
2200	Stockpile & Reclaim	7.99	3.19	11.2
2300	Grinding	66.2	0.0	66.2
2400	Leaching	29.6	0.0	29.6
2500	Elution, Carbon Regeneration & Gold Room	16.2	0.0	16.2
2600	Cyanide Detoxification & Tailings	9.56	0.0	9.56
2800	Reagent Storage & Distribution	9.19	0.0	9.19
2900	Utilities (Air & Water Services)	4.19	0.0	4.19
Process Plant Total		160.9	3.19	164.1
3100	Bulk Earthworks	7.57	0.0	7.57
3200	Power Supply and Distribution	12.0	0.0	12.0
3300	Fuel Storage	0.40	0.0	0.40
3400	Ancillary Buildings	8.85	1.26	10.11
3500	Site Services	2.10	29.9	32.0
3600	Site Water Services	4.97	0.0	4.97
3700	Site Water Management	12.0	0.0	12.0
3800	Tailings Storage and Management Facilities	45.4	34.3	79.7
On-Site Infrastructure Total		93.3	65.5	158.8
4100	Public Road Diversions and Upgrades	2.33	0.0	2.33
4200	Keriens Creek Diversion	10.1	0.0	10.1
Off-Site Infrastructure Total		12.4	0.0	12.4
5100	Temporary Construction Facilities and Services	11.9	0.50	12.4
5300	Spares (Commissioning, Initial and Insurance)	1.24	0.0	1.24
5400	First Fills & Initial Charges	3.86	0.0	3.86
Project Indirects Total		17.0	0.50	17.5
6100	Engineering & Construction Management Services	25.0	2.00	27.0
Project Delivery Total		25.0	2.00	27.0
7000	Owners' Costs	10.6	0.0	10.6
8000	Contingency	44.0	0.0	44.0
	Grand Total	435.1	283.0	718.0

The capital cost estimates are based on the following assumptions and parameters:

- For material sourced in US dollars (5.1% of initial capex), an exchange rate of 1.30 Canadian dollar to 1.00 US dollar was assumed.
- No allowance has been made for exchange rate fluctuations.
- There is no escalation added to the estimate.
- A growth allowance was included.
- Data for the estimates have been obtained from numerous sources, including:
    - Mine schedules;
    - Prefeasibility-level engineering design;
    - Topographical information obtained from the site survey;
    - Geotechnical investigations;
    - Budgetary equipment quotes from Canadian and International suppliers;
    - Budgetary unit costs from several local contractors for civil, concrete, steel, electrical, piping, and mechanical works; and
    - Data from similar recently completed studies and projects.

1.20.2 Operating Costs

The operating cost estimate includes mining, processing, and general and administration (G&A) costs. The overall LOM operating cost is C$1,419 million over 10 years, or an average of C$25.14/t of ore milled in a typical year. Of this total, processing and G&A account for C$521 million and mining accounts for C$898 million. A summary of the operating costs is presented in Table 1-4.

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2022 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.30 Canadian dollar to 1.00 US dollar was assumed.
- Estimated costs for diesel and gasoline are C$1.20/L and C$1.045/L, respectively.
- The annual power costs were calculated using a unit price of C$0.048/kWh.
- Labour is assumed to come from the local area of highly skilled workers in Val d'Or.

Table 1-4: Operating Cost Estimate Summary

Cost Centre	C$/t Milled	C$M
Mining		
Drilling	1.24	69.7
Blasting	2.50	141.1
Loading	1.48	83.8
Hauling & Rehandling	5.43	306.2
Overburden Mining	0.05	2.7
Road & Dump Maintenance	2.09	117.9
Misc. Maintenance	0.92	52.1
Mine General & Admin	2.22	125.0
Mining Subtotal	15.92	898.5
Process Plant		
Reagents	2.13	12.8
Consumables	2.50	15.0
Plant Maintenance	0.42	23.5
Power	1.15	65.2
Laboratory	0.03	1.7
Labour – Process Plant	1.46	87.7
Processing Mobile Equipment	0.06	3.3
Process Plant Subtotal	7.74	442.5
G&A		
Labour – G&A	0.72	40.8
G&A Expenses	0.58	32.6
Site Maintenance	0.08	4.7
G&A Subtotal	1.38	78.1
Total Project Operating Costs	25.14	1,419.0

1.21 Economic Analysis

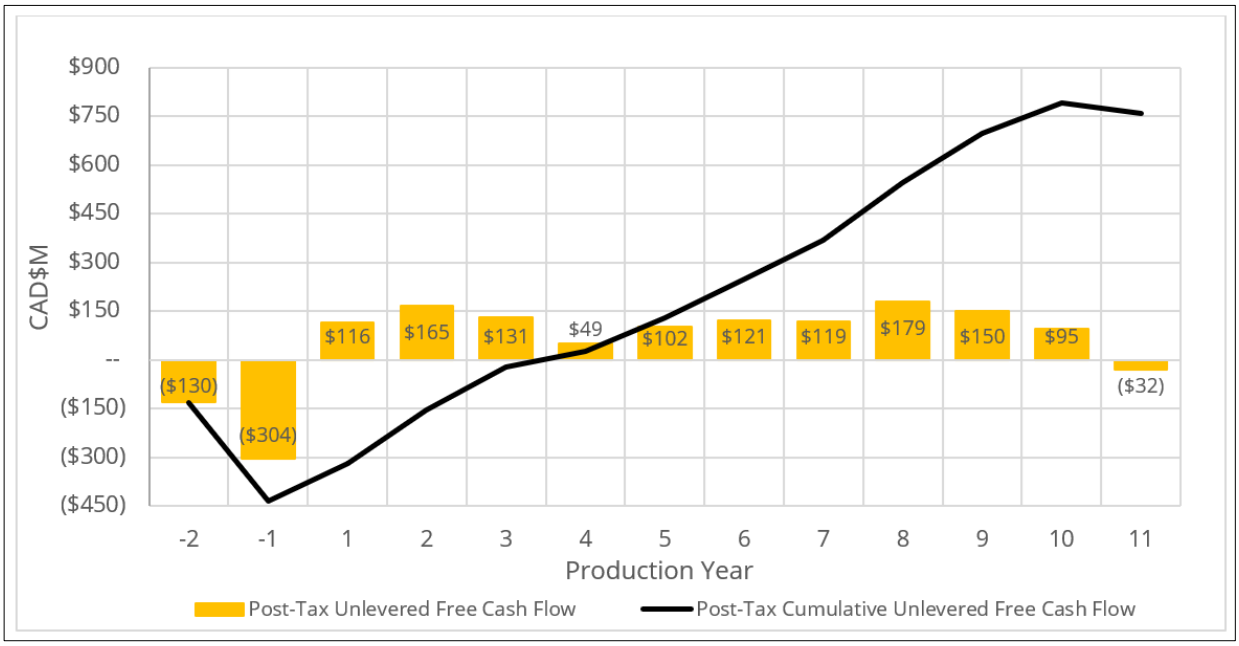
An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess the impact of variations in metal prices, head grades, foreign exchange, operating costs, and capital costs.

The economic analysis was performed using the following assumptions:

- Commercial production start-up in 2026;
- Construction period of 18 months;
- Mine life of 9.6 years;

- Base case gold price of US$1,700/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.77 (US$/C$)
- Cost estimates in constant 2022 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 1.0% Net Smelter Return (NSR;)
- Capital costs funded with 100% equity (i.e., no financing costs assumed);
- Inventory and accounts payable periods of 30 days;
- All cash flows discounted to beginning of construction June 30, 2024;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold doré; and
- No contractual arrangements for refining currently exist.
- The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$775M, the internal rate of return IRR is 30.2%, and the payback period is 2.8 years. On an after-tax basis, the NPV5% is C$463M, the IRR is 23.2%, and the payback period is 3.5 years. A summary of the project economics is shown graphically in Figure 1-5 and listed in Table 1-5.

Figure 1-5: Projected Post-Tax Cash Flow



Source: Ausenco, 2022.

Table 1-5: Summary of Project LOM Cashflow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,700
Exchange Rate ($US:$C)	$0.77
Mine Life (years)	9.6
Total Waste Tonnes Mined (kt)	286,144
Total Mill Feed Tonnes (kt)	56,436
Strip Ratio (waste: mineralization)	5.1
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	0.91
Mill Recovery Rate (%)	94.2%
Total Mill Ounces Recovered (koz)	1,552
Total Average Annual Production (koz)	161
Operating Costs	**LOM Total / Avg.**
Mining Cost (C$/t Mined)	$2.6
Mining Cost (C$/t Milled)	$15.9
Processing Cost (C$/t Milled)	$7.8
G&A Cost (C$/t Milled)	$1.4
Total Operating Costs (C$/t Milled)	$25.1
Refining & Transport Cost (C$/oz)	$2.5
Royalty NSR	1.0%
Cash Costs (US$/oz Au)*	$723
AISC (US$/oz Au)**	$882
Capital Costs	**LOM Total / Avg.**
Initial Capital (C$M)	$435
Sustaining Capital (C$M)	$283
Closure Costs (C$M)	$49
Salvage Costs (C$M)	$10
Financials – Pre-tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$775
IRR (%)	30.2%
Payback (years)	2.8
Financials – Post-tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$463
IRR (%)	23.2%
Payback (years)	3.5

Notes: * Cash costs consist of mining costs, processing costs, mine-level general & administrative expenses and refining charges and royalties. ** AISC includes cash costs plus sustaining capital, closure cost and salvage value.

A sensitivity analysis was conducted on the base case pre-tax and after-tax 5% NPV and IRR of the project, using the following variables: metal price, foreign exchange rate, discount rate, grade, total capital costs, and operating costs. Analysis revealed that the project is most sensitive to changes in metal prices, head grade, and foreign exchange rate, then a lesser extent, to capital costs and operating costs.

1.22 Risks and Opportunities

1.22.1 Risks

1.22.1.1 Geology and Resource Modelling

Risks to the resource estimate include changes to the geologic model, and the following factors which could affect the resource estimate:

- Continuity of mineralization, particularly in new zones such as the Marban Dyke domains;
- Inaccuracies associated with the location of historic underground openings.

1.22.1.2 Mining

The following risks are noted with regard to mining:

- Workforce availability;
- Contractor availability;
- Continued inflationary pressure on equipment and supplies;
- Lengthening of lead times for equipment and materials (specifically due to supply chain issues related to COVID);
- Construction and work delays due to COVID-19 outbreaks; and
- Unmapped/unstable historical underground mining voids require additional safety and mining precautions.

1.22.1.3 Processing

The following risks are noted with regards to the process plant:

- The metallurgical testwork program associated with the design of the leach-adsorption circuit was conducted primarily at grind sizes between 105 and 108 µm, whereas the process plant will be operating at a P_{80} of 85 µm. An additional 1.5% is added to the recovery to account for the adjustment in grind size, but there is a risk associated with the leach circuit reagent addition rates selected and the leach recovery models assumed based on the metallurgical testwork.
- The use of filtered raw water from the Marban pit for process plant raw water presents a risk of equipment scaling or damage, if the water quality deviates from the test data available. This risk will have to be further evaluated in future studies, or an alternate source of raw water found.

1.22.1.4 Tailings Storage Facilities

The following risks are noted with regards to the general site infrastructure:

- If additional resource is discovered in Norlartic deposit, then TSF 2 may not be available for tailings disposal in Year 4.5. There is the ability to raise TSF 1, which would add sustaining capital to the project.
- The underling soils in TSF 1 have a higher permeability then model in the seepage analysis and requires a geomembrane liner along the floor to meet Quebec guidelines for maximum seepage rates from a tailings facility, which would add capital cost to the facility.

1.22.1.5 Water Management

The following risks are noted with regards to the general site infrastructure:

- For waste rock and ore contact water, no chemical water treatment has been included in the PFS design based on the geochemical testwork data available, which does not indicate the likelihood of significant acid generation/metal leaching or exceedances of regulatory guidelines. However, the available data are limited and further testwork is required to confirm this assumption as part of the FS and future phases of the project. Further testwork may indicate additional capital expenditure requirements for water treatment.
- For the tailings facility, there is a general water deficit; therefore, at this time, it is not anticipated that there will be any regular discharge from the facility. Consequently, a treatment facility is not required for surplus water from the tailings facility based on the current available information. Water balance predictions will be better refined during feasibility level design to take into consideration the potential for upset conditions caused by seasonal and extreme weather events as well as taking into consideration the dam raise construction schedule and available freeboard over time. The requirement for occasional treatment of tailings decant water due to the above factors will be revisited and re-assessed as part of the FS and future phases of the project. Refinement of the water balance and tailings dam construction schedule may indicate the requirement for occasional treatment, requiring additional capital expenditures.
- At the decision of O3 Mining, the Keriens Creek diversion channel was sized for the 100-year flood event. The channel would be overtopped in the case of a 200-year or bigger flood event, which could result in the flooding of the Kierens and Norlartic pits and consequential risk to personnel and equipment. Appropriate monitoring and safety mitigations will have to be implemented during detailed design and operations.
- The bridge crossing over the Keriens Creek is significantly undersized, which could cause ponding/flooding upstream of the crossing during relatively major storm events. A detailed hydraulic analysis of the bridge crossing is required as part of the FS study to quantify the existing capacity and to propose a bridge design upgrade according to the local guidelines.

1.22.1.6 Environment and Permitting

The following risks are noted with regards to environment and permitting:

- A residual materials landfill is located within the proposed layout. The landfill may present certain environmental (surface runoff or underground contamination) and public health (gases and vapours) risks. Production activities were carried out at three old mines sites located within the proposed layout, and leachable waste rock backfill, and contaminated soil might still be present although the sites were cleaned. Additionally, some waste materials and potentially hazardous materials were found on private property located within the proposed layout. O3 Mining is currently evaluating these sites and performing additional soil characterization on potentially contaminated lands to evaluate the environmental liability.

- A preliminary evaluation estimates that 226 ha of wetlands and 31 ha of shorelines will be directly impacted by the project and will require authorization and monetary compensation.
- To mine the Norlartic and Kierens pitts, the current Keriens Creek will need to be diverted. The diversion will need to be done with minimum alterations, disruption, or destruction of the fish habitat. The Department of Fisheries and Oceans Canada will require compensation of the fish habitat.
- According to available data and preliminary assumptions, the waste rock storage area is located on one intermittent water course that is a confirmed fish habitat. The encroachment of this water course by the waste rock storage area is considered as a risk because an amendment to the Metal and Diamond Mining Effluent Regulations (MDMER; federal legislative action) will be required.
- Due to habitation proximity to the project's location, emissions of particles (total and fine) and gas, which can cause air quality deterioration and/or health concerns, are a risk. Similarly, noise and vibration from blasting could cause nuisances and damage to infrastructure.
- The predicted underground water table drawdown will possibly have an impact on residential wells located south of the project. Additional investigation will be completed in order to determine the actual condition of the residential wells

1.22.2 Opportunities

1.22.2.1 Exploration

Exploration activities are likely to identify additional mineralization, which could provide additional resources within the known shear zones as well as in additional unknown structures. These efforts could result in changes to the style of mineralization to that currently identified, the scale of the project, and the deleterious elemental issues identified.

1.22.2.2 Resource Modelling

At Kierens-Norlartic, some areas of inferred category blocks were not able to be drilled due to surface access reasons. If these areas could be accessed, then the link between the Kierens and Norlartic pits could be drilled and could result in additional ounces being discovered and incorporated into future mine plans. In addition, if any additional pre-Niogold era drill core can be accessed and reanalysed, this would add value to the block model and improve the confidence of the grade estimate locally.

1.22.2.3 Mining

A fleet of two trucks is currently being used to control overburden and maintain production. More analysis can be completed into better optimizing the fleet to target smaller pits with smaller trucks allowing a redesign of the ramp layouts. This could potentially lead to more aggressive sub pits.

Additional scenarios of pit electrification, trolley systems and autonomous equipment can be looked into in future studies to save costs on fuel, manpower and carbon taxes.

1.22.2.4 Processing

Further cyanide leach testing of Nolartic variability ore samples will refine the recovery forecasts made in the PFS, and may be increased if performance is observed to align to the Marban samples.

1.22.2.5 Tailings

Foundation conditions were assumed in the area of TSF 1. Additional geotechnical investigation in this area may show better foundation conditions, which could reduce sustaining capital costs by the elimination or reduction of the stability buttress.

1.23 Conclusion

Based on the assumptions and parameters presented in the report, the project has a mine plan that is technically feasible and economically viable. The positive financials of the project ($463 M after-tax NPV5% and 23.2% after-tax IRR) support the mineral reserve.

1.24 Recommendations

1.24.1 Drilling and Sampling

Future drilling at Marban should focus on converting the remaining inferred resource to indicated category and on expanding the resource base. The extensions of the Marban, Norlartic, Kierens, North-North, Triple North and Gold Hawk deposits as well as Orion No. 8, Malartic Hygrade, Malartic H and Grano zones provide substantial opportunities to expand the resource of the Marban Engineering Project.

GMS recommends that additional density measurements are gathered in the mineralized zones to be able to compare with the current density measurements taken mostly in barren lithologies. GMS does not expect any significant differences, as the orebodies are not strongly mineralized with sulphides, however silica flooding can occasionally reduce the bulk density of mineralization.

1.24.2 Metallurgy

Sample selection for future mining studies should reflect mineralization that would be treated in the first five years of the mine life. Variability samples are required to understand the responses of the various mineralized zones to grind size, leach kinetics and contaminant correlations.

Additional comminution tests (e.g., SMC, Bond ball work index, and abrasion index) are recommended on material representative of the first 3–5 years of the planned operation to provide more confidence in equipment selection and to ensure that there is sufficient comminution information that is spatially representative of the variability within the various mineralized zones.

The flowsheet selected for the PFS should be validated by selecting a composite sample representative of Year 1 or Year 1.5. This composite sample should undergo gravity-leach testwork, and the tailings should complete cyanide detoxification optimization testwork and vendor thickener tests.

1.24.3 Mining

During the course of the study, items were identified as requiring additional information to further improve precision and information as part of the detailed engineering. Certain risks were also identified that require significant initiatives and continuous monitoring.

- Detailed planning for earthworks and deforestations should detail the tasks performed by owner and contractor fleets to better optimize upfront capital and equipment utilization. Potential overlap can be eliminated and key equipment that would be better owned by the mine can be identified to save costs. The approximate cost of the detailed trade off is $50,000.

- In depth review of the ground water conditions and dewatering plan must take place to ensure that the project is resistant to lifetime rainfall and flooding events. In addition, thorough environmental testing of the ground water should be done to ensure that water sourced from historical stopes are properly treated. The approximate cost of detailed groundwater studies is estimated at $500,000.
- Additional LOM plans can be assessed to explore additional mining options. Smaller pits can be redesigned for a smaller contractor fleet, allowing a larger owner mining fleet for the larger pits. Additional trade-offs between contractor and owner fleets can be assessed. The approximate cost of the additional trade-offs is $80,000
- Additional work on blast rock movements can be done to capture the void loss ore in the models. This material can be properly recovered with better understanding of the post blast movements and underground void structures. Careful work on initial void zones with blast balls and reconciliation programs can potentially recover this ore in the model. The approximate cost of the additional testing is $250,000.

Additional work on blast rock movements can be done to capture the void loss ore in the models. This material can be properly recovered with better understanding of the post blast movements and underground void structures. Careful work on initial void zones with blast balls and reconciliation programs can potentially recover this ore in the model.

1.24.4 Geotechnical

1.24.4.1 Mining Geotechnical

Additional testwork and analysis is recommended prior to the feasibility study to improve information required for the open pit slope design as follows:

- Drilling an additional 9 geomechanical boreholes for a total depth of 2,160 m to support the next phase of work. The program would comprise 1 borehole in Kierens, 4 boreholes in Norlartic and 4 Boreholes in Marban. It is estimated that 2 to 3 of the 9 proposed geotechnical holes will intersect the faults through Norlartic and Marban. All holes are to be televiewer (oriented core) logged and core should be geotechnically logged at the drill sites. Strength profiling by point load testing should also be continued, as per the current practice at the project.
- Surface mapping should be performed for each of the pits. The information should be used to refine the overburden thickness maps and to update engineering properties of the rock based on laboratory testing. Geophysical surveys should also be considered to supplement the overburden thickness database.
- Laboratory testing, including moisture content, gradation, triaxial strength, Atterberg limits, and density, should be performed to classify the saprolite. The site-specific relationship between point load strength and unconfined compressive strength should be refined and shear strength tests should be conducted to determine the basic friction angle for major rock types.
- The structural model should be updated to incorporate all new drilling available at the time the FS is completed. Borehole fault intersections should be traced for continuity to determine if structural control is present in the pit walls of either pit.
- After the new data is available, a new rock engineering analysis should be performed, including:
    - Update the structural fabric database to incorporate all pit areas. Evaluate the data for spatial variation and/or lithologic correlations.
    - Update the rock mass geomechanical parameters and revise design values, as required.
    - Update overburden geomechanical parameters
    - Evaluate hazard posed by upslope overburden deposits relative to pit crests and design conceptual mitigation measures accordingly.

- Perform slope stability analyses to optimize design for each wall for all pits.
- Look at benefits of depressurizing pit slope.

1.24.4.2 Infrastructure Geotechnical

The following recommendations are made for geotechnical site investigation work.

- Completion of 31 (1,240 m) geotechnical boreholes, 58 test pits, and 5 geophysics in the areas of the TSFs, WRSFs, plant site, haul roads, and ancillary roads to investigate and confirm foundation conditions, specifically the extent of the overburden along with depth to groundwater and to bedrock.
- Additional laboratory index testing, including compaction tests, mechanical strength tests, and permeability tests on foundation soils and potential borrow materials.
- Laboratory testing to confirm the physical characteristics of bedrock.
- Using the new data, recommend designs for foundations, borrow sources, construction materials for infrastructure, and tailings and waste rock storage facility exterior slope configurations.
- Perform deterministic and probabilistic local seismic hazard study for the development of design seismic values for infrastructure.

1.24.5 Water Management

It is recommended to plan a water sourcing schedule during different years of the mine life. Pumping and linear water facilities should be designed to source water from the pits and collection ponds on a seasonal basis, as well as during different years.

The following studies are recommended to be considered for the Feasibility Study:

- Review an option to increase the upstream invert elevation of the diversion channel to possibly reduce the excavation quantity. This option would require environmental review to make sure the upstream ponding and shallow water level in the upstream section of the channel does not impact the aquatic habitat.
- Explore modification to the bridge crossing in the downstream, which currently acts as a constriction within the Keriens Creek. Widening the bridge crossing would help lowering the water level and subsequently reduce the inundation extent upstream.
- A more comprehensive hydrological studies including rainfall-runoff modelling and which may improve water management facility design, and reduce estimations of contact water containment volume, excavation, and excess water rates.
- A stochastic hydrologic modelling to account for uncertainties embedded with weather components and other water inflow/loss components.
- A more detailed geochemistry study, including groundwater quality to determine if the water from pit dewatering requires treatment before being released to the environment.
- A more detailed groundwater modelling to provide annually varying estimates of groundwater inflow to the pits.

1.24.6 Geochemistry and Water Treatment

No chemical water treatment has been included in the PFS design based on the geochemical testwork data available, which does not indicate any significant acid generation or exceedances of regulatory guidelines.

However, the available data is very limited in nature. Further testwork is required for the Feasibility Study as follows:

- Additional static test work on a range of samples encompassing all potential geological and mineralogical heterogeneities within the ore and waste rock of the mineral reserve;
- Further kinetic testwork requirements will be defined based on the static test results; and
- Metallurgical test work completed on the ore (planned for H2 2022) can generate process water samples and coarse/fine fractions of tailings that will allow for further test work and characterization/prediction of potential tailings behaviour and water quality.

1.24.7 Tailings Storage Facilities

The following work is recommended to support an FS:

- Review and update meteorological and hydrology information, updating surface water and sediment management for the tailings storage facilities.
- Confirm geochemical characterization of tailings and waste rock from additional waste characterization studies.
- Develop seepage predictions and seepage control measures for the TSFs.
- Update the tailings and waste rock deposition strategy to optimize material handling for tailings facilities.
- The stability model should be reviewed and updated, as required, with consideration of the final deposition plan using updated data about the material properties of the waste rock, tailings, and the foundations for both the TSF 1.
- Perform a liquefaction assessment with consideration of updated information on material properties and the updated stacking plan for the TSF 1.

1.24.8 Environment, Permitting and Community Relations

Recommendations for environment, permitting and community relations to better define the environmental and social impacts are as follows:

- Complete an acquisition protocol.
- Continue with the negotiation and agreements with private and public stakeholders.
- Continue with the consultations and engagement activities, addressing and documenting concerns.
- Continue with the government involvement to ensure that their expectations are addressed throughout the project development.
- Initiate provincial and federal environmental assessment processes in order to receive the Tailored Impact Statement Guidelines.
- Initiate an exhaustive geochemical characterization to support FS and EIA.
- Complete a contaminant dispersion modelling from the tailings in pit storage.

Continue additional soil environmental assessment on potentially contaminated lands.

2 INTRODUCTION

2.1 Introduction

This report was prepared by Ausenco Engineering Canada Inc. (Ausenco) and G Mining Services Inc., (GMS) for O3 Mining Inc. (O3 Mining) to summarize the results of the Prefeasibility Study (PFS) of the Marban Engineering Project.

2.2 Terms of Reference

The report was prepared in accordance with the Canadian disclosure requirements of National Instrument 43-101 (NI 43-101) and Form 43-101 F1, and is prepared using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (CIM, 2014) and the CIM Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines (CIM, 2019).

The report supports disclosures by O3 Mining in a news release dated 06 September 2022 entitled "O3 Mining Completes Prefeasibility Study for Marban Engineering with Post-Tax NPV of C$463 Million, Unlevered IRR of 23.2% And Annual Production Of Over 160Koz Gold."

All measurement units used in this report are SI units unless otherwise noted. Currency is expressed in Canadian dollars (C$).

The Marban property contains six past-producing mines (Marban, Norlartic, Kierens, Malartic Hygrade, Orion No. 8 and Camflo (below 800 m from the surface)), which collectively produced 879,000 ounces of gold between 1959 and 1992. O3 Mining owns 100% of the Marban property.

2.3 Qualified Persons

The Qualified Persons (QPs) are listed in Table 2-1 on the following page. By virtue of their education, experience, professional association, the individuals presented in Table 2-1 are considered QPs as defined by NI 43-101.

2.4 Site Visits & Scope of Professional Inspection

Mr. James Purchase visited the site between September 8, 2021, and September 10, 2021. Drilling activities were ongoing at the time of the visit. Core processing and storage facilities located in Val D'Or were toured, and drill core from Marban, Kierens, and Norlartic deposits was reviewed. Mr. Purchase also reviewed sampling and QA/QC procedures on site, and visited the preferred laboratory (AGAT, Val D'Or) to inspect the sample preparation facilities.

Mr. Ali Hooshiar, representing Ausenco, visited the site on May 27, 2022. The main objective of the site visit was to review the geotechnical conditions of the site for the PFS. Mr. Hooshiar reviewed the general topography and geotechnical surface conditions for the site wide infrastructure to check for any visually apparent geotechnical issues. In addition, Mr. Hooshiar visited the on-site drill core lab and reviewed the drill cores from different deposits and lithologies.

2.5 Effective Dates

The effective date of this report is August 24, 2022.

Table 2-1: Report Contributors

Qualified Person	Professional Designation	Position	Employer	Independent of O3 Mining	Report Section
Renee Barrette	P. Eng.	Principal Metallurgist	Ausenco	Yes	1.1, 1.2, 1.3, 1.4, 1.12, 1.16, 1.17.1, 1.17.2, 1.17.3, 1.18, 1.20, 1.21, 1.22.1.3, 1.22.2.4, 1.23, 1.24.2, 2, 3, 4, 13, 17, 18.1, 18.2, 18.3, 18.4, 18.5, 19, 21.1, 21.2.1, 21.2.3, 21.2.4, 21.2.5, 21.2.6, 21.2.7, 21.2.8, 21.2.9, 21.2.10, 21.2.11, 21.2.12, 21.3.1, 21.4.1, 21.4.3, 21.4.4, 22, 24, 25.1, 25.4, 25.7, 25.10, 25.11, 25.12, 25.13, 25.14.1.3, 25.14.2.4, 25.15, 26.1, 26.6, 27
James Purchase	P. Geo.	Vice-President, Geology and Resources	GMS	Yes	1.5, 1.6, 1.7, 1.8, 1.9, 1.10, 1.11, 1.13, 1.22.1.1, 1.22.2.1, 1.22.2.2, 1.24.1, 5, 6, 7, 8, 9, 10, 11, 12, 14, 23, 25.2, 25.3, 25.5, 25.14.1.1, 25.14.2.1, 25.14.2.2, 26.2, 26.3
Carl Michaud	P. Eng.	Vice-President, Mining	GMS	Yes	1.14, 1.15, 1.20, 1.22.1.2, 1.22.2.3, 1.23, 1.24.3, 15.1, 15.2, 15.3, 15.5, 15.6, 15.7, 15.8, 15.9, 15.10, 16, 21.1, 21.2.2, 21.3.2, 21.4.2, 25.6, 25.11, 25.12, 25.14.1.2, 25.14.2.3, 25.15, 26.4
Ali Hooshiar	P. Eng.	Senior Geotechnical Engineer	Ausenco	Yes	1.17.6, 1.17.7, 1.22.1.4, 1.22.2.5, 1.24.4, 1.24.7, 15.4, 18.7, 18.8, 25.8, 25.14.1.4, 25.14.2.5, 26.5, 26.9, 26.10
Davood Hasanloo	P. Eng.	Senior Water Resources Engineer	Ausenco	Yes	1.17.4, 1.17.5, 1.22.1.5, 1.24.5, 1.24.6, 18.6, 25.14.1.5, 26.7, 26.8
Andréanne Hamel	ing., M.Sc.	Project Director – Environmental Management and Hydrogeology	WSP	Yes	1.19, 1.22.1.6, 1.24.8, 20, 25.9, 25.14.1.6, 26.11

2.6 Information Sources & References

This report is based on internal company reports, maps, published government reports, and public information, as listed in Section 27 of this report. It is also based on the information cited in Section 3.

The authors are not experts with respect to legal, socio-economic, land title, or political issues, and are therefore not qualified to comment on issues related to the status of permitting, legal agreements, and royalties. Information related to these matters has been provided directly by O3 Mining and include, without limitation, validity of mineral tenure, status of environmental and other liabilities, and permitting to allow completion of environmental assessment work. These matters were not independently verified by the QPs but appear to be reasonable representations that are suitable for inclusion in Chapter 4 of this report.

2.7 Previous Technical Reports

The following technical reports have previously been filed on the Marban property:

- Raponi, Tommaso R., Purchase, James, Gariepy, Louis, Vigneau, Sébastien. 2022. NI 43-101 Technical Report and Mineral Resources Estimate for Marban Engineering. Report prepared by Ausenco Engineering Canada for O3 Mining Inc.
- Raponi, Tommaso R., Elfen, Scott, Petrina, Mike, Bird, Sue, Fontaine, René. 2020. NI 43-101 Technical Report & Preliminary Economic Assessment of the Marban Engineering Project, Quebec, Canada. Report prepared by Ausenco Engineering Canada for O3 Mining Inc.
- Belzile, Elzéar, 2016. Updated Mineral Resource Technical Report on the Marban Block Property, Quebec, Canada. Report prepared by Belzile Solutions Inc. for Osisko Mining Inc.
- Gustin, Michael M., and Ronning, Peter, 2013. Updated Mineral Resource Technical Report on the Marban Block Property, Quebec, Canada. Report prepared by Mine Development Associates for NioGold Mining Corporation.

2.8 Unit and Name Abbreviations

Table 2-2: Acronyms and Abbreviations

Acronym	Definition
ALS	ALS Chemex Labs Ltd. / ALS Minerals
AP	acidification potential
ARD	acid rock drainage
BAPE	Environmental Public Hearing Office (Bureau d'audience publique sur l'environnement)
BSI	Belizile Solutions Inc.
BWi	ball mill work index
CCME	Canadian Council of Ministers of the Environment
CDC	map designated claim (claim désigné sur carte)
CEAA	Canadian Environmental Assessment Act
CFE	concentration of frequent effects
CIP	carbon-in-pulp
CLLFZ	Cadillac-Larder Lake Fault Zone
CND	contaminated neutral drainage
CNSC	Canadian Nuclear Safety Commission
COE	concentration of occasional effects
CoG	Cut-off grade
CPC (EO)	contamination prevention criteria (water and aquatic organisms)
CPE	concentration of probable effects
CPP	cumulative probability plots
CRE	concentration of rate effects
CRM	Certified Reference Materials
CTEU-9	Equilibrium extraction water leach test
CV	coefficient of variation
CVAC	Chronic aquatic life criteria

CWi	crusher work index
DD	diamond drilling
DFO	Oceans and Fisheries Canada
DL	detection limit
DTW	down the hole
E-GRG	Extended gravity recoverable gold
ECCC	Environmental and Climate Change Canada (Federal)
EIA	Environmental Impact Assessment
EOP	End of Project
EOY	End of Year
EPCM	Engineering, Procurement and Construction Management
EQA	Environmental quality act
FA-AA	fire assay - atomic absorption
GME	General Mining Expenses
GMS	G Mining Services Inc.
GRG	Gravity recoverable gold
GW	ground water
HARD	half absolute relative difference
HQ	Hydro Quebec
IAA	Impact Assessment Act
IP	induced polarization
KN	Kierens-Norlartic
LG	Lerchs-Grossman
LGS	Low-grade stockpile
LOM	Life-of-Mine
M&I	Measured and Indicated
MABA	Modified acid-base accounting
MARC	Maintenance and Repair Contract
MDMER	Metal and Diamond Mining Effluent Regulations
MELCC	Ministry of the Environment and the Fight Against Climate Change (Ministère de l'Environnement et de la Lutte contre les changements climatiques) (Provincial)
MERN	Ministry of Energy and Natural Resources (Ministère de l'Énergie et Ressources Naturelles)
MFFP	Ministry of Forests, Wildlife and Parks (Ministère des Forêts, de la Faune et des Parcs)
ML	metal leaching
MMTS	Mouse Mountain Technical Services
MNR	Ministry of Natural Resources
MRCVO	Municipalité Régionale de Comté Vallée de L'or (Regional County Municipality of Vallée de L'or
MRE	Mineral Resource Estimate
MTZ	Malartic Tectonic Zone
NNP	net neutralisation point
NP/AP	neutralizing potential / acid potential
NAG	not potentially acid generating
NPI	net profits interest

NPV	net present value
NSP	net smelter price
NSR	net smelter return
NVZ	Northern Volcanic Zone
OBS	Overburden Stockpile
OEM	Original Equipment Manufacturer
OER	environmental discharge objectives
OP	open pit
OVB	Overburden
PEA	Preliminary Economic Assessment
PGA	Potential generator of acid
PFS	Prefeasibility Study
PMF	Probable maximum flood
PMP	Probably maximum precipitation
PNN	net neutralizing power
PN	neutralization potential
QA/QC	Quality Assurance/Quality Control
QP	Qualified Person
RBQ	Quebec Building Authority (Régie du bâtiment du Québec)
RC	reverse circulation
RCM	Regional County Municipality
RES	Resurgence in surface water (Resurgence de eaux de surface)
ROM	Run of mine
RPEEE	Reasonable prospects of eventual economic extraction
RWi	bond rod mill work index
SA	alert threshold (for its acronym in French, seuil d'alerte)
SG	specific gravity
SPLP	synthetic precipitation leaching procedures
SQ	Quebec Security (Sureté du Québec)
SVZ	Southern Volcanic Zone
TCE	threshold concentration producing an effect
TCLP	toxicity characteristic leaching procedure
TSF	Tailings Storage Facility
UG	underground
UCF	undiscounted cashflow
UGAF	fur-bearing animal management unit (Quebec)
W:O	Waste to ore ratio
WRS	Waste Rock Stockpile
WMP	Water Management Pond

Table 2-3: Units of Measurement

Abbreviation	Definition
CAD	Canadian dollar (symbol: C$)
USD	United States dollar (symbol: US$)
°	degrees
oz/ton Au	ounces of gold per ton
g/t Au	grams of gold per tonne
cm	centimetre
dBA	A-weighted decibels
ft	feet
g	gram
g/t	grams per tonne
ha	hectare
km	kilometre
koz	thousand ounces
kt	thousand tonnes
kt/d	thousand tonnes per day
m	metre
mm	millimetre
Mt/a	Million tonnes per annum
Moz	million ounces
Mt	million tonnes
Mt/y	million tonnes per year
t/d	tonnes per day

3 RELIANCE ON OTHER EXPERTS

While the authors have carefully reviewed, within the scope of their technical expertise, all the available information presented to them, 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.

The QPs have relied upon the following for tax advice and tax calculations:

- "RE: Tax Section - Marban Technical Report Ch. 22" received by email sent by O3 Mining on September 7, 2022
- "Marban PFS Financial Model – rev C – Aug 24 for Tax Update.xlsx" received by email sent by O3 Mining on August 24, 2022.

The information has been relied upon in Section 1, 22.3.1, 22.6, 22.8, and 25.13.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Description & Location

4.1.1 Location

The Marban property is located in the western portion of the province of Quebec, Canada, about midway between the towns of Val-d'Or and Malartic (Figure 4-1) and is comprised of 181 mining claims covering 7,701.97 hectares (Figure 4-1 and Table 4-1). The property lies at the junction of Dubuisson, Fournière, Malartic, and Vassan townships, centered at Latitude 48.17°N, Longitude 78.05°W, on NTS sheets 32D/01 and 32C/04.

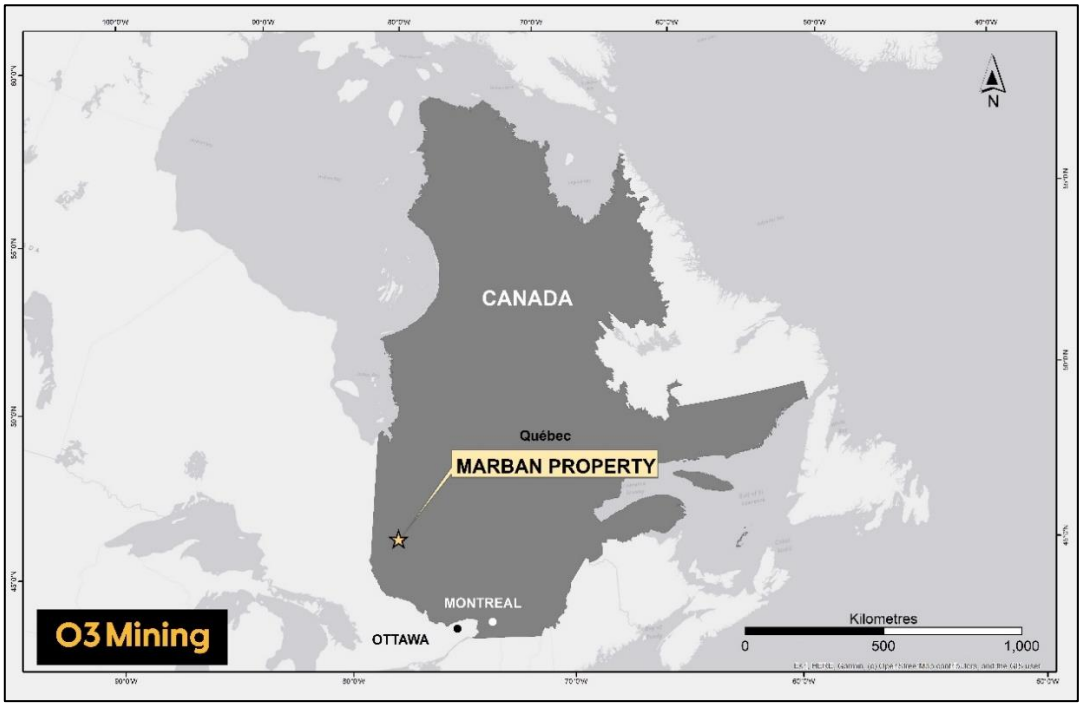
The Marban property is divided into two projects. A first one, Marban Engineering, corresponds to the southeast portion of the property and includes all the infrastructures related to this economic study. A second one, Marban Greenfield, is broader and includes all the north and west parts of the Marban property.

4.1.2 Land Area

Marban Engineering, which contains the resources that are the subject of this report, is the result of the amalgamation of the contiguous Gold Hawk, First Canadian, Norlartic, and Marban historical properties. O3 Mining owns a 100% interest in the Marban property, which hosts the Marban Engineering (Figure 4-2). The remainder of the Marban property is not the subject of this report. On February 25, 2022, as a result of the amalgamation of the companies controlled by O3 Mining Inc., the claims pertaining to the Marban property and previously owned by NioGold Mining Corporation (100% controlled by O3 Mining Inc.), became registered under O3 Mining Inc. Finally, on March 15, 2022, O3 Mining purchased the East-West property from Emergent Metals Corp. (formerly Emgold Mining Corporation). The East-West property contains seven map designated claims totalling 184.18 hectares and contiguous to the Marban property in its southeast corner.

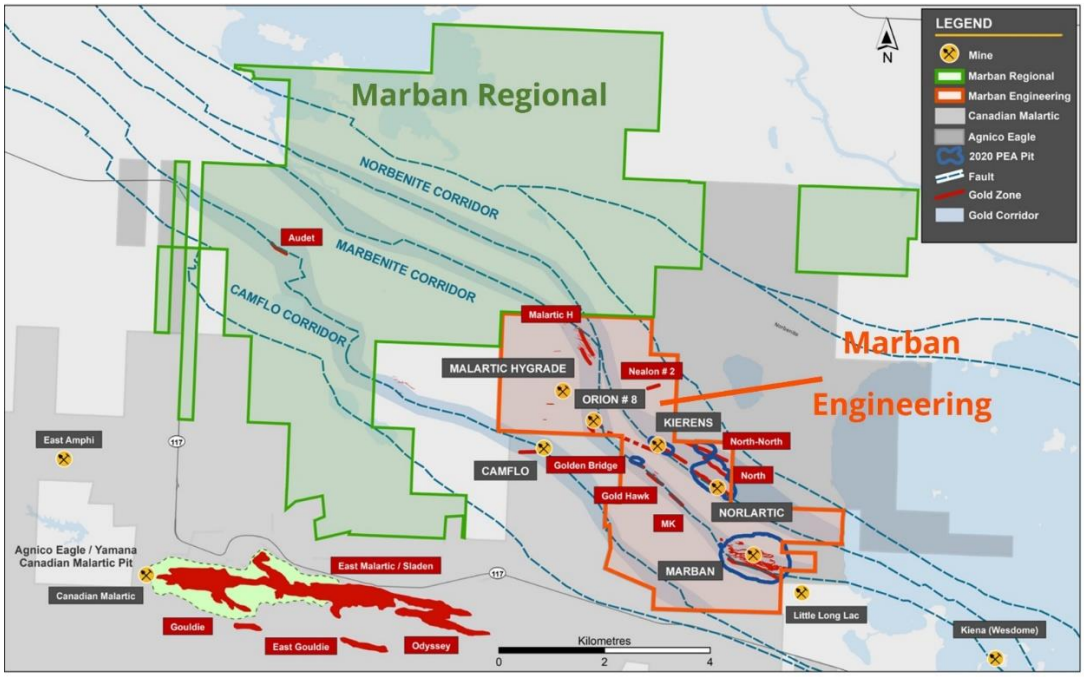
On March 16, 2015, the Marban property stacked claims were converted in new map designated claims. Thus, the property surface was increased slightly increased so that it now covers 7,701.97 hectares (including the East-West property). Also, on October 26, 2021, four mining concessions (CM 474, 486, 512, and 513) were abandoned and adjacent map designated claims were extended to cover the area of those old concessions. Consequently, the Gold Hawk, First Canadian, Norlartic, and Marban original contours do not fit the actual claims boundaries. The Marban historical property, which includes the gold resources of the Marban deposit, covered 718.16 hectares and included three mining concessions; it was formerly jointly owned by Aur Resources Inc. (Au) and McWatters Mining Inc. (McWatters), who each held a 50% interest. The Marban mine, a past producer, is located on the Marban historical property. The past-producing Kierens and Norlartic mines, as well as the Kierens and Norlartic resources discussed in this report, are located on the First Canadian and Norlartic historical properties, respectively; the North and North-North mineralized zones are also located on the Norlartic historical property. The First Canadian and Norlartic historical properties were formerly held by Aur and covered 196.77 hectares. The Gold Hawk historical property covered 61.40 hectares; the Gold Hawk zone, which has experienced limited underground development and mining accessed through the Kierens mine, lies within this historical property. J. L. Corriveau & Associates Inc. surveyed the Gold Hawk, First Canadian, Norlartic, and Marban historical properties boundaries in 2006.

Figure 4-1: Location of the Marban Property



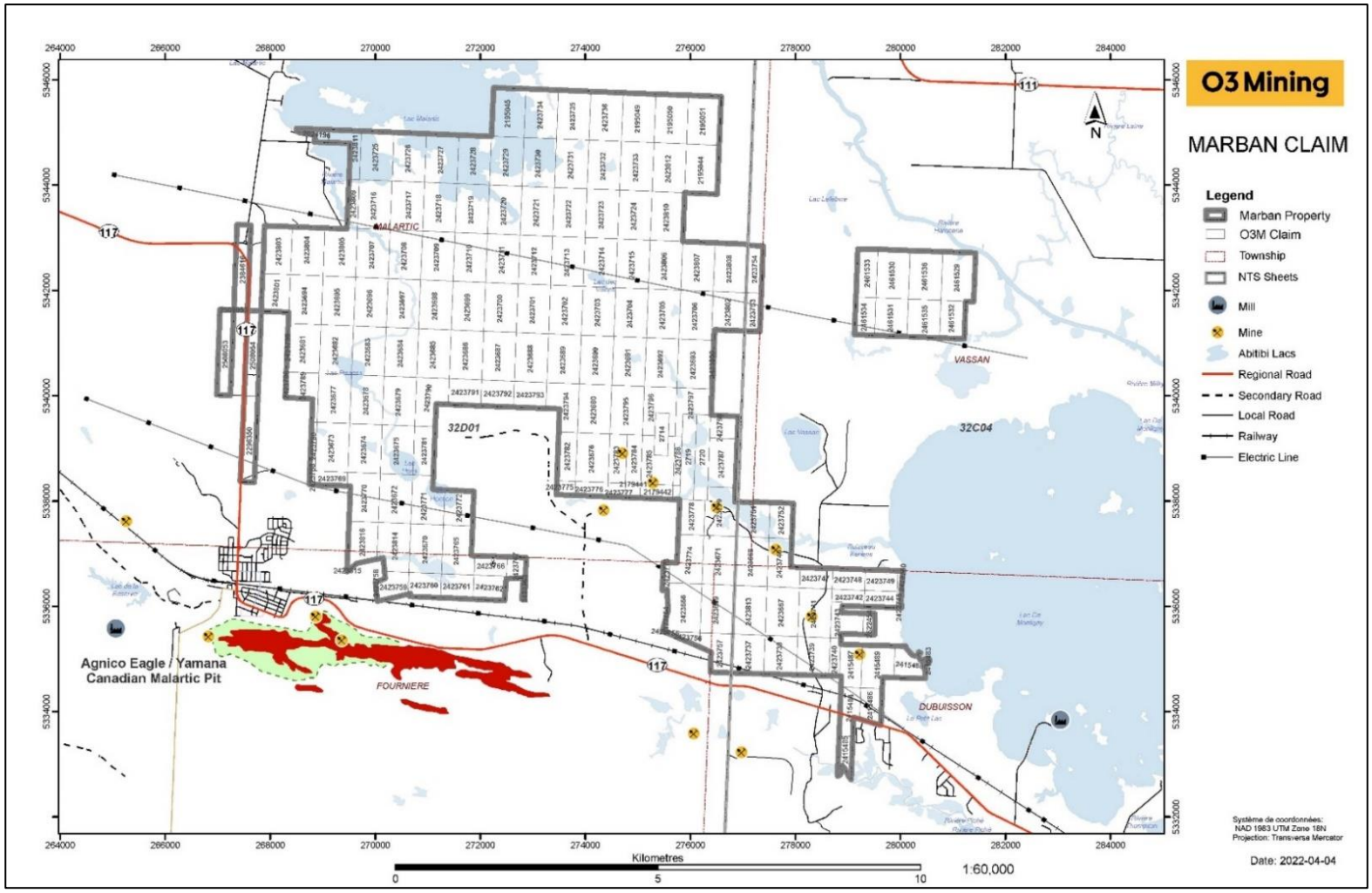
Source: O3 Mining, 2022.

Figure 4-2: Location of O3 Mining Marban Engineering within the Marban Property



Source: O3 Mining, 2022.

Figure 4-3: Marban Property Mining Claims



Source: O3 Mining, 2022.

Table 4-1: Marban Property Claims List

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2714	32D/01	21.07	2024-09-23	2% NSR Osisko Gold Royalties
2719	32D/01	42.54	2024-09-01	2% NSR Osisko Gold Royalties
2720	32D/01	42.54	2024-09-01	2% NSR Osisko Gold Royalties
2179441	32D/01	7.69	2024-02-15	2% NSR Republic Goldfields Inc.
2179442	32D/01	16.40	2024-02-15	2% NSR Republic Goldfields Inc.
2195044	32D/01	57.41	2024-11-19	
2195045	32D/01	57.40	2024-11-19	
2195049	32D/01	57.40	2024-11-19	
2195050	32D/01	57.40	2024-11-19	
2195051	32D/01	57.40	2024-11-19	
2296350	32D/01	43.41	2024-06-14	
2384616	32D/01	42.42	2024-04-23	
2415483	32C/04	0.28	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415484	32C/04	31.25	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415485	32C/04	18.05	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415486	32C/04	25.80	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415487	32C/04	33.45	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415488	32C/04	29.19	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2415489	32C/04	46.16	2024-07-12	1.75% NSR Daniel Pratt, 1.25% NSR Jeremy Mersereau
2423666	32D/01	57.49	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423667	32C/04	57.49	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423668	32C/04	57.49	2024-02-12	2–3% Royal Gold Inc.,1-2% NSR Osisko Gold Royalties,1% Canhorn, 9 2/3% NPI Compressario Corporation, 1/3% NPI Anthony Camisso
2423669	32D/01	57.49	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423670	32D/01	57.48	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423671	32D/01	57.49	2024-02-12	2% NSR NOMAD, 1% NSR OGR, 2-3% Royal Gold Inc.,1% Canhorn, 9 2/3% NPI Compressario Corporation, 1/3% NPI Anthony Camisso
2423672	32D/01	57.47	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423673	32D/01	57.46	2024-02-12	2–3% NSR Royal Gold Inc.,1% NSR Michel Lavoie
2423674	32D/01	57.46	2024-02-12	2–3% NSR Royal Gold Inc.,1% NSR Michel Lavoie, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423675	32D/01	57.46	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423676	32D/01	57.47	2024-02-12	1.5% NSR L. Audet et Jean Robert
2423677	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc.,1% NSR Michel Lavoie, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2423678	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc.,1.5% NSR L.Audet,1% NSR Michel Lavoie, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423679	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423680	32D/01	57.46	2024-02-12	1% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 5% NPR Delfer Gold Mine
2423681	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423682	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423683	32D/01	57.44	2024-02-12	1.5% NSR Léo Audet, 2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423684	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423685	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423686	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423687	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423688	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423689	32D/01	57.45	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423690	32D/01	57.45	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423691	32D/01	57.45	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423692	32D/01	57.45	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423693	32D/01	57.45	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423694	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423695	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc.,1,5% NSR Léo Audet, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423696	32D/01	57.43	2024-02-12	1.5% NSR Léo Audet, 2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423697	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423698	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423699	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423700	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423701	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423702	32D/01	57.44	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423703	32D/01	57.44	2024-02-12	1–3% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423704	32D/01	57.44	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423705	32D/01	57.44	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423706	32D/01	57.44	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423707	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc.,1,5% NSR Léo Audet, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423708	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2423709	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423710	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423711	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423712	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423713	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423714	32D/01	57.43	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423715	32D/01	57.43	2024-02-12	1% NSR Royal Gold Inc.
2423716	32D/01	57.42	2024-02-12	2% NSR Osisko Gold Royalties
2423717	32D/01	57.42	2024-02-12	2% NSR Osisko Gold Royalties
2423718	32D/01	57.42	2024-02-12	2% NSR Osisko Gold Royalties, 1% NSR Jean-Luc Gauthier
2423719	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423720	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423721	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423722	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423723	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423724	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423725	32D/01	57.41	2024-02-12	2% NSR Osisko Gold Royalties
2423726	32D/01	57.41	2024-02-12	2% NSR Osisko Gold Royalties
2423727	32D/01	57.41	2024-02-12	2% NSR Osisko Gold Royalties, 1% NSR Jean-Luc Gauthier
2423728	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423729	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423730	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423731	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423732	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423733	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423734	32D/01	57.40	2024-02-12	
2423735	32D/01	57.40	2024-02-12	
2423736	32D/01	57.40	2024-02-12	
2423737	32C/04	45.46	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423738	32C/04	45.11	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423739	32C/04	44.74	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423740	32C/04	14.44	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423741	32C/04	57.49	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423742	32C/04	17.56	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423743	32C/04	30.65	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423744	32C/04	18.77	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423745	32C/04	2.48	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2423746	32C/04	47.45	2024-02-12	2–3% Royal Gold Inc.,1-2% NSR Osisko Gold Royalties,1% Canhorn
2423747	32C/04	24.41	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423748	32C/04	24.59	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423749	32C/04	24.76	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423750	32C/04	3.38	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423751	32C/04	41.38	2024-02-12	1–2% NSR Osisko Gold Royalties, 9 2/3% NPI Compressario Corporation, 1/3% NPI Anthony Camisso
2423752	32C/04	29.27	2024-02-12	2% NSR Osisko Gold Royalties
2423753	32C/04	21.10	2024-02-12	1% NSR Royal Gold Inc.
2423754	32C/04	26.37	2024-02-12	1% NSR Royal Gold Inc.
2423755	32D/01	0.11	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423756	32D/01	9.21	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423757	32D/01	36.17	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423758	32D/01	0.05	2024-02-12	
2423759	32D/01	27.38	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423760	32D/01	27.99	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423761	32D/01	28.22	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423762	32D/01	27.55	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423763	32D/01	2.70	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423764	32D/01	4.84	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423765	32D/01	53.61	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423766	32D/01	23.00	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423767	32D/01	13.30	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423768	32D/01	2.72	2024-02-12	
2423769	32D/01	16.02	2024-02-12	2–3% NSR Royal Gold Inc.,1% NSR Michel Lavoie
2423770	32D/01	53.47	2024-02-12	2–3% NSR Royal Gold Inc., 3% NSR M. Lavoie, A. Auger et P. Sigouin, 1% NSR Michel Lavoie, 1% NSR Breakwater Resources, 5% NPR Alfer Inc. et René Amyot
2423771	32D/01	51.46	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423772	32D/01	36.71	2024-02-12	
2423773	32D/01	5.09	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
2423774	32D/01	50.53	2024-02-12	2% NSR NOMAD,1% NSR Osisko Gold Royalties, 2-3% Royal Gold Inc.,1% Canhorn
2423775	32D/01	9.45	2024-02-12	1.5% NSR L. Audet et Jean Robert
2423776	32D/01	16.29	2024-02-12	1.5% NSR L. Audet et Jean Robert
2423777	32D/01	8.66	2024-02-12	1.5% NSR L. Audet et Jean Robert, 2% Républics Goldfields Inc.
2423778	32D/01	36.05	2024-02-12	2% NSR NOMAD, 2% NSR Republic Goldfields Inc.
2423779	32D/01	54.89	2024-02-12	2% NSR NOMAD,1% Osisko Gold Royalties, 1.5% NSR L. Audet et J. Robert, 9 2/3% NPI Compressario Corporation, 1/3% NPI Anthony Camisso

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2423780	32D/01	9.76	2024-02-12	
2423781	32D/01	35.98	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423782	32D/01	33.50	2024-02-12	1.5% NSR L. Audet et Jean Robert
2423783	32D/01	30.51	2024-02-12	1.5% NSR L. Audet et Jean Robert, 2% Républics Goldfields Inc.
2423784	32D/01	26.95	2024-02-12	1.5% NSR L. Audet et Jean Robert, 2% NSR Republic Goldfields Inc.
2423785	32D/01	47.77	2024-02-12	1.5% NSR L. Audet et Jean Robert, 2% NSR Republic Goldfields Inc.
2423786	32D/01	12.53	2024-02-12	1.5% NSR L. Audet et Jean Robert, 2% NSR Republic Goldfields Inc.
2423787	32D/01	48.48	2024-02-12	1.5% NSR L. Audet et Jean Robert
2423788	32D/01	0.37	2024-02-12	
2423789	32D/01	34.75	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423790	32D/01	47.27	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423791	32D/01	30.20	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423792	32D/01	30.23	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423793	32D/01	30.26	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423794	32D/01	46.14	2024-02-12	2–3% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423795	32D/01	57.46	2024-02-12	1% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 5% NPR Delfer Gold Mine,2% Républics Goldfields Inc.
2423796	32D/01	46.08	2024-02-12	1% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 5% NPR Delfer Gold Mine
2423797	32D/01	36.45	2024-02-12	1% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 5% NPR Delfer Gold Mine
2423798	32D/01	25.05	2024-02-12	1% NSR Royal Gold Inc., 1.5% NSR L. Audet et Jean Robert, 5% NPR Delfer Gold Mine
2423799	32D/01	0.74	2024-02-12	
2423800	32D/01	4.56	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423801	32D/01	38.11	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423802	32D/01	45.75	2024-02-12	1% NSR Royal Gold Inc., 5% NPR Delfer Gold Mine
2423803	32D/01	48.60	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423804	32D/01	56.48	2024-02-12	2–3% NSR Royal Gold Inc., 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423805	32D/01	56.79	2024-02-12	2–3% NSR Royal Gold Inc.,1,5% NSR Léo Audet, 1% NSR Nyrstar, 5% NPR Alfer Inc. et René Amyot
2423806	32D/01	57.43	2024-02-12	1% NSR Royal Gold Inc.
2423807	32D/01	57.43	2024-02-12	1% NSR Royal Gold Inc.
2423808	32D/01	57.43	2024-02-12	1% NSR Royal Gold Inc.
2423809	32D/01	18.98	2024-02-12	2% NSR Osisko Gold Royalties
2423810	32D/01	57.42	2024-02-12	1% NSR Jean-Luc Gauthier
2423811	32D/01	28.82	2024-02-12	2% NSR Osisko Gold Royalties
2423812	32D/01	57.41	2024-02-12	1% NSR Jean-Luc Gauthier
2423813	32C/04	57.49	2024-02-12	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn

Claim No.	SNRC	Area (ha)	Expiry Date	Royalty
2423814	32D/01	54.13	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2423815	32D/01	0.05	2024-02-12	
2423816	32D/01	44.08	2024-02-12	3% NSR M. Lavoie, A. Auger et P. Sigouin
2461529	32C/04	57.43	2023-10-25	
2461530	32C/04	57.43	2023-10-25	
2461531	32C/04	43.99	2023-10-25	
2461532	32C/04	33.05	2023-10-25	
2461533	32C/04	35.94	2023-10-25	
2461534	32C/04	27.43	2023-10-25	
2461535	32C/04	43.96	2023-10-25	
2461536	32C/04	57.43	2023-10-25	
2508053	32D/01	42.52	2022-12-19	
2508054	32D/01	42.61	2022-12-19	
2622454	32C/04	8.50	2024-10-26	2–3% Royal Gold Inc.,1% NSR Osisko Gold Royalties,1% Canhorn
Total		7701.97		

In Quebec, a claim is a mineral right that gives its holder a 2-year exclusive right to explore a designated territory for any mineral substances that are part of the public domain, with the exception of petroleum, natural gas, brine, sand, gravel, clay and other mineral substances found in their natural state as loose deposits. Mining concessions grant title to land and minerals for mining purposes. Concessions are no longer granted, replaced by mining leases that grant the right to mine mineral substances. Mining leases expire after 20 years if there has been no production from the leased area during that time. Claims can be converted to a lease if the existence of a workable deposit can be demonstrated.

The Ministère de l'Énergie et Ressources Naturelles (MERN) administers the lands in the Marban property area. The Marban mining claims are renewed every 2 years and require the completion of assessment work. The mining claims that comprise the Marban property are listed as being in good standing on MERN's 'Gestim' claim management website.

4.2 Property Ownership Agreements

O3 has currently 100% ownership of the 181 claims that form the Marban property.

4.2.1 Gold Hawk, Kierens, Norlartic, & Marban Claims

On July 5, 2019, Osisko Mining Inc. completed a reverse takeover of Chantrell Ventures. The transaction resulted in, among other things, Osisko Mining transferring certain of its non-core assets to Chantrell in exchange for common shares of Chantrell by way of a plan of arrangement under the Business Corporations Act (Ontario) (the Arrangement) and name change to O3 Mining. Under the Arrangement, Osisko Mining transferred to O3 Mining the Marban property, the Garrison property, exploration properties and a portfolio of selected securities, in exchange for 24,977,898 post-consolidation common shares of O3 Mining, representing approximately 82.2% of the issued and outstanding common shares of O3 Mining.

4.2.2 Purchase of the 50% Northern Star Claims

On November 9, 2020, O3 Mining completed the purchase of the remaining 50% of Northern Star claims (also known as the Virginia claims) for US$150,000 pursuant to the terms of the purchase agreement between Niogold Mining Corporation, a corporation owned and controlled by O3 Mining, and 9265-991 Quebec Inc., the corporation who acquired the mining interests of Northern Star Mining Corporation in 2013. With this purchase O3 Mining now holds 100% ownership of three claims, totalling 106.5 hectares, adjacent to the northwest of the Kierens deposit.

4.2.3 Purchase of East-West Property from Emergent

On March 15, 2022, O3 Mining signed a binding letter agreement with Emergent Metals Corp. (formerly Emgold Mining Corporation) to acquire their East-West property in exchange for i) cash consideration of C$750,000, (ii) 325,000 common shares in the capital of the Corporation; and (iii) the grant of a 1% net smelter returns royalty over the East-West property in favour of Emgold, which will be subject to a buy-back right in favour of O3 Mining. The Buy-Back Right may be exercised until the fifth anniversary from the closing date for a cash payment of C$500,000 until the third anniversary from the closing date and C$1,000,000 until the fifth anniversary from the closing date. The property is adjacent to and east of the Marban property and consists of seven mining claims covering an area of 184.18 hectares.

4.3 Surface Rights

O3 Mining has obtained authorization from landowners to conduct its exploration activities. The company has started to acquire some surface rights as the project develops and has established a plan to acquire the land required to develop the project. O3 Mining does not currently foresee any issues regarding negotiating and/or acquiring the surface rights.

4.4 Royalties within Marban Engineering

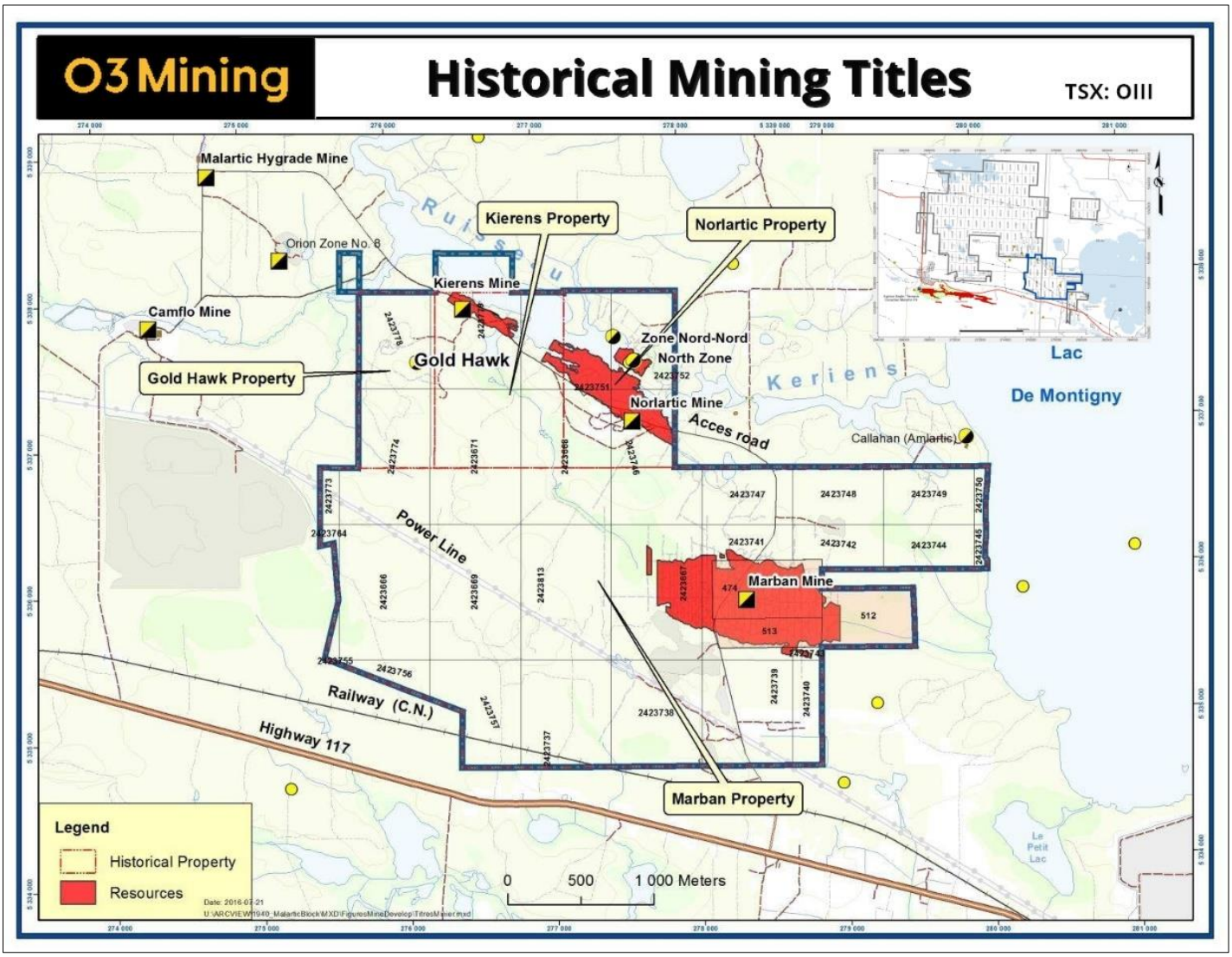
Table 4-2 and Figure 4-4 summarizes the net smelter returns (NSR) and net profits interest (NPI) royalties applicable to production from the Gold Hawk, First Canadian, Norlartic, and Marban historical properties and payable to companies previously involved with the property. In addition to these royalties, the Mining Tax Act of June 6, 2011, enacted a 16% fixed royalty on profits calculated on a mine-by-mine basis, meaning that a loss on one mine cannot be used to reduce the profits generated by another mine. Changes to this royalty have been proposed but not enacted as of the effective date of this report.

As per the option agreement with Aur dated February 3, 2006, upon a decision to proceed to production on any deposit located on the Norlartic, First Canadian, and Marban historical properties, O3 Mining must make a one-time production payment of C$5,000,000 to Aur (now Teck Resources Ltd). On October 19, 2015, Osisko Gold Royalties (Osisko GR) bought the Teck NSR and payment rights.

Table 4-2: Royalties Payable on Marban Engineering

Historical Property	Royalty (C$)	Mineral Resources Subject to Royalty
Norlartic	2.00% NSR payable to Osisko Gold Royalties pursuant to an agreement announced October 19, 2015, between Osisko Gold Royalties and Teck Resources Ltd. with buy-back clause for one-half of the NSR owned by Osisko Gold Royalties in consideration of C$1.0M at any time.	Norlartic
Kierens	1.00% NSR payable to Osisko Gold Royalties pursuant to an agreement announced October 19, 2015, between Osisko Gold Royalties and Teck Resources Ltd. with buy-back clause for one-half of the NSR owned by Osisko Gold Royalties in consideration of C$1.0M at any time.	Kierens
	9⅔% NPI payable to Compressario Corporation (formerly First Canadian Gold Corporation Inc.) pursuant to an agreement dated February 13, 1984, between Aur Resources Inc. (as successor to Brominco Inc.) and Compressario, as supplemented by agreements dated October 23, 1986, and February 25, 1987. This royalty only applies to claims 3357861, 3363141 and 3363142.	
	1/3% NPI payable to Anthony Camisso pursuant to a letter agreement dated February 25, 1987. This royalty only applies to claims 3357861, 3363141 and 3363142.	
Marban	1% NSR payable to Canhorn Mining Corporation pursuant to an agreement dated March 31, 1989, between Aur Resources Inc. and Canhorn. Buy back clause for 100% of the NSR for C$0.5M (source Osisko Mining annual information form Dec 31, 2017).	Marban
	0.5% NSR payable to Osisko Gold Royalties pursuant to an agreement announced October 19, 2015, between Osisko Gold Royalties and Teck Resources Ltd. with buy-back clause for one-half of the NSR owned by Osisko Gold Royalties in consideration of C$1.0M at any time.	Marban
	2–3% NSR on 50% of any production payable to RGLD Gold Canada Inc. pursuant to an agreement dated April 3, 2003, between Barrick Gold Corporation and McWatters Mining Inc. 2% NSR if gold < $350 or 3% NSR if gold > $350. Buy out option on half of royalty for $1.5M if gold > $350 or $1.0M if gold < $350. RGLD acquired the royalty interest from Barrick in 2008.	Marban

Figure 4-4: Historical Mining Properties at Marban Engineering



Source: O3 Mining, 2020.

4.5 Environmental Permits & Liabilities

On crown land, O3 Mining obtained intervention permits to perform its exploration activities delivered by the Ministère des Forêts, de la Faune et des Parcs (MFFP). The permit defines areas within which the vegetation may be cut for the purposes of conducting exploration, such as for drill access roads and pads. The approximate quantity of wood by species of trees encountered in the planned work is calculated, and a fee is paid to the government in an amount that reflects the volume of wood removed.

O3 Mining obtained permits to drill on the Keriens Creek from Fisheries and Oceans Canada and the Ministère de l'Environnement et de la Lutte aux Changements Climatiques (MELCC).

A landfill of residual materials, listed in the land-use planning and development plan of the RMC La Vallée-de-l'Or (MRCVO, revised 2019), is located within the proposed layout. Landfill may present certain environmental and public health risks. Runoff from these sites facilitates the migration of contaminants, while percolation through soil acts in the same way on groundwater. In addition, a landfill can give off gases and vapours.

Three old mine sites are located within the proposed layout, where activities designated under the Land Protection and Rehabilitation Regulation were carried out: Kierens site, Norlartic site, and Marban site. According to O3 Mining, the sites were cleaned of waste materials, but leachable waste rock backfill (Golder, 2013) and contaminated soil might still be present. Some waste materials and potentially hazardous materials were found on a private property located within the proposed layout: garbage, reservoirs, cars and other types of waste (WSP, 2018d). As those properties and lands will have to be acquired for the carrying out of the project, O3 Mining will have to evaluate the need for additional soil characterization on potentially contaminated lands to evaluate the environmental liability.

4.6 Risks and Opportunities

To the extent known, there are no significant factors and risks that may affect access, title, or the right or ability to perform work on the property. There are opportunities to expand the property through negotiations with neighboring mining claims owners.

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

5.1 Accessibility

The Marban Engineering project is located on the north side of Highway 117, approximately 15 km northwest of Val-d'Or, Quebec and immediately east of Canadian Malartic Partnership's Camflo property. The project is accessible by the Gervais Road, a well-maintained all-weather gravel road running north from Highway 117. It provides access to the historic Norlartic, Kierens, and Marban mines. The Camflo road provides access to the western part of the project area. Winter access for snowmobiles and all-terrain vehicles is provided by trails and winter roads.

5.2 Physiography

The Marban Engineering projectis part of the Harricana River watershed, which is part of the James Bay hydrographic basin. The mean elevation of the property is about 300 masl. The property is relatively flat, with local relief of 15 to 30 m. The surface is dominated by swamps and glacial deposits; a north-trending esker lies on the west side of the Gervais Road. The overburden is generally between 10 and 40 m thick, but locally it is very shallow and there are some outcrops (Ducharme et al., 2009).

The De Montigny Lake borders the eastern edge of the Marban Engineering project, while the Keriens Creek trends northwest across the northern Kierens-Norlartic area of the project.

5.3 Climate & Vegetation

The climate is continental and is characterized by relatively dry, cold winters and hot, damp summers. The average daily maximum and minimum annual temperature are 7.5°C and –4.5°C, respectively, with January temperature averaging -17°C and July averaging 17°C (Rivière-Heva Station: MELCC). The temperature is at or below freezing from November to mid-April. The coldest temperature recorded was -43.9°C in 1962 and the hottest temperature recorded was 36.1°C in 1975 (2005 Val-d'Or Station: ECC Canada). The annual rain precipitation is approximately 845 mm and the annual snow precipitation is approximately 240 cm (Rivière-Heva Station: MELCC). Snow falls from November to March and usually remains on the ground until mid-April. Mining and processing activities can proceed 365 days per year.

The project is located in the boreal forest vegetation zone, within the balsam fir/white birch bioclimatic domain. The vegetation is mainly composed of mixed and deciduous stands, and to a lesser extent, of coniferous stands. There are also wooded wetlands and open wetlands. Wetlands also occupy a large area of the proposed implantation site and are located in relatively flat areas and along waterways.

5.4 Local Resources & Infrastructure

5.4.1 Airports, Rail Terminals, & Bus Services

The Marban Engineering project is located midway between the towns of Malartic and Val-d'Or, Quebec. The town of Val-d'Or (population 33,000) is located 15 km east of the project along the provincial Highway 117. Val-d'Or is one of the largest communities in the region and has all major services including an airport with scheduled service from Montreal. Val-d'Or is a major service centre for the mining and lumber industries. A CN railway line crosses the southern part of the property, connecting east through to Montreal and west to the North American rail network. Val-d'Or is located six hours from Montreal by road, and there is daily bus service between Montreal and the other cities in the Abitibi-Témiscamingue region.

The Canadian National Railroad and a power line run parallel to Highway 117 immediately south of the property, and mine sites have access to power and telecommunication systems. A 120 kV powerline crosses through the Marban Engineering

Project property. The communities of Malartic and Val-d'Or, each within 15 km of the property, provide full services for exploration and mining, including a labour force experienced in mining.

5.4.2 Local Labour

According to the 2021 census prepared by Statistics Canada, the population of the Regional County Municipality (RMC) of La Vallée-de-l'Or was 43,648, including 28,022 residents aged 15–64, and a median age of 41.9 years old. Male population accounts for 51.3% of the population, 48.7% is female, and 6.4% is Aboriginal. In 2021, 78.8% of the population participated in the labour force, with 13% of the labour force employed in the natural resources sector (Observatoire de l'Abitibi-Témiscamingue, 2021). This portion of the workforce is experienced in mining operations, as they are currently employed at exploration and gold mines located elsewhere in the region.

5.4.3 Support Services

Local resources also include commercial laboratories, drilling companies, exploration service companies, engineering consultants, construction contractors and equipment suppliers. Val-d'Or services include a hospital, grocery stores, fuel stations, financial institutions and hotels. Val-d'Or has a post office and additional shipping/freight services by several providers. Landline telephone, mobile service, high-speed internet and satellite internet are available in town and the nearby vicinity.

5.5 Community

The project is located in the administrative region of Abitibi-Témiscamingue (08) in the RMC of La Vallée de l'Or. The proposed layout spreads across the territory of two municipalities: Val-D'Or in its eastern portion and Malartic in its western portion. The nearest agglomeration is the Dubuisson district located approximately 600 m from the southern claims limits. Land tenure is a mix of public, private, and municipal property. No federal land is located within the project area. No reserve land is located within the proposed project layout. The project area, however, is located on land that is subject to a comprehensive land claims agreement or a self-government agreement.

O3 Mining has implemented a comprehensive plan to engage with stakeholders. Consultation activities with the local communities are described in details in Chapter 20.

6 HISTORY

6.1 Introduction

This section describes exploration and production history within the area of Marban Engineering prior to creation of O3 Mining in 2019 (see Figure 4-2).

Except where otherwise noted, the information in this section is taken from the original Technical Report (Carrier, 2006) with updated information added from Lefrançois (2006, 2007), Ducharme et al. (2009), Gustin (2013), and Belzile (2016).

The Malartic gold district in the southern Abitibi greenstone belt has produced more than an estimated 15 million ounces of gold, of which 13.7 million ounces is attributable to the Canadian Malartic gold camp (Lehouiller et al., 2020; Arthurion et al., 2021). There are six past-producing mines within Marban Engineering Project — the Marban, Norlartic, Kierens, Malartic Hygrade, Orion No. 8, and Camflo (below 800 m from the surface).

Exploration conducted at the Marban Engineering Project dates back to at least 1940 and includes geologic mapping, sampling, compilation of geological, structural, and geochemical data, geophysical prospecting, trenching, and extensive drilling from the surface and underground. At least 14 different companies explored and/or mined on parts of the property in the period of 1940 through 1994, including Norbenite Malartic Mines, Marbenor Malartic Mines, Little Long Lac Gold Mines Limited (Little Long Lac), Norlartic Mines Ltd., Marban Gold Mines (Marban Mines), East Malartic Mines, Malartic Gold Fields Ltd., First Canadian Gold Corp., Lloyd Harvey, Brominco Inc. (Brominco), Gold Hawk Resources, SOQUEM (Québec Department of Mines), Villebon Resources, Lac Minerals (Lac), Aur, and International Thunderwood Exploration.

The first period of gold production on Marban Engineering was during the 1960s, primarily from the Marban and Norlartic mines. The second phase of production was during the 1980s and early 1990s from the Norlartic, Kierens and Camflo mines. Production from the operations during these two periods totalled almost 900,000 ounces of gold from approximately 5.4 million tonnes of ore. Table 6-1 lists the estimated gold production from the Marban Engineering deposits from 1959 to 1992. The North Zone deposit was accessed through the Norlartic mine workings, the Gold Hawk zone through the Kierens mine workings, and Orion No. 8 through the Malartic Hygrade mine workings.

The grades listed in Table 6-1 are uncertain for most deposits, as some of the production reports combine records for two or more deposits, and it is usually not clear whether the reported grades represent recovered or in situ grades. It is also unclear in some cases whether the reported numbers include or exclude waste from developmental workings.

Table 6-1: Estimated Historical Gold Production from Marban Engineering (1959–1992)

Mine	Company	Years	Tonnes	g/t Au	Au Ounces
Marban	Marban Gold Mines	1961–1974	1,983,000	5.3	330,000
Norlartic and North-North	Norlartic Mines	1959–1966	1,544,000	4.2	209,000
	Aur Resources	1990–1992			
Kierens and Gold Hawk	Norlartic Mines	1965–1966	251,000	6.3	52,000
	Aur Resources	1988–1992			
Malartic Hygrade	Malartic Hygrade	1962–1963	28,000	19.6	18,000
Camflo (on Marban property)	Barrick	1981–1992	1,435,000	5.3	243,000
Orion No. 8	Malartic Hygrade	1987–1990	131,000	5.3	22,000
		Totals	5,372,000	5.1	873,000

Note: Table adapted from Ducharme et al., 2009, with changes by MDA.

Some of the production numbers in Table 6-1 conflict with those presented by Carrier (2006) in Table 6-2 and in the following text. Ausenco has allowed these conflicts to remain in the document in order to reflect the uncertainties of the historic production at Marban Engineering.

6.2 Exploration History

In the following discussion, all tonnages were converted from short tons to metric tonnes, and feet have been converted to metres.

Table 6-2 lists pre-O3 Mining exploration chronologically and references activity at the four historical properties located within Marban Engineering, i.e., the Marban, Norlartic, First Canadian (Kierens) and Gold Hawk groups (Figure 4-3). The property was inactive from 1994 until NioGold consolidated the claim groups and re-initiated exploration in 2006. Aurizon signed an option and joint venture agreement for the Marban property with NioGold on July 5, 2010, and Hecla acquired Aurizon in June 2013, and then terminated the agreement on August 20, 2013. NioGold pursued the exploration effort until 2016 when Oban Mining Corporation (later Osisko Mining) took it over. Osisko Mining briefly explored the property and spun it off into O3 Mining in 2019 to focus on the Windfall property.

Table 6-2: History of Exploration on Marban Engineering (Revised from Carrier, 2006 and Belzile, 2016)

Year	Company - Property	Work Description & Results	Reference
1940	Norbenite Malartic Mines – (?)	Prospecting, mapping and staking. Discovery of two gold-mineralized boulders.	Stuart and Martin (1986)
1941–1956	Marbenor Malartic Mines – Marban	Drilling of 96 holes, resulting in the discovery of two gold-bearing zones: Marban South and Marban North Zone.	Bourget and Bugnon (1986)
Pre-1943	(?) – Norlartic	Five surface drillholes in the NE corner of Lot 2, Vassan Township.	Mannard (1986)
1943–1944	Norbenite Malartic Mines – First Canadian	Gold discovery on the First Canadian property in 1943. Both the Norbenite Gold Horizon and the South Gold Horizon were found as a result of systematic cross- sectional drilling that followed up on the discovery of mineralized float in the area. A total of 36 holes drilled for 5,665 m.	Stuart and Martin (1986)
1943–1946	Norbenite Malartic Mines – Norlartic	Sixty-six drillholes totalling 12,075 metres that resulted in the discovery of the Main Zone of the future Norlartic mine.	Mannard (1986)
1946	Norbenite Malartic Mines – Norlartic	Hole NM-65 was drilled in 1946 that tested the large granodiorite sill located north of the Gold Hawk Horizon. The best assay from this hole was 1.71 g/t Au over 1.5 m.	Stuart and Martin (1986)
1946–1948	Norbenite Malartic Mines – Norlartic	Three-compartment shaft sunk at the Norlartic mine to 165 m. A total of 1508 m of drifting on 38 m, 76 m, 114 m, and 152 m levels, cross-cuts totalling 542 m, and raises totalling 160 m. Underground drilling totalling 4,208 m.	Mannard (1986)
1950	Norlartic Mines Ltd. – Norlartic	Discovery of the North Zone in July during a surface drilling program.	Mannard (1986)
1950–1951	Norlartic Mines Ltd. – Norlartic	Shaft dewatering and resumption of underground exploration at Norlartic mine. Extraction of a 22,680-t bulk sample.	Mannard (1986)
1951	Norlartic Mines Ltd. – Norlartic	Discovery of the North Zone was followed by additional drilling; the deposit was developed via the Main Zone Norlartic mine workings by driving two cross-cuts on the 76 m and -152 m levels. By the time all activities on the Norlartic property closed down during the year, 572 m of drifting, 225 m of cross-cutting, and 3,943 m of diamond drilling had been completed on the North Zone.	Mannard (1986)
1955	Marbenor Malartic Mines – Marban	Marbenor Malartic Mines Ltd. re-organized to form the Consolidated Marbenor Mines Ltd.	Bourget and Bugnon (1986)
1958	Marban Gold Mines Ltd. – Marban	Marban Gold Mines Ltd. company created following an agreement between Consolidated Marbenor Mines Ltd. and Malartic Gold Fields Ltd. in order to put the Marban deposit into production	Bourget and Bugnon (1986)
1958	Norlartic Mines Ltd. – Norlartic	The Norlartic property remained inactive until 1958, at which time Malartic Goldfields gained control of Norlartic Mines Ltd.	Mannard (1986)
1959	Norbenite Malartic Mines – First Canadian	Work was confined to the Norlartic Main Zone, which was developed into a producing mine in 1959.	Stuart and Martin (1986)
1959–1960	Marban Gold Mines Ltd. – Marban	Underground development at the Marban mine, shaft sinking to -250 m, drifting on -61 m, -107 m, -152 m, -198 m, and -244 m levels; production planned from the lower three levels. Malartic Gold Fields (affiliate of Lac Minerals) holds a 75% interest in Marban Gold Mines Ltd.	Bourget and Bugnon (1986)
1959–1966	Norlartic Mines Ltd. – Norlartic	Production at the Norlartic mine and shaft dewatering and deepening to 320 m. Development of the 190 m, 228 m, 266 m, and 305 m levels. Production started in mid-December 1959 for a 7-year period. Total mine production of 1,076,823 t at 4.46 g/t Au yielding 145,610 ounces of gold and 15,189 ounces of silver, including approximately 45,000 tonnes from the Kierens Zone on the First Canadian property [MDA notes that the total mine production exceeds that reported in Norlartic Mines records provided to MDA]. Ore was mined from two shoots (A and B Zones) above the 152 m level. Exploration was conducted below this level but was limited to 9 holes directly below the workings to a depth of 195 m. Only one of these holes hit ore-grade mineralization. An additional 9 holes were drilled east of the workings above the 152 m level. Mining was apparently carried out on a salvage basis in the western portion of the deposit, and it appears that stoping and drifting terminated in high-grade material.	Mannard (1986); Stuart and Martin (1986)
1961–1974	Marban Gold Mines Ltd. – Marban	Production at the Marban mine over this 13-year period of 1,983,112 tonnes grading 5.2 g/t Au and 6.8 g/t Ag. Milling was carried out at the Malartic Gold Fields facilities.	Bourget and Bugnon (1986)
1962–1964	Norlartic Mines Ltd. – First Canadian	Discovery of the Kierens zone during surface drilling in 1962 that totalled 5,055 m in 18 holes. The zone was accessed from the Norlartic -230 m level drift. Limited mining undertaken and 37 exploration holes, for a total of 2,280 m, were drilled from underground.	Stuart and Martin (1986)
1963	Marban Gold Mines Ltd. – Marban and First Canadian	Magnetic survey, geological survey, and a few drillholes.	Bourget and Bugnon (1986) (Wahl, 1963)
1963	Norlartic Mines Ltd. – Norlartic	Hole NS-99 drilled, which apparently cut part or all of the Gold Hawk Horizon, although no drill logs or assays have been preserved.	Stuart and Martin (1988)
1963–1966	Norlartic Mines Ltd. – Gold Hawk	Geological mapping, magnetometer survey, and one diamond-drillhole (427 m).	Martin and Bubar (1988)
1965–1966	Norlartic Mines Ltd. – First Canadian	Approximately 45,000 tonnes were reportedly mined out.	Stuart and Martin (1988)
1966–1973	Norlartic	Ownership changed to Willroy Mines Ltd., and then to K. Wheeler and L. Harvey in 1966. Corvel Securities later optioned the ground.	Mannard (1986)

Year	Company - Property	Work Description & Results	Reference
1973–1975	First Canadian Gold Corp. – Norlartic	Acquisition of Norlartic by First Canadian. Magnetometer survey and 33 surface holes totalling 4,166 m at the North-North Zone.	Mannard (1986)
1975	First Canadian Gold Corp. – First Canadian	Diamond drilling of two holes for 415 m. Tested westward extension of Norlartic North Zone, as well as the central granodiorite south of the Norbenite Horizon. Best assay of 1.37 g/t Au over 0.3 m.	Stuart and Martin (1986)
1975–1980	Lloyd Harvey – Gold Hawk	Prospecting and trenching in the central and southern portion of the property. Discovery on surface of a mineralized east-west shear zone with quartz, chalcopyrite, and pyrite in the west-central part of the property.	Martin and Bubar (1988)
1979–1981	Brominco Inc. – Norlartic	Acquisition of Norlartic by Brominco. Line-cutting and magnetic + VLF surveys. Surface diamond drilling of 62 holes for 6,478 m, including 36 holes (3,757 m) at the Main Zone and 26 holes (2,720 m) at the North-North Zone.	Mannard (1986)
1980	Thunderwood – Gold Hawk	VLF-EM and magnetometer surveys. Diamond drilling of seven holes (1,422 m). No information on the location of the drill core or logs for these holes. Hole 80-1 returned an assay of 18.55 g/t Au over 2.6 m; hole 80-5 returned 16.35 g/t Au over 0.9 m; and hole 80-6 returned 14.40 g/t Au over 1.5 m.	Martin and Bubar (1988)
1981	Gold Hawk Resources Ltd. – Gold Hawk	Diamond drilling of seven holes (874 m).	Martin and Bubar (1988)
1981	SOQUEM – First Canadian	Compilation, mapping, magnetic survey, pole-dipole induced polarization (IP) survey, and diamond drilling of 2 holes for 306 m. Detected two strong IP anomalies on SW part of property, both of which were drilled and encountered barren disseminated sulfides. SOQUEM subsequently dropped their option.	Stuart and Martin (1986)
1982–1983	Villebon Resources Ltd. – Gold Hawk	Diamond drilling of eight holes (611 m).	Martin and Bubar (1988)
1984	Brominco Inc. – First Canadian	Brominco Inc. and Aur Resources Inc. jointly optioned the property from First Canadian Gold Corp. and carried out magnetic, VLF-EM survey, and diamond drilling of five holes (930 m). Surveys done to assist in interpretation of bedrock geology. Tested central granodiorite (3 holes) and South Gold Horizon (2 holes). Best results from South Gold Zone were 2.57 g/t Au over 6.4 m, including 5.42 g/t Au over 1.8 m.	Stuart and Martin (1986)
1984	Brominco Inc. – Norlartic	Line cutting, property-scale surface magnetic and VLF-EM surveys, and three diamond drillholes. Compilation of existing geotechnical data.	Stuart and Martin (1986)
1984–1985	Lac Minerals Ltd – Marban	Work completed includes line cutting (122 m spacing), compilation maps, geology and drill-hole geology at 1:200 and 400 scales, updating of vertical sections, longitudinal projections, and level plans. Detailed geological and structural survey at 1"=50' (survey extended into the summers of 1985 & 1986). Geophysical surveys include magnetics/gradient, VLF, IP, and HEM supplementary line cutting at 61 m spacing. Supplementary magnetic surveys (total field & gradient). Lithogeochemical survey, including major elements and gold.	Desbiens (1992)
1985	Lac Minerals Ltd. – Marban	Geological and structural mapping.	Perreault (1985)
1985–1986	Aur Resources – First Canadian	Brominco was merged with Aur shortly before the 1985–86 program. Aur carried out a 21-hole diamond-drilling program totalling 5,675 m in the northern part of the property. The main purpose of the program was to re-evaluate the economic potential of the Kierens Zone and to establish additional drill-indicated reserves.	Stuart and Martin (1988)
1985–1986	Aur Resources – Norlartic	Three diamond-drillholes for 602 m at the Norlartic Main Zone. All three holes intersected gold grades assaying greater than 6.86 g/t Au over widths greater than 3 m. Hole 2001-48 cored a 6 m interval that returned 16.35 g/t Au.	Mannard (1986)
1985–1986	Aur Resources – Norlartic	A 4,452 m surface drilling program completed at the North Zone. The drilling was concentrated in the area to the east and below the old workings. Several ore-grade intersections were obtained up to 7.85 g/t Au over 3.8 m, but the gold distribution was found to be erratic.	Mannard (1986)
1986	Aur Resources – Norlartic	Single vertical hole, 2001-52, tested Marban shear. The upper part of this hole intersected the footwall portion of the Gold Hawk Horizon, which contained some minor quartz veining with anomalous gold values up to 0.34 g/t Au over 1.5 m. The best assay from this hole was 0.41 g/t over 1.5 m, which occurred in strongly sheared talc-chlorite schists interpreted to be the Marban Shear.	Stuart and Martin (1988)
1986	Aur Resources – Norlartic	The Norlartic shaft was dewatered and rehabilitated in order to re-evaluate the potential of the Main Zone. Drilling of geotechnical holes (2001-53 to 2001-92) to monitor the behaviour of the hanging wall rocks during dewatering. Six holes totalling 356 m were drilled into the North Zone stopes; one of these (2001-75) returned 19.41 g/t Au over 14.9 m (7.6 m true width). Shortly thereafter, nine exploration holes (2001-93 to 2001-101) were drilled, four of which re-tested the area directly west of the old stopes between vertical depths of -76 m and -213 m. These holes returned intersections ranging from 3.12 g/t Au over 1.5 m to 7.2 g/t Au over 3.05 m.	Mannard (1986)
1986	Lac Minerals – Marban	Geological and structural synthesis by Bourget and Bugnon.	Bourget and Bugnon (1986)
1986	Lac Minerals – Marban	Gold distribution study, which suggested that the gold is structurally controlled.	Bourget (1986)
1986	Lac Minerals – Marban	Diamond drill program of 1,862 m in 12 holes completed. Regional compilation map at 1"=1,000'. Definition magnetic contouring at 100 gamma intervals. DDH lithogeochemical sampling for major elements and gold. Major element lithogeochemical interpretation report. Au in ppb distribution summarized in interpretation report. Partial IP survey over Kewagama–Malartic contact. Construction of a 3-D model of the Marban mine.	Desbiens (1992)

Year	Company - Property	Work Description & Results	Reference
1986–1987	Aur Resources – First Canadian	Sinking of 222 m two-compartment exploration shaft at Kierens. Surface diamond drilling of 20 holes for 4,582 m. Underground diamond drilling of 122 holes for 11,417 m. Drifting (7 ft x 8 ft) in ore, with east and west drifts totalling 206 metres. Raising in ore for a total of 37 metres. Sludge sampling program for 132 holes that totalled 322 m. Four individual zones (No. 1, 2, 3, and 6) of quartz–pyrite–gold stockwork that plunge moderately to steeply east, three of which displayed good continuity (No. 1, 2, and 6), were delineated.	Mannard (1987)
1987	Thunderwood – Gold Hawk	Completion of 18,290 m of drilling to test the Gold Hawk Horizon.	Stuart and Martin (1988)
1987	Aur Resources – Norlartic	Hole 2001-106 was drilled on section 20+00W, approximately 365 m west of the nearest existing information, to test the Norbenite Shear at depth. This hole traversed the expected extension of the North Zone at a depth of approximately 122 m, but neither the mafic volcanic package nor the North Zone style of mineralization was encountered.	Mannard (1986)
1987–1988	Lac Minerals Ltd – Marban	Phase I Drill Program: 12,268 ft in 18 holes. Phase II Drill Program: drilling below -244 m level to test depth extensions with 18 holes (3,658 m). Updating of all mine sections with new holes. Revision of regional compilation map, 1"=1,000'. Overburden stripping in eastern part of property. Magnetic compilation of Marban area. Geologic compilation of Marban area (1"=400'). Entry of all old holes into database. Aur bought back an outstanding NSR at Marban from Canhorn Mining Corp.	Desbiens (1992)
1987–1988	Thunderwood and Aur – Gold Hawk	Aur operated the program for International Thunderwood Explorations Ltd. Drilling program of 62 holes totalling 16,000 m, mainly on the Gold Hawk Zone. Detailed compilation, magnetics, VLF-EM, and geological surveys completed.	Martin and Bubar (1988)
1989–-1991	Lac Minerals Ltd – Marban	Exchange of geological information between Aur and Lac. Negotiation between Lac and Aur for the NSR. Lithogeochemical sampling of Marban tonalite. Resampling of holes MBS-88-25 and MBS-88-26. Two memos (04/05/1990) to D.E. Malloy; 'Possible Camflo-type Mineralization within the Marban Property' and 'Marban Vein Type Lode Gold Mineralization.' Major & trace elements geochemistry as assessment work for 1990–1991.	Desbiens (1992)
1989	Thunderwood – Gold Hawk	Report on the mineralogy of gold mineralization.	Hak (1989)
1992	Aur Resources – Marban	Aur optioned the property from Lac Minerals for its bulk-tonnage potential. The Marban "tonalite" was identified as a high-priority target (similar setting to Camflo). The second target identified was a pronounced Z-shaped magnetic embayment or bend.	Cooke (1995)
1993	Aur	A twenty-two overburden drillhole program was carried out to test the favourable Héva-Kewagama contact. Hole 22 returned 5 clast-supported till samples with high visible pristine gold counts. Diamond drilling program of four holes totalling 1061 m.	Cooke (1995)
1994	Aur Resources – Marban	Five overburden holes drilled in a grid-like pattern for follow-up of hole 22 (5 gold counts). Diamond drilling program totalling 4,224 m (10 holes) consisting of 6 holes to test the potential of the Marban Tonalite, 2 holes (501-09 and 501-11) to continue a section across the Héva Formation, and 2 holes (501-13 and 501-14) to test the area around the magnetic inflection of the interpreted Héva-Kewagama contact. All the holes that intersected the tonalite had only minor sulfide mineralization, and although the alteration was fairly strong in these holes, it was most intense in hole 501-08, which returned 675 ppb Au over 0.9 m, the best gold assay of the program. The background gold content of the Marban Tonalite in all holes was low, with almost all samples returning less than 50 ppb Au.	Cooke (1995)
2006	NioGold – All Marban Blocks	High resolution 50 m spacing mag survey by Prospectair Inc.	Belzile (2016)
2006	NioGold – North-North, North, Gold Hawk, Kierens	Approximately 10,000 m in 64 drillholes. This drilling was mainly intended to assess historic "resources" previously outlined.	Davy (2008)
2007	NioGold – Marban	Abitibi Geophysics of Val-d'Or conducted an orientation IP survey designed to detect near-surface gold-bearing disseminated pyrite mineralization intersected in holes MB-06-001 and MB-06-002 in the Marban mine area. One 1,250 m north-south line was tested that passed over the collars of the holes, but the survey did not detect any chargeability response. This lack of response may have been due to thick overburden, and/or the pyrite may be too coarse.	Lefrançois (2007)
2006–2007	NioGold – North-North and Marban	Petrographic study of the gold mineralization in the North-North and Marban (hole MB-06-001). The study of the Marban mineralization indicated that gold grains are intimately linked to coarse pyrite, either as fracture filings in the crystals or as inclusions. Fine-grained pyrite contains far fewer gold grains than coarse-grained pyrite, which is more strongly associated to veins than fine-grained pyrite	Ducharme (2009)
2007–2008	NioGold – Marban	Ninety-four holes and two wedges for a total of over 33,000 m. Little exploration work was conducted in 2009 on the Marban Blocks. Fence of four core holes (1,736 m) 1 km east of the Marban shaft, about 450 m east of any previous holes.	
2008	NioGold – All Marban Blocks	Technologies Earthmetrix conducted a structural study of the Marban Block property using LANDSAT imagery.	Ducharme (2009)
2010	NioGold – Marban	Three (3) holes (905.6 m) 900 m west of the shaft at the Marban deposit, with another two holes (330.3 m) drilled to test the Marban Northeast zone	Camus (2012)
2010–2011	NioGold Aurizon – Marban, Norlartic	Phase I Program that included 146 new holes and extensions of eight holes previously drilled at and just north of the Marban deposit. Plus 24 holes at the Norlartic deposit.	Camus (2012)

Year	Company - Property	Work Description & Results	Reference
2011–2012	NioGold Aurizon – Marban	Phase II Program that included 90 holes and extensions of another 9 previous holes	Camus (2012)
2012–2013	NioGold Aurizon – Kierens, North	Sixteen (16) holes at the Kierens deposit and 17 new holes plus one extension of a previously drilled hole at the North Zone.	Belzile (2016)
2014	NioGold – Kierens, Norlartic	Four (4) holes were drilled at Kierens and 6 at Norlartic (representing 1,646 m and 1,368 m respectively)	Belzile (2016)
2014–-2015	NioGold – Marban	Two hundred forty-six (246) supplementary holes representing 72,489 m were drilled at Marban to complete the drilling pattern to about 25 m by 50 m	Belzile (2016)
2015–2016	NioGold – Norlartic, Marban	Twelve (12) PQ-sized holes totalling 1,692 m, drilled for comminution tests on Norlartic, Kierens, and Marban deposits.	
2016–2017	Osisko Mining	Twenty-six (26) holes testing down dip extension of the Marban, Norlartic and Kierens deposits totalling 15,171 m	

6.2.1 Marban Mine

The Marban mine is located in the Marban Engineering area, south of the Norlartic and Kierens mineralized zones. The Marban mineral resources described in Section 14 are approximately centered on the shaft of the Marban mine. This Subsection 6.2.1 is adapted from Ducharme et al. (2009).

The discovery of two gold-mineralized boulders in the south-central part of the claim group in 1940 led to the first campaign of drilling by Marbenor Malartic Mines Ltd. between 1941 and 1952, which consisted of 96 holes that followed the trend of glacial transport away from these boulders. This campaign led to the discovery of two mineralized zones — the south or Marban zone hosted within the Marbenite shear and the Norlartic zone hosted within the Norbenite shear. Definition drilling on the Marban zone delineated a gold-bearing structure 370 m long in an east-west direction that dips 40° to 60° to the north, with depths of the drilling ranging from 152 to 275 m below the surface.

In 1955, Marbenor Malartic Mines Ltd. reorganized to become Consolidated Marbenor Mines Ltd. This new company signed an agreement with Malartic Goldfields Ltd. in 1958 to form Marban Gold Mines Ltd., which ultimately put the Marban deposit into production. Between 1959 and 1960, Marban Gold Mines, which was 75% owned by Malartic Gold Fields Ltd. (Malartic Gold Fields), sank a shaft to a depth of 260 m. In addition, drifts were initiated on levels at 61, 107, 152, 198, and 244 m below the surface. Only the three deepest levels were extended during production.

Regular shipping of ore to the Malartic Gold Fields mill started in July 1961 and ended in September 1974. During these 13 years of production, a total of 1,983,112 tonnes of ore was processed that yielded 330,027 ounces of gold and 33,726 ounces of silver at an average grade of 5.27 g/t Au and 0.50 g/t Ag. During this time, intense drilling was undertaken from underground drill stations, and underground channel samples were collected.

Little Long Lac controlled the easternmost portion of the Marban zone through its 100% interest in mining concession 512. Little Long Lac drilled this area in 1945 through 1966.

The property history from 1974 to 1984 is incomplete. Records indicate that East Malartic Mines Limited was involved in the Marban claim group from 1975 to the early 1980s, although the nature of the work that might have been completed is not known. An agreement was executed between Consolidated Marbenor Mines Limited and Les Terrains Aurifères Malartic (Quebec) Limitée (Les Terrains) on December 23, 1981, which appears to have transferred Consolidated Marbenor's rights at Marban to Les Terrains. In 1982, Little Long Lac and other companies amalgamated and became Lac Minerals.

A letter dated April 26, 1989, sent by Canhorn Mining Corp. (Canhorn) to Lac states, "We are writing with respect to an agreement dated December 23, 1981, between Les Terrains Aurifères Malartic (Quebec) Ltée and Consolidated Marbenor Mines Limited. We understand that Lac is the successor in interest to Les Terrains Aurifères Malartic. Canhorn, by virtue of an amalgamation in December 1985, acquired all the properties and rights of Consolidated Marbenor Mines Limited." In another letter dated March 31, 1989, Canhorn agreed to sell its interest in the Marban Property to Aur, including a 2.5% NSR.

From 1984 to 1992, Lac worked the property and completed compilations, line cutting, geophysical surveys, lithogeochemical surveys, and drilling of overburden holes. Lac drilled 12 holes totalling 1,877 m into the Marban deposit in 1986 and 26 holes totalling 7,179 m in 1987 and 1988. Aur signed an agreement for a 50% interest in the property with Lac in 1992. In 1994, Lac was incorporated into Barrick Gold Corp. (Barrick). Aur operated the project by completing a drilling campaign in 1993 that consisted of 4 holes (1,061 m); 3 to test the stratigraphy in a northerly direction (up-ice in the opposite direction of glacial transport) from the overburden holes drilled by Lac and 1 to test a magnetic inflection in the Héva-Kewagama contact. Another campaign was conducted in 1994 that included 10 drillholes (4,220 m) — 6 of these holes tested the Marban Tonalite, 2 holes completed a stratigraphic section across the Héva Formation, and 2 holes followed up the magnetic inflection of the interpreted Héva-Kewagama contact.

Table 6-3 summarizes the historic drilling completed at and near the Marban deposit prior to NioGold's involvement.

Table 6-3: Historical Drilling on the Marban Deposit

Hole Name	Company	Type	Length (m)	Number of Holes	Years	Targets
CL-01 to 031	Little Long Lac	Surface	7,997.81	31	1945–1969	Marban deposit and extensions
CU-1 to 26 CU-100	Little Long Lac	Underground	2,397.86	27	Not known	In the mine
MBS-2 to 34 MBS-36 to 119A	Marban Mines	Surface	31,086.82	117	1942–1967	Marban deposit and extensions
MB-1 to 241 245 to 1099 1101 to 1121 1123 to 1177	Marban Mines	Underground	58,704.96	1172	1960–1969	In the mine
8-D-18 to D-6	Marban Mines	Underground	152.70	6	Not known	In the mine
22-1 to 22-5	Lac Minerals	Surface	1,528.29	5	1981–1987	East of the deposit
MBS-86-01 to 12	Lac Minerals	Surface	1,877.75	12	1986	Marban deposit and extensions
MBS-87-01 to 23	Lac Minerals	Surface	5,826.57	23	1987	Marban deposit and Marban NE
MBS-88-24 to 26	Lac Minerals	Surface	1,353.01	3	1988	Marban deposit
501-01 to 14	Aur Resources	Surface	5,280.08	14	1993–1994	Marban property
M-93-0122	Aur Resources	Surface	271.10	22	1993	Overburden RC holes drilled southwest of the deposit
M-94-01 to 05	Aur Resources	Surface	83.40	5	1994	Follow-up overburden RC holes drilled south of the deposit
TOTAL			116,560.35	1,437		

Table 6-4 summarizes historic underground channel sampling by Marban Gold Mines at the Marban mine that are included in the project database; the channel-sample data were added to the Marban Engineering Project database by digitizing the traces of the channels from paper level/plan maps and entering the gold grades indicated on these plans by hand.

No work is reported to have been undertaken on the Marban claim group from 1995–2003. In 2003, McWatters purchased Barrick's 50% interest in the property subject to a 2–3% NSR retained by Barrick. NioGold purchased McWatters' 50% interest in 2004 following McWatters' bankruptcy. In late 2008, Barrick sold its NSR to RGLD Gold Canada Inc. The NSR was subsequently amended in 2006 (see Table 4-2 for current royalty structure applicable to the Marban historical property). NioGold met all obligations and terms of its three-year option agreement with Aur for the remaining 50% interest in the Marban historical property by February 3, 2009, which gave NioGold a 100% interest in the Marban historical property. As per the same agreement, NioGold also acquired a 100% interest in the First Canadian (now Kierens) and Norlartic historical properties. NioGold also met all obligations and terms of its three-year option agreement with Thundermin Resources Inc. (Thundermin) by February 9, 2009, and thereby acquired a 100% interest in the Gold Hawk historical property.

Table 6-4: Historical Channel Sample Data Included in the Marban Database

Channel Name	Company	Mine Level	Type	Length (m)	Number of Channels	Years
EG-N350-1	Marban Mines	350	Underground	12.19	1	1959–1974
EG-N500-1 to EG-N500-84	Marban Mines	500	Underground	816.88	62	1959–1974
EG-N525-1 to EG-N525-10	Marban Mines	525 Sub	Underground	74.66	10	1959–1974
EG-N550-1 to EG-N550-24	Marban Mines	550 Sub	Underground	174.93	24	1959–1974
EG-N575-1 to EG-N575-21	Marban Mines	575 Sub	Underground	117.57	21	1959–1974
EG-N600-1 to EG-N600-51	Marban Mines	600 Sub	Underground	300.61	51	1959–1974
EG-N625-1 to EG-N625-43	Marban Mines	625 Sub	Underground	198.83	43	1959–1974
EG-N650-1 to EG-N650-222	Marban Mines	650	Underground	2,100.01	214	1959–1974
EG-N700-1 to EG-N700-93	Marban Mines	700 Sub	Underground	519.71	93	1959–1974
EG-N725-1 to EG-N725-14	Marban Mines	725 Sub	Underground	54.23	14	1959–1974
EG-N750-1 to EG-N750-176	Marban Mines	750 Sub	Underground	974.22	176	1959–1974
EG-N775-1 to EG-N775-12	Marban Mines	775 Sub	Underground	111.76	12	1959–1974
EG-N800-1 to EG-N800-714	Marban Mines	800	Underground	3,727.81	635	1959–1974
Total				9,183.40	1,356	

6.2.2 Other Marbenite Shear Exploration

The following information is taken from Lefrançois (2005) with additional information as cited. The drill intersections discussed are only the most significantly mineralized, in order to provide the reader with a sense of the potential of the exploration target. The reader is cautioned that these intersections are not representative of the drilling programs as a whole.

In 1984, Brominco (controlled by Aur) drilled 5 holes for a total of 931 m in the Marbenite shear; the best intersection from this drilling was 5.42 g/t Au over 1.8 m (Ducharme et al., 2009). Aur drilled 1 hole in 1985–1986 to test the extent of the Marbenite shear where it dips into the south portion of the Norlartic historical property at depth. The shear was apparently intersected, but no significant gold values were returned. Aur completed 23 holes (5,406 m) in 1987 to test the Marbenite shear on the First Canadian historical property. Significant intersections were obtained from a well-developed vein system at the southeastern end, near the Marban historical property boundary, including 18.31 g/t Au over 0.9 m and 4.66 g/t Au

over 5.2 m. The drilling over this area remains widely spaced. The zone is interpreted to plunge moderately into the Marban historical property.

6.2.3 Norlartic, Kierens, & Related Zones

6.2.3.1 Norlartic Mine (Main, North, & Kierens Zones) (Pre-1985)

The following information is taken from Carrier (2006), who cites Lefrançois (2005). The 'Marban shear' of previous authors is referred to herein as the 'Marbenite shear' to be consistent with more recent usage. The Kierens zone was part of the Norlartic mine in the pre-1985 period.

The Marbenite shear and the Norlartic Main zone (within the Norbenite shear) were discovered during a drilling program in 1943 and 1944 by Norbenite Malartic Mines Ltd. (Norbenite Malartic). A total of 36 drillholes, for 5,664 m, were drilled in this program. The company followed up on the discovery with 66 holes (12,071 m), and in 1946 through 1948 sank a 165 m, three-compartment vertical shaft, completed drifts and crosscuts on the 38 m, 76 m, 114 m, and 152 m levels, and conducted 4,207 m of underground drilling into the Norlartic deposit.

Norlartic Mines Ltd. (Norlartic Mines) was formed in 1950 and 1951, and the company dewatered the mine, resumed underground exploration, and extracted a 22,680-tonne bulk sample. The North Zone was discovered 365 m to the north of the shaft and was reached by crosscuts on the 76- and 152-metre levels. The discovery was followed by drifting and 3,941 m of underground drilling. In 1959, Norlartic Mines dewatered the mine once again, deepened the shaft to 320 m, developed new levels at 190, 229, 267, and 305 m, and initiated production.

The Kierens zone on the First Canadian historical property was discovered one kilometre west of the Norlartic shaft by surface drilling (18 drillholes; 5,054 m) completed in 1963–1964. The Kierens zone was reached by drifting 1,190 m on the 229-m level from the Norlartic mine in 1964–65, and underground development and drilling followed (37 drillholes; 2,281 m).

In the period from 1959–1966, Norlartic Mines mined 1,076,846 tonnes at 4.46 g/t Au. Production came mainly from eight levels on the Main (Norlartic) zone at depths of 30 to 305 m and distances of 60 to 365 m west of the shaft. Stopes averaged 6 m in width. Limited ore (reportedly less than 135,000 tonnes) was mined from the North and Kierens zones as well. Mining of the Kierens zone occurred between the 137- and 229-m levels, while the A- and B- zones at the North Zone were mined from two levels above 152 m. This North Zone production amounted to approximately 90,000 tonnes at 16.46 g/t Au. According to MERN records, the production for the Norlartic Main zone and the North Zone amounted to 1,033,696 tonnes at 4.63 g/t Au. Ore was processed at the Malartic Goldfields mill. According to historic records, Norlartic Mines operated mainly at a loss from mid-1964 to the closure at the end of 1966 due to a combination of low grades, low gold prices, and high operating costs. Limited exploration and development were conducted east of the shaft.

The ownership of the property changed to Willroy Mines Ltd. and then to K. Wheeler and L. Harvey in 1966; it was later optioned to Corvel Securities. In 1973–1975, First Canadian Gold Corp. acquired the property and conducted drilling on the North-North Zone (33 drillholes for 4,166 m). In 1979, the three Norlartic claims lapsed and were staked by Brominco. In 1980–1981, Brominco conducted drilling on the Main zone (36 drillholes; 3,757 m), primarily over untested areas east of the shaft and above the 152 m level, as well as on the North-North Zone (26 drillholes; 2,720 m). During the same period, SOQUEM completed two holes (306 m) on the First Canadian historical property.

Aur acquired a controlling interest in Brominco and the Norlartic historical property in 1983 and optioned the adjacent First Canadian historical property in 1984.

6.2.3.2 Norlartic Mine (1985–1994)

The following information is taken from Carrier (2006), who cites Lefrançois (2005). The drill intersections discussed are only the most significantly mineralized, in order to provide the reader with a sense of the potential of the exploration target. The reader is cautioned that these intersections are not representative of the drilling programs as a whole.

Records indicate that Aur completed a total of 146 surface drillholes from 1985–1989 at the Norlartic historical property, including holes drilled at the North and North-North Zones, discussed below. The Norlartic drilling was financed by Nova-Cogesco. Three holes totalling 603 m tested the Main zone east of the Norlartic shaft above the 152 m level. Nine other holes served mainly as a widely spaced test of the depth potential of the Main zone below the 549 m level. Above this level, tightly spaced underground drilling was completed by Aur from 1986–1990. The Main zone remained sparsely tested below the 549 m level and to the west of the old workings. The remainder of the drilling on the Main zone was performed from underground.

The project was administered by the Kierens Mining Division after 1988. Limited reports are available, although the underground drillhole logs were well preserved in a series of boxes; the underground holes are tabulated in Table 6-5. A new high-grade sub-zone, the Actinolite zone, was discovered east of the Norlartic shaft above the 152 m level.

Table 6-5: Norlartic Zone — Aur Underground Drilling (1986–1990) (from Carrier, 2006)

Hole No.	No. of Holes	Level (m)	Zone	Year
NU-2-1 to NU-2-12	12	76	Main Zone, west of old workings	1986
NU-2-13 to NU-2-47	35	76	Actinolite Zone	1989
NU-5-1 to NU-2-145	151	152	Main Zone, west of old workings 91 m to 305 m level	1986–87
NU-5-146 to NU-5-147	2	152	North Zone, for water	1988–89
NU-8-1 to NU-8-3	3	267	?	1987
NU-13-1 to NU-13-106	107	396	Main Zone, below old workings 305 m to 549 m level	1987–88
NU-13-107 to NU-18-110	4	396	H-Lens	1990
NU-16-1 to NU-16-12	12	488	H-Lens	1990

Aur reportedly mined 511,000 tonnes of ore from the Norlartic mine during the period 1990–1992 and produced a total of 56,000 ounces of gold.

6.2.3.3 Kierens Mine (1985–1994)

The following information concerning the exploration history is taken from Carrier (2006), who cites Lefrançois (2005). The drill intersections discussed are only the most significantly mineralized, in order to provide the reader with a sense of the potential of the exploration target. The reader is cautioned that these intersections are not representative of the drilling programs as a whole.

Records indicate that Aur completed a total of 143 surface drillholes on the First Canadian historical property from 1985–1989. The up-dip extensions of the Kierens zone (three drillholes; 611 m) were tested in 1985–1986. An intersection of 12.00 g/t Au over 4.6 m prompted a second program (18 drillholes for 5,063 m) that tested the Kierens zone over a strike length of 425 m and to a depth of 275 m. High-grade gold intersections were obtained west of the old mine workings. Surface drilling and underground development and drilling followed in 1986 and 1987. This work included the sinking of a 380 m, vertical, two-compartment shaft, 20 surface holes (4,580 m), and 122 underground holes (11,414 m), mainly on the 223 m level. Four individual zones (No. 1, 2, 3, and 6) of quartz–pyrite–gold stockwork that plunge moderately to steeply east were delineated, three of them displaying good continuity (Zones No. 1, 2, and 6).

After 1988, the project was administered by the Kierens Mining Division. The underground holes are tabulated in Table 6-6.

Aur produced approximately 50,000 ounces of gold from the Kierens mine from 1988–1992.

Table 6-6: Kierens: Aur Underground Drilling 1986–1990 (from Carrier, 2006)

Hole No.	No. of Holes	Level	Zone	Year
FU-2-1 to FU-2-5	5	250	Kierens	1989
FU-5-1 to FU-5-5	5	500	Kierens	1986
FU-6-1 to FU-6-9	9	625	Kierens	1989
FU-7-1 to FU-7-121	123	750	Kierens	1986–88
FU-8-1 to FU-8-11	11	803	Kierens	1990
FU-10-1 to FU-10-27	28	1000	Kierens	1988–90
FU-13-1 to FU-13-123	123	1300	Kierens	1987–88
FU-13-124 to FU-13-149	26	1300	Norbenite shear on Norlartic	1988
FU-13-150 to FU-13-I73	24	1300	Gold Hawk Horizon	1988–90

6.2.3.4 North Zone

The following information is taken from Carrier (2006), who cites Lefrançois (2005). The drill intersections discussed are only the most significantly mineralized, in order to provide the reader with a sense of the potential of the exploration target. The reader is cautioned that these intersections are not representative of the drilling programs as a whole.

The North Zone mineralization is located 350 m north of the Norlartic shaft and is hosted within the North shear zone. A limited tonnage of ore in two high-grade lenses above the 152 m level was mined from 1959 through 1966. In 1985 and 1986, Aur completed 16 holes (4,451 m); 10 holes were drilled below the 152 m level, and 5 were drilled along the eastern extent above the 152 m level. In 1998, Aur drilled 7 additional holes (1,579 m) on two sections 30 m apart in order to test the western extent of the North Zone. This was a 60 m step-out from the old mine workings, and the drilling confirmed a moderate westward plunge to the zone. Three holes returned significant intersections, the best being 10.15 g/t Au over 2.0 m, and the zone remained open down-plunge. Ten other holes were drilled to further test the western down-dip extent of the North Zone at 61 m centers and up to 305 m west of the old workings. The holes proved the continuation of the zone, with a notable intersection of 32.00 g/t Au over 1.5 m on the westernmost section at a vertical depth of 380 m. Two holes tested the eastern extent of the North Zone, and an intersection of 10.32 g/t Au over 0.8 m was obtained.

Historic exploration at the North Zone tested a strike length of 730 m; very limited drilling was done below a depth of 365 m.

6.2.3.5 North-North Zone

The following information is taken from Lefrançois (2005).

The North-North Zone is located 450 m north of the Norlartic Main zone and 700 m northwest of the Norlartic shaft on the Norlartic claim group. It was discovered in 1950 as the result of drilling by Norlartic Mines. The deposit was further defined by follow-up drilling in 1963 (6 drillholes) and subsequently by First Canadian Gold Corp. in 1974 (33 drillholes for 4,166 m), Brominco in 1980 (26 drillholes for 2,720 m), and Aur in 1985-86 (32 drillholes for 5,753 m). In their 2013 report, MDA notes that there are other references that cite slightly different numbers of holes and total metres drilled than reported here, but these differences could not be resolved.

The gold-bearing alteration envelope and associated vein stockwork at the North-North Zone were traced by drilling over a strike of 600 m and to a vertical depth of 300 m. Limited drilling has been done on strike to the east, and only three holes were drilled below a depth of 230 m.

6.2.3.6 Gold Hawk Horizon

The following information is taken from Carrier (2006), who cites Lefrançois (2005).

The Gold Hawk horizon traverses the Gold Hawk, First Canadian, and southwestern corner of the Norlartic historical property in a west-northwest direction. Surface and underground drilling during the 1980s outlined several generally thin high-grade gold veins within the horizon.

Following the discovery of high-grade gold veins on the Gold Hawk claims by Thunderwood Explorations Ltd. (Thunderwood), Gold Hawk Resources Ltd., and Villebon Resources Ltd. (22 holes for 2,906 m) between 1980 and 1983, Thunderwood completed an additional 62 holes (16,000 m) in 1987 using 30 m drill-section spacings to a vertical depth of 400 m. Veins/vein systems No. 1, No. 2, and No. 3 were identified by this program. Visible gold in veins was observed in 40% of the 1987 drillholes. Vein No. 1, which is located on the southern contact of the Gold Hawk horizon in ultramafic rocks, demonstrated the best continuity.

Under an agreement with Thunderwood, Aur drilled, accessed, and developed Vein No. 1 in 1988 via a 500 m crosscut driven from the 230 m level of the Kierens mine. From 1989–1991, Aur conducted small-scale mining of Vein No. 1 within the Gold Hawk historical property on the 230 m level and 218 and 187 m sub-levels. At the end of October 1991, 8,273 tonnes with an average head grade of 13.95 g/t Au (3,711 oz/ton Au) were mined from Vein No. 1. Pillars were recovered in November 1991, but production numbers are not available. Stopes averaged 1.2–1.5 m in width. The mined-out section of Vein No. 1 is localized within a fold of the footwall ultramafic contact, which apparently plunges 45º to the east. This was considered to be the main structural control to Vein No. 1.

Concurrently, Aur tested the Gold Hawk horizon by surface drilling on the adjoining First Canadian and Norlartic historical properties over a strike of 1.2 km. The drilling was mainly completed on 60 m section spacings. Limited underground drilling was also completed from a cross-cut driven from the 400 m level of the Kierens mine.

A total of 20 surface holes (19 drilled by Aur) were completed along the southeastern extent of the Gold Hawk horizon on the Norlartic and First Canadian historical properties. Some of these holes were drilled to test the Marbenite shear to the south. The 20 holes tested the Gold Hawk horizon over a strike of 550 m and to a vertical depth of 300 m. The zone was further tested by 24 underground holes from the 395 m level of the Kierens mine.

6.2.3.7 Work Completed by NioGold (2006–2016)

Since first gaining interests in the property in early 2006, NioGold, including the work of the Aurizon-NioGold joint venture, has carried out the following activities at the Marban property, mostly on the historical Marban, Norlartic, Kierens and Gold Hawk properties:

- Construction of three-dimensional computer models of the historic underground workings;
- Completion of high-resolution airborne magnetic surveying;
- Orientation induced-polarization surveying;
- Petrographic studies of the gold mineralization at the North-North and Marban zones;
- Structural study using LANDSAT imagery; and
- Drilling of 954 holes on the entire property, representing 281,217 m of drilling.

6.2.3.8 Work Completed by Osisko Mining (2016–2019)

Following the NioGold takeover, Osisko Mining completed 26 drillholes totalling 15,171 m in 2016 and 2017 into the extensions at depth of the Marban, Norlartic and Kierens deposits. The drillholes were widely spaced at some 400 m aiming at testing the concept of a large volume low grade ore body at depth. The deepest drillhole reached 1,475 vertical metres.

6.3 Historical Mineral Inventory Estimate (1993)

A number of estimates of mineral inventory at the Marban property were completed prior to involvement in the project. There are insufficient details available on the procedures used in these estimates to determine if any of the estimates comply with NI 43-101. Details are available in Carrier (2006), however they are deemed to be not relevant for this report and have been excluded accordingly. Only the most recent post-production historical mineral inventory is described in Section 6.3.1

6.3.1 Aur Mineral Inventory Estimates

Table 6-9 summarizes historical mineral inventory remaining in 1993 as estimated by Aur following the closure of the Norlartic and Kierens mines. Davy (2008) reported that these remaining Reserves on the Norlartic and First Canadian historical properties as estimated by Aur totalled 1,354,249 tonnes averaging 5.33 g Au/t. No mineral inventory was calculated for the Marban deposit at that time.

Accordingly, these figures are presented here merely as an item of historical interest with respect to the exploration targets and should not be construed as being representative of actual mineral resources or mineral reserves (under NI 43-101) present at the Marban property. Insufficient work has been done by a qualified person to classify these historic estimates as current mineral resources or mineral reserves, and O3 Mining does not consider them as current estimates. These historic mineral resource estimates are superseded by the current mineral resource estimate described in Section 14.

The use of the terms "Resource" and "Reserve" in the following discussions and tables is not consistent with National Instrument 43-101. Ausenco knows little of the techniques and parameters used in these estimates, and therefore Ausenco is uncertain if any of these estimates were prepared in full compliance with the provisions of National Instrument 43 101.

All tonnages were converted from short tons to metric tonnes, and feet have been converted to metres.

No resource category was assigned to the remaining historical mineral inventory, therefore the relative confidence of these tonnages and grades cannot be ascertained.

Table 6-7: 1993 Remaining Historic Mineral Inventory

Zone	Tonnes	g/t Au	oz Au	Description
Kierens	261,269	4.80	40,320	Undeveloped drill-defined ore below the 395 m level (1300 ft)
Norlartic	252,197	5.11	41,422	Undeveloped drill-defined ore east of shaft
	59,511	2.54	5,710	L Lens stope muck
	2,449	2.40	189	Actinolite Zone 201 stope muck
	3,175	2.74	280	Actinolite Zone 503 stope muck
North	136,078	6.86	30,000	Undeveloped, defined by surface drilling
North-North	385,554	3.63	45,050	Undeveloped, defined by surface drilling
Total			162,971	
Other Resources				
Gold Hawk	254,016	8.57	70,000	Defined by widely spaced drilling

Note: From Carriers, 2006, citing Aur internal report, remaining resources in 1993 by P. Pelz, P.Geo.

6.3.2 Recent Historical Mineral Inventory Estimate (2020)

Since the closure of the mines, numerous studies and historical mineral resources have been released. Belzile Solutions Inc (BSI) completed mineral resource estimates for the Marban property (called the Malartic property in MDA's 2010 Technical Report) that were described in previous Technical Reports (Gustin, 2007, 2010 and 2013). SGS completed an updated Resource estimate for the Marban deposit that included Phase I drilling by the Aurizon-NioGold joint venture, described in a 2012 Technical Report (Camus et al., 2012). BSI completed mineral resource estimates for the Marban property in BSI's 2016 Technical Report. For the purposes of this report, only details of the most recent historical mineral inventory estimate are deemed relevant, which is summarized below.

Insufficient work has been done by a qualified person to classify these historic estimates as current mineral resources or mineral reserves, and O3 Mining does not consider them as current estimates. These historic mineral resource estimates are superseded by the current mineral resource estimate described in Section 14

In 2020, Ausenco Engineering Canada Inc. prepared a Preliminary Economic Assessment (PEA) report for O3 Mining Inc. on the Marban Engineering Project which is considered the most recent historical estimate. The resource estimate in this PEA was under the responsibility of Sue Bird, P.Eng of Moose Mountain Technical Services (MMTS). The mineral resource estimate was based on a drilling database where drillholes before 1980 were excluded due to precision issues with the gold assays. Wireframes were based primarily on gold grades, and the regional shears were also accounted for in the interpretation. Assay gold grades were capped by estimation domain, and varied between 10 g/t and 100 g/t, and an additional high-grade restraint of 2 to 10 m was applied to specific domains to restrict the influence of high-grades. Compositing was undertaken on 1.5 m run-lengths, and gold grades were estimated into the block model using ordinary kriging. The block model used 5 m x 5 m x 5 m block sizes, and was validated using global bias checks with nearest neighbour, and a change of support to assess the overall "smoothing" within the model. Pit optimisations were run on the block model using a US$1,800 gold price and an overall ore-based cost of C$16. The underground mineral resource was constrained using a grade shell representing a 3.5 g/t grade shell. Tonnage and grade information is shown in Table 6-8.

Table 6-8: Mineral Resource Estimate disclosed in the 2020 PEA by MMTS

Class	Source	Tonnage (kt)	Au (g/t)	Au Metal (oz)
Measured	Marban Pit	65	1.32	2,792
	Kierens-Nolartic Pit	450	1.03	14,900
Indicated	Marban Pit	46,260	1.03	1,536,671
	Kierens-Nolartic Pit	6,646	1.15	246,430
	Marban UG	220	7.77	54,982
	Kierens-Nolartic UG	510	3.57	58,504
Measured + Indicated	All	54,151	1.10	1,914,249
Inferred	Marban Pit	6,465	1.09	227,226
	Kierens-Nolartic Pit	6,299	1.42	286,724
	Marban UG	304	8.73	85,317
	Kierens-Nolartic UG	119	3.02	11,560
	All	13,187	1.44	610,827

These previous estimates are superseded by the one described in Section 14, which includes all drilling at Marban Engineering as of the Effective Date of this report.

7 GEOLOGICAL SETTING AND MINERALIZATION

Information on the geology, mineralization and deposit types of the Marban property has been described in previous NI 43-101 Technical Reports (Carrier, 2006; Gustin, 2007, 2010, 2013; and Camus et al., 2012) filed by the NioGold. The information provided herein is modified from public scientific literature, O3 Mining geological staff observations, and from those reports.

7.1 Geological Setting

7.1.1 Regional Geology

The Marban Engineering Project is located in the Malartic mining district, which is 15 km northwest of the Val-d'Or mining district, within the Precambrian Canadian Shield in western Quebec.

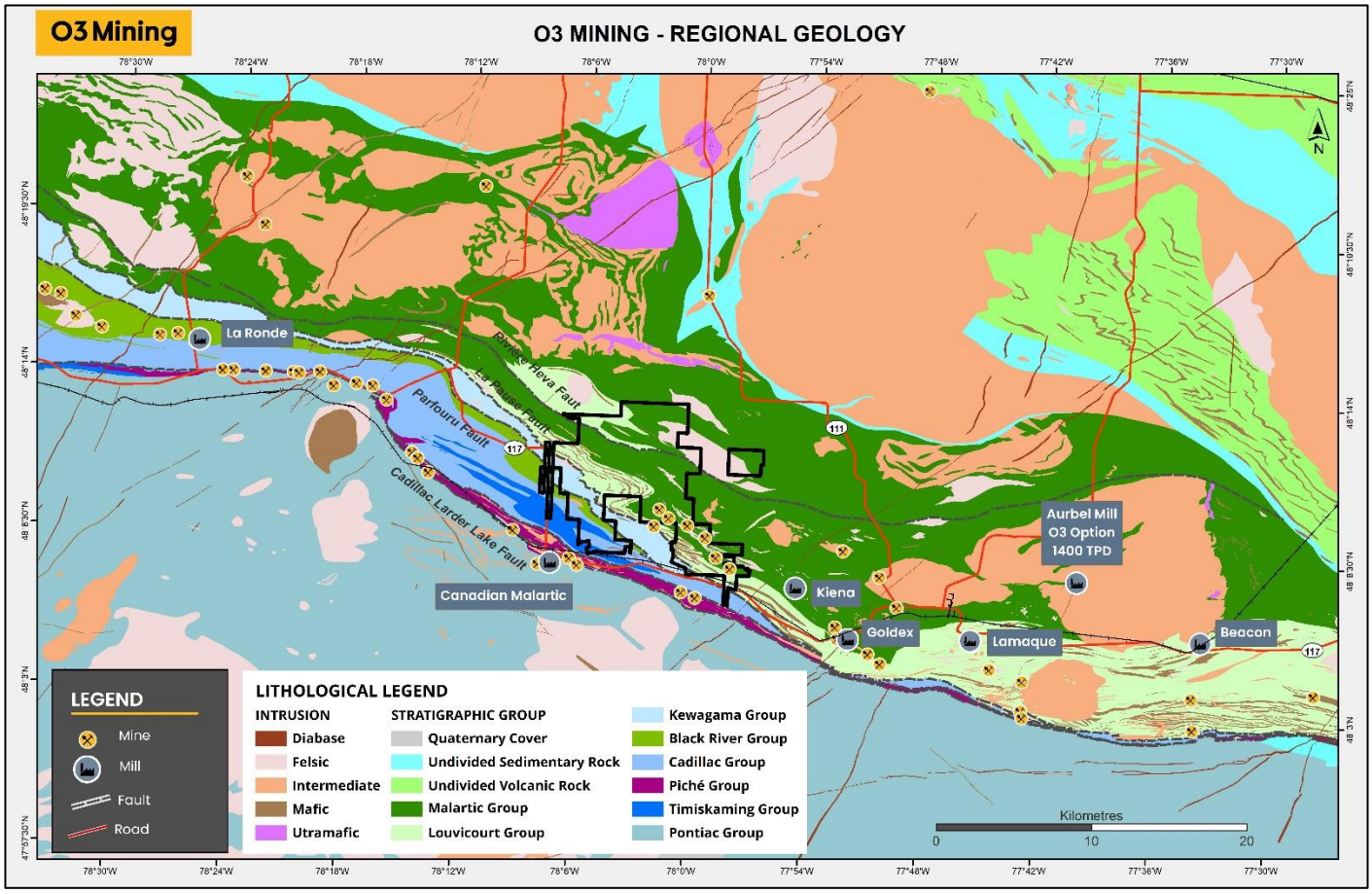
Rocks of the Malartic gold district belong to the Archean Abitibi greenstone belt of the Superior Province, Quebec. The Abitibi greenstone belt consists of east-trending alternating volcanic-plutonic and sedimentary belts that are bounded by crustal-scale faults (Figure 7-1). Based on the different tectonic, plutonic, and volcanic histories, the Abitibi belt has been divided into a Northern Volcanic Zone (NVZ) and a Southern Volcanic Zone (SVZ) (Chown et al., 1992). The NVZ (2730–2705 Ma) (Mortensen, 1993a, b) is interpreted as an intact arc segment, subdivided into 1) a Monocyclic volcanic segment with an extensive basalt plain with scattered felsic to bimodal mafic-felsic effusive complex, interstratified or overlain by volcaniclastic sedimentary basins and 2) a Polycyclic volcanic segment, characterized by a second mafic-felsic cycle, sedimentary assemblage and local shoshonitic volcanic rocks (Chown et al., 1992; Mueller et al., 1996; Daigneault et al., 2002). The SVZ, where the Marban property sits, is defined to its western part by tholeiitic basalts considered as an oceanic island arc, overlain by mafic to felsic calc-alkaline sequence, all of which are included in the Blake River Group (2703–2698 Ma) (Dimroth et al., 1982; Mortensen 1993a; Daigneault et al., 2002). Divided from the western part by the Parfouru fault, which is bordered by the turbiditic graywacke and mudstone of the Kewagama (De Souza et al., 2020), the eastern part of the SVZ is composed of a homoclinal oceanic floor to complex volcanic-arc assemblage (Dimroth et al., 1982; Daigneault et al. 2002). It is composed of komatiite, basalt, and intermediate to felsic volcanic and volcaniclastic rocks of the Malartic Group (2714–2706 Ma) (Pilote et al., 1999; De Souza et al., 2020) and the basalt, andesite and intermediate to felsic volcaniclastic rocks of the Louvicourt Group (2714–2698) (Pilote et al., 1998; Guay et al., 2018; De Souza et al., 2020). The SVZ is limited to the south by the Cadillac-Larder Lake Fault Zone (CLLFZ), straddled by the clastic-volcaniclastic Cadillac Group (Daigneault et al., 2002). The CLLFZ represents the limit between the Abitibi Subprovince and the Pontiac metasedimentary Subprovince and is characterized by the presence of the Piché Group, that consists of a thin but continuous unit of ultramafic rocks with minor mafic to felsic units. The Piché Group thickness varies from few metres up to a kilometre. The NVZ and SVZ are separated by the Destor-Porcupine-Manneville fault, interpreted as marking the collision of two oceanic arcs (Chown et al., 1992; Mueller et al., 1996; Daigneault et al., 2002). The regional metamorphism is mostly at the greenschist facies and can locally reach the amphibolite facies due to the contact metamorphism surrounding late tectonic intrusions and locally a deeper erosion level.

The tectonic evolution of the sector has been well documented by Daigneault et al. (2002) and the proposed model was divided into six major events. Over them, three main phases of ductile deformation led to the different structural fabrics that can be recognized in the Malartic mining district. The accretion of the NVZ and the SVZ which is commonly describe as the first episode of deformation (D_1). This deformation as produced isoclinal overturned F_1 fold with a subtle S_1 schistosity, 2) the accretion of Abitibi and Pontiac Subprovinces (D_2), that developed steeply east-plunging F_2 folds that overprinted the F_1, and a highly penetrative S_2 east-west schistosity (Bertrand-Blanchette, 2016; De Souza, 2020). The Malartic mining district was thereafter submitted to a late dextral transcurrent shearing along lithologic contacts and associated folding. The D_3 structural elements correspond to Z-shape F_3 folds, east–northeast S_3 crenulation cleavage or schistosity and dextral shear zones (Daigneault et al., 2002; Samson, 2019; De Souza et al., 2020).

Proterozoic diabase dykes strike across all previous lithologies in north to east–northeast directions. Segments of the dykes have been displaced by late faulting. North-trending dykes are traceable over hundreds of kilometres and range in thickness from 15 cm to 50 m.

The Abitibi is one of the most gold-rich of the worldwide occurrences of Archean greenstone belts, with a high concentration of gold deposits found in the SVZ, Major gold districts within the SVZ are primarily localized at flexures along the Destor-Porcupine-Manneville and Cadillac-Larder Lake structural zones.

Figure 7-1: Regional Geologic Map



Source: O3 Mining, 2022.

7.1.2 Local Geological Setting

The Marban property is located in the southern portion of the SVZ, where the Parfouru fault and the associated sedimentary rocks of the Kawagama Group separate the Blake River segment to the west from Malartic segment to the east (Daigneault et al., 2002). The western portion of the property contains the eastern end of the Blake River Group, composed of basalt and andesitic basalt of the Hébécourt Formation (2703–2702 Ma) (McNicoll et al., 2014) which consist of a north-dipping panel with faulted contacts bordered with the sedimentary units of the Kewagama to the north and Cadillac to the south (De Souza et al., 2020). To the east, the Malartic segment is subdivided into the Malartic Group and the Louvicourt Group. The Malartic Group consists of plume-derived komatiitic-tholeiitic marine-plain volcanic assemblages, divided from north to south, into the La Motte-Vassan, Dubuisson, and Jacola formations. In contrast, the overlying Louvicourt Group

represents an arc-type complex. The Louvicourt Group is subdivided into the Val-d'Or Formation, a transitional to calc-alkaline volcanic complex, and the Héva Formation, characterized by geochemically distinct iron tholeiites.

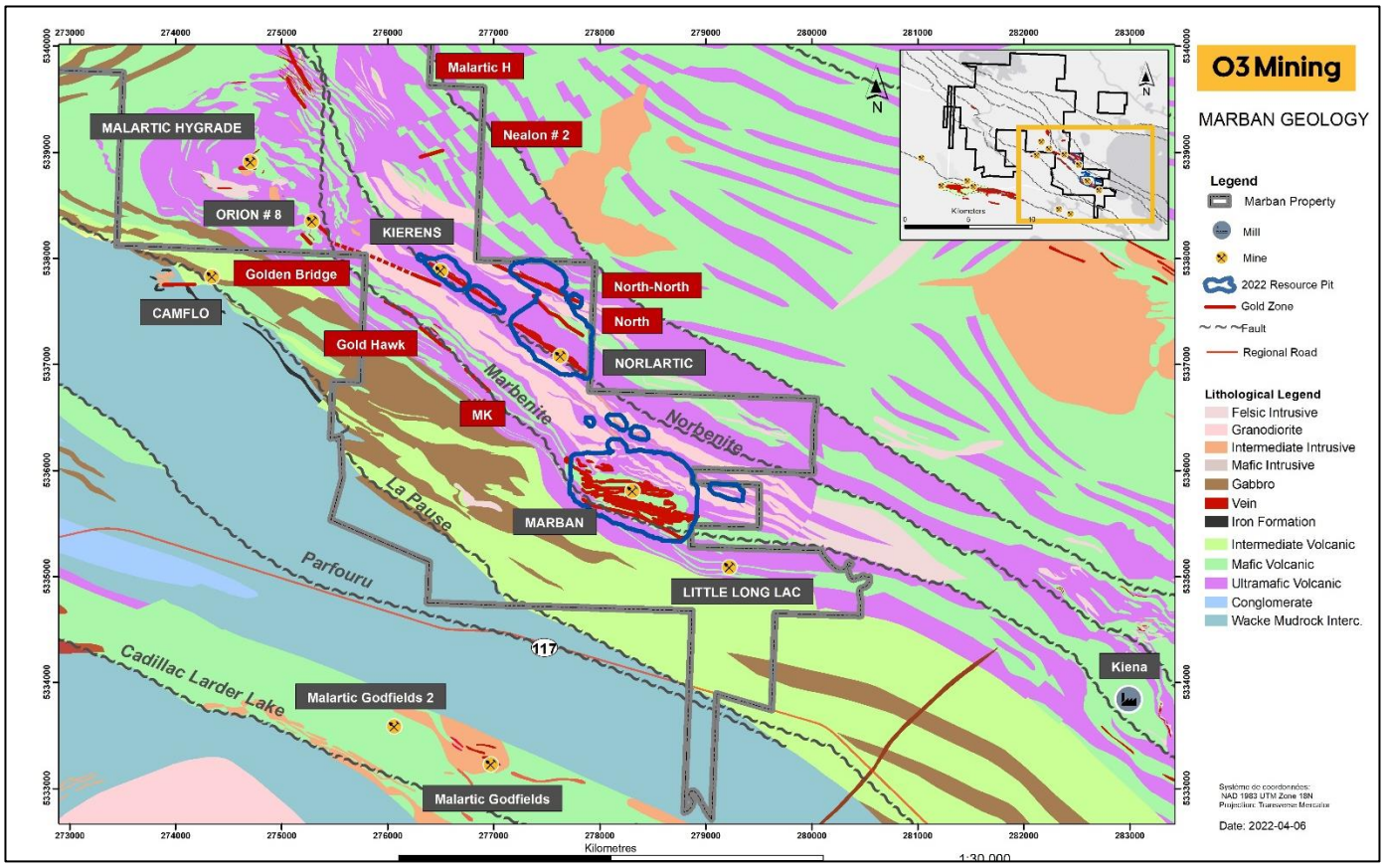
The Canadian Malartic Mine, owned by the Agnico Eagle-Yamana Gold Partnership which includes the historical mine production of the East Malartic, Barnat-Sladen, Canadian Malartic, and Malartic Goldfields gold deposits, occur within a 900 m wide band of highly sheared and faulted rocks named the Malartic Tectonic Zone (MTZ), part of the 300 km long Cadillac-Larder Lake Tectonic Zone. Two other northwest-striking fault systems are recognized to the north of the MTZ, namely the Norbenite and Marbenite shear zones. These two shear zones are important controls on the localization of gold within the Malartic district, as numerous deposits occur along these structural zones, including Gold Hawk, Kierens, Norlartic, Marban, Kiena, Shawkey, Wisik, and Goldex. Stratigraphic units in the area of the Malartic district generally strike at 300° azimuth, and their contacts are usually sheared and faulted. Two other faults have been documented bordering the Kewagama group to the northeast and southwest, respectively, the La Pause Fault and Parfouru Fault.

The geologic setting of the Marban property is illustrated in Figure 7-2. The units are mainly oriented northwest-southeast and dip to the northeast, although they are oriented more west-northwest–east-southeast in the southeastern portion of the property with a dip to the north. The rocks become younger to the southwest, forming an overturned sequence on the south limb of the La Motte-Vassan Anticline. The northeast half of the property is underlain by komatiites and basalts belonging to the Jacola Formation of the Malartic Group. They appear to be in conformable contact with the mafic pyroclastics, volcanics and volcanoclastic rocks belonging to the Héva Formation that belong to the Louvicourt Group. These in turn are in contact with sedimentary rocks of the Kewagama Group. Dykes and sills of various compositions intrude all these units.

The volcanic rocks of the Jacola Formation contain three major NW-SE oriented deformation corridors, from northeast to southwest, the North, Norbenite and Marbenite shears. The North Shear affects the basalts and parts of the komatiites of the North Zone. The full extent of this shear zone is unknown. The Norbenite Shear affects the komatiites and dioritic dykes that form the Kierens and Norlartic deposits. The Norbenite is an important northwest-trending deformation zone with a moderate to steep dip to the northeast and up to 150 m width on the property. The Marbenite Shear hosts the Marban deposit and the Gold Hawk Horizon. It is oriented east-west to west-northwest–east-southwest with a moderate to steep dip to the north and northeast and up to 450 m width. All the deformation corridors are going through the property from east to west. The rocks are also affected by late-east-west structures including schistosity and northeast-southwest-trending faults.

Two major granodiorites and one tonalite sills intrude the Jacola Formation subparallel to stratigraphy. The northernmost intrusion, known as the North-North Granodiorite, hosts the North-North Zone. The second intrusion named the Central Granodiorite is located between the Norbenite and the Marbenite shears. Several decimetric to metric mafic to intermediate dykes intrude all the previous units. To the west of the Marban Engineering sector, the Camflo mine is partly hosted in a porphyritic monzonite, the Camflo stock (U-Pb on zircon 2685 +10/-8 Ma, Zweng et al., 1993; U-Pb on titanite 2680 ±4 Ma; Jemielita et al., 1990), that is injected within the northern part of the Kewagama, at the contact with the Jacola.

Figure 7-2: Geologic Map of the Marban Property



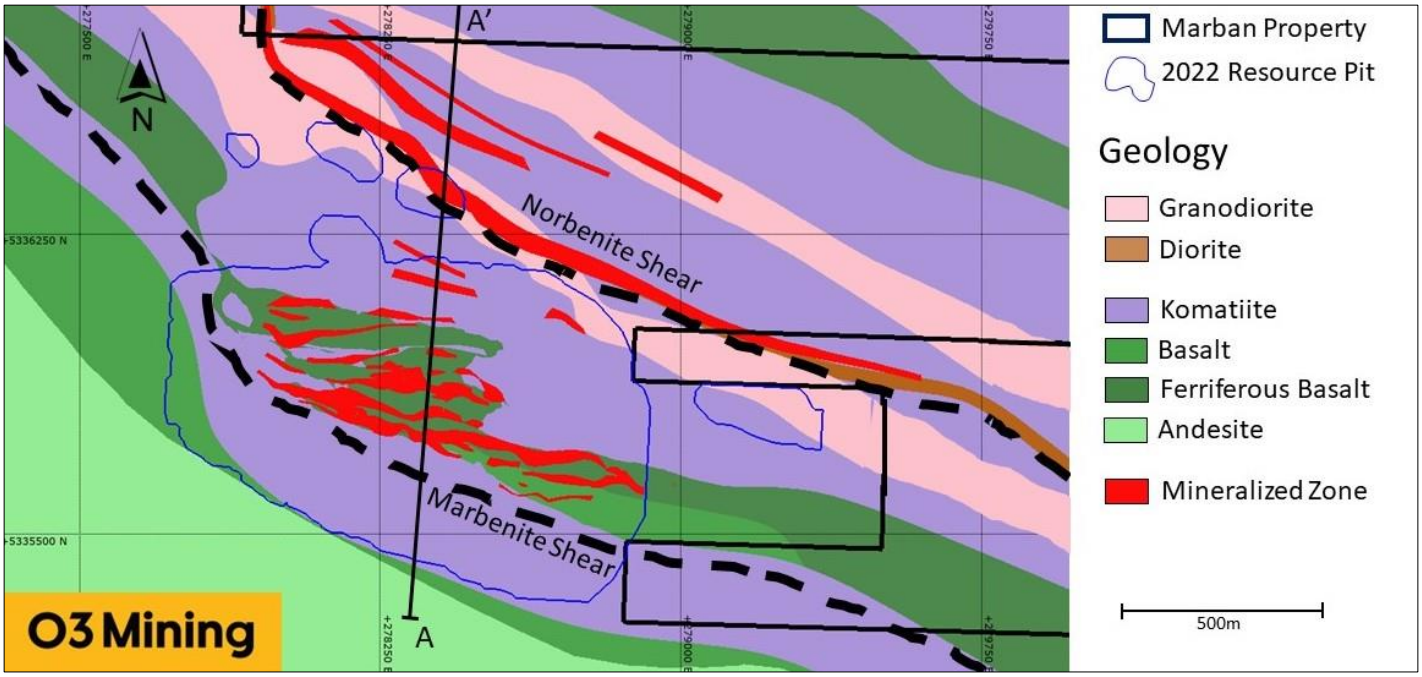
Source: O3 Mining, 2022.

7.1.2.1 Litho-Structural Model of the Marban Deposit

The Marban deposit is located at the Marbenite shear and extend in the hanging wall rocks for several hundreds metres. It sits immediately to the east of a curvature in the Marbenite which is trending more east-west in this particular area. The southernmost part of the mineralisation is hosted by a strongly sheared komatiite unit within the Jacola Formation. Going north, the Mine Sequence corresponds to basaltic volcanics hosting most of the gold mineralization of the deposit. The mafic unit in the Marban deposit presents a significant thickening, compare to the correlated basalt horizon east and west, due to a multi-phased folding that shows a doubly plunging fold axis interference pattern which implies an early fold phase overprinted by subsequent east-west folding. The basaltic unit can be geochemically divided into two different basalts. A Mg-rich basalt to the south that overlay a ferriferous basalt to the northern part of the Mine Sequence.

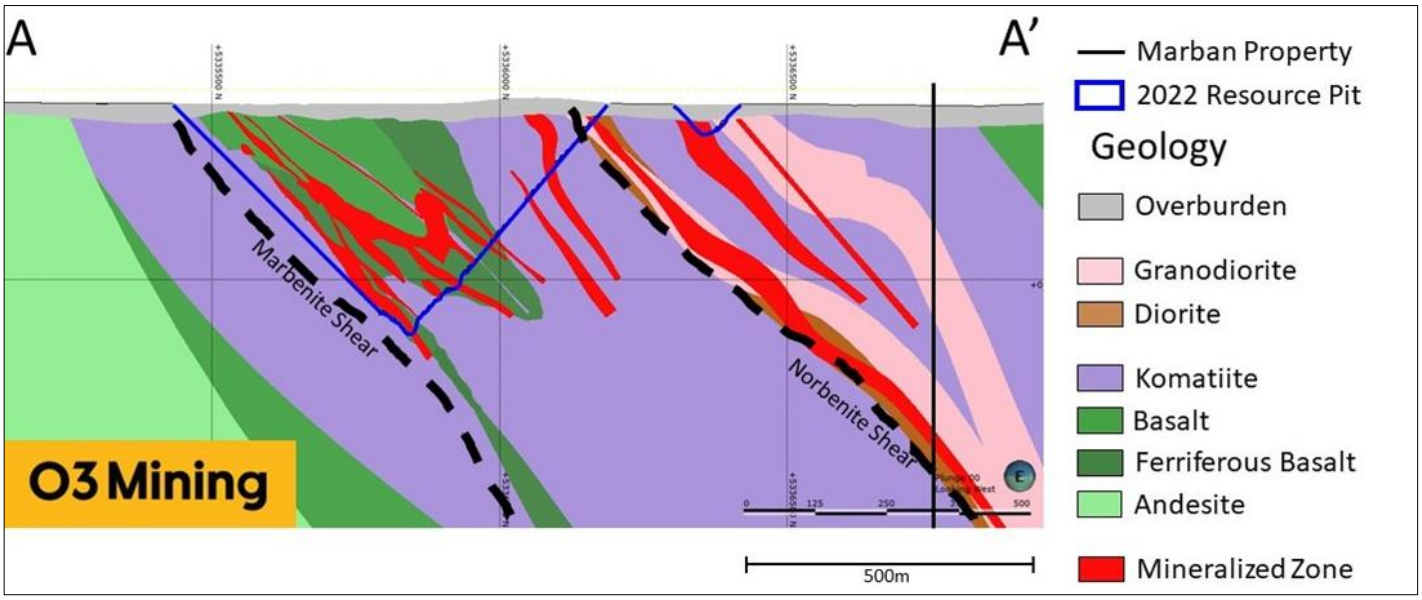
The Marban deposit is also characterized by minor cross-cutting shear that link early fold hinges together. Those shears are plunging 40–70 degrees to the north and often host gold zones. North of the basaltic units, the komatiite is injected by multiple felsic dykes, namely the Marban Dyke area. Those dykes a locally strongly altered and sheared ang contain gold mineralisation.

Figure 7-3: Geologic plan view of the Marban pits area, at 260m below surface



Source: O3 Mining, 2022.

Figure 7-4: Geologic Cross-section of the Marban Pits Area



Source: O3 Mining, 2022.

7.1.2.2 Kierens & Norlartic Resource Areas

The Kierens and Norlartic zones are located in the northern part of the west-north-west striking, north-dipping shear zones related to the Norbenite shear. The mineralization is related to a swarm of mafic to intermediate sills and dykes that intrude the shear zones (Lefrançois, 2005). The Norbenite shear is comprised of highly altered and deformed units of ultramafic volcanics and mafic to intermediate dykes. These units tend to be discontinuous due to the effects of superimposed shearing and drag folding and are thereafter cut by relatively undeform felsic dykes. The shear zone ranges from 45–90 m in width and is approximately conformable with encompassing hanging wall and footwall ultramafic units. The different lithological units present within the Norbenite deformation zone were well described by Stuart and Martin (1986). Some of the rock names have been modified in terms of new interpretations, but the descriptions are still accurate.

The mafic rocks, previously interpreted as volcanics and later as mafic dykes, make up 20–30% of the mineralized horizons and may occur anywhere within the zone. Thicknesses of individual units vary from a few centimetres to tens of metres. These rocks may be either relatively unaltered or intensely altered by albitization and carbonatization.

Relatively unaltered mafics are medium to dark green, non-magnetic, fine-grained, and massive to moderately foliated. Abundant fine-grained chlorite and carbonate are common. The altered mafic rocks are a distinct sub-unit representing up to 30% of the mafic unit. Those rocks are characterized by intense albitization, carbonatization, a quartz and quartz-carbonate stockwork is related to pyrite dissemination and generally contain economic gold mineralization. The matrix of the host rock within these altered zones is dark grey-green, fine grained, and probably recrystallized.

The komatiite makes up approximately 50% of the mineralized horizons, occurring in units varying from less than 1 metre to over 30 m in thickness. These rocks are typically blue-grey or green, have a soapy texture, and are very highly schistose and incompetent. They are composed of 50–70% fine-grained talc and chlorite, with 30–50% calcite and magnesite. The carbonates occur as greenish-white disjointed veins up to one centimetre wide that have a characteristic preferred orientation parallel to a well-developed, and commonly folded, schistosity.

The komatiites typically contain 1–2% medium to coarse-grained disseminated euhedral pyrite. Gold appears to be erratically distributed within this unit. The blue-grey komatiites are moderately to strongly magnetic and are the altered and deformed equivalents of the hanging wall and footwall komatiites. The green facies are non-magnetic, but their origin is probably similar to that of the blue-grey units, the green colour being due to the presence of abundant magnesian chlorite.

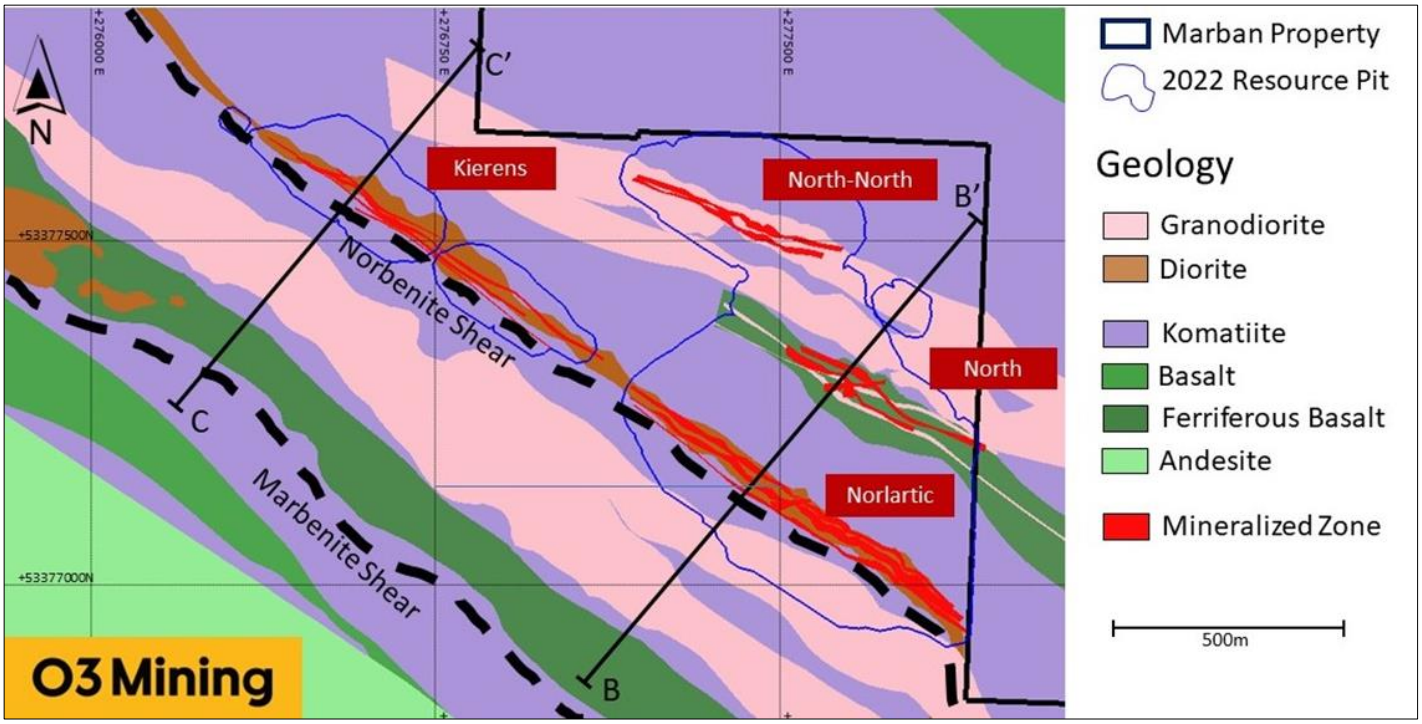
The intermediate dykes generally make up between 20% and 30% of the mineralized horizons and are an important host for mineralization in the Kierens and Norlartic areas. They range in thickness from a few centimetres to tens of metres, although rarely exceed 10 m, and are oriented parallel to the principal schistosity.

The intermediate dykes are typically medium to pale grey, rarely reddish grey (hematite staining), fine-grained to aphanitic, and have a massive to weakly foliated texture. They are usually strongly albitized and variably carbonatized, with primary textures often obliterated. In some cases, apparent recrystallization has taken place, which produces a medium- to coarse-grained, allotriomorphic, granular rock composed almost entirely of carbonate. The dykes also commonly contain 5–15% randomly oriented carbonate and quartz-carbonate veins.

The intermediate dykes usually contain between 2 and 5% fine-grained disseminated pyrite and can also contain significant associated gold values. In general, gold only occurs in trace amounts where the dykes are relatively homogenous, weakly or moderately altered, and contain fine disseminated pyrite, but the dykes host more important mineralization where they are strongly altered and are cut by numerous quartz-carbonate veins.

The mafic and intermediate dykes related associated with the Norlartic and Kierens mineralization are mainly calc-alkaline dykes that stands out of the tholeiitic nature of the Jacola Formation.

Figure 7-5: Geologic Plan View of the Norlartic Pits Area, at 40m below surface



Source: O3 Mining, 2022.

7.2 Mineralization

There are six past-producing mines within the Marban property — the Marban, Norlartic, Kierens, Malartic Hygrade, Orion No. 8, and Camflo (below 800 m from the surface) mines. The first five former mines are related to three major shear zones that have been traced across the property; from north to south, these are the North, Norbenite, and Marbenite shears. Seven sub-parallel gold-mineralized horizons are recognized; from northeast to southwest; these include the Triple North Zone, the North-North Zone, the North Zone (along the North shear), the Norlartic and Kierens zones (within the Norbenite shear), and the Gold Hawk horizon and the Marban deposit (within the Marbenite shear). The current mineral resource estimates focused on the Marban, Norlartic, Kierens, North and North-North deposits.

7.2.1 Marban Resource Area

The Marban mine is located in the southeast portion of the property and is situated within the northern part of the Marbenite shear zone. The mineralization can be followed over a strike length of 1.1km and down to a vertical depth of 650m.

Gold mineralization is hosted primarily within the Mine Sequence basalts and mostly within the ferriferous basalt or at its folded contact with the magnesian basalt. Within the basalt, the mineralization consists of quartz and quartz–carbonate–chlorite veins and veinlets with disseminated pyrite and pyrrhotite mostly within the chloritized and albitized wall rock. Veins and veinlets vary in thickness from one centimetre to few metres, but they form stockworks that can be up to 70 m thick. At the core scale, the tight folding of veinlets can be observed. Thicker quartz veins often have angular chloritized fragments of wall rock. This texture is characteristic of gold bearing veins of the Marban project and is found at Marban, Orion, Norlartic, and Kierens deposits. The folded nature of the veinlets is reflected into the shape of many historical mine stopes that drawn close fold pattern with an east-west subhorizontal plunge. This folded nature is also

reflected at a broader scale where the main stockwork envelope form a 'Z' shaped fold in cross-section with an east-west subhorizontal plunge. The mineralization is thicker in the fold hinges but still present along the limbs.

Mineralization is also hosted in local east-west shear zones that pass through the basaltic units. Those shear zones dips at 45–70° toward the north. The aspect of the mineralization is similar than within the stockwork envelope but some strongly sheared slivers of komatiite and komatiitic basalt are present in those shears.

The sulphide content related to gold consist of pyrite and pyrrhotite, with free gold occurring locally. Gold bearing pyrites are anhedral to euhedral and fine- to medium-grained and can also occur as subhedral to anhedral clusters mobilized by the deformation (Beauchamp, 2010). Pyrite also appears as coarse-grained sub to euhedral dissemination in the host rocks. The pyrrhotite is mainly visible as fine-grained anhedral clusters often elongated along shear planes. Gold may also be disseminated within the mafic volcanic rocks are within quartz veins. Although free gold is observed, the gold generally occurs as inclusions or fillings in cracks in pyrite (Renou, 2007).

North of Mine Sequence basalt, gold mineralization is related to granodiorite dykes hosted in ultramafic flows, in an area called the Marban Dyke. The mineralization is related to a stockwork of centimetric quartz veinlets within the dykes. The dykes could be almost unaltered or strongly sheared and altered in sericite, fuchsite and at a lesser extent, chlorite with centimetric clusters of pyrite disseminated within the altered dykes. But the gold grade for the Marban Dyke area seems correlated to the intensity of quartz veining.

At Marban and along the Marbenite, a very high-grade component is observed as in hole MB-11-170, were two centimetric veins of gold were intercepted (3% gold over 0.5m; press release NioGold 25 May 2011). Those spectacular grades are probably related to late remobilization along discrete structures. Isolated high-grade intercepts outside known mineralized were not model for the current MRE.

Out of the very high-grade component, the gold grade distribution inside the mineralized zones is predictable with rakes along the sub-horizontal axial plunges.

7.2.1.1 Marban Mineralized Envelopes

Following the creation of the Marban litho-structural 3D, O3 Mining decided to review the Marban resource-related wireframes, and to reinterpret the mineralized envelopes within their respective litho-structural context.

The premises used are intimately tied to O3 Mining's general observations as laid out above. For instance, the mineralized envelopes are designed to honour the geological model. In particular the lithological contacts, shear zones, fold hinges and the geometry of the existing stopes has served as guides to build the mineralized wireframes. Obviously, the gold grade continuity was also used as a guide using a 0.3 g/t Au cut-off grade and 3 m minimum thickness. The current interpretation recognizes a total of 28 mineralized envelopes. The mineralized shell created are grouped into six (6) distinct groups based on their geometry and relative location.

The first category of envelope corresponds to the large, folded stockwork envelope that follows the hinges and limbs of a more continuous mineralized sector associated with an asymmetrical west-northwest-trending fold that affects the mine sequence. The second category, also linked to the main folding, was designed as more tabular zones to complete the selection of mineralized intersect especially in the continuation of the fold limbs. Along the south limb, those zones are generally northwest-trending and shallowly dipping. They show the greatest vertical extent; likely consequence of being located along the long limb of the asymmetrical fold. The zones along the northern short limb, show a lesser vertical extent and appear to be more discrete and more widely spaced. A third group of mineralized shells correlate with planar, shallow dipping shear zones that slightly dislocate the flanks of the main folded structure.

To the south of the mine sequence, the sheared komatiites host couple mineralized zones within the Marbenite deformation zone, which represent the fourth group of zones.

To the north, two ultimate groups of zones were defined in the present work. The fifth type of mineralized envelopes is spatially associated to the felsic intrusions found in the northern ultramafic rocks. Individual zones of this group are

co-planar with the northwest-trending felsic intrusions. The last type corresponds to the southeastern extension of the Norbenite shear and hosts some mineralization that can be compared to the mineralization of the Norlartic zone to the west along the same structure.

7.2.2 Norlartic and Kierens Resource Areas

The Kierens and Norlartic deposits are hosted by the Norbenite deformation zone. Therefore, a low-grade dilution shell was designed along the structure and link the western part of the Kierens deposit up to the eastern part of Norlartic deposit.

7.2.2.1 Norlartic Deposit

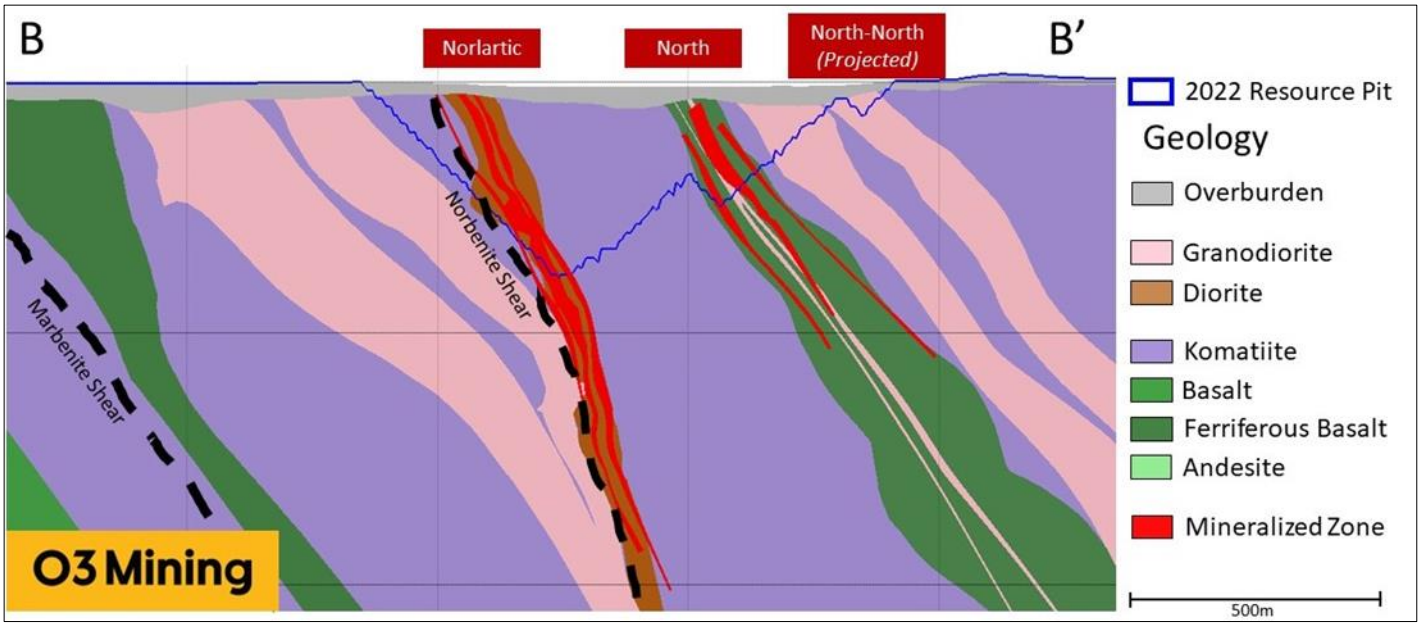
The Norlartic deposit can be trace over a strike length of 800m down to a vertical depth of 800m. Gold was mined from two principal zones: the Main Zone and the Actinolite Zone, which are both now referred to as the Norlartic deposit. These zones are hosted by the Norbenite shear. The Main zone consists of a 5 m to 15 m wide intermediate sill or dyke that occurs within a large, sheared envelope of mafic rocks in the core of the Norbenite shear (Sauvé et al., 1993). Above the 500 level of the Norlartic mine, the main sill seems to change into a swarm of smaller dykes in 'horse tail' style. Gold is usually directly associated with the intrusion(s). The intermediate and mafic intrusions in the Norlartic zone have been strongly albitized and pyritized and have undergone intense brittle fracturing and quartz-carbonate-pyrite vein injection. Economic gold concentrations are found in quartz-calcite-pyrite vein stockworks that occur in closely spaced tabular intermediate intrusions. Hosting sheared komatiite also contain disseminated pyrite and are associated with gold, creating a wide auriferous envelope. On the east side of the historical shaft, the main target was the Actinolite Zone, where similar mineralization is hosted by a set of dioritic dykes in the hanging wall of the Main Zone. The higher percentage of actinolite in the ultramafic rocks hosting these mineralized dioritic dykes gave the zone's name. The Norlartic mineralized zones were defined as three northwest-trending planar conformable with the Norbenite deformation corridor. A 0.3 g/t Au cut-off grade was used for the in-pit material while the cut-off for the under pit was raised to 2.0 g/t Au. A west-southwest planar structural component seems to intersect the zones in the old mine sector and while it doesn't seem to displace the zone, the structure carries some continuous mineralisation, so a fourth zone was created to follow this structure.

7.2.2.2 Kierens Deposit

The Kierens deposit can be trace over a strike length of 900m down to a vertical depth of 700m. Stuart and Martin (1986) have documented that gold mineralization at Kierens is primarily associated with disseminated and fracture-controlled pyrite and occurs in a strongly albitized and carbonatized zone that ranges in thickness from 3–30 m (Figure 7-5 and Figure 7-7). The alteration zone generally occupies the central portion of the Norbenite shear and is broadly conformable with it. The alteration zone affects sheared ultramafic rocks as well as more massive intermediate to mafic planar intrusions that appear as boudins and dislocated lenses that are difficult to correlate vertically. The main control on gold mineralization within the shear zone appears to be a westerly plunging drag fold. Gold is concentrated in the hinge zone of this fold structure, which may have acted as a trap for gold-bearing hydrothermal fluids (Stuart and Martin, 1986).

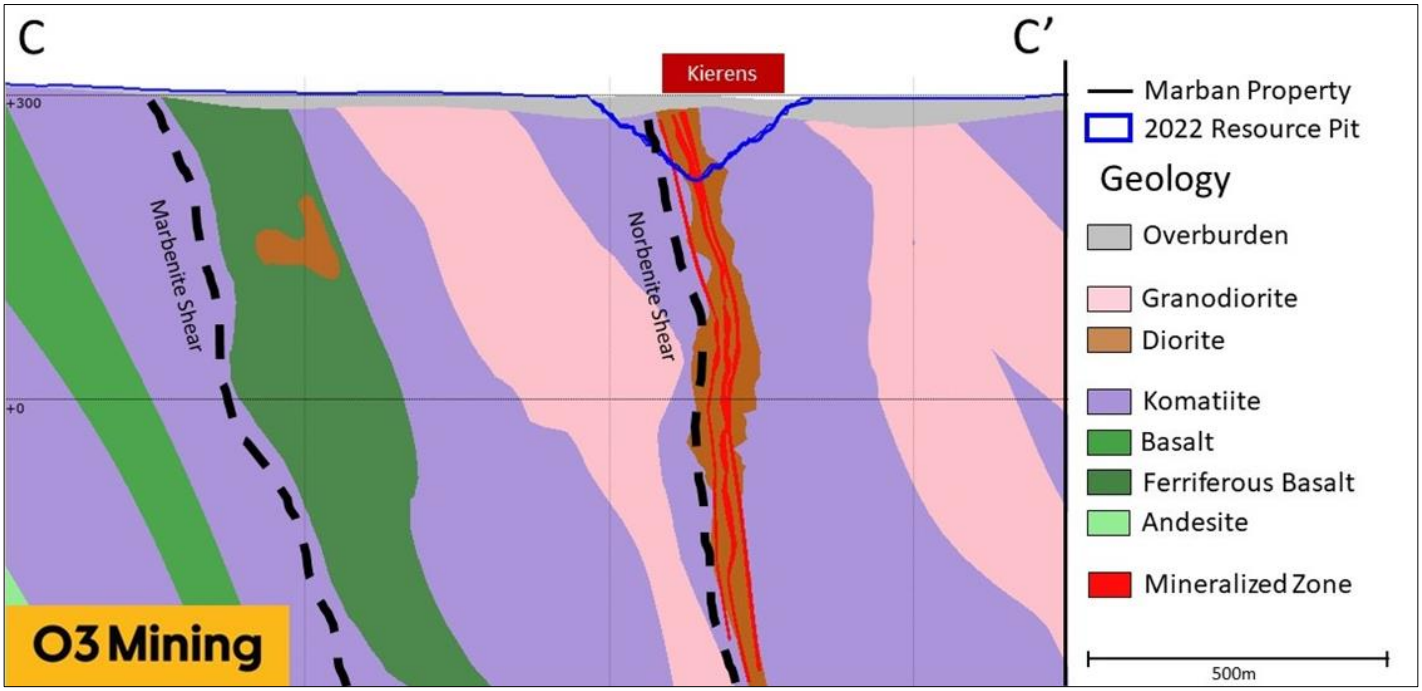
At the Kierens deposit, three sub-planar zones parallel to the Norbenite shear were modelized. Those zones are lightly offset by an east-northeast structure on the eastern part of the zone. An east-northeast structure also characterizes the western part and seems to mostly terminate the division between three different zone for the deposit (Figure 7-6). As for the Norlartic deposit, the Kierens mineralized shell were designed following a 0.3 g/t Au cut-off grade for the in-pit material while the cut-off for the under pit was raised to 2.0 g/t Au.

Figure 7-6: Geologic Cross-section of the Norlartic Pits Area



Source: O3 Mining, 2022.

Figure 7-7: Geological Cross-section of Kierens Pit Area



Source: O3 Mining, 2022.

The modelling parameter use for all the further described zone was the same. A relatively wide mineralized dilution envelope was designed using a 0.3 g/t Au cut-off grade. To limit the dilution of higher-grade material, some thinner 2.0 g/t Au cut-off grade shell zones were also created within the larger one where it applies.

7.2.2.3 North-North Zone

The North-North Zone is located 700 m northwest of the Norlartic mine shaft (Figure 7-5 and 7-6) and is a near-surface intrusive-hosted deposit with mineralized quartz-tourmaline stockwork. The North-North has a strike length of 850m with a vertical extend of 400m.

Gold mineralization is confined to a conformable quartz-albite-carbonate-pyrite alteration envelope with quartz-tourmaline-carbonate vein stockwork localized in the central portions of a 60 m wide granodiorite sill. The sill was emplaced within a sequence of deformed ultramafic and mafic volcanic rocks. The sill and alteration envelope strike northwesterly and dip 40–55º to the northeast. Based on very limited drilling below 200 m, the dip of the granodiorite apparently abruptly steepens at depth, and the alteration envelope becomes thinner and less developed. The two zones are somewhat irregular, with an average thickness of 4.5–6 m. It can be followed on strike for about 800 m and to a depth of about 150 m, with deeper exceptions (Ducharme, 2007).

The best grades and thicknesses of North-North Zone mineralization occur in a near-surface (45 m to 105 m depth) northeast-dipping zone of gold-bearing quartz-tourmaline stockwork mineralization hosted within the North-North granodiorite intrusive sill. A core orientation instrument was utilized in holes NL-06-001 to NL-06-007 in order to establish structural controls on this alteration-stockwork zone. All kinematic indicators recorded confirm a reverse movement on the controlling shears.

Sulfide content — mainly pyrite with minor chalcopyrite and pyrrhotite — occurs as disseminations and coarse clusters. Gold typically occurs as isolated particles averaging two mm in diameter in both the quartz-tourmaline veins and the altered wall rocks of the veins. Although gold may occur throughout the altered zone, it appears to be most abundant in areas of highest pyrite content and quartz-tourmaline veining (Mannard and Bubar, 1986b). Visible native gold is locally observed in the veins. There is a strong correlation between quartz-tourmaline veins and the occurrence of gold, whereas quartz-chlorite veins are seldom gold bearing (Ducharme, 2007 and 2008). Proximity of sheared and carbonatized mafic dykes and ultramafic slivers also characterize the North-North Zone.

7.2.2.4 North Zone

The North zone has a lateral extend of 700m with a vertical extend of 550m. The North Zone mineralization is associated with the North shear, which is located 350 m north of the Norlartic shaft and strikes northwesterly across the property (Figure 7-5). Gold mineralization are emplaced as three sub-parallel gold-mineralized zones dipping 60º to the northeast are recognized within the North shear and are confined to zones of quartz-carbonate veining and pyrite alteration within sheared iron-rich tholeiitic basalt (Stuart and Martin, 1988). Intermediate dykes across the deposit seem to be spatially correlated with the different zones but the gold still mostly remain in the hosting mafic volcanic. Some of the highest grades seem to occur where an east-northeast planar structurally controlled zone seems to intersect the three others. The mineralization plunges steeply to the west and seems to follow the western limit of the basaltic unit. The zone may extend eastward on the portion of the property to the north-east of the Marban deposit.

7.2.3 Other Mineralization not Included in the Mineral Resource Estimate

7.2.3.1 Triple North

The Triple North zone has a strike length of 100m down to a vertical depth of 150m. The zone is located to the north of the North-North intrusion and hosted in a mafic volcanic assemblage in the northern sector of the Jacola Formation within the property. Few holes intersected the zone recently and the preliminary observations lead to interpret the zones as east–west structures, slightly dipping to the north. A distinct lithological association with felsic dykes, with pervasive albitisation is

noticeable at least in one of the two zones. The sheared structures are oblique with the stratigraphical trend but seems to follow the dykes orientation. Strong crenulations are found within the basalt immediately north of the felsic dyke. Gold occurrences are in association with fine to medium pyrite dissemination, along with quartz-carbonate veinlets that are concordant with the structure.

7.2.3.2 Gold Hawk Zone

The Gold Hawk zone has a strike length of 300m down to a vertical depth of 500m. It lies within the Marbenite shear and is exposed northwest of the Marban deposit. The geologic setting of the gold mineralization for the Gold Hawk zone has been described in detail by Stuart and Martin (1988). The economic-grade gold mineralization is almost exclusively confined to narrow quartz veins within a 60 m to 90 m thick mafic volcanic and intrusive sequence. The volcanic rocks are relatively massive, although they lie directly adjacent to the Marbenite shear. While veins occur throughout these rocks, the gold-bearing veins tend to occur in areas where a relatively high density of veining exists and appear to fill tensional openings in the host rocks. At least one, and commonly two or more, such zones of high-density veining are generally recognized throughout the Gold Hawk zone, and these most commonly occur on the hanging-wall side of the mafic volcanic package. Individual gold-bearing quartz veins range from 4–50 cm wide. The only exception to this was encountered in hole 2001-112, where a zone approximately 6 m wide of almost continuous quartz was intersected. In this case, economic gold grades were only obtained within the first 0.3 m of the vein.

Gold most commonly occurs as coarse visible flecks within the quartz or in chloritic seams along the vein boundaries. In addition, the veins typically contain between 1 and 5% pyrite ± pyrrhotite ± chalcopyrite. Gold may have an association with chalcopyrite, as the best grades appear to correlate with veins containing 3% or more chalcopyrite.

7.2.3.3 Golden Bridge

The Golden Bridge zone has a strike length of 200m down to a vertical depth of 300m. It is hosted within the stratigraphic conformable Central granodioritic intrusion, previously mentioned, in its westernmost portion. This sill reaches close to 150 m thickness and was injected within the ultramafic assemblage of the Jacola Formation, in between the Norbenite and Marbenite deformation zones. Golden bridge mineralization is hosted within a west-northwest-trending zone that is locally accompanied by a pervasive weak to strong albitization. The zone is discordant with Norbenite and Marbenite shears. The gold is related to a stockwork of quartz and quartz carbonate veinlets that show locally dissolution void. The sulfide present is mostly pyrite dissemination and locally molybdenite selvages are related to best gold grade.

7.2.3.4 Orion

The Orion deposit has a strike length of 200m down to a vertical depth of 500m. It is hosted in the pillowed facies of a mafic volcanic sequence in the upper portion of the Jacola. It is located at or close to the parasitic Z-shaped fold hinge. Two principal zones are included in the Orion deposit: No. 8 and No. 10, with Orion No. 8 being the most developed. The mineralization is localized along a northwest trending shearing and is subparallel with the stratigraphy. Bertrand-Blanchette (2016) gives a detailed description of Orion No. 8. It is slightly dipping to the northeast at 45° at surface but become steeper (80°) at a depth of 300 m. The hanging wall is a fine-grained basalt along with ultramafic flow, while the foot wall is composed of aphanitic basaltic flows, with visible pillow and breccia. The mineralization occurs in a strong association with quartz-chlorite-albite veins and veinlets. The veining can be developed as massive quartz vein along the main shear plane and as quartz-albite stockwork veining in a fractured basalt. Strong albitization and pyritization of the hosting rock is also common, 5–10% of pyrite. Within the mafic volcanics, the pyrite is dominantly present in the pillow edges. Visible gold is frequent in the mineralized zones associated with veining.

7.2.3.5 Malartic Hygrade

The Malartic Hygrade deposit has a strike length of 200m down to a vertical depth of 250m. It comprises different mineralized zone localized in the synform-anticlinal fold hinge of the regional Z-fold, northwest of Orion. Two different main

orientation of mineralization were identified in the Malartic Hygrade area and are roughly subparallel with the stratigraphy, in both limbs of the fold. The gold bearing zones are often hosted in the basalts of the Jacola Formation, associated with quartz veining (Trudeau et Raymond, 1992). Several intermediated porphyritic intrusive rocks cut the Jacola volcanic sequence, mostly as planar dyke, but a cigar shape intermediate stock that plunge north-northeast is also injected in the fold hinge. The gold bearing quartz veins are spatially associated with those intrusions, the veins are taking place mostly in more competent unit in the fracturing caused by the deformation. The quartz veining is filling the fractures and within shear zones.

7.2.3.6 Malartic H

The Malartic H deposit has a strike length of 600m down to a vertical depth of 500m. It is located about 1km to the north of Malartic Hygrade, on the other end the short limb of the regional Z-fold, close to the antiform-synclinal hinge. The deposit is hosted by an alternance of mafic and ultramafic volcanic sequence of the Jacola. As visible in the Hygrade area, numerous complex intermediate dyke system is injected in the volcanic sequence. The deposit is associated with a northwest-trending deformation zone that dip around 70° to the northeast. The mineralization occurs as a pyrite dissemination with local visible gold in quartz veins and veinlets and disseminated in the host rock. Zones are spatially correlated with contacts between mafic and ultramafic volcanics or with the intermediate dykes. All zones are roughly subparallel.

7.2.3.7 Camflo

The Camflo shaft is located on the Canadian Malartic partnership property, outside O3 Mining's claims. But the mineralization is mainly hosted into a cigar shape intrusive plug that plunge to the northeast and enters the Marban property at about 800 m below the surface. The Camflo plug correspond to an alkaline porphyritic monzonite to quartz monzonite with K-feldspar phenocrysts. The intrusion is characterized by the presence of fluorine crystals. The mineralization is mainly restricted to the plug with a strike length of 200m with a vertical extend that exceeds 1.6km. The mineralization consists of a stockwork of quartz–carbonate–feldspath veins and veinlet and disseminated pyrite within the intrusion. The surrounding sedimentary rocks of the Kewagama Group can also be altered and mineralized. A secondary zone, called Diorite Ore, is hosted within a diorite planar dyke that took place along the contact between the Kewagama sediments and the Louvicourt lava flows.

8 DEPOSIT TYPES

The various mineralization styles present on the Marban property can be characterized as sub-types of the orogenic class of gold deposits, most commonly shear-zone hosted gold mineralization within greenstone terrains.

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 terrains 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). They are also known as mesothermal, orogenic. 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.

In the case of Marban property, the three major shear zones previously described in Section 7 are the principal fluid conduits, with second or third-order structural traps being prospective for the accumulation of gold mineralization. Typical mineralization consists of quartz and quartz–carbonate–chlorite veins and veinlets within mafic and intermediate rocks, with a low sulphide content (<2%).

Although the mineralization at Marban shares multiple characteristics of orogenic gold deposits, namely the association with major shear, the carbonite and albite alteration and the lack of significant base metal or silver content suggests they are much older than the orogenic gold deposits of the Val-d'Or mining district (Sigma-Lamaque and others). They then can be classified as pre-main deformation event Archean orogenic gold deposits (Couture et al., 1994; De Souza et al, 2020).

In addition to the typical shear zone-hosted gold mineralization, tonalite and granodioritic intrusions in the Malartic region can be prospective host units for mineralization and exhibit different alteration minerals due to the chemical interaction between mineralising fluids and the host rocks. This type of mineralization tends to be associated with quartz–carbonate–tourmaline veins surrounded by albite alteration halos. They are very similar to the vein systems found at Sigma-Lamaque and Goldex mines in the Val D'Or mining camp. The veins are spatially associated with discrete shears, but the veins itself are relatively undeformed.

9 EXPLORATION

Since the creation of O3 Mining in 2019, exploration work on the Marban property has focused mainly on exploration drilling. The significant exploration results that are material to this Technical Report were obtained by surface core drilling. This work and resulting interpretations are summarized in Sections 10 and 14 of this Technical Report.

In May 2021, an air-borne drone magnetometric survey was completed covering the Camflo, Orion, Malartic Hygrade, and Malartic H areas. Vision 4K inc. performed the survey using the AIM-LOW™ system. Flight lines were spaced at 25 m and the total survey length was 394.5 linear-km. This first survey was a test to plan a broader second survey using the same technology which covered the entire property using high-resolution surveys. The second drone survey wascompleted in March 2022 and covered all the western portion of the property with flight lines spaced at 50 m; in total, 712 linear-km were flown.

In November 2021, Abitibi Geophysique Inc. undertook an Orevision® induced polarization (IP) orientation survey on nine lines covering the Marban, Norlartic, Kierens, Gold Hawk, and Orion deposits. The purpose of the survey was to validate whether the known mineralization present at those deposits had distinguishable IP responses. Lines were between 400 and 1,000 m apart and 1 to 3 km long. It successfully detected the Norlartic and Kierens zones with the highest chargeability values of the survey, around 9 mV/V. It also detected the Orion and Malartic Hygrade zones with lower chargeability and 5 to 6 m V/V.

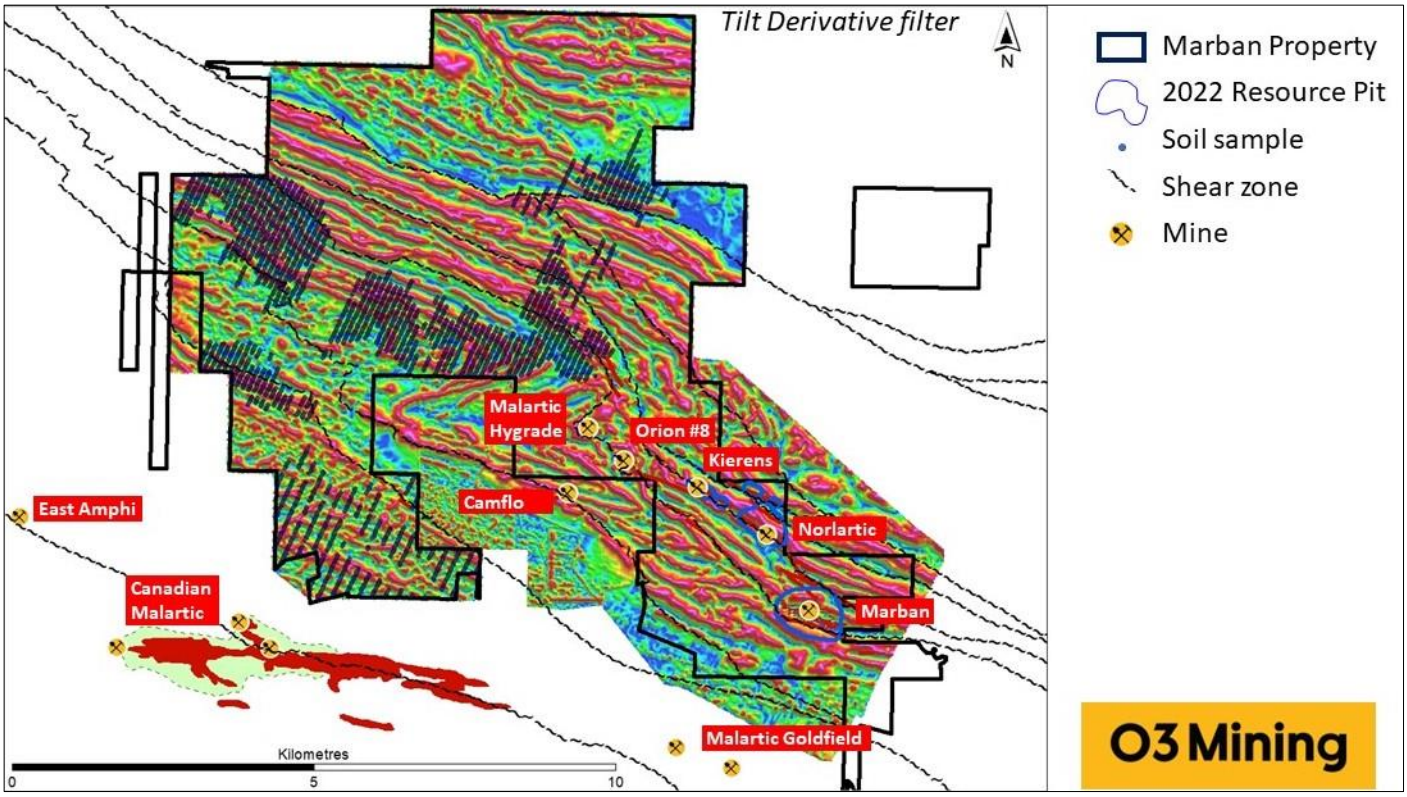
In July 2021, a soil and spruce bark orientation survey was carried out over mineralized and barren areas. The survey covered Marban, Norlartic, Kierens, Gold Hawk, and Orion deposits. Eighty-five soil samples and 74 bark samples were collected along five lines with a sample spacing of 25 m along the lines. The soil samples were analysed with two different methods, one using a standard aqua regia digestion analysed with an induced coupled plasma mass spectrometer (ICP-MS) and also using a Mobile metal ion (MMI) leaching. The aqua regia assays successfully detected the gold mineralization underneath sampling location using the raw gold values and using residual gold values after applying a multi-variate regression. The bark samples assays returned gold only near the Orion deposit in a sub-outcropping area.

In July 2021, the Orion No. 8 outcrop was sampled by 12 channel that totalled 136 samples covering 131 m. The surface map of Bertrand-Blanchette (2016) has been updated. Channel sample descriptions confirm that the gold is associated with a metric quartz vein with angular chloritic clasts and a quartz veinlets stockwork hosted in a strongly albitized and pyritized basalt. The amount of sulphide reaches 10% pyrite, higher than at the other gold deposits on the property. The best channel sample returned 21.8 g/t Au over 11.2 m. The Orion ore body has been mined out up to about 10 m below this outcrop. For this reason, the results are considered not material and therefore were not published.

From June 2022 to September 2022 a soil survey and prospection were carried out on the western portion of the property. In total, 4,135 soil samples, 284 channel samples and 50 grab samples were taken. All assays are pending for that campaign.

See Figure 9-1 for an illustration of the magnetic data overlying the property and location of the 2022 soil sample campaign.

Figure 9-1: Tilt Derivative Filter of the Magnetic Data overing the Property and Location of the 2022 Soil Sample Campaign



Source: O3 Mining, 2022.

10 DRILLING

The drilling database for the entire Marban property contains 7,593 holes representing a total of 1,099,817 m. The database covers the entirety of the Marban property, and a significant amount of drillholes were outside the areas covered by the current MRE. Table 10-1 shows the breakdown of those drillholes, the period in which they were drilled, and their consideration in the current MRE. All drillholes outside the Marban, Norlartic, Kierens, North, and North-North deposits were excluded from the MRE. All underground test holes, muck samples were also excluded from the MRE. Historical drilling (pre-1986) with a detection limit (DL) above or near the open-pit cut-off were also excluded from the MRE, as discussed in Chapter 12. After these revisions, 1,743 holes totalling 401,178 m were considered in the MRE, including data up to January 13, 2022. The database provided consisted of a Leapfrog Geo™ project and CSV files exported from O3 Mining's Geotic™ database. The dataset includes collar location, downhole survey, assay, lithologic, alteration, mineralization, geotechnical, and structural data. All projected coordinates use the Universal Transverse Mercator (UTM) projection and the NAD83 datum Zone N18. All azimuth data used in the field were in degrees related to True North. A value of 2.28° was added to all azimuth data to reflect the difference between the True North azimuth and the North azimuth for UTM projection at the Marban project location.

Table 10-1: Summary of Drilling Database

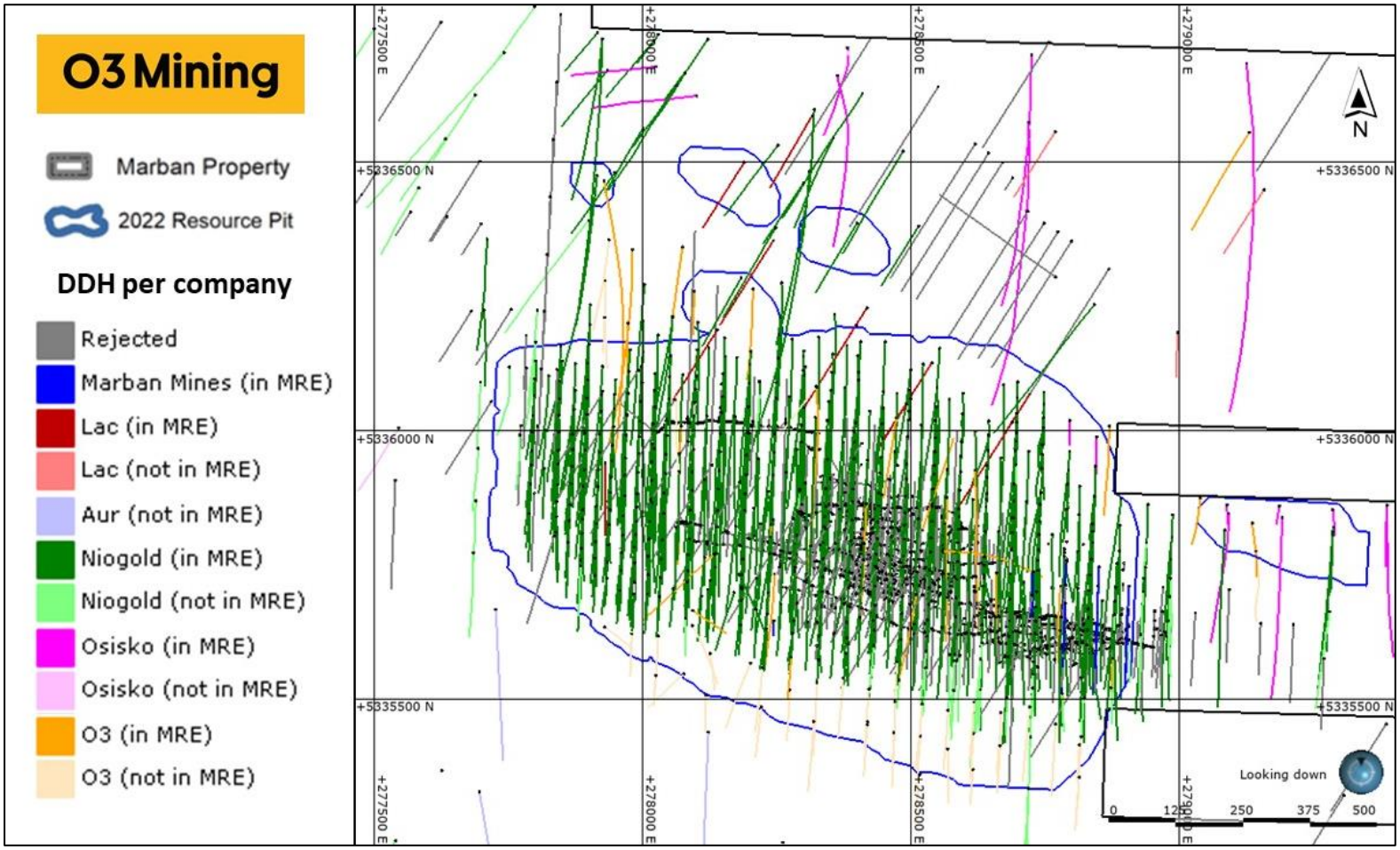
Owner	Period	Type	Excluded from MRE			Included in MRE		
			Number of holes	Total length (m)	Assayed length (m)	Number of holes	Total length (m)	Assayed length (m)
Various	Pre-1994	Surface and Underground	5,400	572,938	144,291	914	140,428	98,177
NioGold	2006-2015	Surface	314	75,901	52,021	640	205,315	177,920
Osisko	2016-2017	Surface	10	5,295	4,006	16	9,876	6,724
O3	2020-2022	Surface	126	44,506	31,332	173	45,558	40,504

Surface drilling was mainly oriented from 180–220° with a dip generally between 45° and 65°. This orientation was more or less perpendicular to the mineralisation main trend. Some infill drillholes were oriented 0–40° due to access and terrain constraint. The historic underground drillholes have highly variable azimuths and dips, as they were drilled in series of fans from individual drill stations, which created multitude of angles between drillholes and mineralization. Some of them are more or less parallel to the mineralization and may not be representative of true mineralized width. However, this may be mitigated by the modelling technique that constrains all intercepts within interpreted domains with respect to true mineralized widths.

Figure 10-1 illustrates all the drillholes within the Marban areas used in the MRE coloured by year drilled, and also shows the Marban resource pit outline for reference. Drilling from 1986 to the present shows that there remains good drill density throughout the Marban deposit.

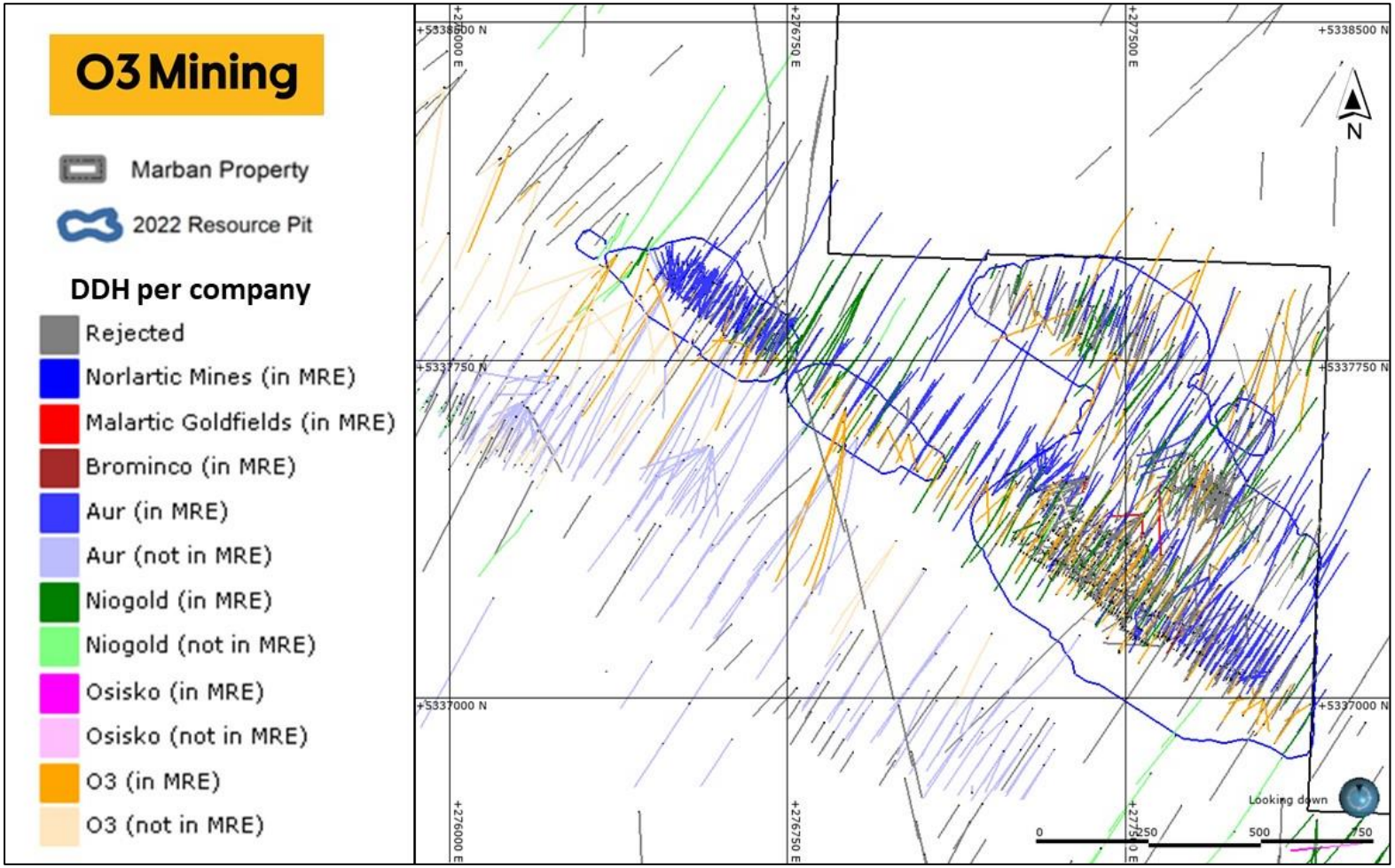
Figure 10-2 illustrates all the drillholes within the Kierens-Norlartic areas that were used in the MRE, coloured by year drilled. The Kierens, Norlartic, Gold Hawk and North Zone resource pit outlines are also shown for reference.

Figure 10-1: Plan View of Drillhole Database by Company – Marban Area



Source: O3 Mining, 2022.

Figure 10-2: Plan View of Drillhole Database by Company – Norlartic Area



Source: O3 Mining, 2022.

A typical section from Marban is presented in Section 7 (Figure 14-5) and an isometric view of the drilling on the Marban deposit is presented in Section 14 (Figure 14-3).

Typical geological cross-sections from Kierens-Norlartic is presented in Section 7 (Figure 14-5 and Figure 7-7) Isometric view of the drilling on the Norlartic and Kierens deposit is presented in Section 14 (Figure 14-4) and well as a typical cross-section of wireframes used for the MRE (Figure 14-5).

10.1 Drilling Considered in the MRE

The provided database was divided in two sub-databases; a first one covering the Marban deposit and the southeastern extension of the Norlartic deposit, and a second one covering the Norlartic, Kierens North, and North-North deposits. The current subsection describes the different drilling campaigns and the origin of the core used in this MRE. Table 10-2 and Table 10-3 show the various drilling campaigns subdivided by year.

Data on core size, dates, and drilling contractors for the pre-NioGold holes are derived primarily from copies of the original geological logs of drillholes. The summary for historic holes is based on the audit completed for the 2013 report (MDA, 2013).

NioGold's first drilling program at the Marban property was undertaken from July 4 through November 26, 2006. This program included 64 core holes (including one wedge) for a total of 10,196 m, that tested the Marban (2 holes), Kierens (10 holes), Norlartic (2 holes), Gold Hawk (15 holes), North, and North-North (35 holes) zones.

Table 10-2: Summary of Marban Database used for MRE

Year (approx.)	Owner	Type	Hole Series	Core Size	Number of Holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
1960	Marban Mines	Surface	MBS		11	3,267	2,665	1,028
1986	Lac Minerals	Surface	MBS	BQ	7	1,299	991	364
1987	Lac Minerals	Surface	MBS	BQ	5	2,089	1,947	1,158
1988	Lac Minerals	Surface	MBS	BQ	3	1,353	1,273	493
2006	NioGold	Surface	MB	NQ	2	1,116	1,007	-
2007	NioGold	Surface	MB	NQ	26	11,603	10,327	-
2008	NioGold	Surface	MB	NQ	59	22,870	18,008	-
2009	NioGold	Surface	MB	NQ	3	1,436	935	
2010	NioGold	Surface	MB	NQ/BQ	43	12,884	11,486	-
2011	NioGold	Surface	MB	NQ/BQ	86	27,542	24,535	-
2012	NioGold	Surface	MB	NQ/BQ	84	30,467	27,756	-
2013	NioGold	Surface	MB	NQ	4	1,251	1,129	-
2014	NioGold	Surface	MB	NQ	34	12,453	11,170	-
2015	NioGold	Surface	MB	NQ	188	56,536	48,738	-
2016	Osisko	Surface	MB	NQ	9	8,016	5,011	-
2017	Osisko	Surface	MB	NQ	7	1,860	1,713	-
2021	O3	Surface	O3MA	HQ/NQ/BQ	27	9,871	9,070	-
Total					598	205,913	177,763	3,043

Table 10-3: Summary of Norlartic-Kierens Database used for MRE

Year (approx.)	Owner	Type	Hole Series	Core Size	Number of holes	Length (m)	Assayed Length (m)	Re-assayed length (m)
1958	Malartic Goldfields	Underground	N		5	987	763	116
1964	Norlartic Mines	Surface	NS		8	2,299	1,352	458
1980	Brominco Inc	Surface	BV		2	208	152	126.
1986	Aur	Surface	1902	BQ, NQ	72	16,803	12,810	1,592.
1986	Aur	Surface	2001	BQ	152	36,021	28,415	1,025
1988	Aur	Underground	NU	BQ	313	41,074	25,334	4,258
1990	Aur	Surface	OVB		21	324	34	-
1990	Aur	Underground	FU		315	34,704	22,440	4,012
2006	NioGold	Surface	FC	NQ	10	1,465	1,145	-
2006	NioGold	Surface	NL	NQ	37	6,707	4,862	-
2007	NioGold	Surface	NL	NQ	5	1,104	695	-
2011	NioGold	Surface	NL	NQ	22	4,523	4,017	-
2012	NioGold	Surface	NL	NQ	5	1,504	1,380	-
2013	NioGold	Surface	FC	NQ	15	7,162	6,697	-
2013	NioGold	Surface	NL	NQ	13	3,047	2,710	-
2014	NioGold	Surface	FC	NQ	4	1,646	1,323	-
2020	O3	Surface	O3MA	NQ	12	6,714	6,503	-
2021	O3	Surface	O3MA	NQ	134	28,973	24,930	
Total					1,145	195,264	145,562	11,587

All 2006 holes were drilled with NQ-size (47.75 mm) core (Lefrançois, 2006; 2007) by Forage Mercier of Val-d'Or using an FM-1500 fully hydraulic core rig in a padded shelter. Drill sites were located in the field using re-surveyed Aur drillhole casings as control points, as well as re-established field-grid baselines.

Downhole azimuth, inclination, and magnetic-field measurements were taken using the FLEXIT Survey System every 30 m (50 m for longer holes) while drilling using a downhole single-shot instrument, and subsequently every 3 m after completion of the hole with an up-hole multi-shot instrument, both operated by the drilling contractor (Davy, 2006, 2008; Lefrançois, 2007). The drillhole collars were surveyed by J. L. Corriveau and Associates Inc. using a real-time high-precision GPS unit. Drillhole casings were left in place when the sites were abandoned.

The drill core was taken by NioGold personnel to a core shack at the Val-d'Or office for washing, photographing, logging, and sampling (Lefrançois, 2007). RQD and recovery information was then measured and compiled into the database. The core was logged by NioGold geologists using GeoticLog® software before sampling. Various drilling parameters, including downhole surveys, were also compiled into the database.

NioGold's Phase II drilling took place from April 24 to September 13, 2007, and from October 25, 2007 to November 1, 2008. This program included 90 holes drilled on the Marban deposit, 3 holes, and 2 wedges at Norlartic, and 1 hole at Gold Hawk.

Eighty-eight of the Marban deposit holes were drilled in a 50 m by 50 m pattern. Two other holes (MB-07-021 and MB-07-022) were drilled to investigate gold anomalies found in pre-Osisko holes at Marban Northeast; this drilling revealed a narrow but high-grade system, which NioGold interprets as the extension of the Kierens-Norlartic trend (Ducharme et al., 2009). The hole drilled at Gold Hawk tested two structural targets. The three Norlartic holes and two wedges tested northwestward extensions of the Norlartic mineralized system.

Forage Mercier was again the drilling contractor for NioGold's 2007–2008 drilling. The drill was an HD 3000, and the holes were drilled with NQ core. Core-handling procedures were as described for the 2006 drilling. NioGold rented and operated the equipment to survey the collars from J. L. Corriveau and Associates Inc. The precision of the surveys is reported by NioGold to be about 3 cm for the easting and northing and about 30–50 cm for the elevation. As in the 2006 program, downhole surveying was conducted by the drillers using the FLEXIT Survey System. Single-shot measurements were taken every 30 m during hole drilling, and up-hole multi-shot measurements were taken every 3 m when the hole was finished. In addition to the second NioGold drilling phase, some intervals not previously sampled in drillholes MD-06-001 and MD-06-002 in 2006 were sampled in 2007.

Four core holes were completed late in 2009 to test the eastern extension of the Marban zone.

NioGold drilled 50 holes at the Marban deposit in 2010, with another 103 new holes and 8 extensions of previous holes completed in 2011. In addition, they drilled 24 new holes at the Norlartic deposit in 2011 and 1 additional hole in 2013. NioGold completed 88 new holes and 9 additional extensions of previous holes at the Marban deposit in 2012 as well as 17 new holes and an extension of a 2006 hole at the North Zone between December 2012 and February 2013. They also drilled 16 additional holes at the Kierens Resource in 2013.

Forage Mercier of Val-d'Or was again the drilling contractor for all the drilling from 2009 through 2012, using up to four drills. All drilling was NQ size, except for holes that traversed underground workings, which were completed using BQ size. NioGold continued completing in-house drillhole collar surveys. From 2006 to May 2011, collars were surveyed with a Leica Geosystems instrument with real-time correction and an accuracy of 0.075 m. Since July 2011, they have used a Trimble Geo XH with a Zephyr antenna and with real-time correction; accuracy is 0.1 m.

A total of 239 out of the 241 holes drilled at the Marban deposit in 2010, 2011, and 2012 have downhole survey data, as do the 41 holes drilled at Norlartic and North Zone during the same time period. The surveys were completed by the drillers using a Reflex instrument, with a single shot at every 30 m and multi-shot at every 3 m. NioGold personnel entered the downhole survey data into the project databases and then carefully reviewed the results. Abrupt changes in azimuth and, less commonly, dip, were flagged in the database as "invalid."

In 2014, 4 holes were drilled at Kierens and also four at Norlartic (representing 1,646 m and 1,368 m respectively). Also, in 2014–2015, 246 supplementary holes and 1 extension representing 72,672 m were drilled at Marban to complete the drilling pattern to about 25 m by 50 m. Forage Spektra of Val-d'Or was the drilling contractor for this program.

In 2015 and 2016, NioGold drilled 9 PQ-size holes on the Marban deposit and 3 PQ-size holes on the Norlartic deposit. Holes were twinning previous NioGold holes and were drilled for comminution testing. The contractors were Orbit-Garant Drilling and G4 Drilling. Procedures for surveying and downhole measurements remained the same as in 2012–2013.

Osisko drilled 16 drillholes between 2016 and 2017 on the Norlartic and Marban deposits. The drilling contractor was Forage Hébert. Downhole azimuth, inclination, and magnetic-field measurements were taken using the FLEXIT Survey Systems at every 50 m in single-shot mode during the drilling and at every 3 m in multi-shot mode after completion of the hole. The instrument was operated by the drilling contractor. The drillhole collars were surveyed by Osisko's personnel using a Trimble GeoXH with a Zephyr antenna and real time correction. All drillhole casings were left in place.

O3 Mining started drilling on Marban Engineering in August 2020 and was continuing as of the date of closure of the database. This drilling program totalled 287 holes; 9 of them were wedged and 2 were extended, for a grand total of 80,559 m. The first phase of the program targeted extensions at depth in the North, Gold Hawk, Orion No. 8 and Marban zones. Drillholes were generally spaced by 100 m to expand the known mineralization. After completion of the PEA, an important

infill drilling phase was carried out within the pit designs, and also between the Norlartic and Kierens pits. Drilling spacing varied between 30m and 40 m during the campaign. A third phase consisted of 14 geotechnical drillholes, which were also within the proposed pits outlines. A fourth phase was completed in December 2021 to infill the Gold Hawk PEA proposed pit. As most assay results from the fourth phase were pending at the closing date of the database, the holes from this campaign were excluded from the MRE.

To carry out this exploration program, four contractors were involved in the field; they were Forage Dami-Or, Orbit Garant, Spectra Drilling, and RJLL. Downing Drilling was in charge of the geotechnical drilling. Most of O3 Mining's drillholes were drilled in NQ-size core. Some of them were drilled in HQ and then telescoped to NQ and occasionally BQ, although BQ was only used when multiple historical mining openings had to be traversed. The inland drillhole collars were surveyed by J. L. Corriveau and Associates Inc. using a real-time high-precision GPS unit. The on-barge drillholes were surveyed using a SX-Blue precision GPS. All drillholes deviations were measured using EZ-Gyro or EZ-Trac of Reflex, depending on the magnetism of the rocks drilled. Drillhole casings were capped using a metal cap with a tag. When the drillholes were flowing with artesian water, they were capped with a valve casing cap.

The planning, core logging, data validation and supervision of the 2020–2021 drilling program were supervised by O3 Mining staff.

The core was logged by O3 Mining geologists using GeoticLog software before sampling. Rocks were described in terms of their different aspect, namely lithology, alteration, ductile and fragile deformation, veining and rock quality. In addition to those descriptions, samples for lithogeochemistry characterization were taken. For each of those samples, three portable X-ray fluorescence (pXRF) analysis and magnetic susceptibility measurements were taken to validate the geological and alteration codes described. Logging was done at the company's core shack, located at 11 Finlay of the O3 Mining office in Val-d'Or.

10.2 Sampling Methods

10.2.1 Pre-NioGold Sampling

The following subsection on pre-NioGold sampling procedures has been sourced from a prior Technical Report (Carrier, 2006).

There has not been any systematic reviews of the sampling method and approach used during the historical assessment work, but sampling techniques have varied at different times and different stages of activity on the Property. Sampling methods used during underground operations and exploration programs included different core sizes (AQ, BQ and NQ), chips, muck, and sludge samples. It should be noted that the resources discussed herein are based on sampling of drill core only, and underground channels, muck, or sludge samples were excluded in the MRE.

10.2.2 Kierens Sampling

This section describes the sampling methods at Kierens as described by Mannard (1987).

Diamond drill (DD) core was systematically sampled across the entire width of the altered zone. The sample lengths were variable due to the rapid alternation in lithologies and prevalence of narrow mineralized structures observed within the altered zone. DD core was split with a hydraulic core splitter or sawed with an electric diamond rock saw. The rock saw was systematically cleaned on a daily basis and DD core was thoroughly washed with water before being sent to the assay lab. These precautions were regimentally adhered to in order to prevent potential contamination from the rock saw cuttings.

Chip samples were taken from each successive drift and raise round across the full width of the face. Samples were systematically taken at waist height regardless of geology. The samples were obtained using a chisel and hammer and collected from a tarpaulin spread on the drift floor or raise staging. Care was taken to procure large, representative, and unbiased samples measuring 4 inches in width and 2–4 ft in length.

Muck samples were taken from each successive drift and raise round. Drift muck was transported from the ore drift to the shaft by one yard scoop trams and hoisted to surface. One shovel full of muck was sampled from each 0.7-yd^3 bucket of muck dumped on surface. This resulted in a sample for each round that weighed approximately 100 pounds. This sample was crushed on surface using a jaw crusher with a 3/8-inch jaw spacing. The crushed sample was split, using a Jones type riffle to produce three 10-pound samples which were sent for assay. Assay results from each sample split were averaged to produce the final muck grade.

The crusher's jaws and sample pans were cleaned with a wire brush between each sample and crushed rejects were stored in burlap bags which were left on the corresponding, numbered muck piles in the surface stockpile.

Raise mucks were sampled from the muck heap immediately below the raise portals then crushed and split following the same procedure.

Sludge samples were taken at right angles to the ore drifts at 10-ft intervals along the entire length of both walls of the ore drifts. Sampling was performed using a Jackleg Stoper with carbide button bits. Drill cuttings were collected in plastic bags mounted on a Penguin muck shovel. Each hole penetrated 8 ft into the wall of the drift and three samples were procured from each hole (two 2-ft samples followed by a 4-ft sample).

10.2.3 NioGold Sampling

The following summary has been sourced from the 2016 Technical Report. Details on sampling methods during NioGold's drilling were provided by NioGold personnel (Yan Ducharme, 2016) and various news releases.

The NQ-sized core from the drill was taken to either the NioGold office in Val-d'Or (2006–2007) or to NioGold's core facility located at the Norlartic property (2007–2015). The boxes were then opened by NioGold geologists and technicians. The core was photographed, measured, and geologic and geotechnical logging was completed. All sections of the drill core deemed to be potentially mineralized were identified and samples to be taken clearly identified and tagged. Their lengths ranged between 0.5 and 1.5 m.

Sampling was done by sawing the core in half, with one half placed with a tag in a labelled bag and then sealed, and the other half stored in the core shed as a witness sample for future reference. Transport to the laboratories was done by NioGold personnel or, when possible, by laboratory personnel.

10.2.4 Osisko Sampling

The NQ diameter core from the 2016 and 2017 drilling program was placed in core boxes and sealed by the drilling contractor, before transport to the NioGold core shack. The boxes were then opened by the technicians. The core was photographed, measured, and described in detail. All samples, with lengths ranging from 0.5–1.5 m, were clearly identified and tagged with unique sample numbers. Sampling was done by sawing the core in half, with one half placed with a tag in a labelled bag and then sealed, and the other half stored in the core shed for future reference.

10.2.5 O3 Mining

The BQ, NQ, and HQ diameter core from the 2020 and 2021 drilling program was placed in core boxes and sealed by the drilling contractor, before transport to O3 Mining's core shack. The remainder of the sampling protocol remains similar to the Osisko Period.

10.3 Summary

The QP believes that the drilling and sampling procedures implemented at the Marban and Kierens-Norlartic deposits are consistent with best industry practice. The chain of custody has been clearly documented for all activity pre-O3 Mining, and the current procedures ensure the integrity of the sample database. The QP is satisfied that the information retained in the MRE is of sufficient quality, and that questionable data has been appropriately excluded.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Historic Sample Preparation, Analysis, and Security

The original Technical Report (Carrier, 2006) reported that there have been no systematic reviews of sample preparation, analysis, security, or QA/QC used by operators prior to NioGold.

The 2013 Technical Report (Gustin, 2013) reports that the only laboratory NioGold could identify as having been used in historic Kierens and Norlartic drilling, other than information reported in the original Technical Report and described below, was Abilab Inc. from Val-d'Or, which is no longer in operation. MDA found evidence of fire assays with gravimetric finishes from Abilab Inc. for two of Aur's holes drilled in 1987 and 1990. The detection limit was 0.034 g/t Au (0.001 oz/ton Au) for the Abilab assays, according to the database.

Mannard (1987) described sample preparation and analytical methods used by Aur at the Kierens mine during its 1986–1987 underground exploration program. For core drilling, the split core was analyzed by fire assaying one-assay-ton aliquots. Samples containing quartz-vein material and visible gold were analyzed by metallic-sieve methods. The remaining core was retained for reference in a permanent core-storage facility located adjacent to the surface facilities at the Norlartic mine site. Assays on core samples from this underground program at Kierens were performed by Bourlamaque Assay Laboratories Ltd. (Bourlamaque) in Val-d'Or and Bell-White Assay Laboratories (Bell-White) in Haileybury, Ontario, both independent to Niogold, using standard fire assays. Assays were also done at Chimitec in Quebec City, with some overflow sent to Bondar-Clegg in Ottawa. Several composite samples of drill-core rejects from the North-North Zone were sent to Lakefield Research Laboratories (Lakefield) for total-gold extraction analysis (Mannard, 1986, as cited by Carrier, 2006). Chip samples from the Kierens underground program were analyzed using standard fire assay or metallic-sieve/fire-assay techniques on one-assay-ton aliquots. Muck and sludge samples from Kierens were also assayed for gold using fire assay.

For assaying of their 1985–1986 drill samples from the North-North Zone, Aur used Bourlamaque and Bell-White (Mannard and Bubar, 1986b). Gold was analyzed by conventional fire-assay methods using one-assay-ton aliquots and, in some cases, ½-assay-ton. Selected samples observed or suspected to contain coarse native gold were analyzed by metallic-sieve methods. Several composite samples were prepared from core sample rejects in selected holes and were assayed by a total-gold extraction technique at Lakefield.

Based on copies of original laboratory certificates used in the process of auditing, MDA reported that Brominco used Bourlamaque for analyses of the samples from their BV-series of holes drilled in 1980 and 1981 at Norlartic, and Lac used Minera Lac Ltd. Regional Lab for their holes drilled at Marban.

The detection limit of the Brominco analyses performed by Bourlamaque was 0.343 g/t Au (0.01 oz Au/ton). Examination of the gold analytical data from Little Long Lac and Marban Mines from Marban drillholes, and data from Malartic Gold Fields, Norbenite Malartic, Norlartic Mines, and SOQUEM from Kierens-Norlartic holes, infers that the detection limit for these assays was also 0.343 g/t Au (0.01 oz Au/ton). The precision above the detection limit appears to have been 0.343 g/t Au as well, which led to a very large number of assays with grades that are multiples of 0.343 g/t Au.

11.2 NioGold Sample Preparation Analysis and Security

The following information on NioGold's assaying procedures is taken from the original Technical Report (Carrier, 2006) and is updated with information from NioGold; Davy 2006, 2008; Lefrançois, 2006, 2007, and personal communication, November 6, 2006; Ducharme et al., 2009; and other references as cited.

For the 2006 drilling, gold assaying of the drill core from the North-North and North Zone holes was done by ALS Chemex Labs Ltd. (now called ALS Minerals [AL]) in Val-d'Or, which is ISO:9001:2000 /IEC17025 certified. Gold assaying on drill core from holes in the Kierens and Norlartic areas was performed by Bourlamaque in Val-d'Or. Samples from the Marban deposit and some from the Gold Hawk zone were assayed by Techni-Lab in Sainte-Germaine-Boulé (ISO 17025 certified); other drill

samples from Gold Hawk were analyzed by ALS. Unusually high turnaround times for assaying forced NioGold to use these three different laboratories. Samples from the North-North Zone and Gold Hawk zone were assayed systematically using the total metallic-sieve assay method, as were intervals with visible gold in holes on the North, Marban, Norlartic, and Kierens zones. All other samples were analyzed by fire assay using an atomic absorption (FA-AA) finish on 30 g or 50 g pulp splits of 1,000 g pulps.

For the 2006 drilling, coarse-reject fractions were systematically re-assayed using 50 g FA-AA for samples that originally assayed from 2 to 5 g/t Au, and 50 g fire assay with a gravimetric finish for samples originally assaying over 5 g/t Au.

During the 2006 drilling program, drill core was transported by NioGold personnel from the drill site to a secured core facility located at their office in Val-d'Or. The half-core samples to be assayed were bagged, sealed, and delivered by NioGold staff to ALS for the holes in the North-North and North Zones, and to Bourlamaque for the holes in the Kierens zone, Norlartic Main zone, and the Marban mine area. Both laboratories are located in Val-d'Or. Samples from the Gold Hawk zone were picked up by Techni-Lab staff and delivered to their laboratory in Sainte-Germaine-Boule.

Activation Laboratories Ltd. ("Actlabs"), which has ISO/IEC 17025 certification, was the primary lab for the 2007–2008 drilling. Each sample was crushed to the point where at least 90% passed a 2 mm sieve. After homogenization, an approximately 250 g split was crushed to 85% passing a 75 µm sieve. Cleaner sand was inserted between all samples. Samples were analyzed by 50 g FA-AA. For every sample with a result greater than 2 g/t Au, a second pulp from 250 g split was re-assayed by fire assay with a gravimetric finish. When visible gold was seen in a sample during logging, the lab was sometimes asked to perform a metallic-sieve analysis using more than 1,000 g of material.

ALS performed sample preparation and analyzed the drill samples from the 2009 through 2013 drilling programs (Camus et al., 2012; NioGold written communication, May 28, 2013). Each sample was crushed to the point where at least 90% passed a 2 mm sieve. After homogenization, an approximately 250 g split was crushed to 85% passing a 75 micron sieve. Samples were analyzed by FA-AA on 50 g pulp splits, with re-assaying of samples exceeding 2 g/t Au using fire assay with a gravimetric finish (Aurizon news releases, November 4, 2010; April 14, 2011; May 25, 2011; June 22, 2011; September 19, 2011; June 12, 2012). In addition, pulps assaying over 0.5 g/t Au were sent to Bourlamaque for check assaying.

The 2007–2013 core was cut and sampled at NioGold's core shack located at the Marban Engineering Project. Transport Manitoulin transported the samples from the project site to Actlabs; samples analyzed by Bourlamaque or ALS were transported directly to the labs by NioGold staff. Specific gravity measurements were obtained on selected samples of core from mineralized intervals.

11.3 O3 Mining Sample Preparation Analysis and Security

For the 2020 and 2021 drilling, gold assaying of the drill core from all zones was completed by AGAT Laboratories Ltd. in Mississauga, which is ISO/IEC17025:2017 certified and independent to O3 Mining. Each sample was crushed to the point where at least 75% passed a 2mm sieve. After homogenization, an approximately 250 g split was crushed to 85% passing a 75 µm sieve. Samples were analyzed by FA-AA on 50 g pulp splits, with re-assaying of samples exceeding 10 g/t Au using fire assay with a gravimetric finish. Samples containing visible gold were assayed systematically using the metallic-sieve assay method using a 1,000 g pulp, with triplicate in the fine fraction. Most of the samples were also analyzed for 43 elements with a 4-cid digestion and an ICP-OES finish. In addition, 10% of the samples in mineralized zones were randomly selected and a second 250 g pulp were pulverized and assayed for gold by AGAT.

The 2020–2021 core was cut and sampled at O3 Mining's core shack located in Val-d'Or. The half-core samples to be assayed were bagged, sealed and picked-up by AGAT personnel directly at O3 Mining's core shack.

11.4 NioGold Quality QA/QC Programs

According to Lefrançois (2006; 2007), both NioGold and the laboratories they used for assaying implemented QA/QC programs to monitor the precision, accuracy, and reproducibility of the analytical method and results. For the 2006 drilling,

certified reference material (CRM) were inserted every 20–25 samples in the sample batches sent to the laboratories, with a minimum of one CRM per batch. At the beginning of the project, three different CRMs were used, with two other CRMs added later to bring the total to five (Davy, 2008). No blanks or duplicates were used in this program. In addition, coarse-reject fractions were systematically re-assayed using 50 g FA-AA for samples that originally assayed from 2–5 g/t Au, and 50 g fire assay with a gravimetric finish for samples originally assaying over 5 g/t Au.

If the laboratory returned more than a 15% deviation for the inserted CRM samples, complete re-assaying of the sample batch was requested from the laboratory. This was required for some batches for the Gold Hawk holes assayed by Techni-Lab. About 12–15% of pulps were re-assayed at a second laboratory for duplicate assays.

Quality control practices were enhanced for the 2007–2008 and all subsequent drill programs. Originally, CRM, blank, and duplicate-core samples were inserted into the stream at every 10 to 15 samples. Starting in February 2008, CRM were inserted alternately at every 20th sample, with a minimum of one CRM per batch; all sample numbers ending in 20, 40, 60, 80, and 00 were a CRM. When assay results were received, the values obtained for every CRM were checked to see if they fit into the range of two standard deviations from the certified values, which was calculated with the information on the certificate of the CRM. If the assay results were not acceptable, the lab re-assayed the whole batch of samples in which the CRM was inserted. Blank samples were inserted among the samples of a suspected mineralized zone. The blanks consisted of sawed half-core samples from project core deemed to be unmineralized and, later, core from a limestone quarry. All results on blanks were checked and the lab consulted if there were problems. Duplicate-core samples were taken in suspected mineralized zones and consisted of one-half or one-quarter of the original core. The duplicate core was inserted later in the sequence of sample numbers. The practice of taking field duplicates was stopped in February and resumed with hole MB-08-076 in the first half of July 2008.

Before February 2008, the labs returned approximately 5% of the drill sample pulps chosen randomly by the labs to NioGold. For the remainder of 2008, Actlabs was instructed to return every pulp with an assay greater than 0.5 g/t Au. These pulps were then retagged; blanks and standards were inserted into the re-tagged numbering sequence, and the samples were sent to Bourlamaque for check assaying. While check assaying has remained a part of all subsequent drilling programs, protocols for choosing the sample pulps to be checked have varied. Results of the NioGold QA/QC programs are presented in Section 11.6.

11.5 O3 Mining QA/QC Programs

O3 Mining and AGAT Laboratories implemented QA/QC programs to monitor the precision, accuracy, and reproducibility of the analytical method and results. One CRM (Certified Reference Material) supplied by OREAS and one course gravel blank were inserted every 18 samples within the sample sequence. Samples were sent to the lab in 72-sample batches to fit the fire assay oven capacity of 84 at AGAT. The 12 remaining samples in the oven consisted of CRM, blanks, and duplicates inserted by AGAT into the sequence. Five different CRMs were used by O3 Mining at the same time the ensure that the variability of gold grades was covered. The selected CRMs were composed of either primary volcanic or intrusive rocks from greenstone belt with a similar matrix to rock from the project. On a daily basis, results from the CRM and blanks assays were validated.

Gold re-assays were performed on 10 samples below and above the problematic CRM when its assays result varied more than two standard deviations from the certified value in mineralized zone, or three standard deviations outside. If within the same batch, two or more CRMs had problematic results, the entire batch was re-assayed. Quarter-split drill core was taken and assayed when a sample inversion or contamination was suspected. In addition, 10% of samples within mineralized zones were randomly selected and their pulp were re-assayed at ALS Laboratories in Val-d'Or.

11.6 NioGold/Osisko Era QA/QC Data Analyses

Drilling commenced in both the Marban area and the Kierens-Norlartic area in 2006 under NioGold Mining Corporation and continued in 2016 and 2017 under Osisko Mining Inc. All data from 2006 through 2017 is considered to be modern era and

consistent with contemporary QA/QC practices in terms of blanks, CRMs, and duplicates, blindly inserted with respect to the analyzing laboratories. All CRMs were obtained from Rocklabs Ltd. of New Zealand.

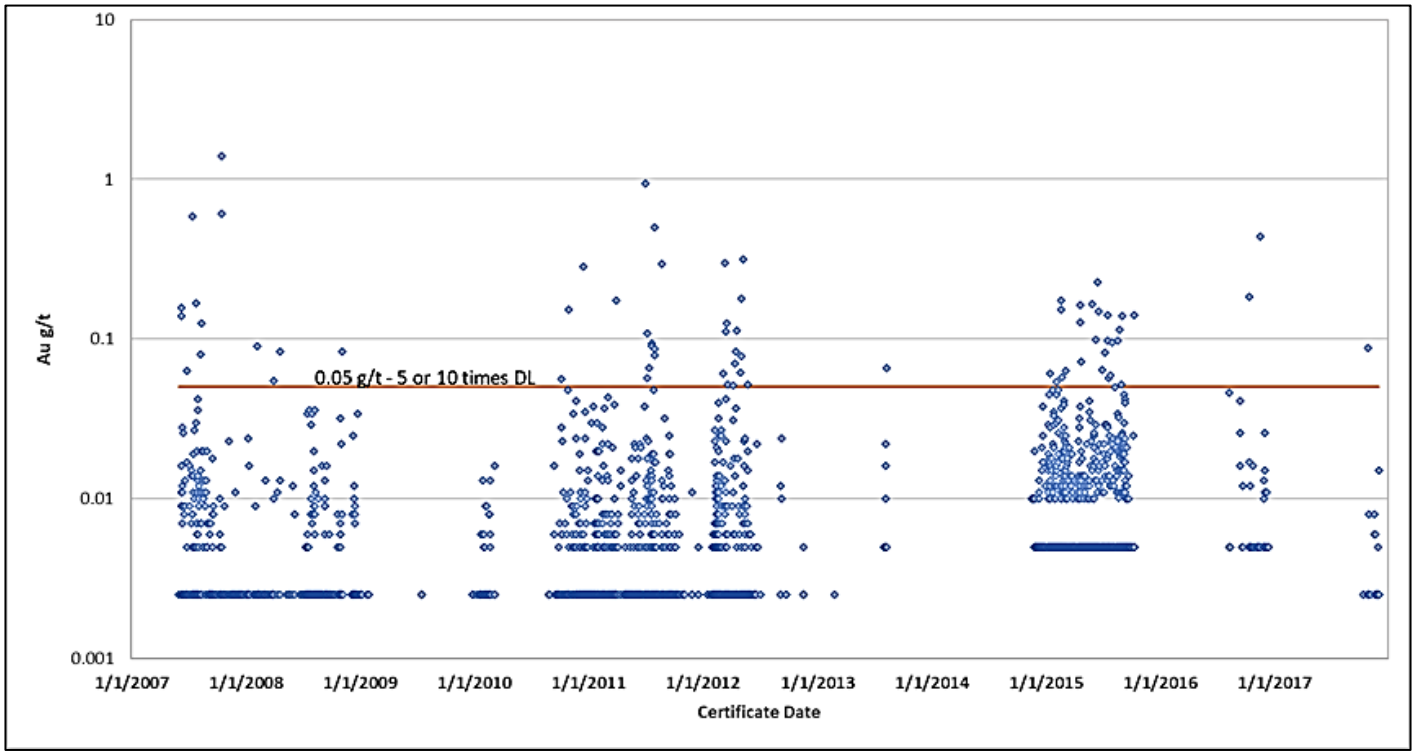
11.6.1 Marban Blanks

The QA/QC sample database includes 3554 blindly inserted samples of material believed to contain little or no gold which are reported both as sawed core samples from a limestone quarry (NioGold Activity Report, 2009) and store-bought gravel of sedimentary composition (2017 Activity Report). Internal and previous reports indicate that failed blank samples are investigated within the contracted lab and that re-assays may have been requested on occasion; however, the QA/QC database contains numerous failures and does not show a record of re-assay and replacement.

Figure 11-1 shows the assay values of the plotted blanks in the order of the certificate date. Even at a generous failure threshold of 0.05 g/t (5 or 10 times the detection limit), 73 samples, or 2.1%, are shown to fail. Of the six fails with the highest value, only three follow assays of high value, indicating minor contamination at the laboratory may not be the only reason for the high failure rate; it may be that the blank material, not being a certified blank material, occasionally contains some gold.

The relatively high failure rate of the blanks is not believed to materially affect the resource model at this stage, but future reviewers may require re-assays and that amendments be made to the assay database.

Figure 11-1: Marban Blanks 2007–2017



Source: MMTS, 2020.

11.6.2 Marban CRMs

The database provided of assays from 8558 blindly inserted CRMs indicates the targeted rate of 1 per group of 20 assays was met. Analysis results of CRMs with more than 20 insertions, in order of increasing grade is presented in Table 11-1 below. It should noted that the lowest grade CRM is 0.583 g/t, and normally CRMs with lower grade would be included. It is observed that the % error, coefficient of variation, and % of failed samples (at the +- 3 SD level) is greater than expected for a significant number of CRMs. What makes this less of a concern, for the purposes of resource reporting, is that the trend of the error is mostly negative, meaning that the assays of the CRMs are mostly below the expected value, indicating that the error is in the conservative direction. In fact, the overall error is a negative 0.8%.

The 2013 Technical Report (Gustin 2013) indicates NioGold considered CRM assays acceptable when they fell within four standard deviations of the expected values. Additionally, it is reported that failed samples and neighboring samples in the sample stream were submitted for re-assay primarily only when significant gold values were encountered in the sample sequence. MDA reports reviewing the records of re-assayed selection. (Gustin 2012) Spot checking of six significant fails found only one re-assay. Confidence in the data improves when there are fewer decision points surrounding re-assays and the record of CRMs is shown to have fewer fails.

The statistics describing the accuracy of the assays are plotted in Figure 11-2 in the order of the year that use of CRMs began, and by grade in Figure 11-3. It is observed that the greatest deviations occurred in 2006 and 2007, and that accuracy improves over time. It is also seen that two of the highest grade CRMs (18.104 g/t and 8.543 g/t) had some of the largest inconsistencies, which is not of great significance, as only 0.3% of assay data is greater than 7.0 g/t.

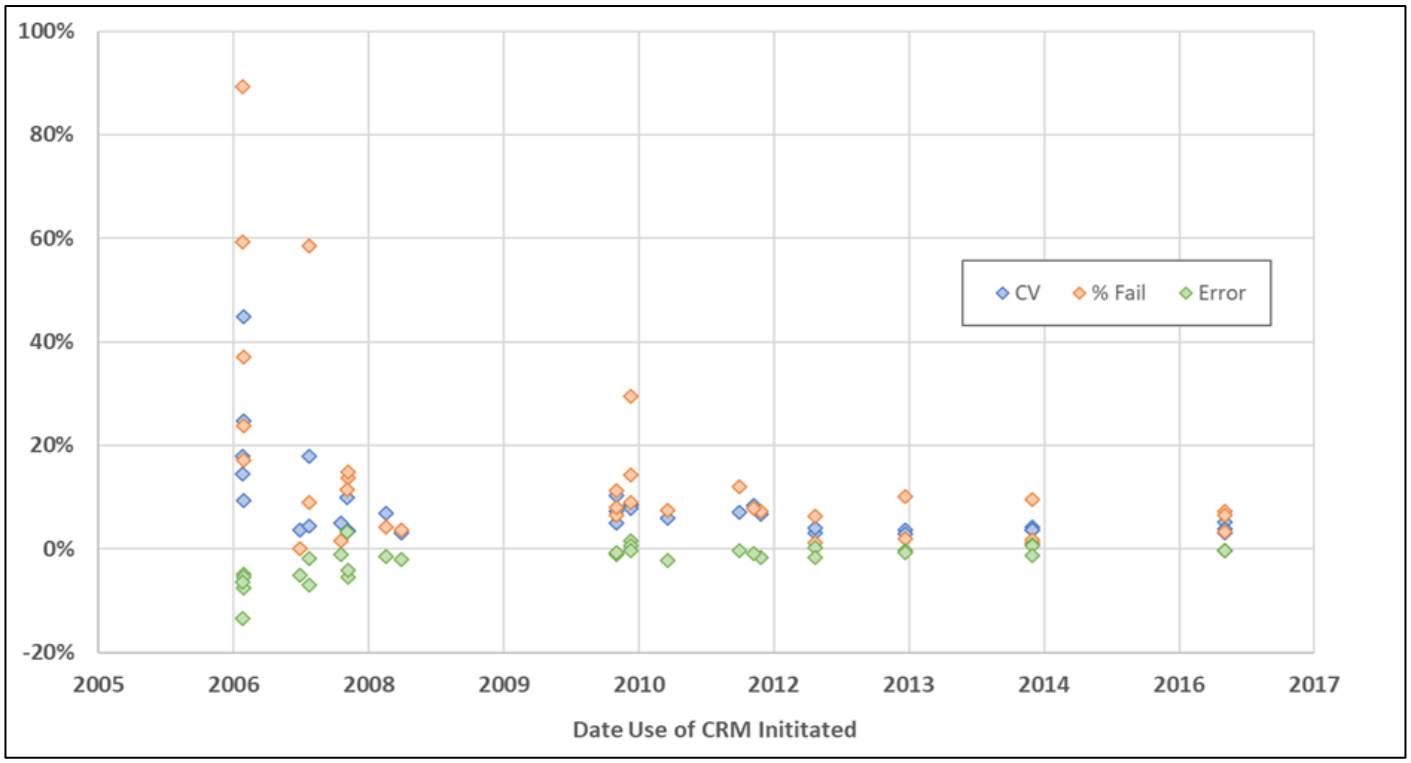
Table 11-1: Marban CRM Results

CRM	Total Insertions	Au Expected Value (g/t)	Au Average Assay (g/t)	SD Au Assay (g/t)	Error (%)	CV (%)	Total Fails (+- 3 SD)	PercentFail (%)	Used From	Used To
1001/SE19	34	0.583	0.555	0.021	-5.0	3.6	0	0.0	2007	2007
1038/SE86	55	0.595	0.593	0.031	-0.3	5.2	4	7.3	2016	2017
1014/SE29	29	0.597	0.566	0.021	-5.4	3.5	4	13.8	2007	2008
1034/SE68	734	0.599	0.605	0.026	0.9	4.3	70	9.5	2014	2017
1023/SE44	177	0.606	0.615	0.052	1.5	8.6	16	9.0	2010	2011
1025/SE58	439	0.607	0.594	0.036	-2.2	6.0%	33	7.5	2011	2012
1015/SF30	78	0.832	0.860	0.083	3.2	10.0	9	11.5	2007	2008
1020/SF45	46	0.848	0.840	0.062	-0.9	7.3	3	6.5	2010	2010
1028/SF57	566	0.848	0.834	0.058	-1.7	6.8	41	7.2	2012	2013
1010/1016/ SG31	291	0.996	0.977	0.044	-1.9	4.5%	26	8.9	2007	2011
1036/SG84	61	1.026	1.024	0.040	-0.2	3.9	2	3.3	2016	2017
1035/SG66	765	1.086	1.093	0.040	0.6	3.7	9	1.2	2014	2017
1011/SH35	183	1.323	1.310	0.065	-1.0	4.9	3	1.6	2007	2011
1006/SH24	29	1.326	1.265	0.594	-4.8	44.8	5	17.2	2006	2007
1037/SH82	61	1.333	1.329	0.042	-0.3	3.1	4	6.6	2016	2017
1019/SH41	486	1.344	1.329	0.139	-1.2	10.3	55	11.3	2010	2011
1032/SH69	829	1.346	1.341	0.049	-0.4	3.6	84	10.1	2013	2016
1030/SH65	32	1.348	1.353	0.042	0.4	3.1	2	6.3	2012	2013
1026/SH55	383	1.375	1.370	0.097	-0.4	7.1	46	12.0	2011	2012
1005/SI25	59	1.801	1.589	0.262	-13.3	14.5	35	59.3	2006	2007

CRM	Total Insertions	Au Expected Value (g/t)	Au Average Assay (g/t)	SD Au Assay (g/t)	Error (%)	CV (%)	Total Fails (+- 3 SD)	PercentFail (%)	Used From	Used To
1031/SJ63	49	2.632	2.616	0.077	-0.6	2.9	1	2.0	2013	2014
1024/SJ53	635	2.637	2.653	0.221	0.6	8.4	187	29.4	2010	2012
1017/SJ39	168	2.641	2.602	0.180	-1.5	6.8	7	4.2	2008	2011
1007/SJ32	35	2.645	2.460	0.248	-7.5	9.4	13	37.1	2006	2007
1008/1013/ SK33	59	4.041	3.831	0.997	-5.5	24.7	14	23.7	2006	2008
1027/SK62	445	4.075	4.037	0.340	-0.9	8.3	35	7.9	2012	2013
1018/SK43	214	4.086	4.003	0.126	-2.1	3.1	8	3.7	2008	2011
1022/SK52	548	4.107	4.094	0.322	-0.3	7.8	78	14.2	2010	2011
1021/SL46	62	5.876	5.841	0.293	-0.6	5.0	5	8.1	2010	2010
1012/SL34	27	5.893	5.660	0.208	-4.1	3.5	4	14.8	2007	2008
1033/SL76	830	5.96	5.886	0.219	-1.3	3.7	14	1.7	2014	2017
1004/SN26	37	8.543	8.025	1.524	-6.5	17.8	33	89.2	2006	2007
1029/SN60	77	8.595	8.462	0.344	-1.6	4.0	1	1.3	2012	2015
1009/SP27	29	18.104	16.930	3.231	-6.9	17.8	17	58.6	2007	2008

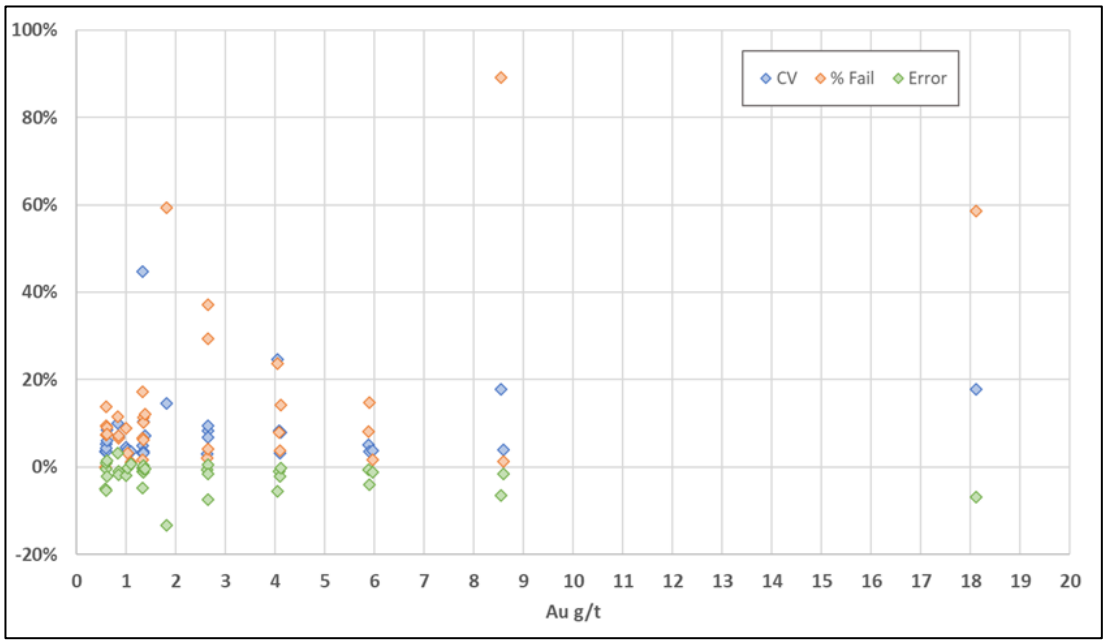
Note: SD = Standard deviation; CV = coefficient of variation.

Figure 11-2: Marban CRM Statistics by Year



Source: MMTS, 2020.

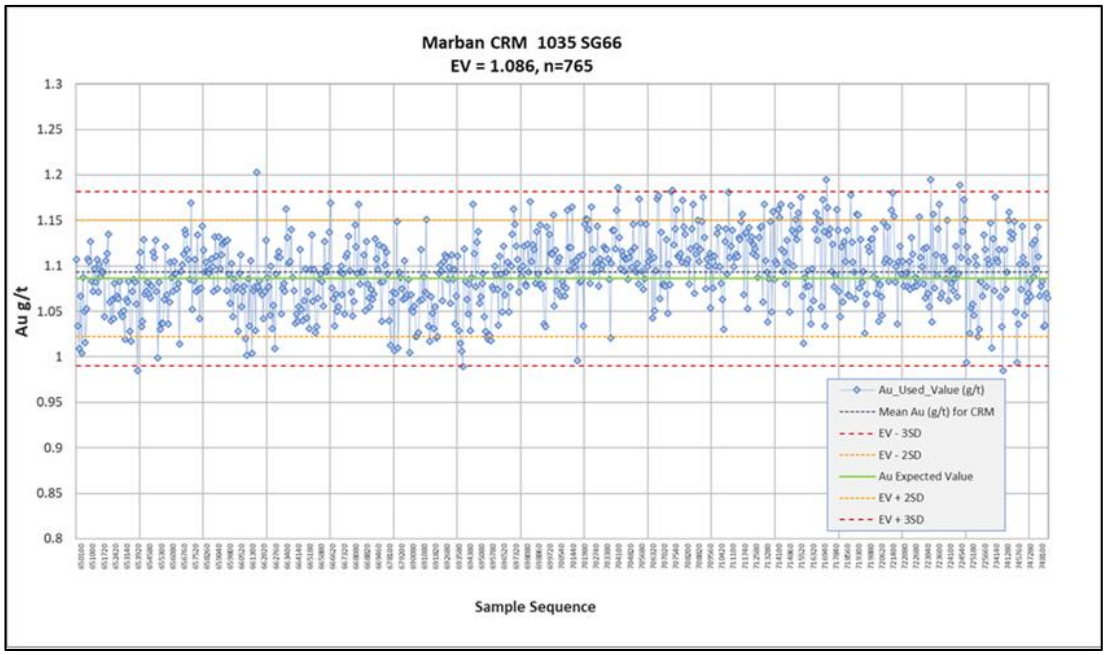
Figure 11-3: Marban CRM Statistics by Grade



Source: MMTS, 2020.

An example of a process control chart is presented in Figure 11-4 for CRM 1035/SG66 with an expected grade of 1.086 g/t, used between 2014 and 2017. This standard performed relatively well over 765 insertions with only 9 fails.

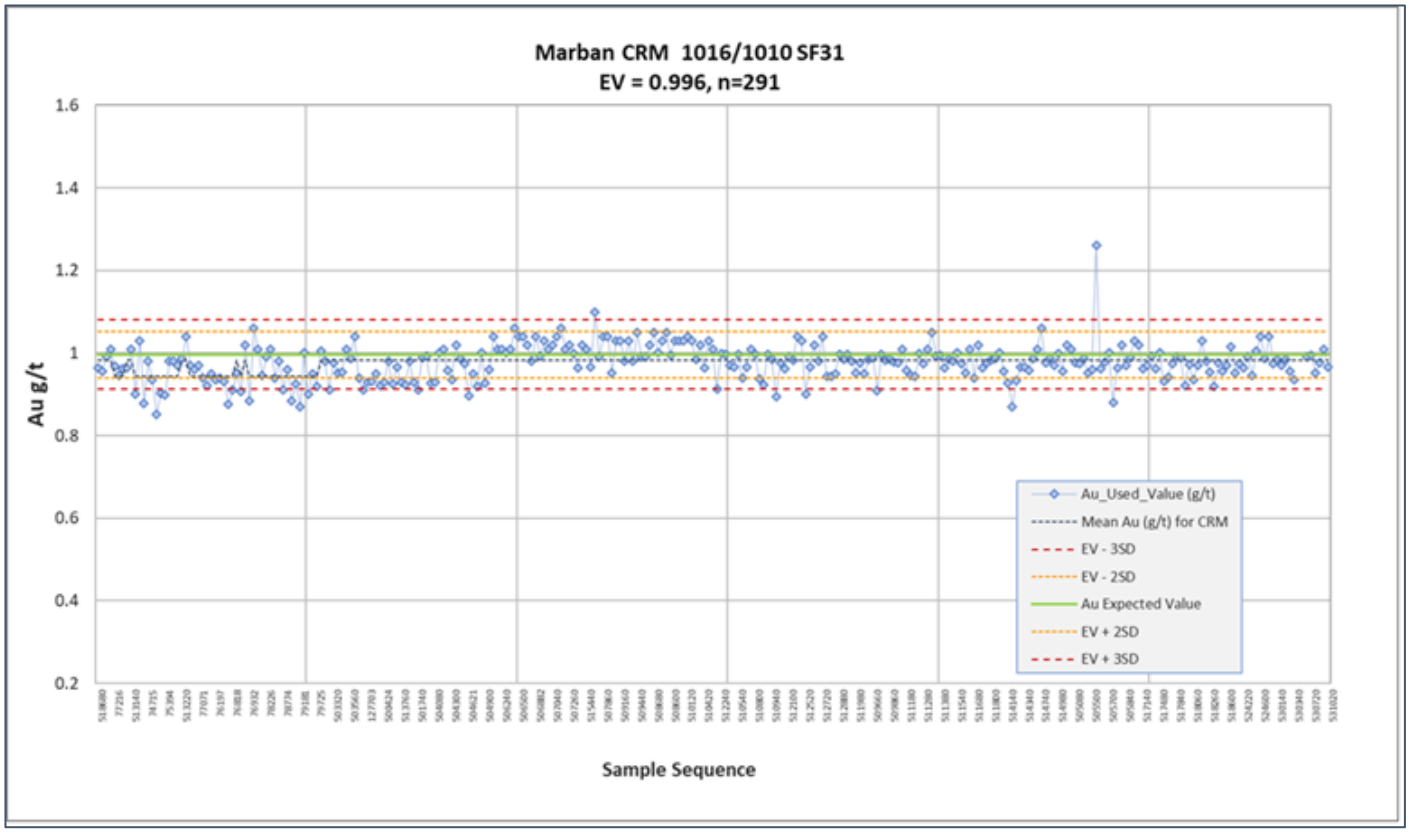
Figure 11-4: Marban Process Control Chart for CRM 1035/SG66, EV = 1.086 g/t



Source: MMTS, 2020.

Another example is given in Figure 11-5 for CRM 1016/1010/SF31, used from 2007 to 2011. This CRM has almost 9% fail at the +- 3 SD level. It is possible some are mislabelled as the range of the failed sample values does overlap other standards in use at the same time.

Figure 11-5: Marban Process Control Chart for CRM 1016/1010/SF31, EV = 0.966 g/t



Source: MMTS, 2020.

11.6.3 Marban Duplicates

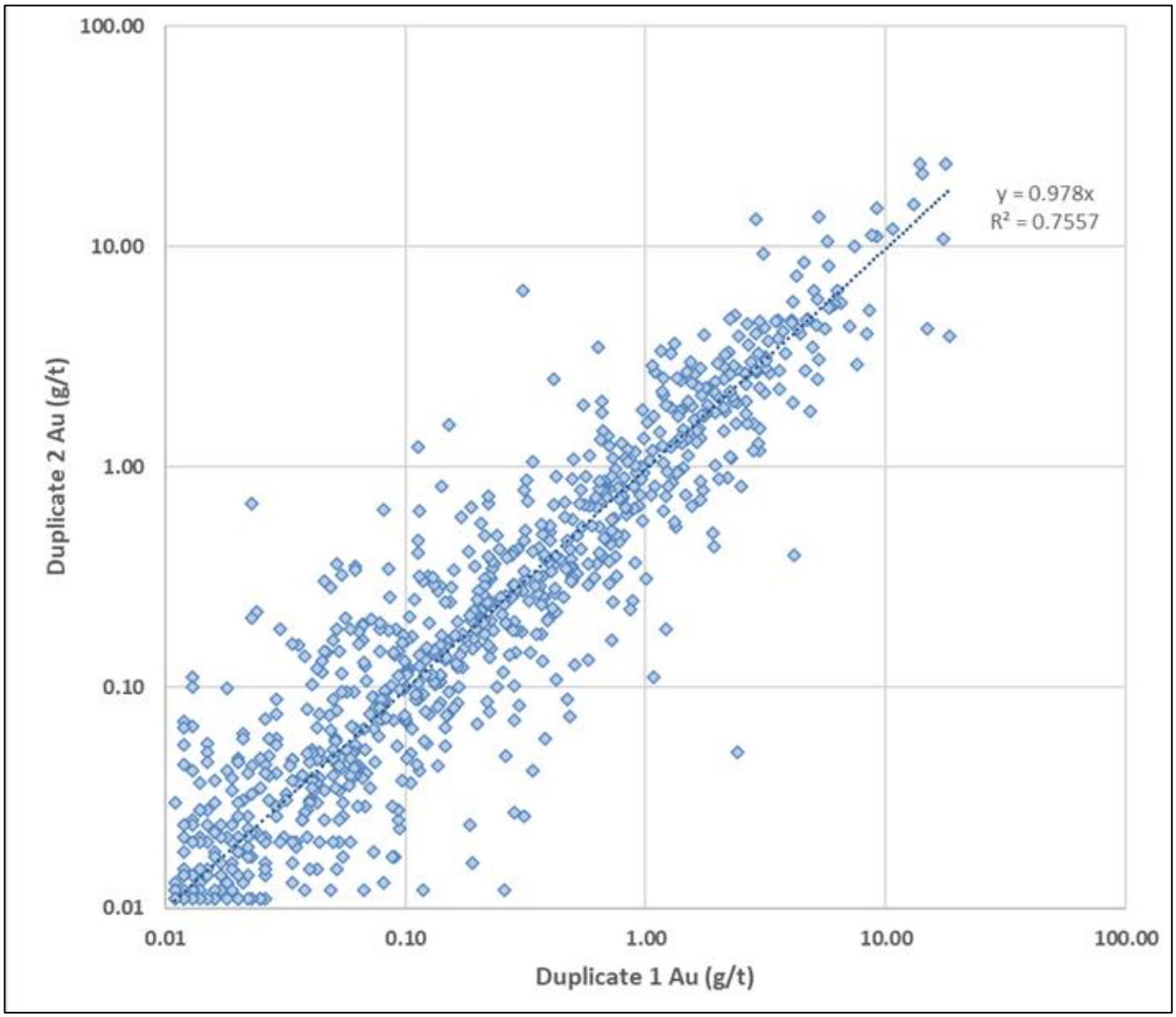
Duplicate assays of the Marban samples are available as field duplicates and pulp duplicates by external and internal laboratories. Analysis of the field duplicates shows strong heterogeneity in samples of split core typical of "nuggety" gold deposits. Analysis of external duplicates gives little indication of bias at a level that would affect the resource estimate. Analysis of internal duplicates shows good repeatability at the primary laboratories.

11.6.3.1 Marban Field Duplicates

Field duplicates are reported to have been half core splits before 2009 and quarter core splits in the following years (Gustin, 2013). For this purpose, all splits are considered together, despite having potentially different variability. The Marban database includes 1271 field duplicate samples collected between the years 2006 to 2017.

Figure 11-6 shows a scatter plot of gold grades from the paired duplicates. The plot excludes samples below 0.01 g/t and four outliers of high grade with large differences. A best fit line for the remaining 798 pairs shows reasonable trend along the 1:1 line.

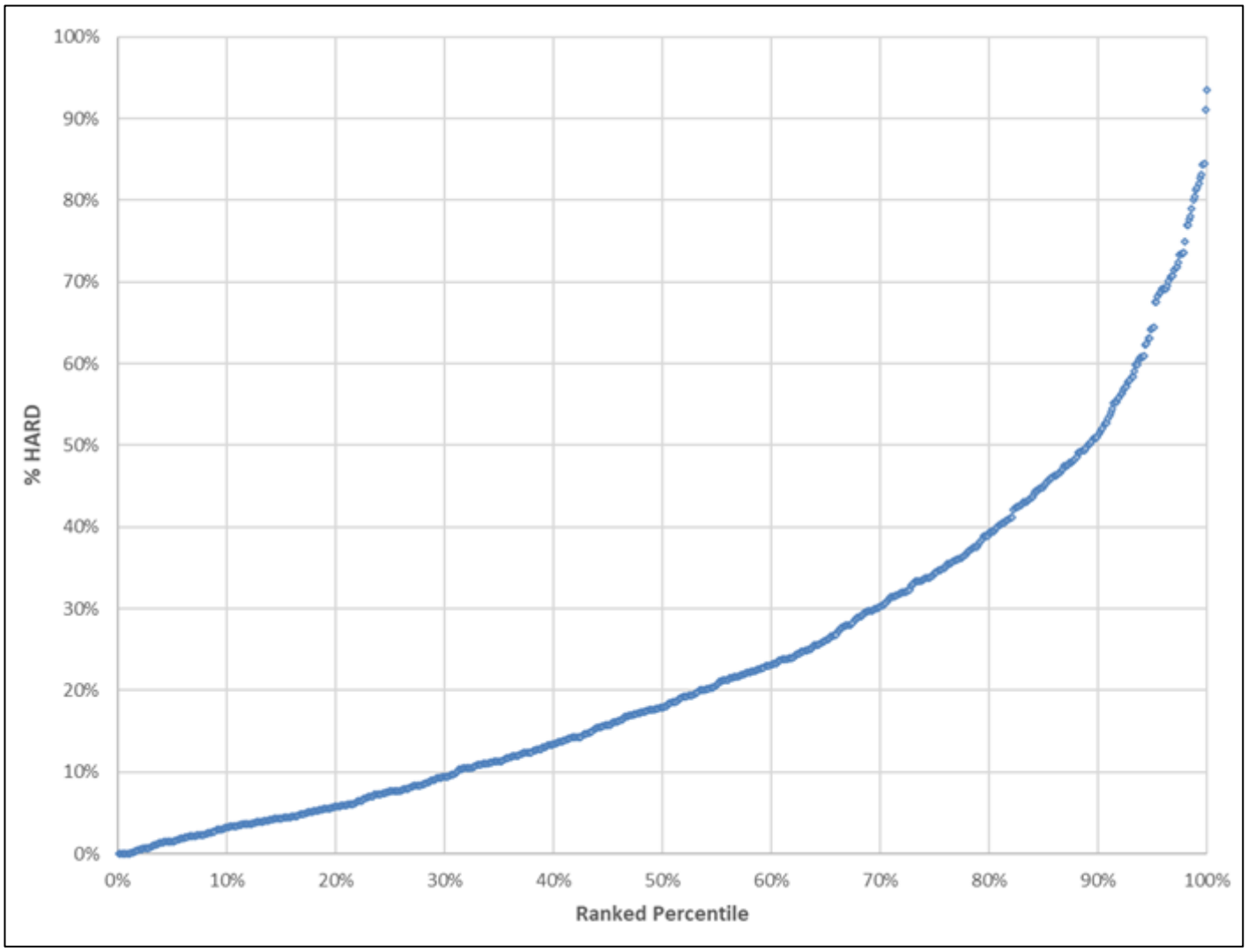
Figure 11-6: Marban Field Duplicates 2006–2017



Source: MMTS, 2020.

Figure 11-7 shows the ranked half absolute relative difference (HARD) plot which shows approximately 70% with greater than 10% HARD. Field duplicates for disseminated deposits not typical of gold would show approximately 30% greater than 10%. This implies that the nature of gold mineralization is highly variable, or "nuggety".

Figure 11-7: Marban Field Duplicates Ranked HARD



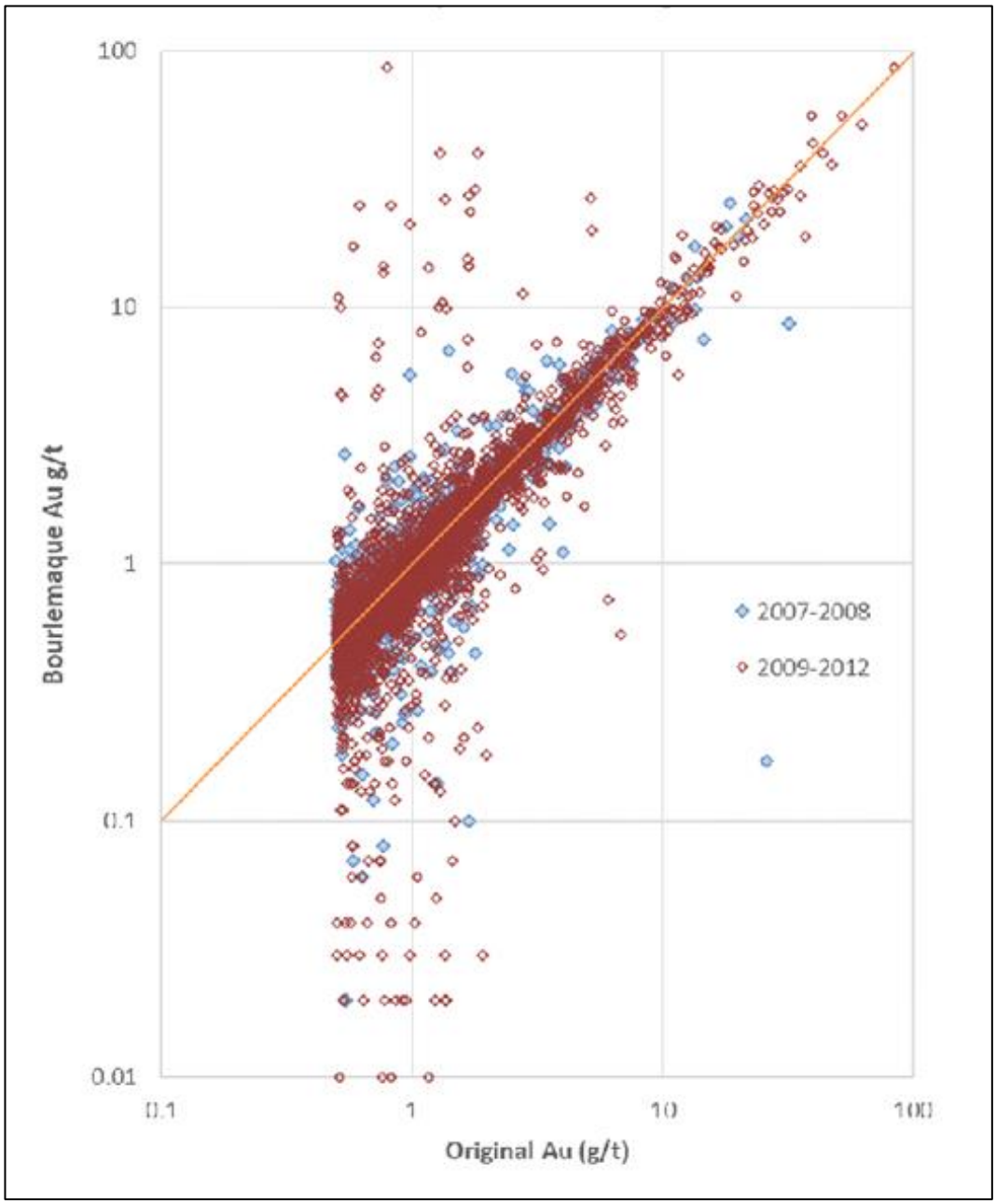
Source: MMTS, 2020.

11.6.3.2 Marban External Duplicates

Bourlamaque was selected to perform additional checks on Actlabs pulps from 2007 and 2008 and ALS was selected to perform additional check on pulps from 2009 through 2012 that had gold grades above 0.5 g/t. An additional 573 samples below 0.5 g/t were sent for a total of 4,543 pairs available in the database. Only 11 of the below threshold set are from 2009 to 2012. Because the dataset is not a natural distribution of samples, as it has been truncated in the original set at the 0.5 g/t level, it is not appropriate to analyze this data as a single set.

The pairs with original assay at 0.5 g/t and up are shown in Figure 11-8. The data shows concentration along the 1:1 line with not unexpected scatter for gold deposits. The mean of the original assays at 1.98 g/t is slightly lower than the mean of the re-assays at 2.04 g/t, indicating an error of almost three percent. The median of both sets is 1.03 g/t. For the purposes of resource reporting, this error is not considered material as it is relatively small and is in the conservative direction.

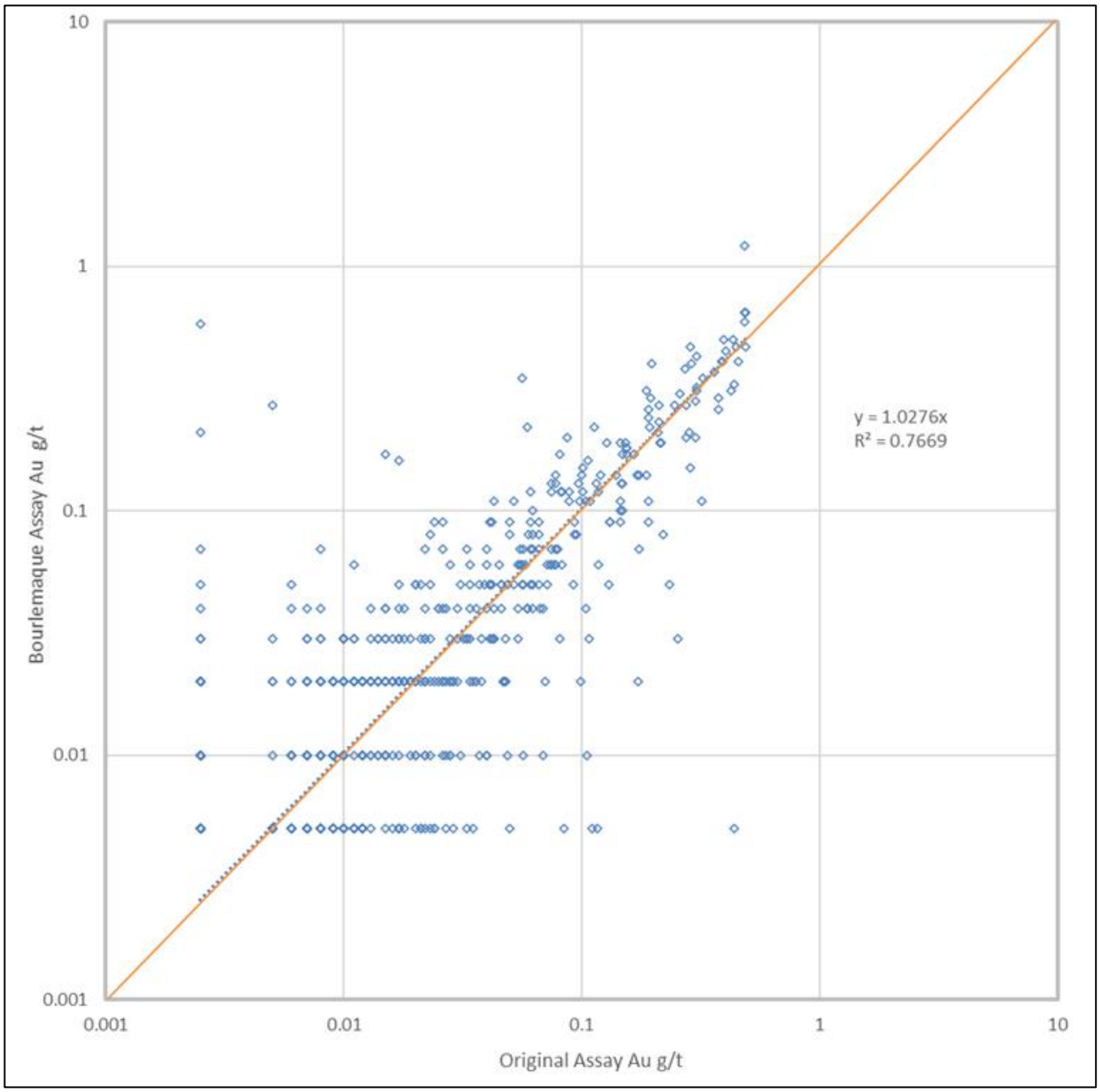
Figure 11-8: Marban External Duplicates ≥0.5 g/t, 2007–2012



Source: MMTS, 2020.

Figure 11-9 shows the external duplicates from 2007–2009 below the 0.5 g/t threshold. Reasonable concentration is shown along the 1:1 line above 0.1 g/t, and the best fit line shows a small slope above 1. The data below 0.1 g/t highlights the granularity of the Bourlemaque data, where the detection limit was either 0.01 g/t or 0.02 g/t compared to the original detection limit of 0.05 g/t. Of the data with mean assay >0.02 g/t, the average of the original assay is 0.11 g/t and the Bourlemaque average is 0.12 g/t, confirming again that the Bourlemaque re-assay data is slightly higher.

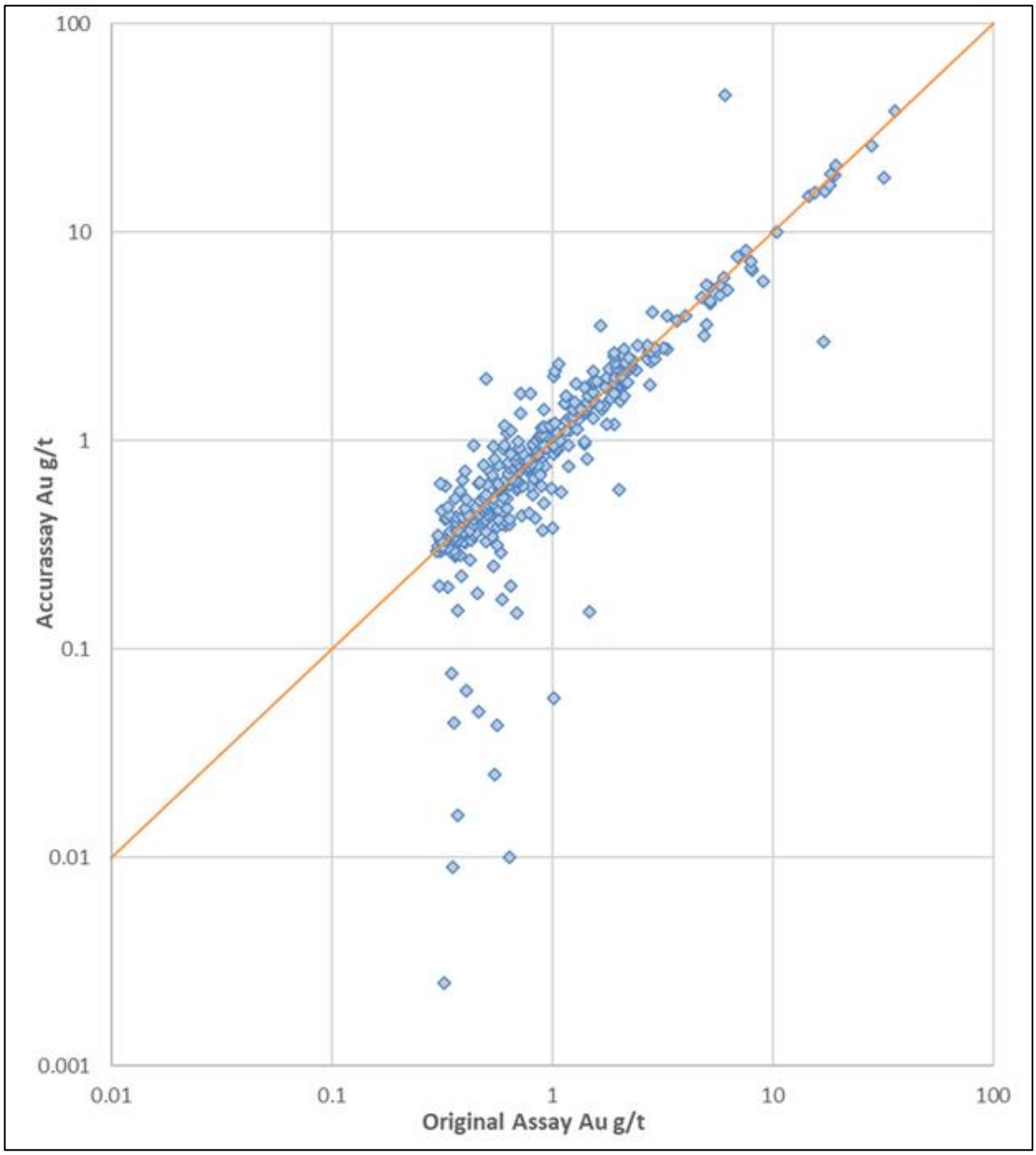
Figure 11-9: Marban External Duplicates <0.5 g/t, 2007–2009



Source: MMTS, 2020.

A set of 345 pulps with greater than 0.3 g/t assay at the primary laboratory was sent to Accurassay in 2016 for checks. Figure 11-10 shows a scatter plot of these paired assay results. The data is seen to cluster along the 1:1 and has some outliers. Analysis of the average of the pairs gives 1.94 g/t for the original and 1.95 g/t for the re-assay, the medians are 0.79 g/t and 0.80 g/t.

Figure 11-10: Marban External Duplicates, 2014–2015 ≥ 0.3 g/t



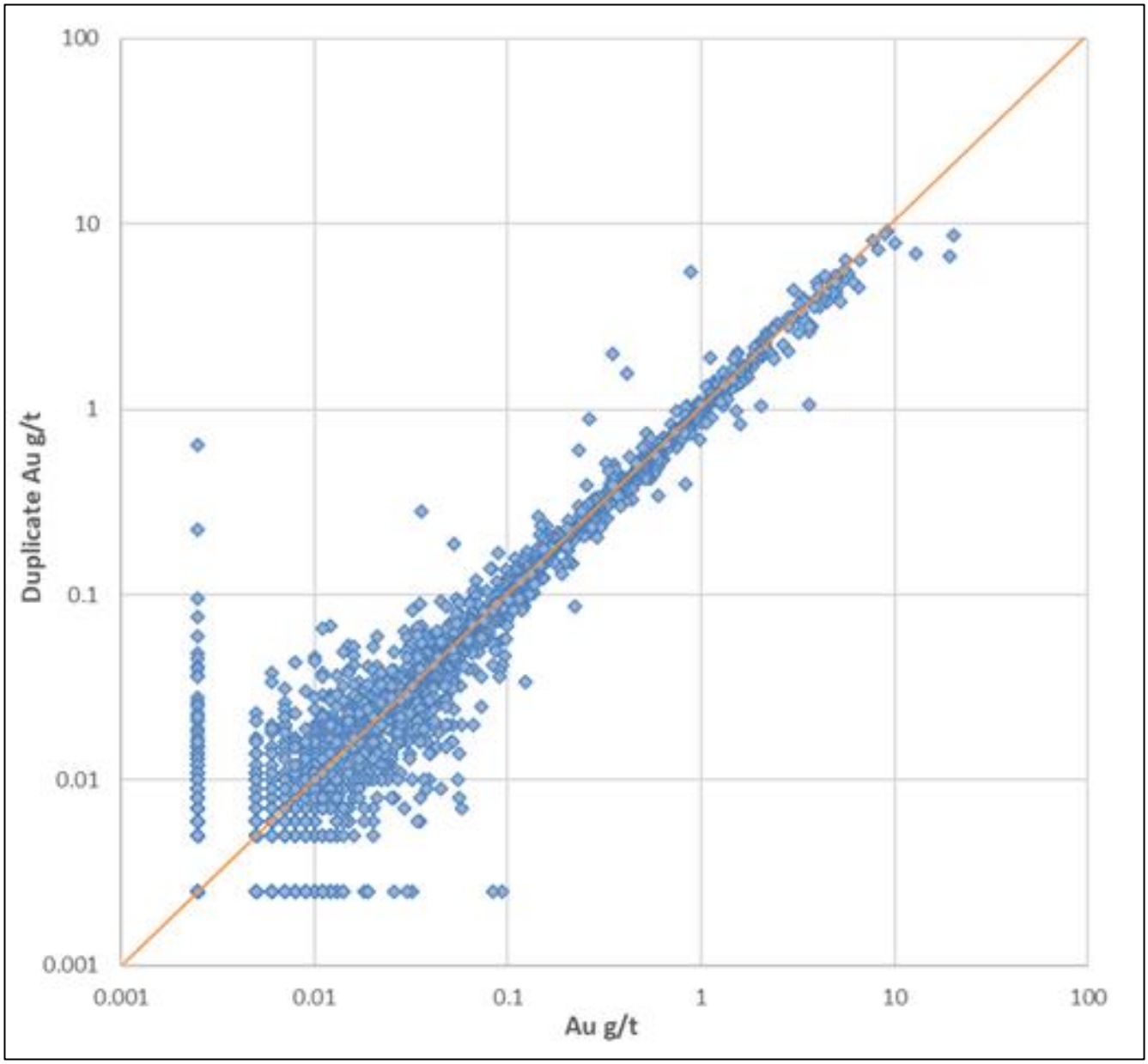
Source: MMTS, 2020.

Analysis of these external duplicates provides confidence in the assay database and gives no indication that the primary laboratory is biased in a manner that would affect the resource estimate.

11.6.3.3 Marban Internal Laboratory Duplicates

Internal duplicates at the primary laboratories (ActLabs and ALS) from 2006–2012 are presented in Figure 11-11 below. The data shows good correlation along the 1:1 line indicating the laboratories have acceptable repeatability.

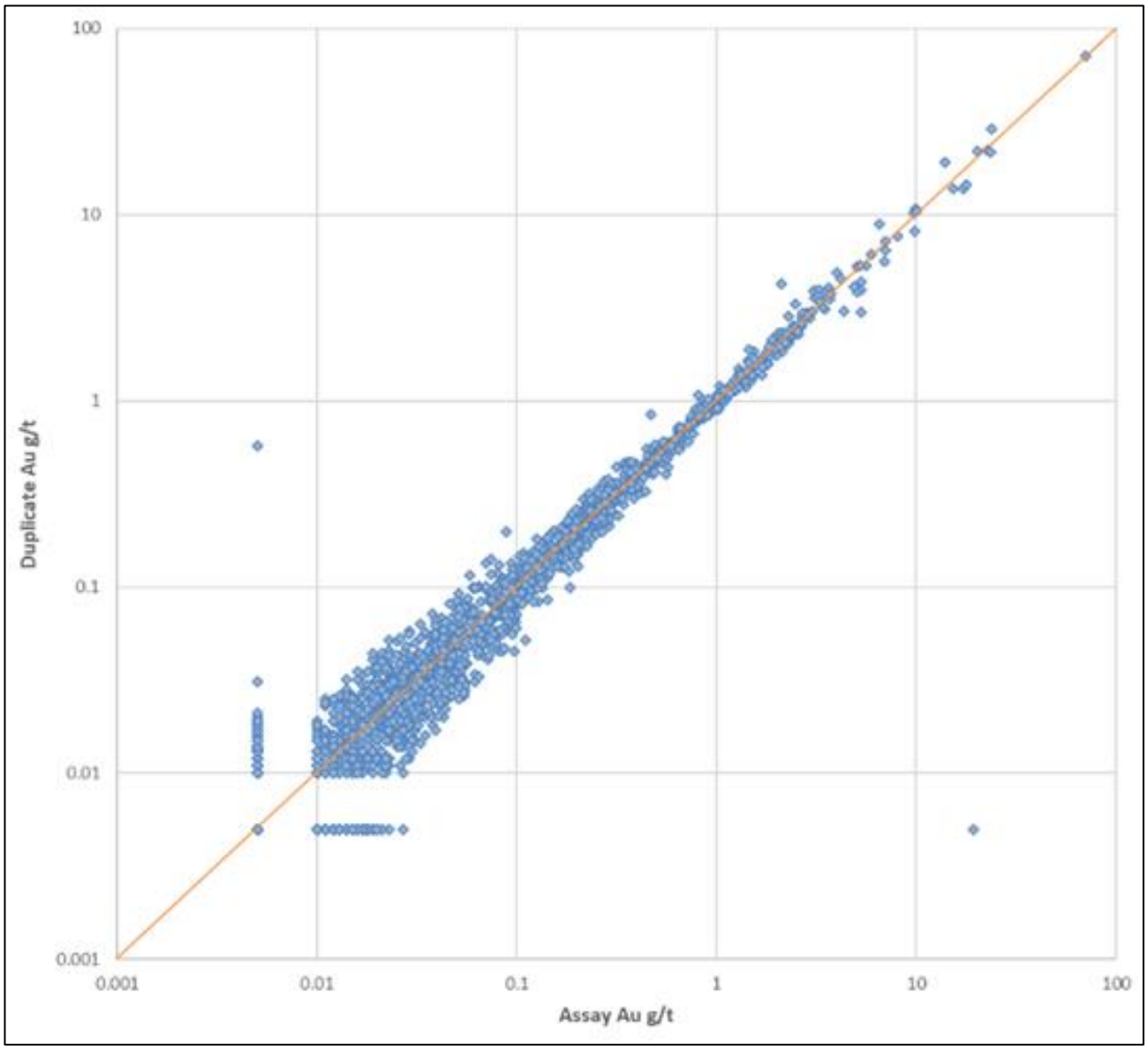
Figure 11-11: Marban Internal Duplicates, 2006–2012



Source: MMTS, 2020.

Figure 11-12 shows internal laboratory duplicates performed at TechniLab during 2014–2015. The data clusters along the 1:1 line with a few outliers, indicating the laboratory itself has acceptable repeatability.

Figure 11-12: Marban Internal Duplicates, 2014–2015

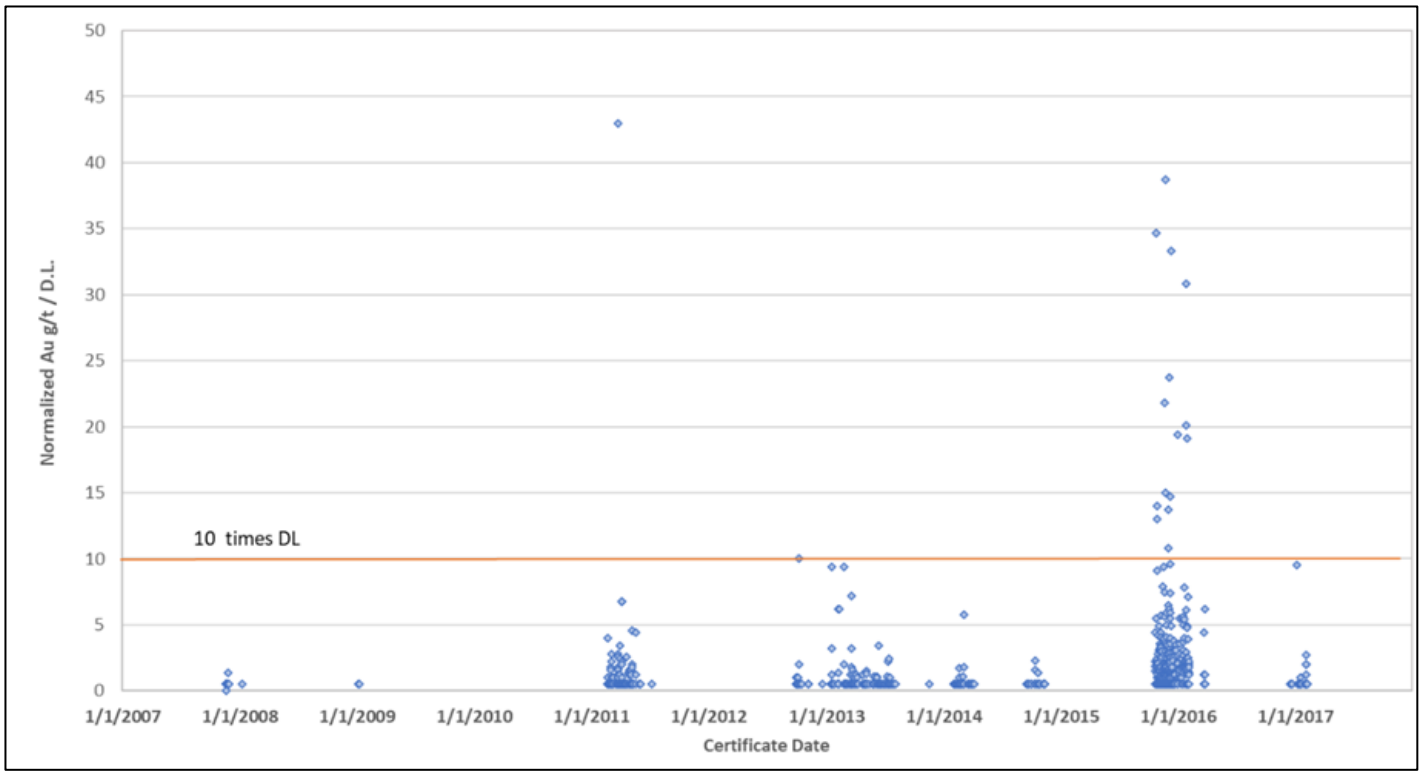


Source: MMTS, 2020.

11.6.4 Kierens-Norlartic Blanks

The QA/QC database includes 709 assays of blank samples assayed between 2007 and 2017. In this data, three different detection limits are reported between the assayers, 0.005 g/t, 0.01 g/t and 0.02 g/t. For this reason, the data is normalized by the appropriate detection limit and presented in Figure 11-13 against a criterion of 10 times the detection limit (DL). The percent of samples failing these criteria is 2.3%, while 6.8% fail against a more stringent criteria of 5 times the DL. This is a higher rate of failure than would normally be considered acceptable. The eight highest fails are all found to follow samples of high assay value, indicative of a problem with contamination at the laboratories. No re-assays were discovered of these most problematic samples. These fails are concentrated around a period in 2016.

Figure 11-13: Kierens-Norlartic Blanks, 2007–2017



Source: MMTS, 2020.

11.6.5 Kierens-Norlartic CRMs

The assay database includes 2283 blind insertions of CRMs in the Kierens-Norlartic zone and indicates the targeted rate of 1 in 20 assays was met. Analysis of CRMs with more than 20 insertions, in order of increasing grade is presented in Table 11-2. It is observed that the % of failed samples at the +- 3 SD level is greater than normally expected for many samples, although the coefficient of variation (CV) value and error is only concerning for two CRMs; 1005/SI25 and 1000/SL20 both used in 2006 with a low number of insertions. The reasonable error and acceptable CV values indicate some consistency despite the high number of fails in the remaining CRMs. There are no significant trends by year or grade to discuss.

It is possible that in some cases the samples have been mislabelled, increasing fail rates. The overall error indicates no directional bias. In general, the performance of the standards is acceptable at this level of resource reporting.

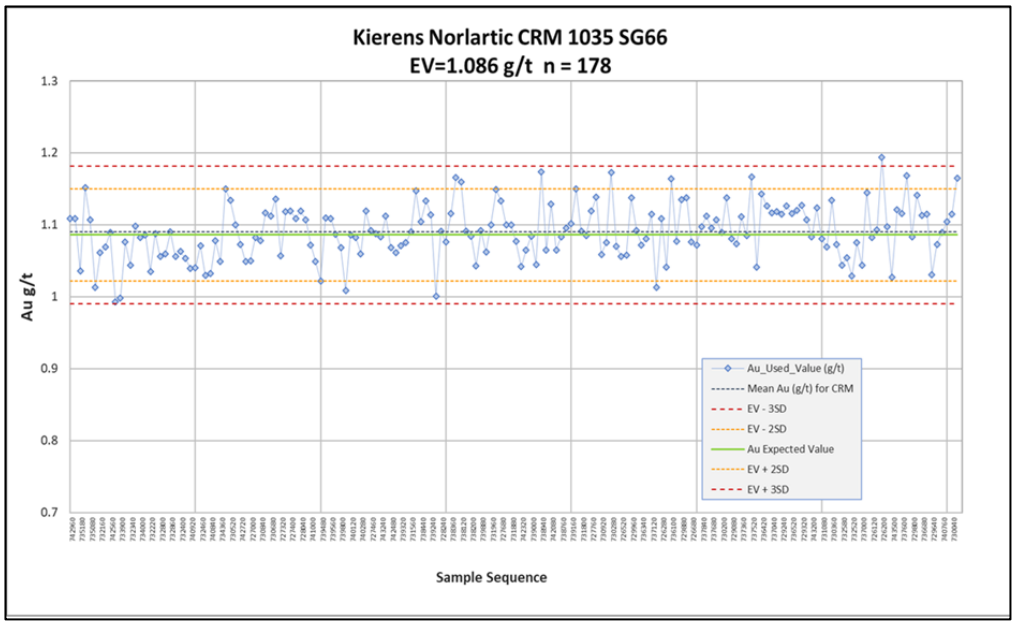
Table 11-2: Kierens-Norlartic CRM Results

CRM	Total Insertions	Au Expected Value (g/t)	Au Average Assay (g/t)	SD of Au Assay (g/t)	Error (%)	CV (%)	Total fail (+- 3 SD)	Percent Fail (%)	Used From	Used To
1001/SE19	21	0.583	0.582	0.030	-0.2	5.1	0	0.0	2006	2007
1038/SE86	30	0.595	0.598	0.018	0.4	3.0	0	0.0	2016	2017
1034/SE68	189	0.599	0.607	0.021	1.4	3.5	18	9.5	2015	2017
1025/SE58	62	0.607	0.596	0.034	-1.8	5.7	5	8.1	2011	2011
1003/SF23	32	0.831	0.834	0.022	0.4	2.7	1	3.1	2006	2006
1028/SF57	21	0.848	0.816	0.037	-4.0	4.5	2	9.5	2013	2014
1036/SG84	30	1.026	1.023	0.026	-0.3	2.5	1	3.3	2016	2017
1035/SG66	178	1.086	1.090	0.038	0.4	3.5	1	0.6	2015	2016
1037/SH82	28	1.333	1.326	0.037	-0.5	2.8	0	0.0	2016	2017
1032/SH69	299	1.346	1.326	0.047	-1.5	3.6	35	11.7	2013	2016
1030/SH65	221	1.348	1.348	0.060	0.0	4.4	26	11.8	2012	2013
1005/SI25	25	1.801	1.694	0.375	-6.3	22.1	4	16.0	2006	2007
1031/SJ63	126	2.632	2.573	0.107	-2.3	4.1	20	15.9	2013	2014
1024/SJ53	74	2.637	2.639	0.155	0.1	5.9	19	25.7	2011	2011
1027/SK62	155	4.075	4.041	0.190	-0.8	4.7	8	5.2	2012	2013
1022/SK52	68	4.107	4.160	0.155	1.3	3.7	6	8.8	2011	2011
1000/SL20	27	5.911	6.174	0.984	4.3	15.9	3	11.1	2006	2006
1033/SL76	231	5.96	5.860	0.216	-1.7	3.7	4	1.7	2014	2017
1004/SN26	34	8.543	8.430	0.345	-1.3	4.1	5	14.7	2006	2007
1029/SN60	352	8.595	8.469	0.314	-1.5	3.7	11	3.1	2012	2015

Note: SD = Standard deviation; CV = coefficient of variation

An example of a process control chart is given for CRM 1035/SG66 in Figure 11-14. It was used in assays in 2015 and 2016 and it is shown that the mean of assays is slightly above the expected value and has only one failure at the +- 3 SD level. There are also some observable trends over time which can result from calibration issues at the lab.

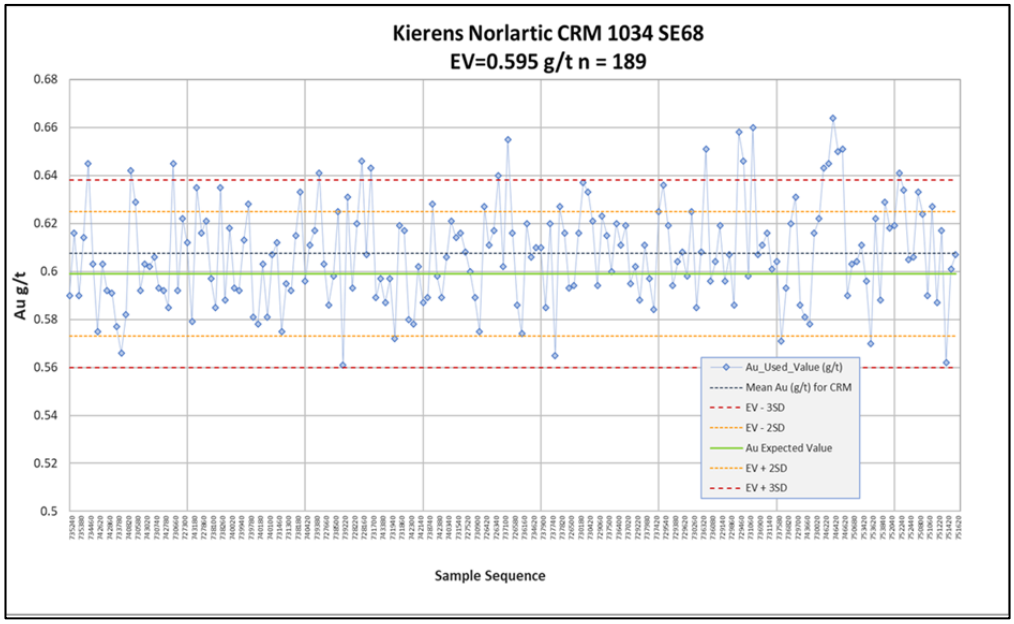
Figure 11-14: Kierens-Norlartic Process Control Chart for CRM 1035/SG66, EV = 1.086 g/t



Source: MMTS, 2020.

Another example of a process control chart is given in Figure 11-15 for CRM 1034/SE68. Here it is observed that the mean of samples is above the expected value and there are multiple fails at the +- 3 SD level. Normally, few fails would be seen at this level, as the samples and surrounding assays samples would have been re-assayed and replaced in the database.

Figure 11-15: Kierens-Norlartic Process Control Chart for CRM 1034/SE68, EV = 0.595 g/t



Source: MMTS, 2020.

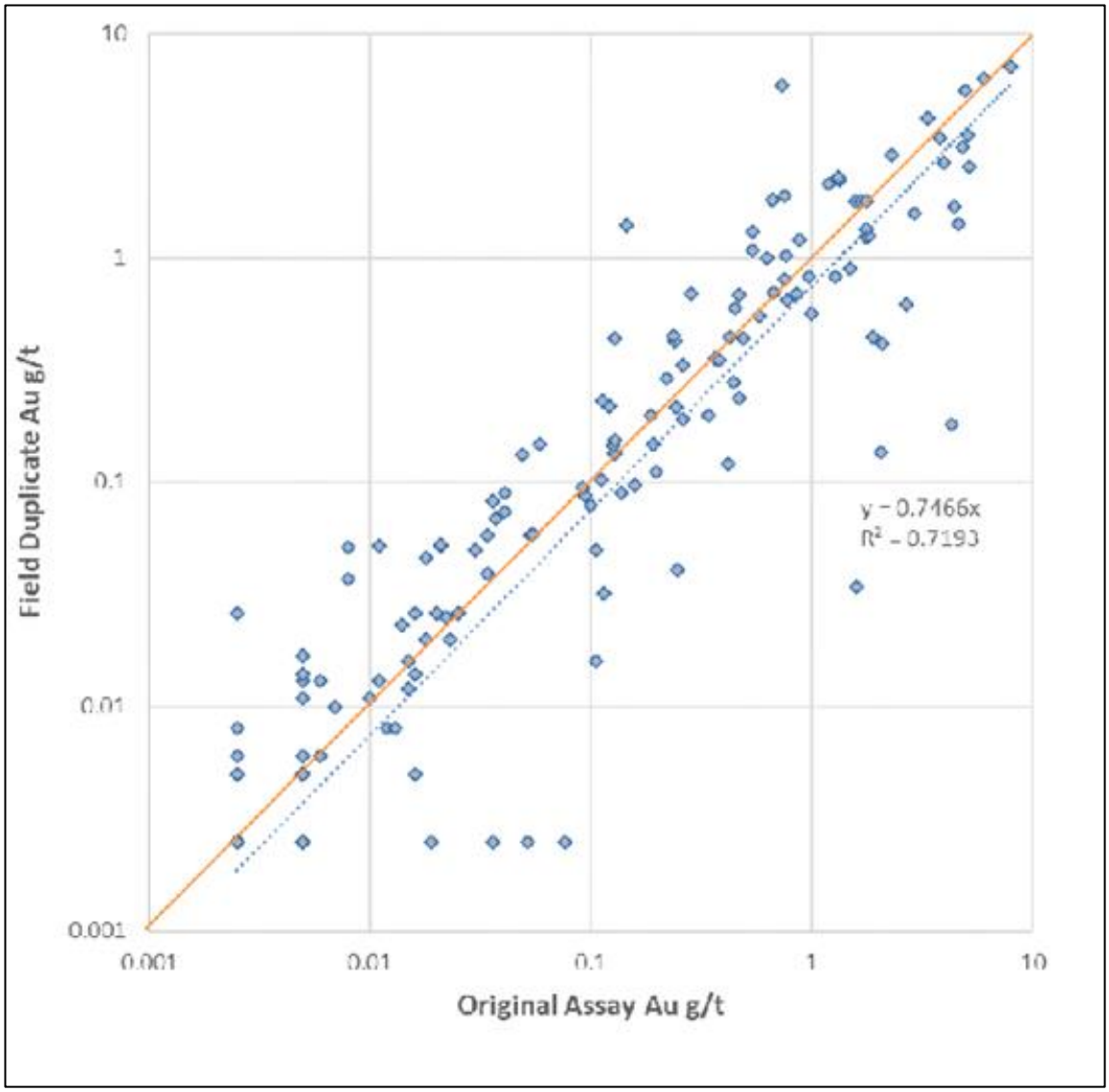
11.6.6 Kierens-Norlartic Duplicates

Duplicates pairs in the NioGold/Osisko Era drilling in Kierens-Norlartic are available as field duplicates and internal laboratory duplicates for analysis.

11.6.6.1 Kierens-Norlartic Field Duplicates

A set of 155 field duplicates sampled between 2007 and 2014 are available for review. Figure 11-16 shows a scatter plot of the duplicate pairs. A best fit line gives a slope less than 1.0 indicating that the assays of the field duplicates are less than the original assay, although the scatter is high, and the best fit line is influenced by some outliers. Natural variation between the field duplicates could be to blame.

Figure 11-16: Kierens-Norlartic Field Duplicates, 2007–2014



Source: MMTS, 2020.

A comparison of means and medians of subsets of the field duplicate data is presented in Table 11-3. The mean and medians agree more closely as the data set is restricted to exclude the extreme pairs by mean assay, but the indicator of error, being negative, continues to imply that the duplicate assay is lower than the original assay. However, because the sample set is small (155 samples, representing 0.6% of core samples from this era), limited conclusions can be obtained and the nugget effect could be a factor.

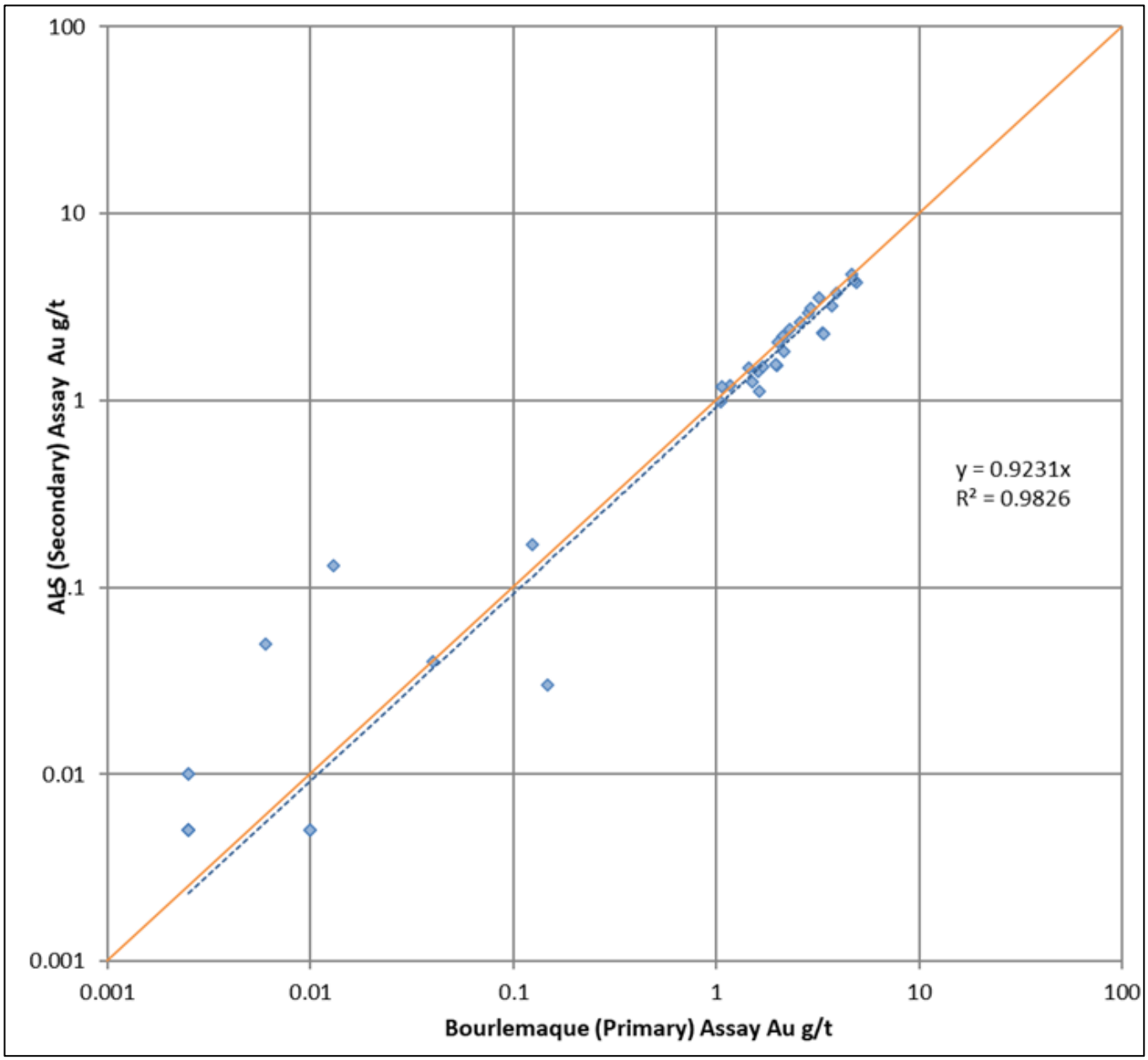
Table 11-3: Kierens-Norlartic Field Duplicates Subset Statistics 2007–2014

Subset	Statistic	Original Assay (g/t)	Field Duplicate (g/t)	Error (%)
All pairs	mean	0.724	0.627	-15.3
	median	0.106	0.090	
Above 0.01 mean assay	mean	0.934	0.809	-15.4
	median	0.186	0.207	
Above 0.01 and below 2.0 mean assay	mean	0.453	0.428	-5.7
	median	0.146	0.146	
Above 0.01 and below 1.5 mean assay	mean	0.336	0.313	-7.3
	median	0.129	0.132	

11.6.6.2 Kierens-Norlartic External Duplicates

External Duplicates for Kierens-Norlartic are available for drilling in years 2006 and 2007. In 2006, Bourlemaque was the primary laboratory and in 2007, ALS was the primary laboratory. Figure 11-17 shows the 41 duplicate pairs, most of them over 1 g/t sent for re-assay at ALS. Although the best fit line indicates a slightly lower value at the second laboratory, the values cluster the 1:1 line well and are not a large enough sample set to be of concern.

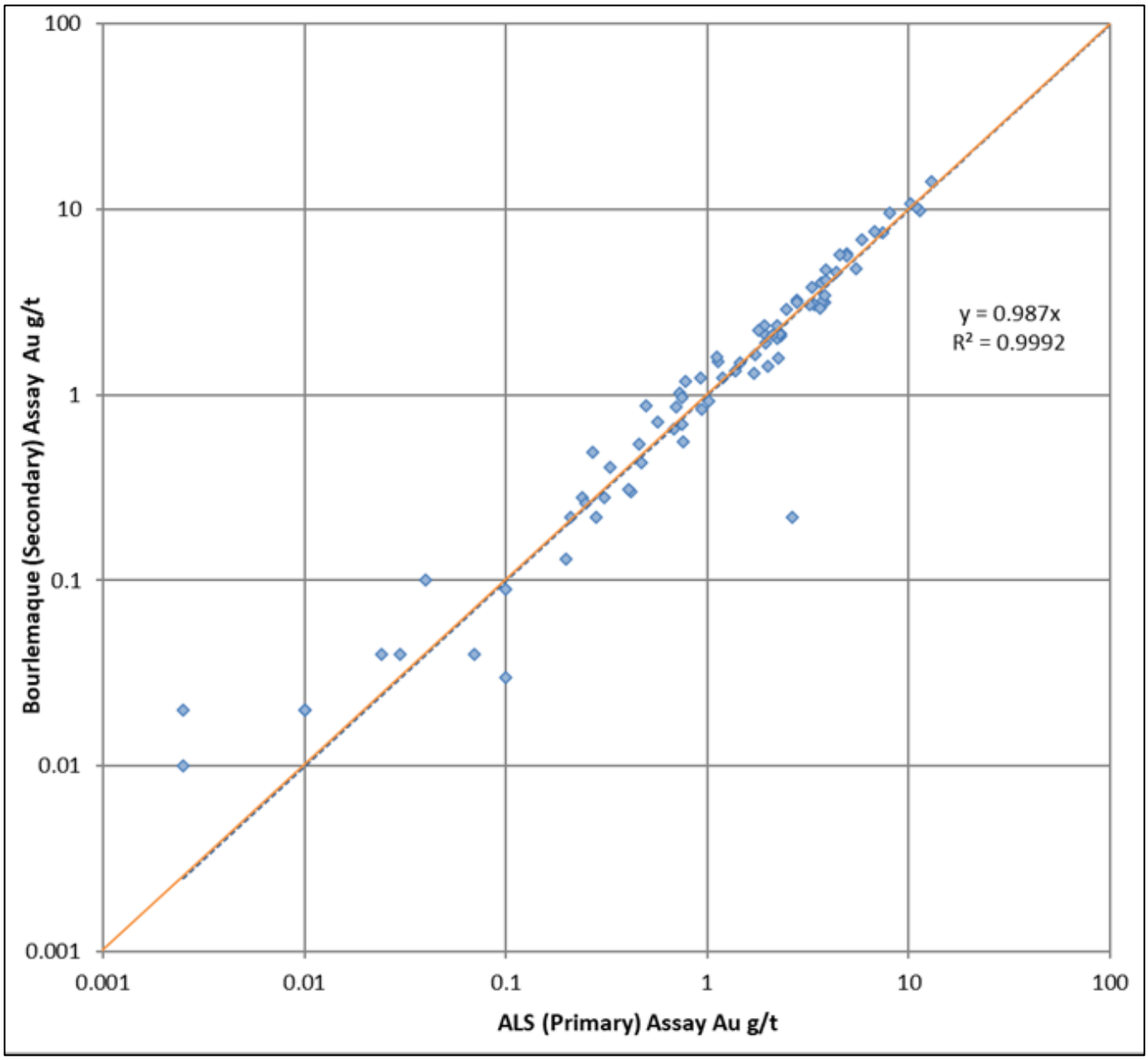
Figure 11-17: Kierens-Norlartic External Duplicates, 2006



Source: MMTS, 2020.

Eighty-one duplicates pairs of 30 g AA-FA assays from 2007 with primary lab ALS and secondary lab Bourlemaque are shown in Figure 11-18. The data approximates the 1:1 line well and the best fit line confirms a strong correlation between the two sets of data.

Figure 11-18: Kierens-Norlartic External Duplicates, 2007

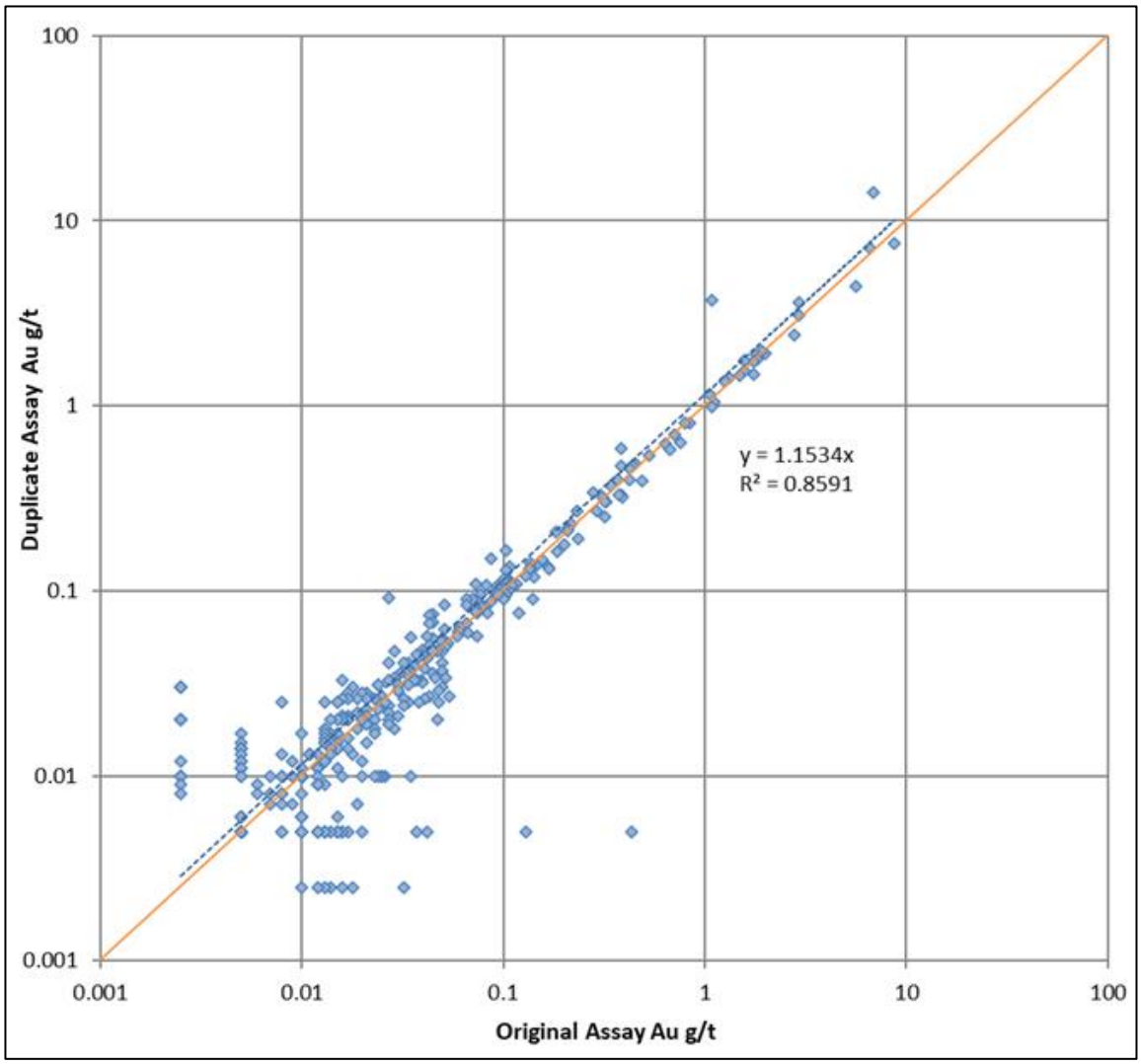


Source: MMTS, 2020.

11.6.6.3 Kierens-Norlartic Internal Duplicates

The QA/QC database includes internal pulp duplicates performed at the primary laboratory for drilling between 2011 and 2014. These 732 pairs are presented in Figure 11-19 and show reasonable clustering along the 1:1 line. A best fit line indicates the values are higher in the duplicate samples. Analysis of the basic statistics of the paired data in Table 11-4 indicates the error is likely due to the values at the extremes as the means and medians between 0.1 g/t and 10.0 g/t mean assay value match well.

Figure 11-19: Kierens-Norlartic Internal Duplicates, 2011–2014



Source: MMTS, 2020.

Table 11-4: Norlartic Kierens Internal Duplicate Statistics 2011–2014

Samples	Statistic	Original (g/t)	Duplicate (g/t)	Error (%)
All	average	0.115	0.126	8.7
	median	0.010	0.008	
Between 0.1 g/t and 10 g/t mean assay	average	0.944	0.954	1.1
	median	0.386	0.381	
Between 0.01 g/t and 10 g/t mean assay	average	0.275	0.278	1.0
	median	0.041	0.037	

11.7 O3 QA/QC Data Analyses – 2020 and 2021

GMS reviewed the analytical quality control data produced by O3 Mining between 2020 and 2022 to confirm that the analytical results were reliable for informing the mineral resource. Blanks and CRM data were provided by O3 Mining in Microsoft Excel spreadsheets. From June 26, 2020, to March 16, 2022, there were a total of 3,567 CRMs and 3,583 blanks submitted to AGAT Laboratory. The control samples represent approximately 6% of the total number of samples submitted for assaying. Analyses of data from CRM and blank samples are normally illustrated in time-series plots to identify extreme values (outliers) or trends that may indicate issues with the overall data accuracy and precision.

11.7.1 Blanks

Blank materials were considered failed when the returned gold value exceeded 10 times the lower detection limit (DL = < 0.002 g/t Au) of the analytical method. The results are considered acceptable since more than 99% of the control samples show no signs of contamination (see Table 11-5).

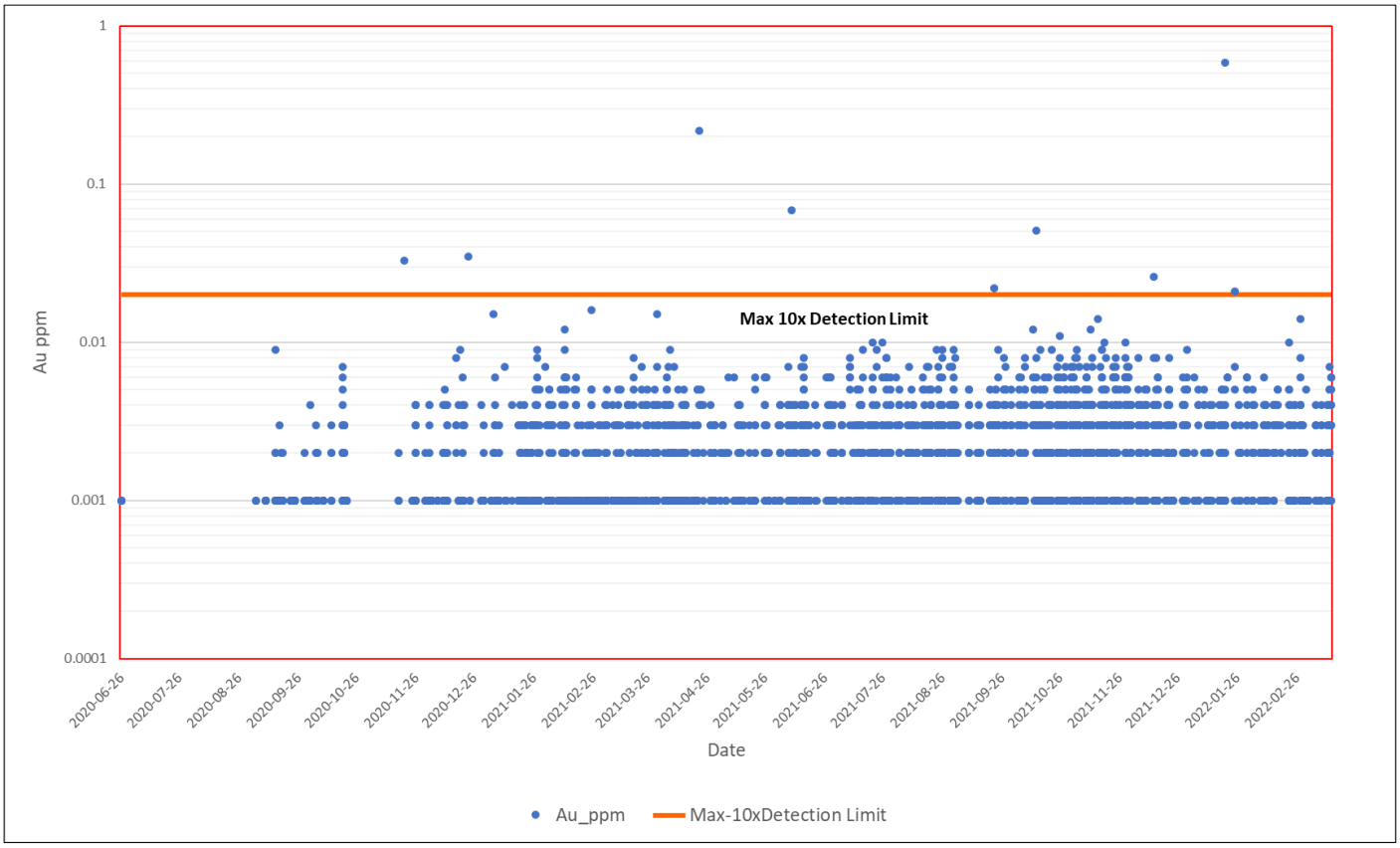
Table 11-5: Summary of Blanks Analytical Results as implemented by O3 Mining – 2020–2022

Material	Lower Detection Limit (ppm)	Number of Samples	Number of Failures	Percent Failure (%)
Blank material	0.002	3583	11	0.31

Figure 11-20 shows the performance of the blank samples within the control limits submitted by O3 Mining during the recent drilling programs for the Marban Engineering Project.

Overall, the performance of control samples inserted with the assay data submitted for Marban and Kierens-Norlartic deposits is considered by GMS as good. Inspection of time series charts for blank samples indicate that the AGAT laboratory performed well without showing contamination issues over the analysis, with only 0.31% (11/3,583) blank samples assaying above 10 times the detection limit. Some of the failures are likely caused by an inversion of a blank by swapping a blank with a CRM, or and erroneous entry in the database. O3 Mining has a robust system in place to identify QA/QC failures, and re-assays are always requested when an abnormal value is returned from the laboratory.

Figure 11-20: Blanks Analytical Results – 2020–2022 Drilling Program – Marban Engineering Project – Gold (g/t)



Source: GMS, 2022.

11.7.2 Standards

The analytical quality control data produced by the CRM samples used during 2021–2022 sampling program is summarized in Table 11-6.

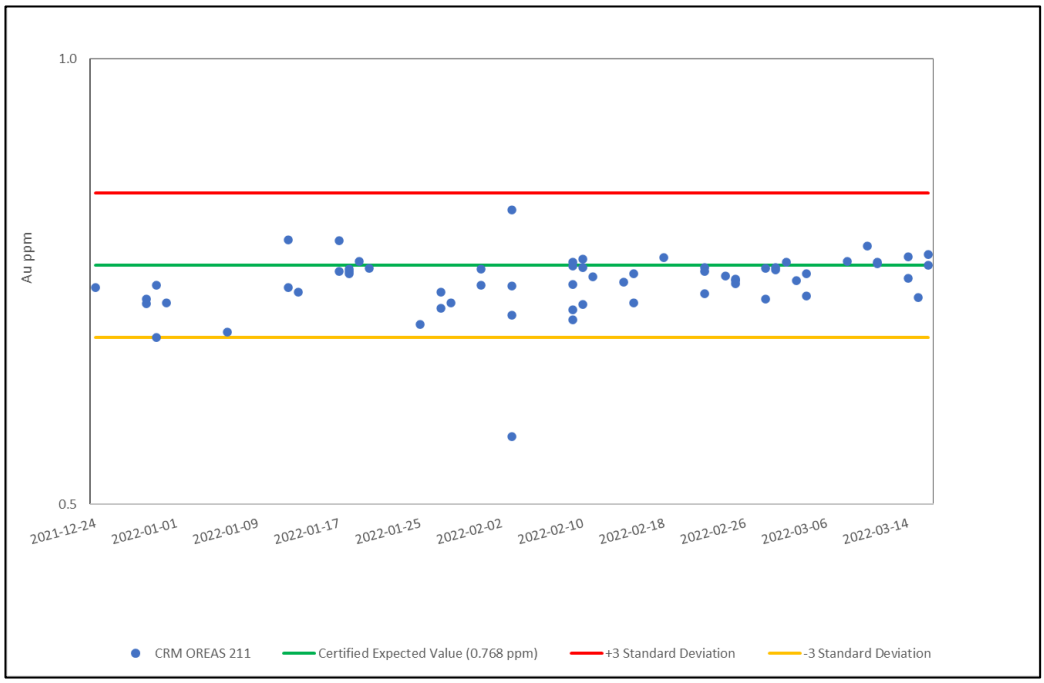
All of the failures observed in the CRM results were investigated by O3 Mining and replaced by the re-analysed value when it was considered necessary. Overall, the results tabulated below indicate that the CRMs included as quality control samples are acceptable and minimal bias regarding the precision of the expected values vs. the analysed value observed through the assaying period. OREAS 215 exhibited strong variation compared with other standards and was subsequently replaced by another standard (OREAS 214 and 239) with a similar grade.

Table 11-6: Summary of CRM Analytical Results as Implemented by O3 Mining on the Marban Drilling Program (2020–2022)

CRM	Recommended Gold Value (ppm)	Standard Deviation (ppm)	Number of Samples	Number of Failures	Percent Failure (%)
OREAS 211	0.768	0.027	64	1	1.6
OREAS 214	3.03	0.082	123	1	0.8
OREAS 215	3.54	0.097	216	33	15.3
OREAS 216b	6.66	0.158	263	3	1.1
OREAS 219	0.76	0.024	478	3	0.6
OREAS 221	1.062	0.036	68	0	0.0
OREAS 224	2.154	0.053	70	0	0.0
OREAS 229b	11.95	0.288	318	3	0.9
OREAS 231	0.542	0.015	225	7	3.1
OREAS 235	1.59	0.038	779	14	1.8
OREAS 237	2.21	0.054	384	8	2.1
OREAS 239	3.55	0.086	222	4	1.8
OREAS 242	8.67	0.25	314	9	2.9
OREAS 243	12.39	0.306	43	1	2.3
Total			3,567	87	2.4

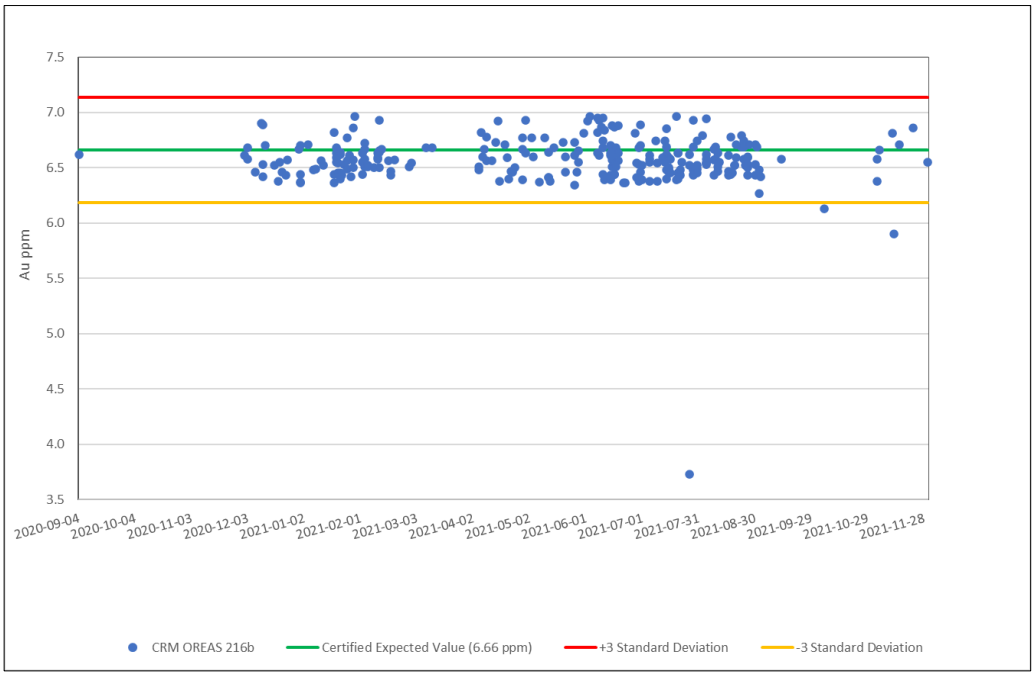
The performance of the CRMs was also validated on time-series control charts to monitor for analytical drift and abnormal assay batches. After verification, the CRMs generally performed well over time, and the majority of the results are within the control limits of ± 3 times standard deviation (3SD) of the certified recommended value. Some of the control limits of CRMs are illustrated in Figure 11-21 to Figure 11-24.

Figure 11-21: Sample Control Chart of CRM 211 – 2021–2022 Drilling Program



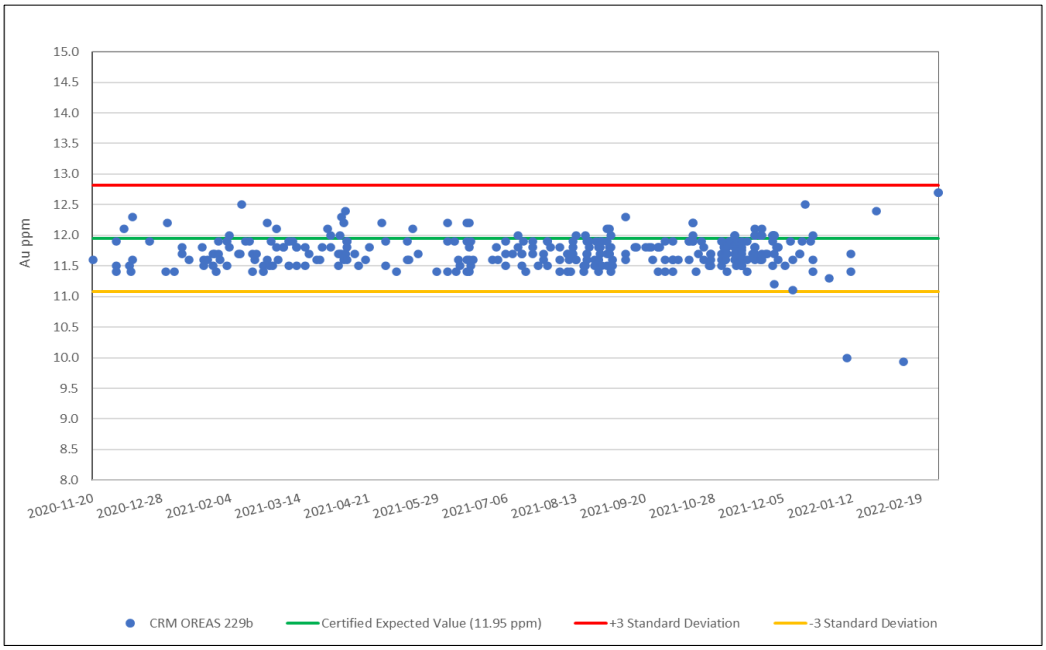
Source: GMS, 2022

Figure 11-22: Sample Control Chart of CRM 216b – 2020–2022 Drilling Program



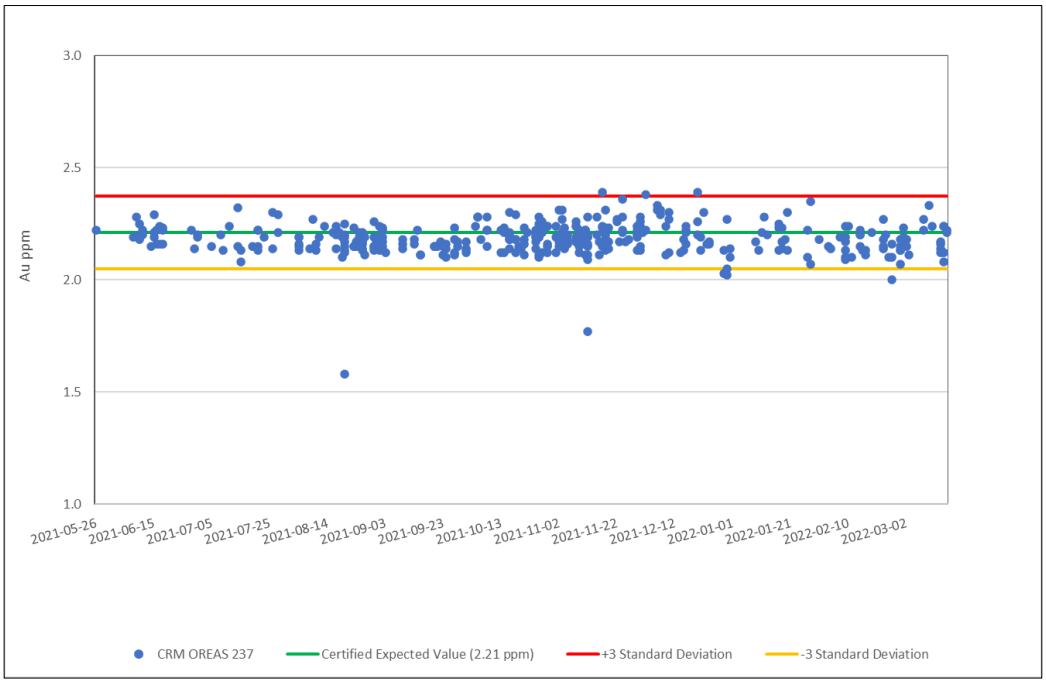
Source: GMS, 2022

Figure 11-23: Sample Control Chart of CRM 229b – 2020–2022 Drilling Program



Source: GMS, 2022

Figure 11-24: Sample Control Chart of CRM 237 – 2021–2022 Drilling Program

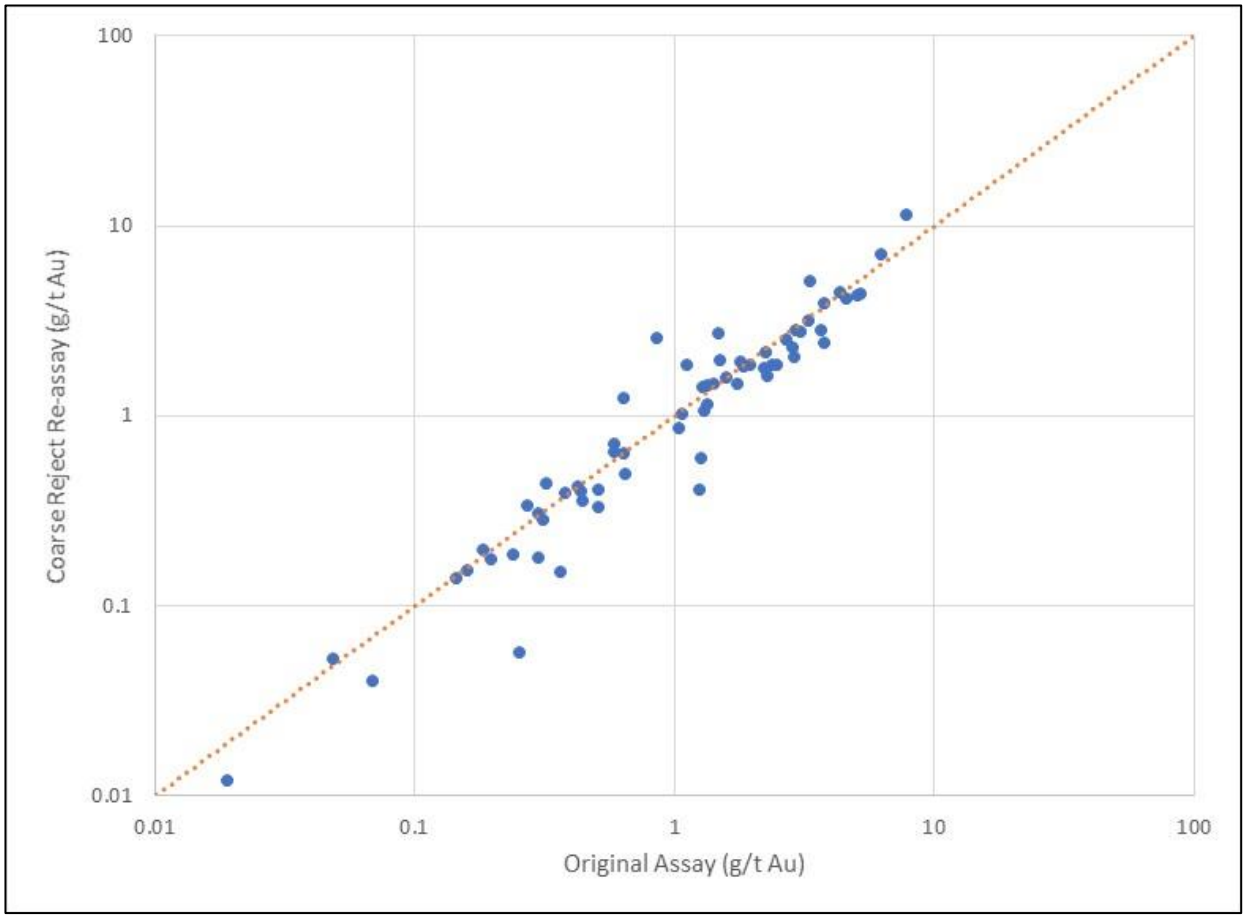


Source: GMS, 2022.

11.7.3 Coarse Reject Duplicates

As part of the QA/QC protocol, O3 Mining re-analysed 65 coarse duplicates from the 2020–2022 drilling program to test for variability within the assaying process, and to identify any potential nugget effect present in the deposit. The results are presented in Figure 11-25. The coarse duplicates have reproduced the original assay value well, with no bias observed.

Figure 11-25: Coarse Reject Duplicates – 2020–2022 Drilling Program



Source: GMS, 2022.

11.8 Conclusions and Recommendations

The QP concludes the sample preparation, analysis, and security procedures used by NioGold and Osisko are adequate. Documentation of the procedures employed in the pre-NioGold drilling programs is lacking; however all operators were recognized mining companies, many of which actively mined at the project site. The data lacking QA/QC has been validated through re-assays.

The results from QAQC sampling during the Niogold era, especially the failure rates of CRMs and blank, are at times questionable and show high variation. In addition, there is some uncertainty surrounding the reanalysis protocols applied at the time. Subsequent umpire analysis of pulps and data verification has shown that no bias has been introduced, and the QP remains satisfied that this information can be used for mineral resource estimation purposes.

In the opinion of the QP, sample preparation, analysis, and security procedures implemented by O3 Mining are to best industry standards, and robust controls are in place to ensure the integrity of the assay database. QA/QC results obtained by O3 Mining between 2020 and 2022 show no material errors. The QP believes that the drilling database is sufficiently reliable for mineral resource estimation.

The QP recommends that future drilling and QA/QC protocols include CRMs of lower grades, such as 0.15 g/t, 0.25 g/t, and 0.35 g/t Au. At the time of writing, the lowest grade standard is 0.76 g/t Au.

12 DATA VERIFICATION

Some information in this section relating to the verification of historical drilling has been sourced from the 2020 Technical Report by MMTS, with some additions by GMS regarding the site visit and additional data verification work.

12.1 Site Visit

James Purchase, P.Geo, Vice-president of Geology and Resources at GMS, visited the site between September 8, 2021, and September 10, 2021. Drilling activities were ongoing at the time of the visit. Core processing and storage facilities located in Val D'Or were toured, and drill core from Marban, Kierens, and Nolartic deposits was reviewed. GMS also reviewed sampling and QA/QC procedures on site, and visited the preferred independent laboratory (AGAT, Val D'Or) to inspect the sample preparation facilities.

Drill collars were visited and independently verified by the QP using a handheld GPS. The comparison between the results and the database is shown in Table 12-1. Some examples of drill collars are shown in Figure 12-1.

Table 12-1: Drill Collar Verification

Hole ID	Database Coordinates (NAD83, Zone 18N)		Handheld GPS Coordinates (NAD83, Zone 18N)		Difference in Coordinates (m)	
	x	y	x	y	x	y
MB-08-50	278493.1	5335937.6	278494	5335939	-0.9	-1.4
MB-10-118	278288.3	5335850.3	278286	5335854	2.3	-3.7
MB-15-445	278491.8	5335909.6	278490	5335912	1.8	-2.4
NL-11-041	277362.5	5337306.1	277360	5337309	2.5	-2.9
O3MA-21-178	278240.4	5335825.3	278241	5335828	-0.6	-2.7
O3MA-21-173	278472.3	5336024.1	278471	5336032	1.3	-7.9

Figure 12-1: Example of Drill Collars Located and Verified by the QP

Source: GMS, 2021.

Figure 12-2: Secure Core Storage Facility (top) and Core Cutting Facilities (bottom) at the O3 Site in Val D'Or

Source: GMS, 2021.

12.2 QP Samples (Check Assays)

Considering the Marban and Kierens-Norlartic deposits have been past gold producers and the quantity of reanalyses undertaken by previous operators (NioGold and O3), the QP is satisfied that there is no requirement for additional independent sampling. When comparing drill core observations (veining, alteration, structure) of mineralized intervals with gold grades, mineralisation is can be visually identifiable and correlates well with the assayed intervals. The QP has no concerns regarding the validity of the 2021 drilling campaign, or the chain of custody.

12.3 Drillhole Database Verification

The QP checked 10% of assay certificates from the 2021 drilling campaign against the drilling database for accuracy and no errors were found.

The QP conducted an audit of the assay database against the original assay certificates to verify the reliability of the data provided by O3 Mining. Approximately 10% of the assay data was audited, and no discrepancies were noted on the gold value data entry. During the validation, GMS verified the final gold (Au) value recorded in the assay table and noticed that the value is compares well the original laboratory values.

The QP noted that below detection values have been assigned a zero in the database. These could be replaced by half of the DL rather than the zero value as noted in the assay database.

In the opinion of the QP, the Marban and Kierens-Norlartic assay data is reliable and free of material data entry errors.

12.4 Historic Database Verification – MMTS

In the 2020 PEA Technical Report, MMTS described in detail the validation of the historical drilling from the 1960's and 1980's. GMS has reviewed this information and is in agreeance with their findings and recommendations. A summary is provided in the following sections.

NioGold performed re-assays of available core samples of historic drilling in both the Marban and Kierens-Norlartic zones with the intent of validating the historic drill data. Analysis and conclusions by MMTS are presented in the following subsections.

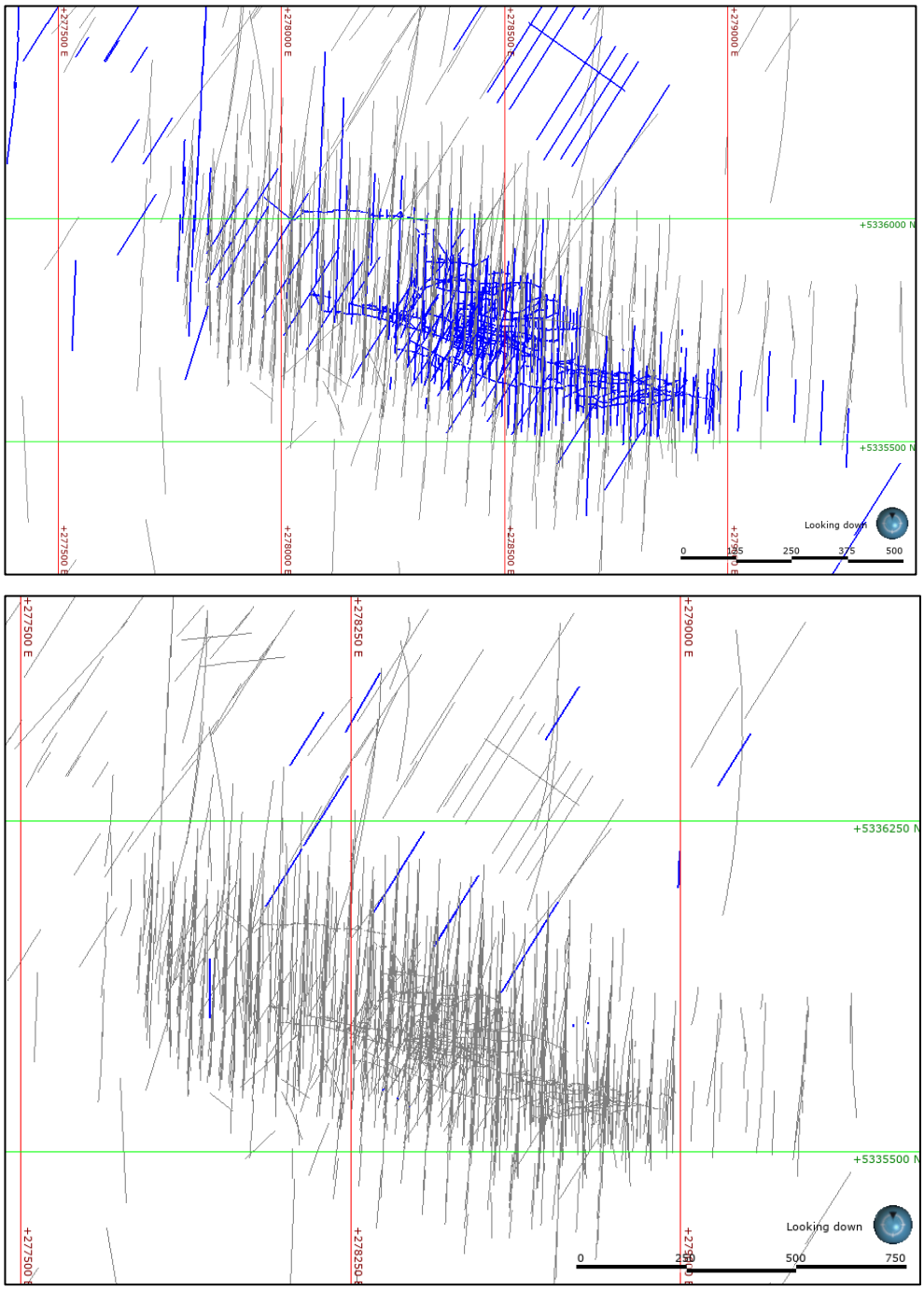
A plan of historical drilling at Marban from the 1960's and 1980's is shown in Figure 12-3.

12.4.1 Marban Historic Verification

12.4.1.1 Marban Historic 1960s and 1980s Data – Surface and Underground drilling

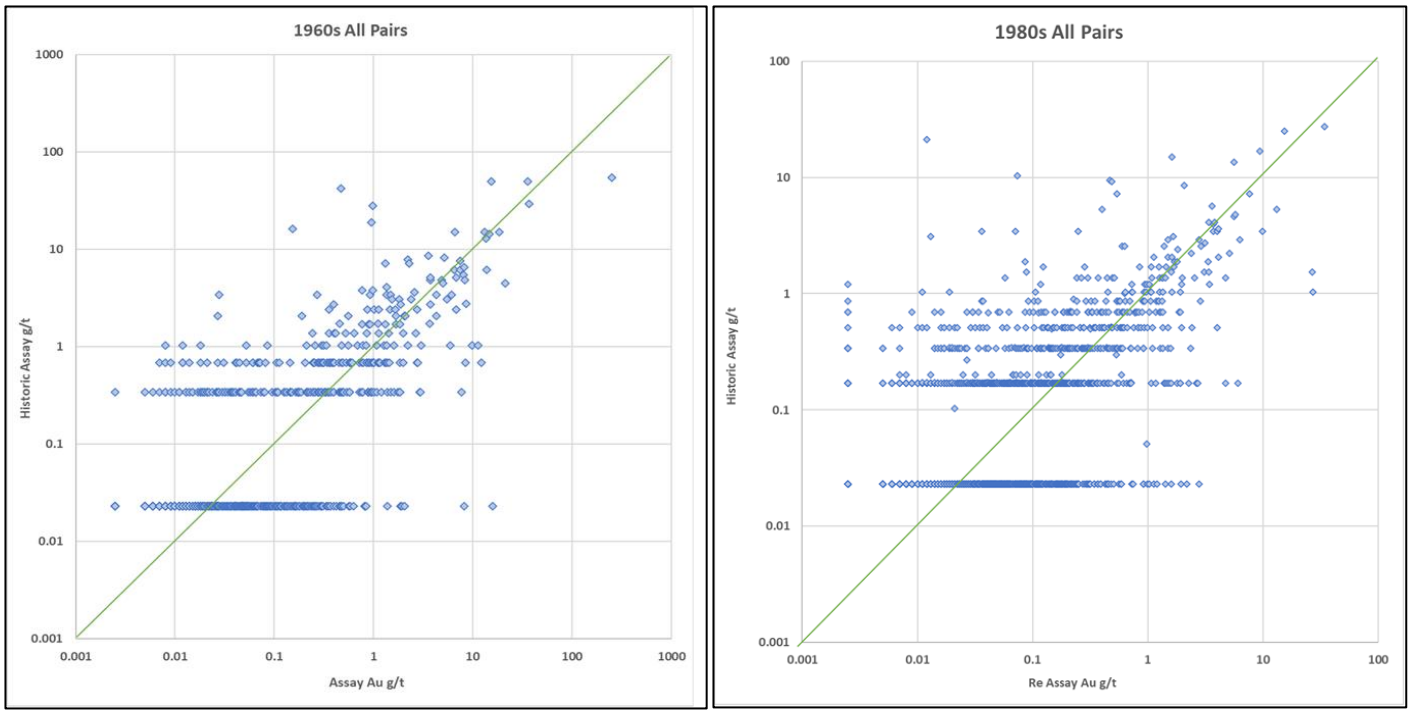
Scatter plot of the duplicate pairs of the 1960s and 1980's assay data are provided in Figure 12-3. The paired analysis shows the impact of the historic DL of 0.01 opt (0.34 g/t), at the time irrelevant in relation to the historical underground operation, but critical in the framework of the current open-pit scenario. The lack of precision of the historical data below 1 g/t is apparent, and the strong horizontal lines in historic value correspond at intervals of the DL (0.34 g/t) and are clearly visible up to 1.37 g/t, or four times the DL.

Figure 12-3: Plan Views of Historical Drilling in Blue – 1960's (top) and 1980's (bottom)



Source: GMS, 2022

Figure 12-4: Marban Scatter Plot of 1960's Assays vs. Re-assays (left), and 1980's Assays vs. Re-assays (right)



Source: MMTS, 2020.

MMTS produced binned statistics for the 1960's and 1980's data, which show a clear positive bias in the historical assays to overestimate the gold grades in the 0.0–1.0 g/t range due to the aforementioned DL issue. In the opinion of the QP, historical data from the 1960's cannot be included in the MRE as it would likely result in a considerable overestimation of tonnage and grade in the block model. Considering the 1980's drilling at Marban represents less than 1% of the drilling at Marban and a significant portion was re-assayed, the QP considers that its inclusion in the mineral resource database is warranted. Eleven holes from the 1960's were also sufficiently re-assayed for inclusion in the mineral resource database.

12.4.2 Kierens-Norlartic Historic Verification

According to MMTS, a lack of certificates and QA/QC for assay data prior to 2000 resulted in 13,849 re-assays being completed and subsets of these paired data have been analyzed by others, with the conclusion that the historic data is biased high (Gustin 2013, Belzile 2016). As was noted in these reports, the historic data showed a significant bias, but the data was not considered on an individual year basis. This analysis confirms that the data from Kierens-Norlartic prior to 1986 is significantly biased both at grades near DL and at high grades above about 10 g/t. It is therefore concluded that this data should not be used in the mineral resource estimate. The QP considers all re-assay data and drillhole data after 1986 as acceptable and that it should be included in the MRE. In addition, 17 pre-1986 drillholes were considered sufficiently re-assayed for inclusion into the mineral resource database.

Comparison of mean and median values for all relevant years of re-assays to historic assays is presented in Table 12-2. Although the means of all samples compare well, the medians are significantly different. The table illustrates significant differences by year showing why it is important to consider each group separately.

Table 12-2: Comparison of Kierens-Norlartic Re-assay and Historic Means and Medians by Year

Year	Number of Re-assays	Re-assay Mean (g/t)	Historic Assay Mean (g/t)	% Diff. (mean)	Re-assay Median (g/t)	Historic Assay Median (g/t)	% Difference (median)
1958	116	0.46	0.90	49%	0.09	0.34	73%
1964	420	1.19	1.34	11%	0.31	0.34	9%
1980	112	1.02	1.22	17%	0.16	0.34	52%
1981	185	0.72	1.51	52%	0.01	0.17	94%
1986	809	0.75	0.83	10%	0.17	0.17	-2%
1988	2,688	1.02	0.91	-11%	0.36	0.34	-6%
All samples	4,330	0.96	0.97		0.28	0.34	

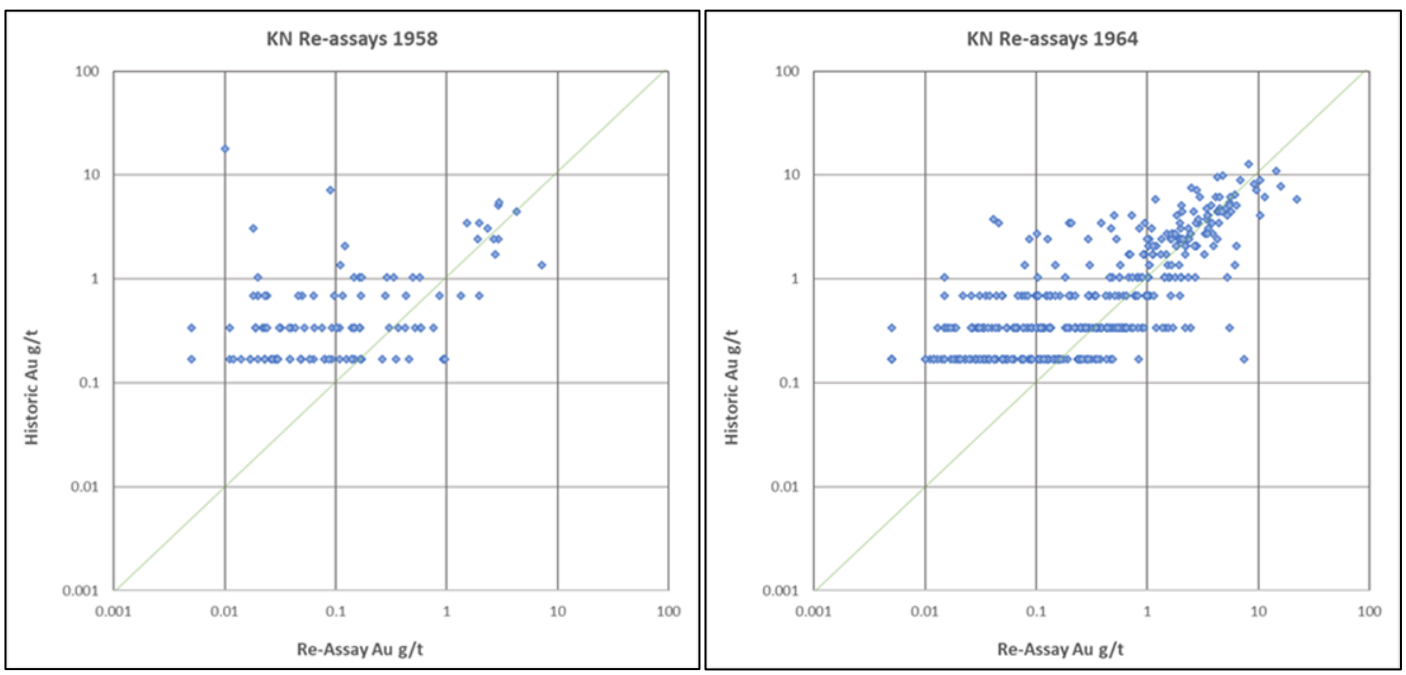
Source: MMTS, 2020.

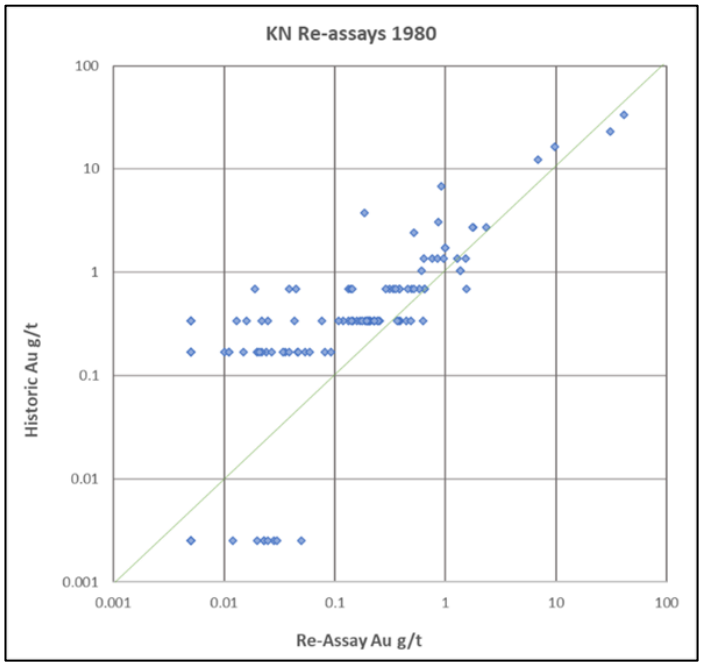
12.4.2.1 Kierens-Norlartic Pre-1986 Data

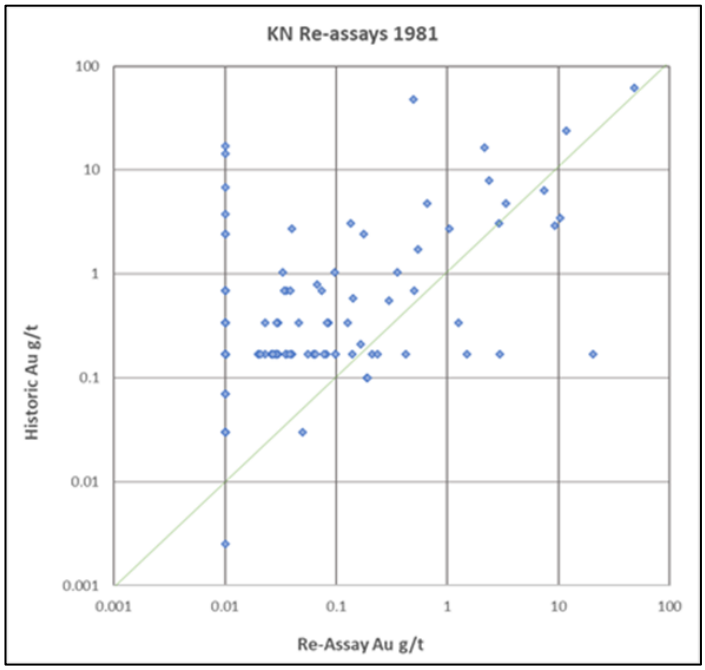
Of the samples surrounding the pit area, the historic assays before 1986 have an original gold assay DL of 0.01 oz/ton (0.34 g/t). In all of the analyses in this section, the historic assay data reported as 'tr' has been replaced with one-half the DL, or 0.17 g/t.

Figure 12-5 shows various scatter plots of the re-assays with the original assay on the y-axis and the re-assay on the x-axis. It is clearly seen that the data is clustered above the 1:1 line shown in green, indicating the historic assay data is biased high as confirmed by mean and median values presented earlier in Table 12-2.

Figure 12-5: Scatter Plots for 1958, 1964, 1980, and 1981 Datasets at Kierens-Norlartic







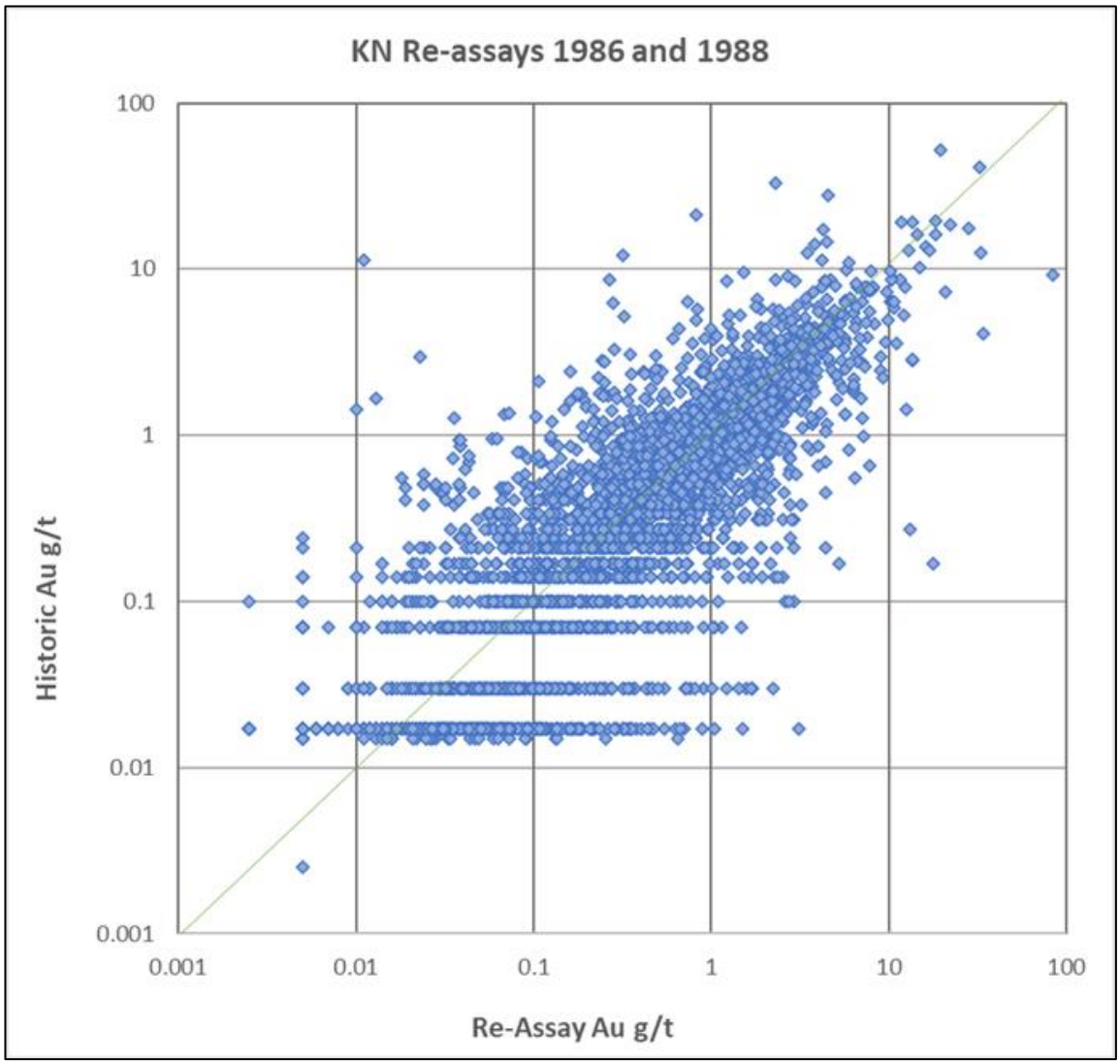
Source: MMTS, 2020.

12.4.2.2 Kierens-Norlartic - Later 1980s Data

In addition to reassays for the pre-1986 data, NioGold reanalysed pulps from holes drilled in 1986 and 1988. The DL of the later 1980s data is significantly lower than the early 1980s data, at 0.001 oz/ton or 0.034 g/t. Although this DL is still high when compared to today's standards, no bias is observed in the scatter plot (Figure 12-6), and it appears balanced around the 1:1 line.

Based on this observation, the QP believes that the assay data from 1986 and 1988 can be included in the mineral resource estimate.

Figure 12-6: Kierens-Norlartic Scatter Plot of 1986 and 1988 vs. Re-Assay

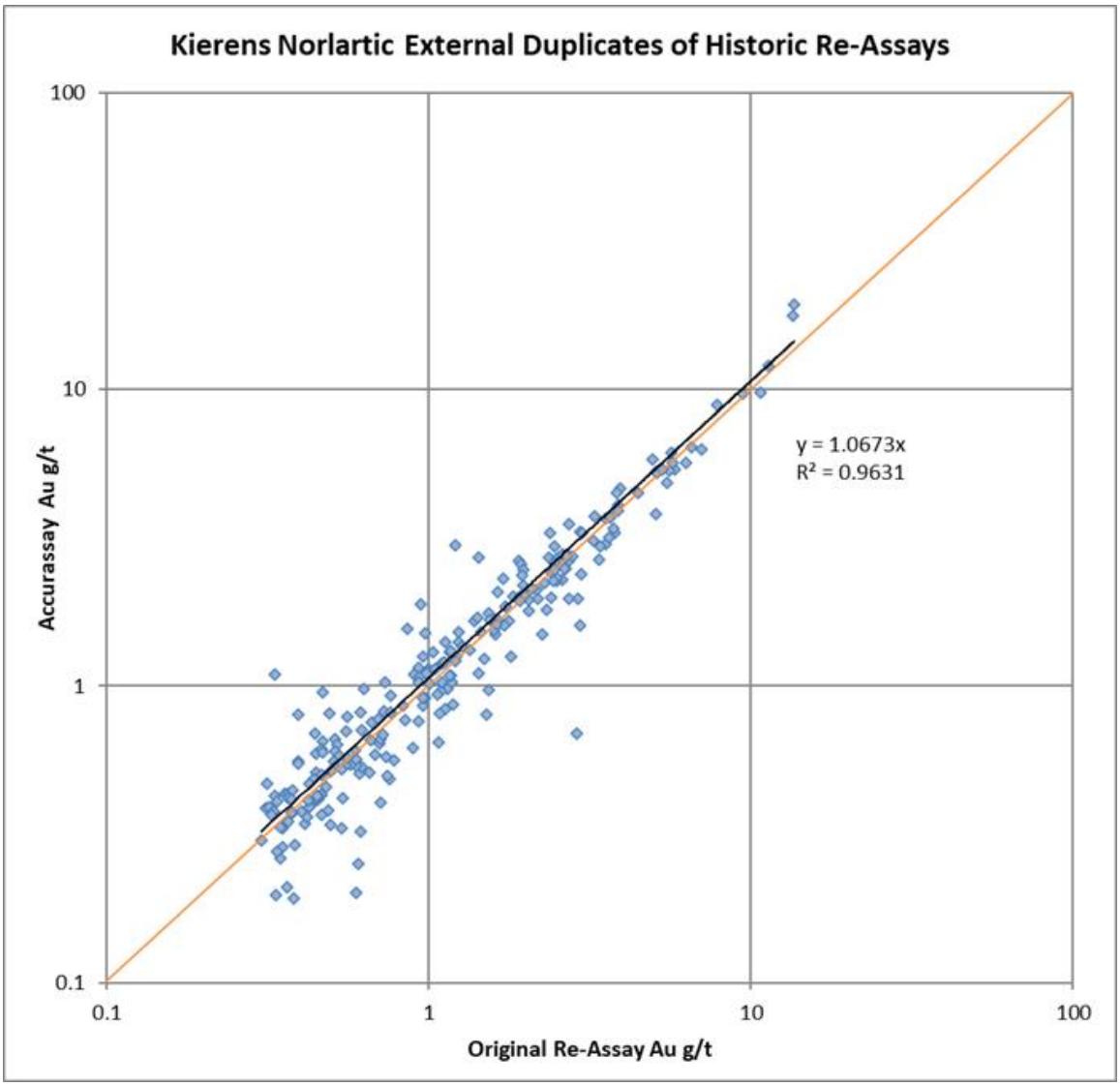


Source: MMTS, 2020.

12.4.2.3 Kierens-Norlartic Historic Verification - External Duplicates

An additional set of 248 external duplicate re-assays by Accurassay was completed in 2016 of pulps of historic re-assays with assays greater than 0.3 g/t. Figure 12-7 shows a scatter plot of the duplicates with the two pairs with greater than 10 g/t difference removed from the dataset. The data show reasonable correlation along a 1:1 line. The mean of the original historic re-assay data set is 1.78 g/t and the Accurassay set is 1.82, an error of 2.6% higher which the QP considers acceptable.

Figure 12-7: Kierens-Norlartic External Duplicates of Historic Re-Assays

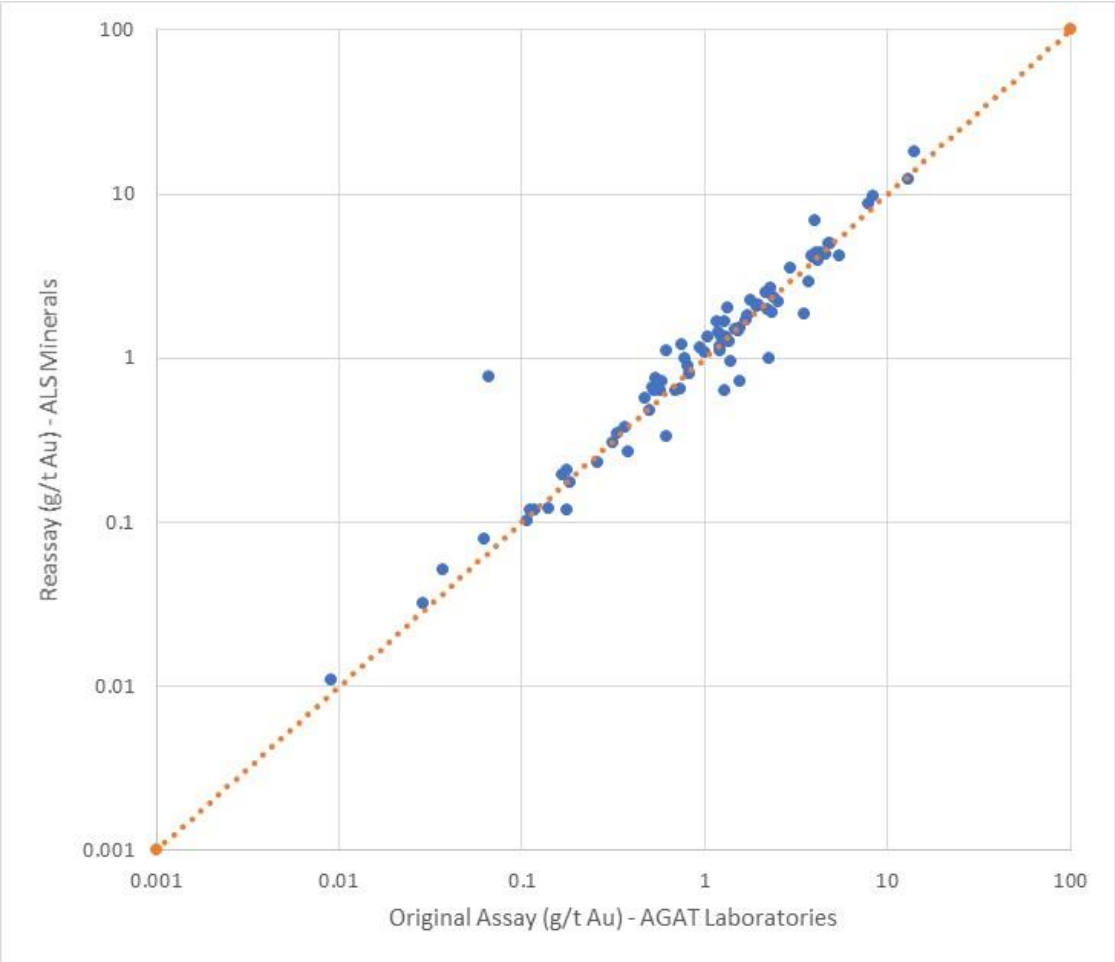


Source: MMTS, 2020.

12.5 Umpire Assays – 2021 Drilling Campaign

In addition to the previous data verification work outlined in the 2020 Technical Report, the QP reviewed the umpire assays undertaken by O3 Mining using a second laboratory (ALS Minerals). Eighty-two pulps were re-analysed, and the results are presented in Figure 12-8. The reassays confirmed the original assay values, and no bias was observed.

Figure 12-8: Umpire Assays of Pulps Undertaken at ALS Minerals – 2021 Drilling Campaign



Source: GMS, 2021.

12.6 QP Conclusions

The QP has reviewed all available information and has come to the following conclusions:

- The Marban historic data from the 1960s and 1980s should not be included in the mineral resource estimate due to the possibility of introducing a material bias and an overestimation of the mineral resource.
- The Kierens-Norlartic historic data earlier than 1986 cannot be relied upon due to the same precision issues as observed at Marban. This data should not be included in the mineral resource estimate.
- The Marban data since the 1980s and the Kierens-Norlartic data from 1986 forward is considered of acceptable quality and accuracy for resource modelling.
- All assay data acquired in 2021 is of a high quality and has undergone significant checks to ensure it is robust. It is fit for inclusion into the mineral resource estimate
- Drilling, sampling and QA/QC procedures are to best industry practice, and the QP has no concerns in regard to chain of custody.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Introduction

The objective of the metallurgical testing program was to quantify the metallurgical response of ores from deposits in the Marban project. The program was designed with the intent to confirm the parameters for process design criteria for comminution, gravity concentration, leaching, and cyanide detoxification. The metallurgical program was completed in Q1 2022 at Base Metallurgical Laboratories Ltd. (BaseMet Labs) in Kamloops, BC as Project BL0886, and was performed on composites from Marban, Norlartic, Kierens, North Zone, and Gold Hawk Zone deposits.

13.2 Historical Testwork

The sources of technical and project information summarized in Table 13-1 were referenced in developing the process plant design for the Prefeasibility Study.

Table 13-1: Historical Testwork Programs and Reports

Document Name	Issuer	Date	Description
Revised Preliminary Metallurgy Study Marban report 13333-001	SGS Mineral Services	March 1, 2012	Head analysis, mineralogy, gravity separation, cyanidation, flotation testing on gravity tailings, and whole ore cyanidation.
Gravity Circuit Modelling and Scale-Up Report	FLSmidth	October 20, 2015	Gravity testing campaign to estimate E-GRG recovery.
The Recovery of Gold from Marban Engineering Project Samples report 14947-002	SGS Mineral Services	February 26, 2016	Comminution testing and cyanidation testing examining the effect of pre-aeration on gold extraction and reagent consumption.
The Grinding Circuit Design Based on Ten Bench-Scale Grindability Test results from the Marban Engineering Project report 149647-003	SGS Mineral Services	May 24, 2016	Comminution testing.
Développement du circuit de concentration – Project Marban No. T1947	FLSmidth	June 20, 2016	Flash flotation testwork.

Source: Ausenco, 2022.

Summaries of the historic testwork listed above can be found in previous technical report on the Marban Engineering Project:

- NI 43-101 Technical Report & Preliminary Economic Assessment of the Marban Project by Ausenco Engineering Canada Inc., October 23, 2020.

The 2020 PEA design was based on testwork programs conducted on mineralized samples between 2015 and 2015. The testwork programs were performed on samples from both the Marban and Norlartic deposits. The programs investigated comminution testing, head analysis, mineralogy, gravity separation and cyanidation and flotation testing on gravity tailings and whole ore cyanidation on composites from the Marban and Norlartic deposits. The direct gold head grades were

1.24 g/t to 4.59 g/t for two Marban composites, respectively. Problematic elements such as copper and arsenic are at low concentrations and will not pose any metallurgical issues. While total organic carbon "TOC" is present, leach test results show no preg-robbing issues. Overall gold recovery for the gravity concentration and cyanide gold processing plant was estimated as 93.9% for the 2020 PEA design. Testwork from a 2016 comminution testwork program was used to develop the parameters for the crushing and grinding circuit design in the 2020 PEA. An Axb value of 31.8 was selected for the design, along with a Bond Ball Mill Work Index of 13.2 kWh/t.

13.3 Prefeasibility Study Metallurgical Testing

13.3.1 Sample Descriptions

Samples were selected from available drill core and coarse rejects for the Marban and Norlartic open pits to assemble primary composites for flowsheet development. The Marban pit was split into two parts to investigate any variations from the north to the south end of the pit. Samples from the Kierens, North Zone, and Gold Hawk Zone deposits were selected as variability samples. Twenty-two samples were utilized for the testing program.

Drill core samples from the Marban and Norlartic pits were selected by O3 Mining geologists using the following criteria provided by Ausenco for the primary Marban and Norlartic composites:

- Provide spatial representation of the Marban and Norlartic open pits;
- Reflect the predominant lithological units; and
- Provide an overall average grade aligned with the average resource grade for their respective pits while maintaining the grade distribution.

Additional Marban and Norlartic samples were selected to cover the expected range of head grades from the cut-off grade up to the maximum grades over a quarterly or semi-annual basis. Three Marban samples were selected to represent the two main lithological units, basalt and dyke, and the komatiite (ultramafic) lithological unit.

Comminution testing samples were selected based on continuous intervals from drillholes, typically 5 m. Three of the four Marban samples were from the basalt lithological unit and one from the dyke lithological unit. Table 13-2 shows the samples used and its acronym.

Table 13-2: Marban PFS Metallurgical Samples

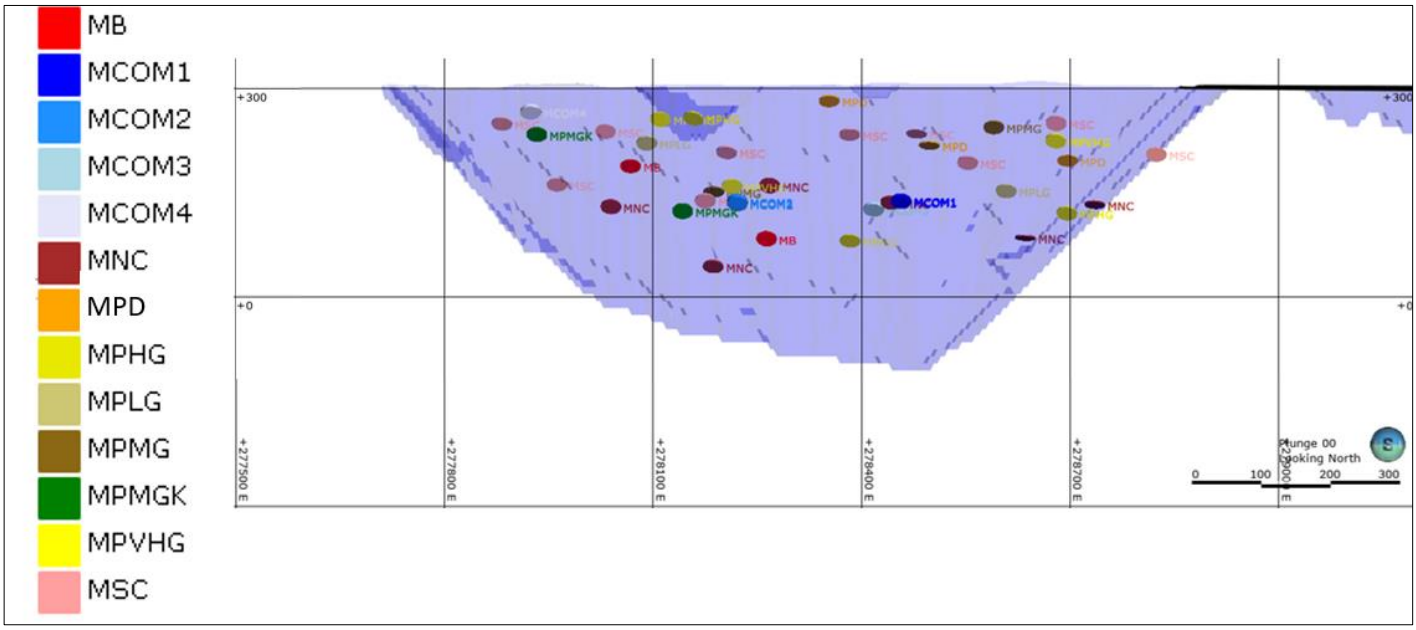
Sample	Acronym
Norlartic Pit Composite	NPC
Marban Pit South Composite	MSC
Marban Pit North Composite	MNC
North Zone Composite	NZC
Marban Pit Low Grade Composite	MPLG
Marban Pit Medium Grade Composite	MPMG
Marban Pit High Grade Composite	MPHG
Norlartic Pit High Grade Composite	NPHG
Marban Pit Very High-Grade Composite	MPVHG
Marban Komatiite (Ultramafic) Sample	MPMGK

Sample	Acronym
Norlartic Pit Low Grade Composite	NPLG
Norlartic Pit Medium Grade Composite	NPMG
Norlartic Pit Very High-Grade Composite	NPVHG
Marban Pit Dyke Composite	MPD
Marban Pit Basalt Composite	MB
Goldhawk Zone Composite	GMZC
Kierens – Norlartic Medium Grade	KNMG
Marban Comminution Sample 1	MCOM1
Marban Comminution Sample 2	MCOM2
Marban Comminution Sample 3	MCOM3
Marban Comminution Sample 4	MCOM4
Norlartic Comminution Sample 1	NCOM1
Norlartic Comminution Sample 2	NCOM2

Source: BaseMet, 2022.

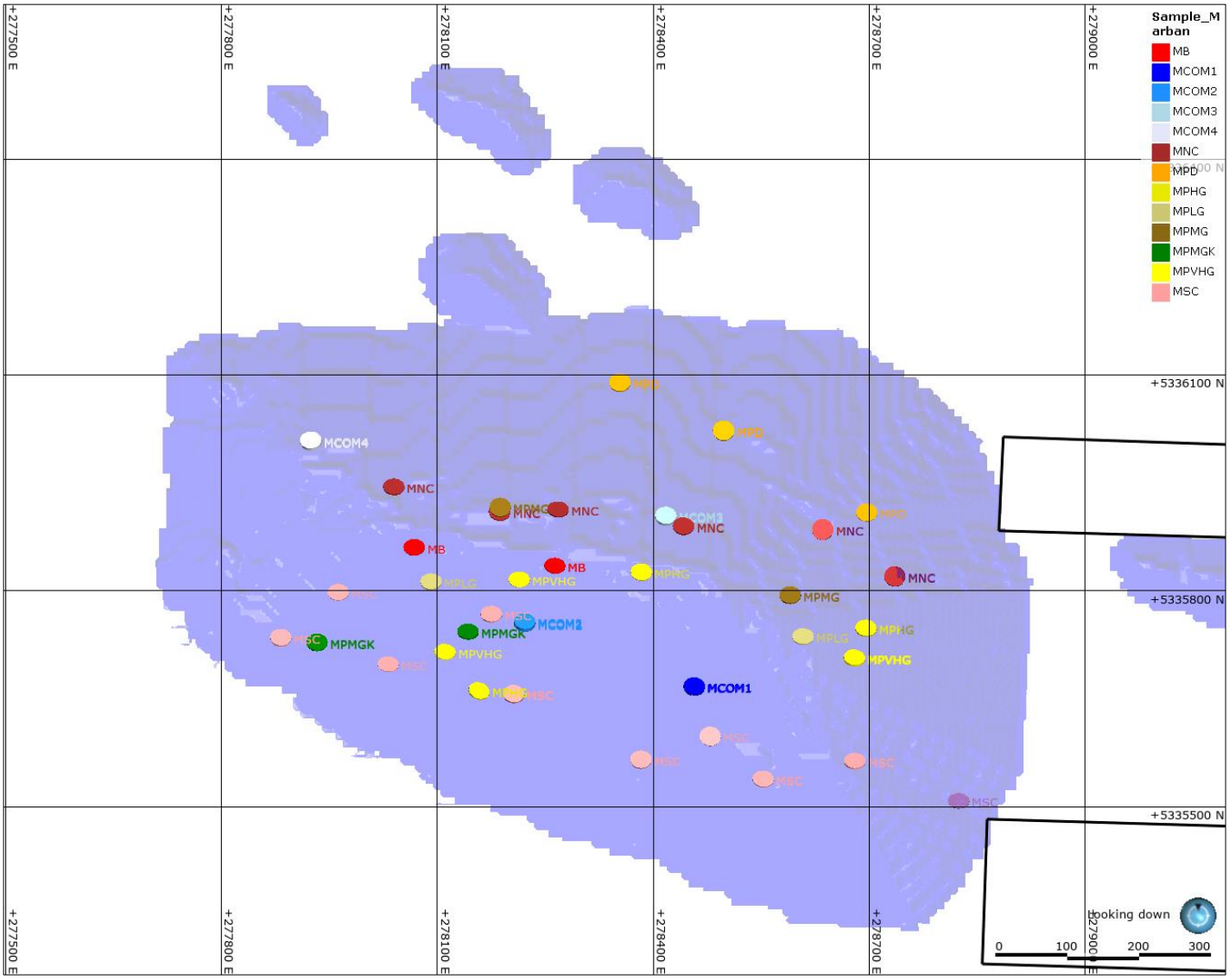
Figure 13-1 demonstrates the depths of the drillholes and samples selected for the metallurgical testing program and Figure 13-2 shows the spatial distribution.

Figure 13-1: Depths of PFS Metallurgical Samples



Source: O3 Mining, 2022.

Figure 13-2: Location of PFS Metallurgical Samples



Source: O3 Mining, 2022.

13.3.2 Sample Assays and Mineralogy

Screened metallics gold assays were conducted on the 17 composites to evaluate the occurrence of coarse free gold, as noted by the extent of gravity-recoverable gold in previous testing. Aliquots of approximately 500 g from each composite were pulverized and then screened at 106 µm with the oversize and undersize fractions assayed separately. The head grade was calculated from the weighted assays from the two fractions. The results are shown in Table 13-3. Most of the samples contained coarse gold within the coarse fraction that was significantly above the mass in the oversize fraction. This is indicative of free gold in the samples that are amenable to gravity concentration.

Table 13-3: Marban PFS Samples Screened Metallics Head Grades

Sample	Head Grade (g/t Au)	Distribution in +106 µm Fraction	
		Au (%)	Mass (%)
NPC	1.29	12.1	6.0
MSC	1.61	34.4	4.6
MNC	1.02	14.1	5.8
NZC	1.41	7.1	5.2
MPLG	0.41	9.7	6.0
MPMG	0.88	1.5	5.5
MPHG	3.90	8.0	4.6
NPHG	2.76	6.7	5.4
MPVHG	12.2	12.1	6.0
MPMGK	0.77	19.5	5.2
NPLG	0.37	15.4	5.3
NPMG	1.38	9.4	4.6
NPVHG	8.92	9.9	4.5
MPD	0.87	14.4	4.8
MB	1.13	6.3	6.0
GMZC	1.15	3.8	6.0
KNMG	1.20	13.0	4.2

Source: BaseMet, 2022.

The head analysis of the samples is shown in Table 13-4 and the bulk mineralogy of the three Marban and Norlartic primary composites is shown in Table 13-5. Sulphur occurs primarily as sulphide sulphur and is associated predominantly with pyrite. Pyrrhotite is also found in the Marban pit samples. Copper concentrations were low and below where they would be considered cyanide consuming. Arsenic was present in minor amounts. The Master Composite sample is a blend of 50% MSC, 30% MNC and 20% NPC samples used for the bulk leach test for cyanide detoxification testing.

Table 13-4: Marban PFS Sample Assays

Sample	Element					
	Au (g/t)	Ag (g/t)	S (%)	$S^{=}$ (%)	As (g/t)	Cu (g/t)
MNC	0.96	0.2	0.85	0.80	7	123
MSC	0.67	<0.1	0.39	0.38	8	91
NPC	1.00	0.3	0.46	0.45	6	48
Master Comp	0.70	0.2	0.66	0.61	-	-
MPD	1.11	<0.1	0.45	0.43	4	31
MB	1.23	<0.1	0.44	0.41	2	85
NPLG	0.49	0.3	0.24	0.22	1	81
MPLG	0.28	<0.1	0.45	0.37	2	87
MPMG	0.72	<0.1	0.40	0.37	2	77
MPMGK	0.92	0.3	0.64	0.59	7	46
KNMG	0.96	<0.1	0.30	0.29	<1	40
NPMG	2.40	<0.1	0.70	0.68	1	70
NPHG	3.21	<0.1	1.28	1.26	1	43
MPHG	4.06	<0.1	1.05	1.02	<1	218
NPVHG	13.1	0.6	1.14	1.11	5	55
MPVHG	10.9	0.9	1.28	1.26	<1	91
KC	1.62	0.5	0.70	0.66	5	41
NZC	1.09	0.6	0.30	0.28	7	14
GHZC	0.88	0.2	0.36	0.32	< 1	23
MCOM 1	2.64	0.2	0.69	0.66	6	106
MCOM 2	0.52	<0.1	0.30	0.27	100	74
MCOM 3	1.00	<0.1	1.41	1.38	11	123
MCOM 4	1.07	1.6	0.28	0.16	5	53
NCOM 1	1.46	0.1	0.88	0.85	8	34
NCOM 2	1.02	0.2	0.43	0.40	2	25

Source: BaseMet, 2022.

Table 13-5: Marban PFS Samples Mineral by Mass Percentage

Mineral	Primary Composite Sample Mineral Abundance (wt.%)		
	MNC	MSC	NPC
Pyrite	0.82	0.56	0.83
Pyrrhotite	0.73	0.31	0.00
Other Sulphides	0.01	0.02	0.00
Quartz	16.2	19.9	7.70
Plagioclase	29.0	20.6	49.5
Sericite/Muscovite	0.07	0.48	0.29
Biotite	0.47	0.39	0.52
Chlorite	33.8	33.5	24.2
Amphibole	3.02	6.53	1.45
Talc	0.02	0.35	3.56
Clays	1.29	0.91	0.21
Other Silicates	0.92	2.61	0.21
Oxides	1.31	0.98	1.44
Calcite	11.4	5.95	8.89
Dolomite	0.19	5.52	0.33
Ankerite	0.63	1.13	0.13
Apatite	0.23	0.16	0.68
Other	0.05	0.01	0.03
Total	100	100	100

Source: BaseMet, 2022.

13.4 Comminution Testing

13.4.1 Historical Testing

Comprehensive comminution testing was completed in the 2016 SGS report referenced at the beginning of this section. Although the title states that 10 samples were tested, 16 samples (10 Marban and 6 Norlartic) were tested throughout the two phases of the program. Testing included:

- JK Drop Weight and Steve Morrell Comminution (SMC) tests;
- Bond low-energy impact (crusher work index or CWi) tests;
- Bond rod mill work index tests; and
- Bond ball mill work index (BWi) tests.

The locations of the drillholes used for the samples in this program were reviewed against the 2020 PEA open pit shells. Six samples were found to fall in the Marban open pit and six withing the Norlartic open pit. The results for these samples were suitable for use in the PFS with the exception of the bond BWi test results, which were conducted with too fine a closing size for the grind selected.

13.4.2 PFS Comminution Testing

The PFS comminution testing program was designed to supplement the 2016 results. The testing included BWi and bond abrasion index tests. In total, nine samples were tested. The four Marban and two Norlartic comminution samples were tested for both tests while samples KC, NZC, and GHZC were tested only for BWi. The standard BWi test was performed each composite with a closing screen aperture of 150 µm. A summary of the results is shown in Table 13-6.

Table 13-6: Marban PFS Comminution Testing Summary

Test	Units	Average	75th percentile	Minimum	Maximum
Abrasion Index	g	0.194	0.346	0.038	0.419
Bond Ball Mill Work Index	kWh/t	13.2	14.1	9.6	14.6

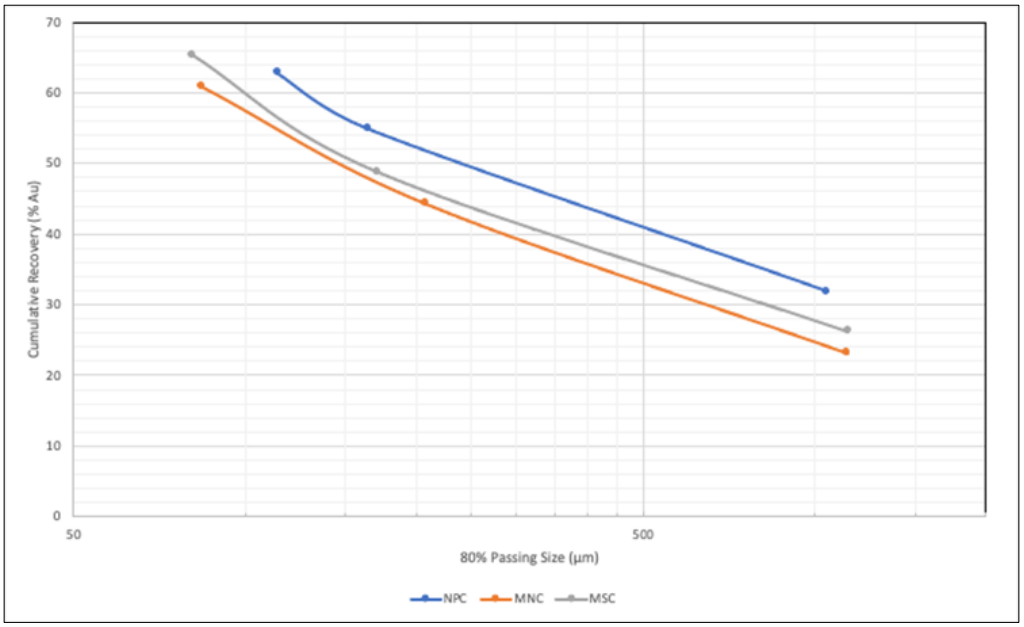
Source: BaseMet, 2022.

The BWi results are characterized as medium hard (75th percentile) while the abrasion index results are characterized as moderately abrasive.

13.5 Gravity Separation

Extended gravity recoverable gold (E-GRG) tests were completed on the two Marban and Norlartic primary composite samples. The three stage test passes sample initially crushed to –1.7 mm and then passed through a 76 mm diameter Knelson concentrator. This is repeated twice with the concentrator tailings ground to target values of 80% passing (k_{80}) = 250 µm and 75 µm in each stage. The test results showed high levels of gravity recoverable gold (GRG) which is indicative of plant scale gravity gold amenability. The E-GRG test results shown in Figure 13-3. The E-GRG test results demonstrate that samples are amenable to gravity concentration in the grinding circuit to remove coarse free gold prior to leaching.

Figure 13-3: Cumulative E-GRG Recovery as a Function of Grind Size



Source: BaseMet, 2022.

Gravity separation testwork program was also completed on all samples (primary composites, variability samples and master composite) to investigate the efficacy of gravity-concentration and provide gravity tailings for leach testing. The laboratory gravity separation testwork equipment included a Knelson concentrator followed by a Mozley table to upgrade the concentrates at a k_{80} of 150 µm for each sample. The target mass recovery of 0.05% aligns with typical full scale plant recovery.

Results indicated highly variable gravity recoveries with range spanning from 9% to 41%, with an average gold recovery of 26%.

13.5.1 Primary Composites Leach Optimization Tests

Initial leach testing examined the impact of grind and retention time on gravity tailings. Conditions included:

- Cyanide (NaCN) concentrations of 0.5, 1.0, and 2.0 g/L;
- Slurry density 40% w/w solids;
- pH range 10.5 to 11.0;
- 48-hour retention time (kinetic samples at 4, 12, 24, and 48 hours); and
- Grind targets (k_{80} values) from 50 to 125 µm.

Optimum conditions included grinding to k_{80} of 105 µm, 24 hours retention time and cyanide concentration of 1.0 g/L. Tests at the lower concentration of 0.5 g/L resulted in either reduced leach kinetics or lower recovery. Pre – aeration was found to not reduce reagent consumptions nor improve recoveries and was not carried forward. Averages of gravity/leach test results are shown in Table 13-7 for the primary composite leach test results at the optimized conditions.

Table 13-7: Gravity/Leach Test Results for Primary Composites

Sample	Size P_{80} (µm)	Regent Cons. kg/t Feed		Recovery (% Au)		Residue Au (g/t)	Head Grades (g/t Au)	
		NaCN	CaO	Gravity	Total		Calc.	Direct
NPC	108	0.38	1.27	24.1	91.2	0.09	0.99	1.00
MNC	107	0.76	1.29	21.7	95.8	0.03	0.81	0.96
MSC	108	0.67	1.30	25.9	96.7	0.02	0.64	0.67

Source: BaseMet, 2002.

13.5.2 Variability Sample Testing

The variability samples were all tested using the optimized conditions from the primary composites. The results are shown in Table 13-8.

Gravity recoveries are substantial for each sample indicating the inclusion of gravity concentration in the plant flowsheet will yield consistent high recoveries. The results all show moderate to low cyanide and lime consumptions. In general, leach residue grades increased with increasing head grades.

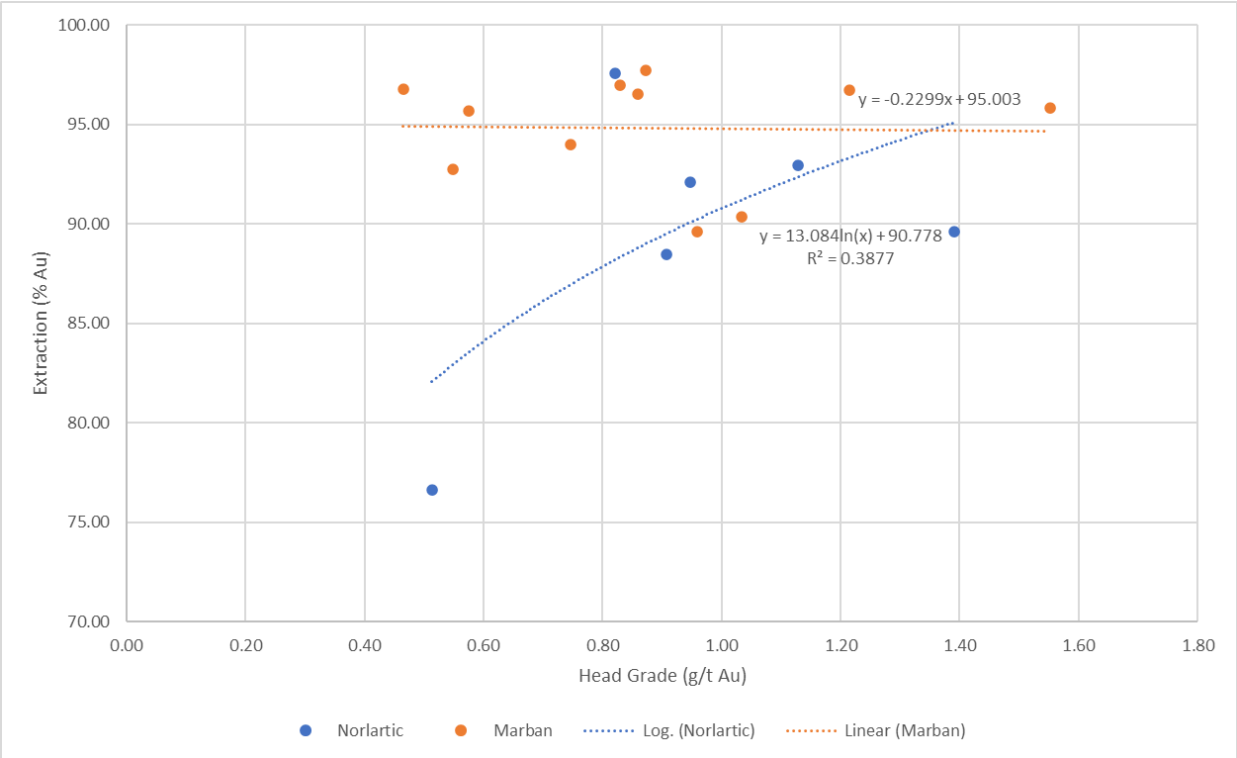
The results for all leach tests are shown in Figure 13-4 with leach residue grades shown as a function of head grade. The results show a strong relationship between these parameters. The Norlartic samples result in higher leach residue grades than the Marban samples for similar head grades. Cyanide and lime consumptions are low.

Table 13-8: Marban and Norlartic Variability Samples Gravity/Leach Test Results

Sample	Size P_{80} (µm)	Regent Cons. kg/t Feed		Recovery (% Au)		Residue Au (g/t)	Head Grades (g/t Au)	
		NaCN	CaO	Gravity	Total		Calc.	Direct
MPHG	105	0.14	0.75	36.4	94.7	0.16	2.93	4.06
MPMG	105	0.20	0.78	19.9	90.4	0.10	1.03	0.72
MPMGK	105	0.23	0.82	25.4	96.7	0.04	1.21	0.92
MPLG	105	0.17	0.79	15.5	92.7	0.04	0.55	0.28
MB	105	0.13	0.86	32.4	95.8	0.07	1.55	1.23
MPD	105	0.11	0.72	33.5	89.6	0.10	0.96	0.11
MPVHG	105	0.18	0.75	40.8	94.4	0.48	8.47	10.9
KNMG	105	0.17	0.66	26.9	97.6	0.02	0.82	0.96
NPVGH	105	0.13	0.81	17.5	89.2	0.65	5.97	13.1
NPHG	105	0.14	0.8	28.5	94.0	0.20	3.31	3.20
NPMG	105	0.14	0.68	34.2	89.6	0.15	1.39	2.40
NPLG	105	0.14	0.77	20.4	76.6	0.12	0.51	0.49

Source: BaseMet, 2022.

Figure 13-4: Gravity/Leach Test Results for All Samples



Source: Ausenco, 2022.

13.6 Cyanide Detoxification

Cyanide destruction tests were performed using the SO_2/air process. Process development testing for the SO_2/air process is completed in two stages. The first stage is batch testing, followed by second stage continuous testing. A 1 L reactor was used for both batch and continuous tests. For the continuous tests, an overflow nozzle on the reactor transferred treated slurry to a storage tank.

The cyanide detox program was conducted in two parts. One part was completed with a target CN_{WAD} concentration of <25 mg/L CN_{WAD}, whereas the other targeted a CN_{WAD} concentration of <1 mg/L CN_{WAD}.

The first part of the program included eight runs to evaluate retention time, and reagent addition rates. All tests were run at a slurry density of 40% w/w solids. The optimal results were achieved in the final two runs with the following results.

- Retention time = 20 – 21 minutes;
- SO_2:CN_{WAD} addition rate = 2.5:1 equivalent;
- Hydrated lime addition rate = 0.4 – 0.8 CaO:CN_{WAD} equivalent;
- Copper addition = 25 mg/L Cu^{2+} solution concentration; and
- Discharge average cyanide concentrations: 7.9 mg/L CN_{WAD} and 13.8 mg/L CN_T.

The second part of the program included seven runs to evaluate retention time, and reagent addition rates. All tests were run at a slurry density of 40% w/w solids. The optimal results were achieved in the final two runs with the following results.

- Retention time = 83 –107 (average 95 minutes) minutes;
- SO_2:CN_{WAD} addition rate = 7.5:1 equivalent;
- Hydrated lime addition rate = 4.7 – 6.3 CaO:CN_{WAD} equivalent;
- Copper addition = 50 mg/L Cu^{2+} solution concentration; and
- Discharge average cyanide concentrations: 0.6 mg/L CN_{WAD} and 0.9 mg/L CN_T.

13.7 Solid Liquid Separation

Thickener testwork was performed on Master Composite detoxified final leach tailings. Both static and dynamic tests were performed.

Dynamic settling tests were conducted to determine thickener sizing parameters for the project. Feed characterization is presented in Table 13-9.

Table 13-9: Thickener Feed Sample Characterization

Sample	Solids SG	k_{80}(1)	k_{80}(2)	<5 µm(2)
Master Composite	2.85	105	91	11%

Notes: 1. Measured by sieve analysis, 2. Measured by laser analyzer. Source: BaseMet, 2022.

Flocculant AN913SH (manufactured by SNF) was selected in static screening tests and used for dynamic settling tests. These tests were all performed initially targeting a natural pH and using 15% w/w solids concentration for the feed slurry. The highest underflow density achieved was 59.7% w/w solids, with a settling rate of 0.5 t/m²/h at natural pH and 30 g/t flocculant addition, with an acceptable overflow clarity of 139 mg/L. Dynamic settling results are shown in Table 13-10.

Table 13-10: Master Composite Dynamic Settling Test Results

Parameter	Test No.					
	D1-A	D1-B	D1-C	D1-D	D1-E	D1-F
Unit Area (t/m²/h)	0.5	0.7	1.0	0.7	0.7	0.5
Rise Rate (m/h)	3.1	4.3	6.2	4.4	4.3	3.1
Flocculant Dosage (g/t)	30	30	30	40	20	40
pH	7.8	7.8	7.8	7.8	7.8	7.8
Underflow Density (%w/w solids)	59.7	57.8	53.9	57.5	55.9	49.9
Overflow Clarity (mg/L)	139	150	100	64	148	100

Source: BaseMet, 2022.

13.8 Gold Recovery

The gravity leach test results were analyzed to provide a recovery model for use with the mine production schedule to provide gold recovery and production data. In addition to the predicted extraction, plant losses were estimated at 0.5% of head gold, including soluble gold solution, fine carbon losses to tailings.

Recoveries for material from the Marban and Kierens pits are estimated as 94.9% of feed gold based on the recovery model derived from the gravity leach test results. The Norlartic pit recoveries are estimated based on the equation presented below using a fixed tails grade of 0.09 g/t Au.

$$Norlartic\ Recovery = \frac{Head\ grade\ (g/t) - 0.09}{Head\ Grade} * 100 + 1.5\% - 0.5\%$$

A grind sensitivity trade-off was performed to determine the optimal grind size by accounting for the change in recoveries. A regression analysis on leach testwork conducted at multiple grind sizes estimated that the plant will recover an additional 2.1% of the gold present in Norlartic material at 85 µm compared to 100 µm. A summary of this analysis is presented in Table 13-11. The selected grind size for the Marban Project was a P_{80} of 85 µm. A lower value of 1.5% was added to the Norlartic recovery formula above for the PFS due to the limited data available.

Table 13-11: Grind Size Optimization Trade-off Result

Description	65 µm		85 µm		100 µm	
	Marban and Kierens	Norlartic	Marban and Kierens	Norlartic	Marban and Kierens	Norlartic
Average Recovery (%)	95.1	95.8	94.5	93.1	94.1	91.0
Cyanide Consumption (kg/t)	1.1	0.8	1.2	1.0	1.2	1.1
Lime Consumption (kg/t)	1.8	1.7	1.3	1.4	0.8	1.0

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

The MRE of the Marban and Kierens-Norlartic deposits presented herein represents an update from the previous MRE issued in the 2020 Marban Preliminary Economic Assessment, completed by Moose Mountain Technical Services (MMTS).

This MRE is based on the updated drillhole database, which includes additional data from the 2020 exploration and 2021 infill drilling programs completed since the previous MRE. Most of the drilling was dedicated to infill drilling to convert inferred resources to indicated category for inclusion in the Prefeasibility Study (PFS).

The MRE update was produced by Mr. James Purchase, P. Geo of GMS, Vice-president of Geology and Resources, an independent QP as defined in National Instrument 43-101. Mr. Purchase visited the Marban Engineering Project in September 2021 to review the geological data, infill drilling program, and sampling protocols.

The MRE was prepared in accordance with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) Definition Standards for Mineral Resources and Mineral Reserves (2014), and in accordance with CIM Guidelines (2019) for Estimation of Mineral Resources and Reserves.

The effective date of the mineral resource estimation is 27 February 2022 and the MRE statement is listed in Table 14-1.

Table 14-1: Mineral Resource Estimate – Effective Date February 27, 2022

Deposit		Indicated			Inferred		
		Tonnes (kt)	Gold (g/t)	Ounces (koz)	Tonnes (kt)	Gold (g/t)	Ounces (koz)
Open Pit (OP)	Marban	52,437	1.03	1,736	1,038	0.97	32
	Kierens - Norlartic	14,795	1.22	582	1,068	1.42	49
	Total	67,232	1.07	2,318	2,106	1.20	81
Underground (UG)	Marban	162	4.47	23	860	4.43	123
	Kierens - Norlartic	297	3.36	32	182	3.36	20
	Total	460	3.75	55	1,043	4.25	142
Combined Mineral Resources - OP and UG		67,692	1.09	2,374	3,149	2.21	223

Notes: **1.** The mineral resources described above have been prepared in accordance with the CIM Standards (Canadian Institute of Mining, Metallurgy and Petroleum, 2014) and follow Best Practices outlined by the CIM (2019). **2.** The QP for this mineral resource estimate is James Purchase, P. Geo of G Mining Services Inc. Mr. Purchase is a member of L'Ordre des Géologues du Québec (#2082). **3.** The effective date of the mineral resource estimate is 27 February 2022. **4.** The lower cut-off used to report open-pit mineral resources is 0.30 g/t Au. Underground mineral resources have been reported using a 3.0 g/t lower cut-off at Marban, and a 2.5 g/t lower cut-off at Kierens-Norlartic. **5.** The Marban and Kierens-Norlartic deposits have been classified as indicated and inferred mineral resources according to drilling spacing and estimation pass. No measured resource has been estimated. Underground mineral resources have been categorized manually to remove isolated areas and have been reported using a 3 m minimum thickness. **6.** Known underground works have been incorporated into the block model, and zero density has been assigned to the blocks located within the voids. **7.** The density has been applied based on measurements taken on drill core and assigned in the block model by lithology. **8.** In general, a minimum thickness of 3 m was used when interpreting the mineralized bodies. **9.** The MRE is based on subblock models with a main block size of 5 m x 5 m x 5 m, with subblocks of 2.5 m x 2.5 m x 2.5 m. **10.** Tonnage have been expressed in the metric system, and gold metal content has been expressed in troy ounces. **11.** The tonnages have been rounded to the nearest 1,000 tonne and the metal content has been rounded to the nearest 1,000 ounce.

The open-pit mineral resources are stated using a lower cut-off grade of 0.30 g/t Au, using a whittle shell at a US$1900/oz Au price. The underground mineral resources are reported using a 3.0 g/t lower cut-off at Marban, and a 2.5 g/t lower cut-off at Kierens-Norlartic. At these cut-offs, the total indicated mineral resource is estimated at 67,692 thousand tonnes (kt) at a

grade of 1.09 g/t Au for a total of 2,374 thousand ounces (koz), and inferred mineral resource is estimated at 3,149 Kt at a grade of 2.21 g/t Au for a total of 223 koz.

Mineral resources are not mineral reserves and have not demonstrated economic viability. There is no certainty that all or any part of the mineral resource will be converted into mineral reserve.

The QP is not aware of any factors or issues that materially affect the MRE other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors and additional risk factors regarding indicated and inferred resources.

The database used to estimate the mineral resources of Marban and Kierens-Norlartic deposits was validated by the QP. The QP considers that the current drilling information is sufficiently reliable to interpret with confidence the boundaries for gold mineralization. and that the assay data are sufficiently reliable to support mineral resource estimation and block modelling.

The following sections describes the mineral resource estimation methodology and summarizes the key assumptions considered by the QP.

14.2 Estimation Methodology

The mineral resources reported herein have been interpolated into a sub-block model using the modelled mineralized zones for each deposit, Marban and Kierens-Norlartic.

The resource estimation methodology is summarized with the following procedures:

- Drillhole database validations and selection of the drillholes for the resource estimation database;
- 3D modelling of mineralized wireframes based on mineralization style and gold grades greater than 0.3 g/t Au;
- Geostatistical analysis: capping, compositing and variography;
- Block modelling and grade estimation;
- Resource classification and grade interpolation validations; and
- Cut-off grade sensitivities.

14.3 Resource Database

In order to complete an updated MRE for the Marban Engineering Project, a database comprising a series of comma-delimited spreadsheets containing information for the Marban project was provided to GMS on 14 January 2022. The database included drillhole collar information (NAD83/UTM, Zone 18), surveys, assays, lithological, alteration, structural and density data.

The current MRE is derived exclusively from an updated version of the resource database which excludes historical data exhibiting DL and bias issues, as described in the recommendations outlined in Section 12.

The total number of holes completed for the project is presented in Section 10.

The vast majority of this MRE is covered by drillholes spaced 25–50 m apart. The MRE is based on 1,743 drillholes totalling 401,178 m, of which 323,325 m were assayed. A breakdown by operator is provided in Table 14-2

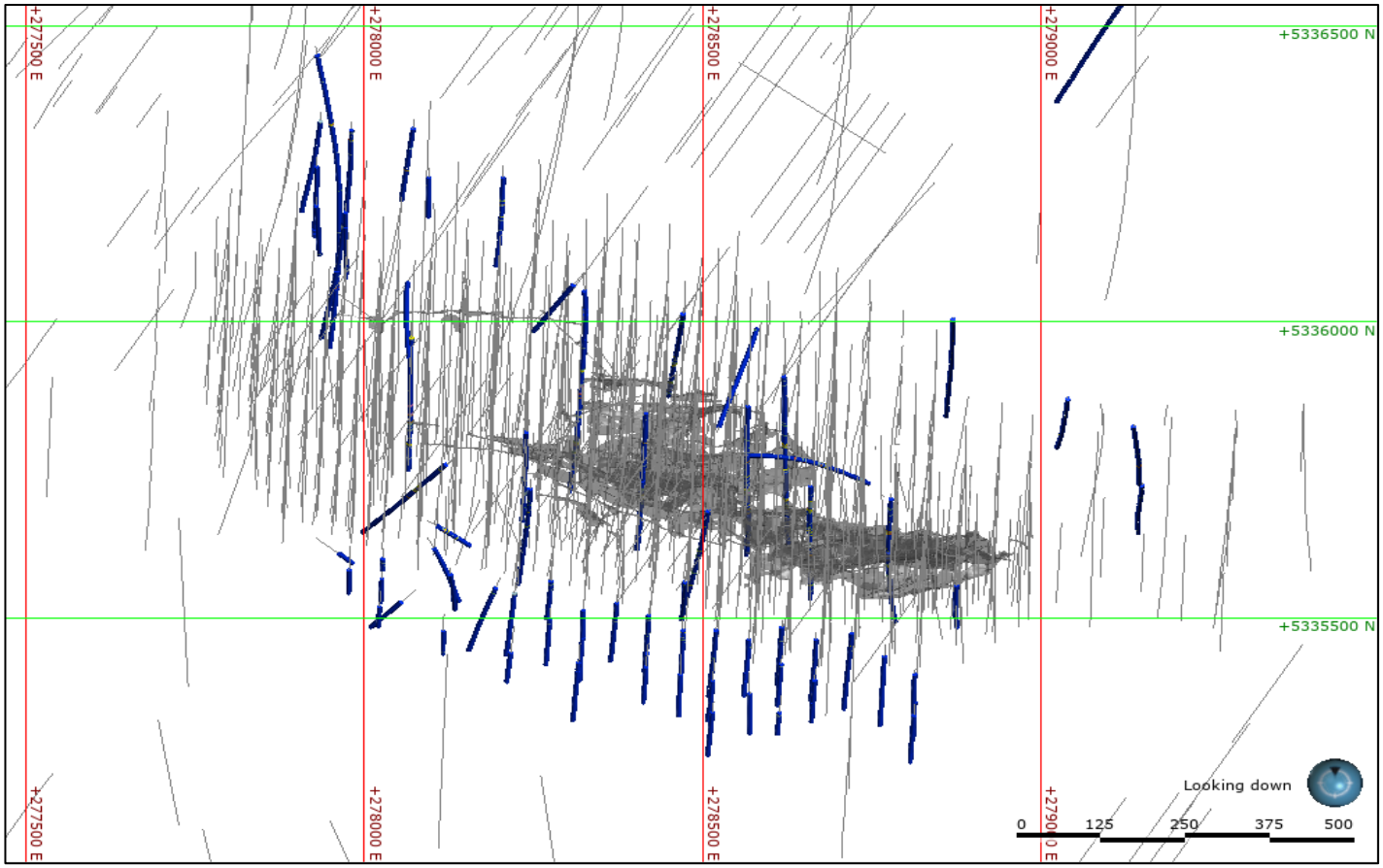
Since the previous MRE, O3 Mining has completed a comprehensive infill drilling program which was primarily designed to convert inferred resources outlined in the 2020 PEA into indicated category. The 2021 drilling program completed by O3 Mining on the Marban and Kierens-Norlartic deposits are shown in Figure 14-1 and Figure 14-2, respectively.

Table 14-2: Summary of Drillholes and Assays used for the Marban Resource Estimate

Owner	Period	Type	Excluded from MRE			Included in MRE		
			Number of holes	Total length (m)	Assayed length (m)	Number of holes	Total length (m)	Assayed length (m)
Various	Pre-1994	Surface and Underground	5,400	572,938	144,291	914	140,428	98,177
NioGold	2006-2015	Surface	314	75,901	52,021	640	205,315	177,920
Osisko	2016-2017	Surface	10	5,295	4,006	16	9,876	6,724
O3	2020-2022	Surface	126	44,506	31,332	173	45,558	40,504

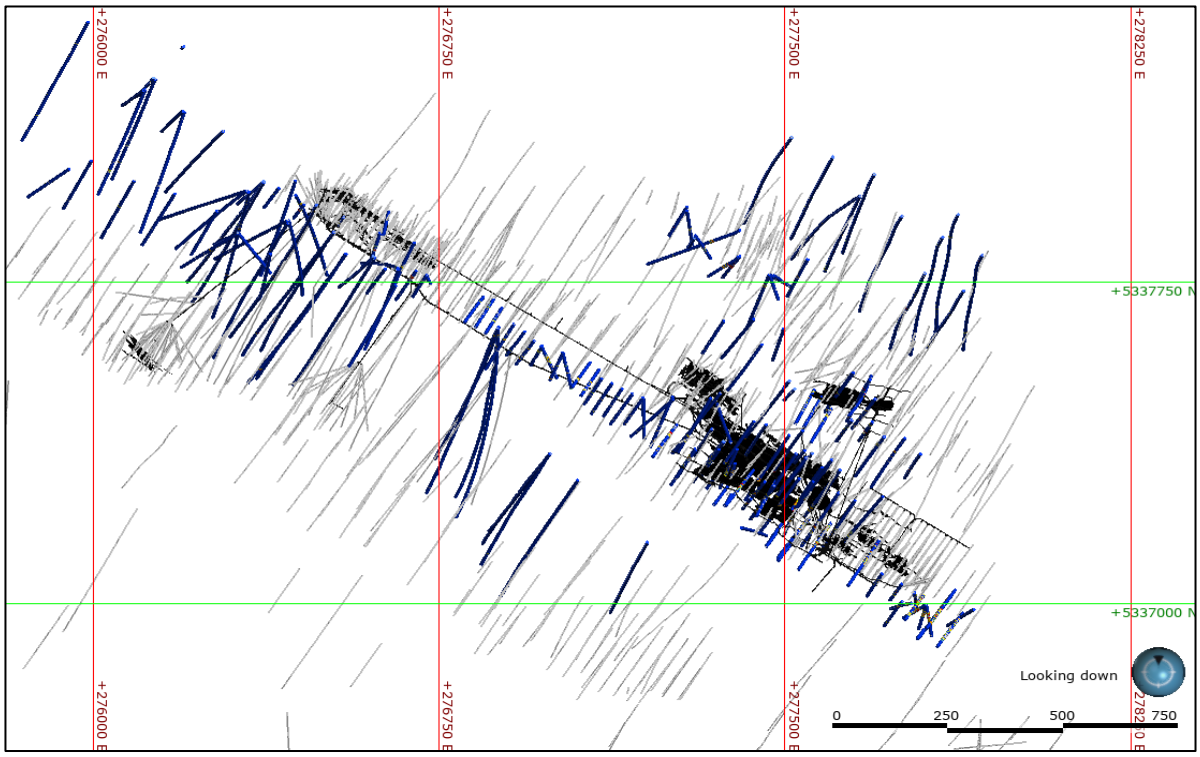
Note: Included in MRE drillholes are those which intersect at least one of the wireframes. Excluded from MRE are holes that either do not traverse any wireframes, or were excluded from the MRE due to data bias issues described in Section 12.

Figure 14-1: Plan View of 2021 Drilling Campaign and Existing Historical Stopes – Marban



Source: GMS, 2022

Figure 14-2: Plan View of 2021 Drilling Campaign and Existing Historical Stopes – Kierens-Norlartic



Source: GMS, 2022

14.4 Modelling

14.4.1 Geological Model

The updated MRE is based on 29 mineralized zones in the Marban deposit and 18 mineralized zones in the Kierens-Norlartic deposit. All wireframing and geological interpretation was prepared by using Leapfrog GEO™.

For both deposits, a lithology model was provided by O3 Mining which was subsequently reviewed by GMS and used to code the appropriate bulk densities into the block model according to the respective lithological unit.

The overburden surface was created using logged drillhole intercepts in the drilling database. All mineralized wireframes were constrained by the base of overburden material.

The mineralization wireframes were created through a joint interpretation with by both O3 Mining and GMS geological teams to reflect the folded nature of the mineralization at Marban, and in accordance with the shape of the historical mining stopes and the historical underground mapping.

Most of the mineralization at Marban consists of quartz–calcite–chlorite veinlets associated with disseminated pyrite, pyrrhotite and locally visible gold. Mineralization is hosted within or near iron-rich basalts.

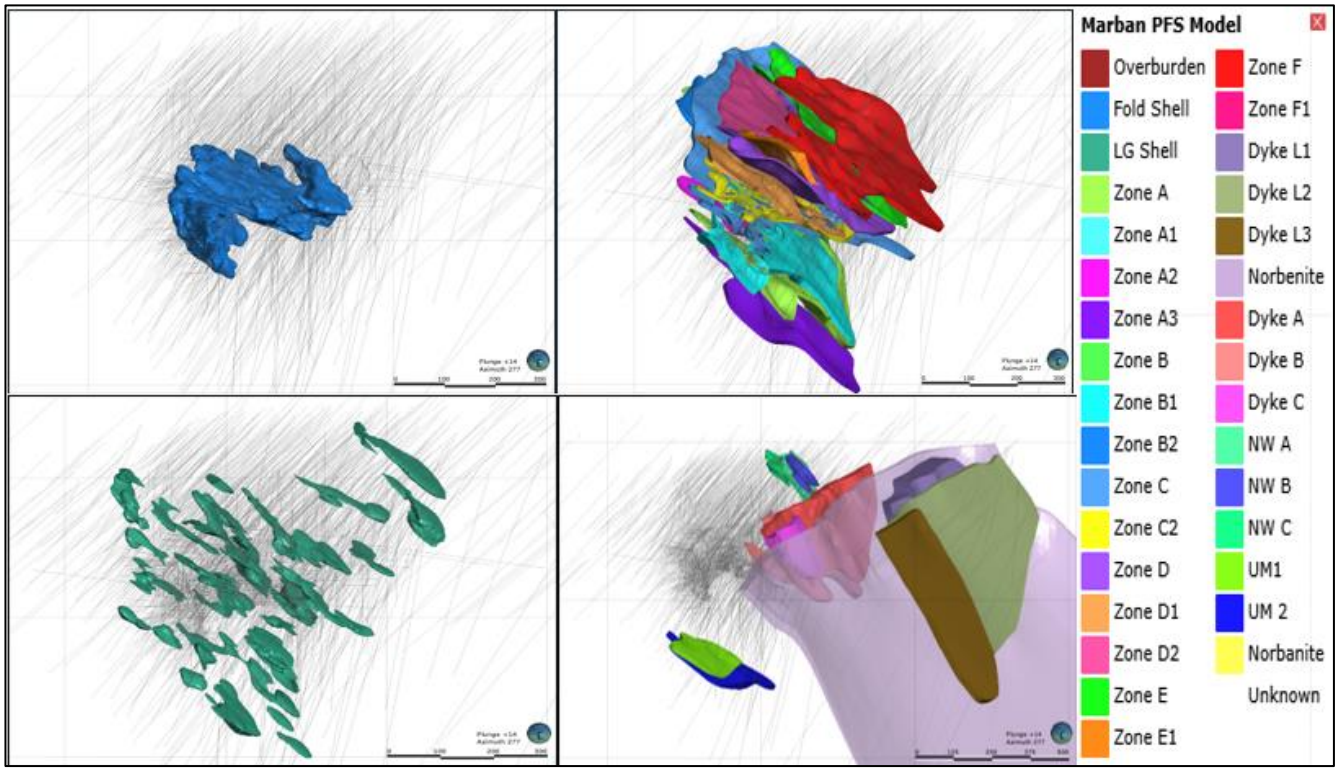
Mineralization along the Kierens-Norlartic zones is hosted within or near an albitized and chlotitized intermediate dyke swarm. The North-North zone mineralization consists of quartz–tourmaline veinlets within a felsic albitized dyke.

For both deposits, efforts were made with the new geological interpretation to minimize internal dilution within wireframes, whilst generally applying a minimum thickness of 3 m.

The Marban mineralized domains are described below, and shown in Figure 14-3:

- Fold Shell — the folded and higher-grade portion of the deposit, which encompasses the mined-out portion of the deposit. This domain ensures that data is shared between the various flanks of the folds during the interpolation process.
- Planar Zones — represents the more extensive shear zones that generally follow the axial planes of the various folds.
- Low-grade Shell — represents mineralization which exhibits continuity over a minimum of three drillholes over two sections.
- Periphery Zones — represents the Marban Dyke domains to the north of the deposit, and some deeper ultramafic domains below the pit outline.

Figure 14-3: Isometric Views of the Three Groups of Mineralized Domains, Looking West



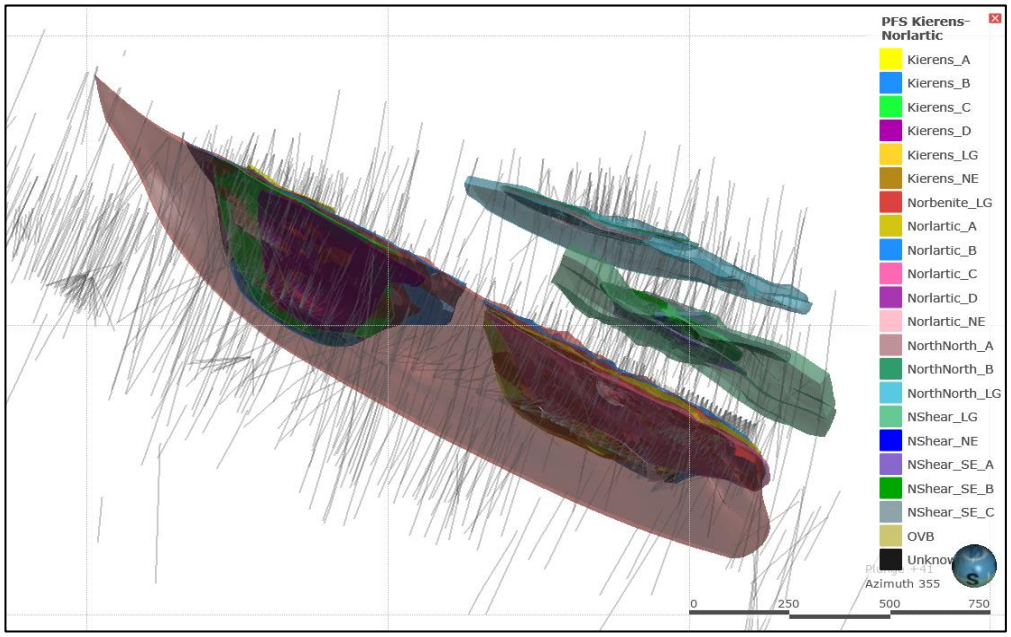
Note: Top Left: Fold Shell, Top Right: Planar Zones, Bottom Left: Low-Grade Shell, Bottom Right: Periphery Zones. Source: GMS, 2022

The Kierens-Norlartic mineralized domains are grouped as follows:

- Kierens A, B, C, D, and NE Domains;
- Norlartic A, B, C, and NE Domains;
- North-North A and B;
- North-Shear NE, SE-A, SE-B, and SE-C; and
- Low-grade (LG) Domains Norbenite, North-North, and North-Shear.

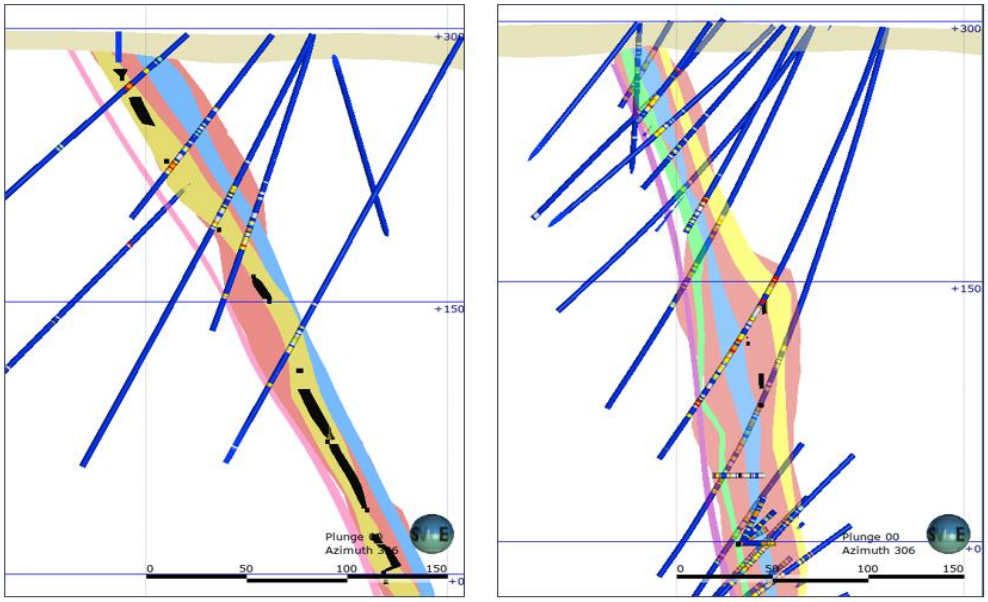
The Kierens-Norlartic mineralization domains were re-interpreted in 2022 and are illustrated in Figure 14-4 and Figure 14-5. It was possible to model narrower, higher-grade domains (> 2.0 g/t Au) at both Kierens and Norlartic, with a lower-grade envelope used in the background to capture any lower-grade intersections.

Figure 14-4: 3D View of Mineralized Domains Model – Kierens-Norlartic



Source: GMS, 2022

Figure 14-5: Typical Section of Kierens (right) and Norlartic (left) Mineralized Domains



Source: GMS, 2022

14.4.2 Topography Surface

A topographic surface was generated from LiDAR data provided by O3 Mining. Surface drillholes were draped onto the surface to ensure historic holes and recent holes honour the LiDAR data.

14.5 Assays, Capping, and Compositing

Gold assay capping was undertaken by GMS before compositing original assay lengths for each domain. GMS used a combination of Parrish Analysis (decile analysis) and cumulative probability plots (CPP's) to determine capping levels. Parrish analysis was used first to determine if capping was warranted for each grouped domain, and if warranted, capping thresholds were chosen from CPPs.

In total, 46 and 56 samples for the Marban and Kierens-Norlartic deposits respectively were capped. The metal loss factor (based on a length-weighted accumulation of grade) obtained from the capped gold grades is 16.4% for Marban and 3.9% for Kierens-Norlartic. At first glance this appears high for Marban, but a single high-grade assay (2,610 g/t) is responsible for the majority of the metal loss at Marban. Capping levels are generally more conservative than those used in the previous MRE; however, GMS does not intend to apply a high-grade restraint during interpolation. Table 14-3 and Table 14-4 present a summary of the mean, coefficient of variation (CV), and capping values applied by GMS on raw gold assays for each mineralized domain.

Table 14-3: Assay Capping Levels and Metal Loss for the Marban Deposit – All Wireframes

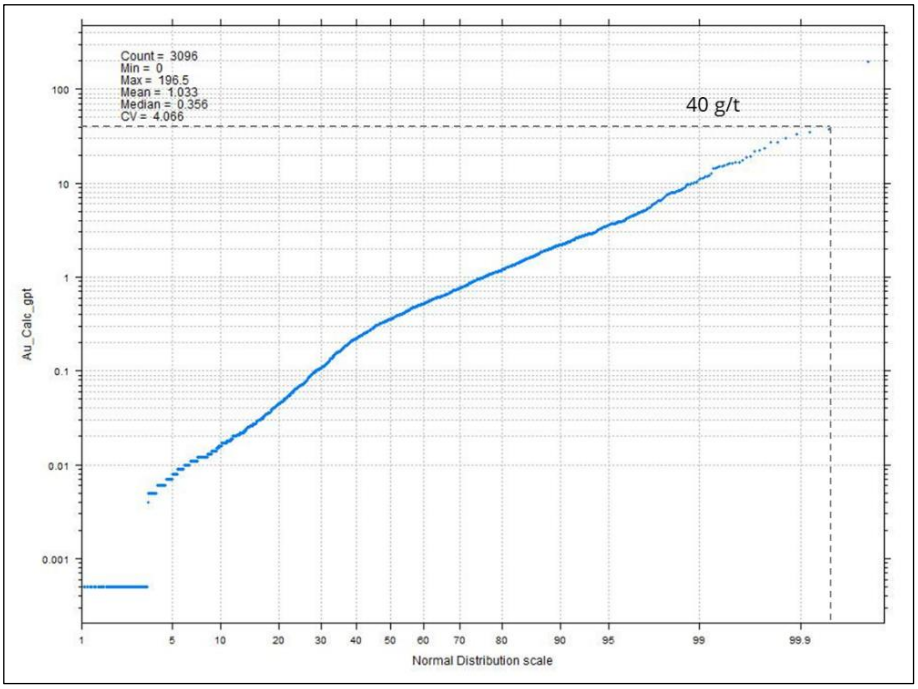
Domain	Num. of Assays	Au Uncapped (g/t)			Num. of Assays Capped	Au Capped (g/t)			Metal Loss (%)
		Max	Mean	CV		Max	Mean	CV	
Fold Shell	5,034	440.00	1.08	6.26	4	40.00	1.02	2.47	5.7
Zone A	3,096	196.50	1.03	4.08	1	40.00	0.97	2.47	5.4
Zone B	2,490	87.17	0.94	2.74	1	40.00	0.92	2.25	2.2
Zone C	2,794	60.62	0.97	2.70	3	40.00	0.96	2.53	0.8
Zone D	2,102	313.11	1.18	6.36	3	40.00	1.06	2.84	9.8
Zone D1	879	37.03	0.66	2.74	3	10.00	0.60	1.91	9.8
Zone E	1,542	259.71	1.66	6.11	5	40.00	1.22	3.10	26.9
Zone F	1,748	58.49	0.64	3.04	1	20.00	0.62	2.32	3.2
LG Shell	2,416	2,610.00	1.74	30.64	9	25.00	0.64	3.22	63.0
Dyke A	1,409	29.25	0.47	3.55	4	14.33	0.43	2.58	7.4
Dyke B	898	31.80	0.70	3.11	3	19.20	0.66	2.58	6.1
Dyke C	423	2.45	0.16	2.43	-	2.45	0.16	2.43	0.0
Dyke L1	561	7.32	0.28	2.42	-	7.32	0.28	2.42	0.0
Dyke L2	177	1.92	0.12	2.55	-	1.92	0.12	2.55	0.0
Dyke L3	24	4.36	0.46	1.96	-	4.36	0.46	1.96	0.0
Norbenite	1,477	64.10	0.32	5.89	4	10.00	0.27	3.03	15.6
NW A	472	123.53	0.83	7.09	2	20.00	0.59	2.98	28.7
NW B	222	35.27	0.63	4.15	-	35.27	0.63	4.15	0.0
NW C	169	6.14	0.50	1.98	-	6.14	0.50	1.98	0.0
UM 1	145	12.45	0.92	2.03	-	12.45	0.92	2.03	0.0
UM 2	231	37.10	1.61	2.62	3	20.00	1.42	1.95	12.0
Total	28,309	2,610.00	0.98	16.61	46	40.00	0.82	2.78	16.4

Table 14-4: Assay Capping Levels and Metal Loss for the Kierens-Norlartic Resource Area

Domain	Num. of Assays	Au Uncapped (g/t)			Num. of Assays Capped	Au Capped (g/t)			Metal Loss (%)
		Max	Mean	CV		Max	Mean	CV	
Kierens A	1,367	463.54	2.00	7.86	6	100.00	1.81	5.06	9.8
Kierens B	3,177	346.29	1.85	4.46	5	70.00	1.75	2.94	5.2
Kierens C	1,578	111.39	1.42	2.96	4	27.00	1.38	2.18	2.9
Kierens D	617	22.86	0.64	3.11	-	22.86	0.64	3.11	0.0
Kierens NE	676	194.71	1.98	4.95	3	50.00	1.83	3.28	7.6
Norbenite LG	14,028	83.60	0.28	4.19	5	21.00	0.28	3.14	2.2
Norlartic A	6,122	192.00	1.44	2.99	5	34.00	1.40	1.94	2.6
Norlartic B	3,965	84.10	1.14	3.15	4	50.00	1.13	2.98	1.2
Norlartic C	1,514	55.30	0.61	3.50	3	20.00	0.60	2.62	2.7
Norlartic NE	455	18.56	1.35	1.72	-	18.56	1.35	1.72	0.0
North-North A	566	39.05	1.28	2.55	2	25.00	1.27	2.31	1.3
North-North B	575	126.00	1.22	5.24	3	25.00	0.92	2.93	24.1
North-North LG	2,486	41.70	0.22	5.54	4	10.00	0.21	3.84	6.1
North-Shear LG	4,878	63.60	0.19	7.60	5	30.00	0.19	6.26	2.0
North-Shear NE	70	3.94	0.25	2.89	-	3.94	0.25	2.89	0.0
North-Shear SE A	276	39.02	0.60	4.68	1	20.00	0.58	3.41	3.0
North-Shear SE B	637	81.89	1.32	5.07	6	40.00	1.22	4.24	7.0
North-Shear SE C	498	16.46	0.36	3.22	-	16.46	0.36	3.22	0.0
Total	43,485	463.54	0.75	6.18	56	100.00	0.72	4.21	3.9

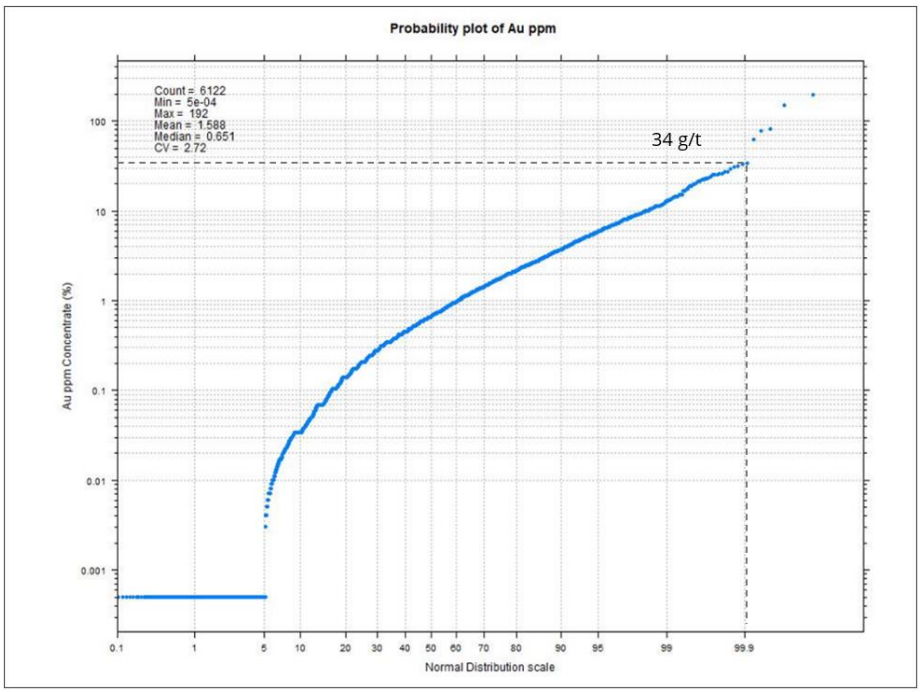
Probability plots were generated for each mineralized domain to establish the capping limits. Figure 14-6 and Figure 14-7 show cumulative probability plots (CPP) of the grouped domain Zone A of Marban deposit and Norlartic A of Kierens-Norlartic deposit.

Figure 14-6: CPP – Grouped Domain A – Marban



Source: GMS, 2022.

Figure 14-7: CPP – Norlartic A – Kierens-Norlartic

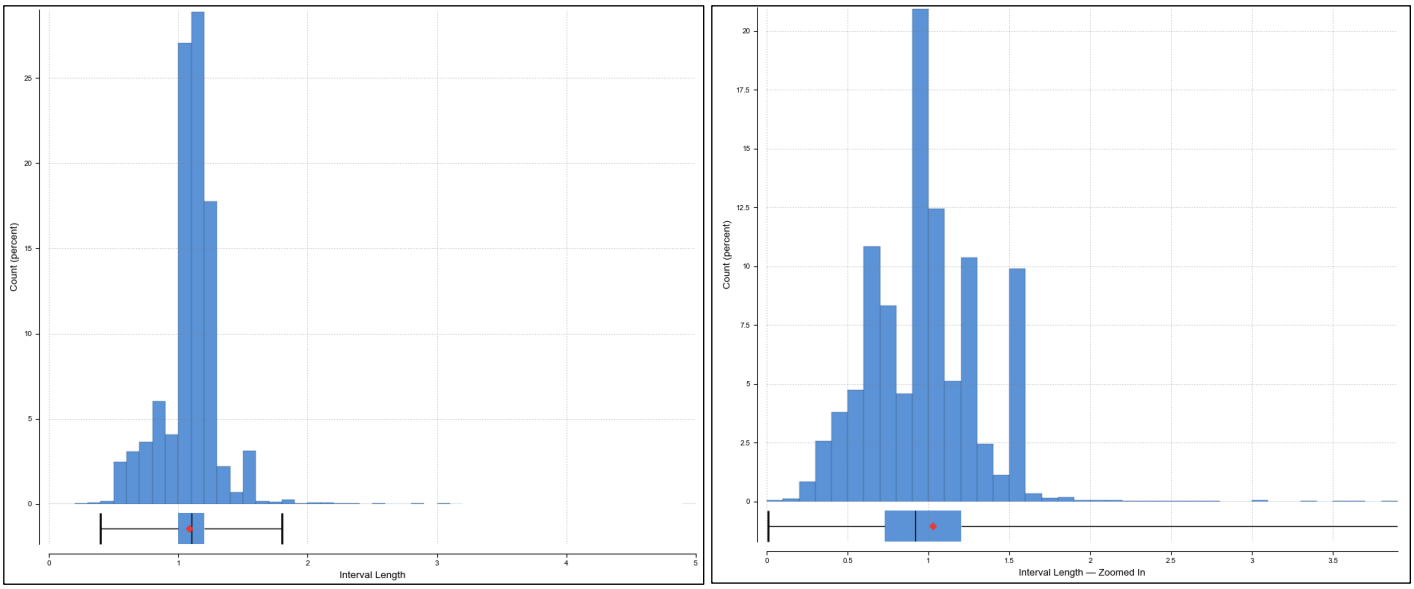


Source: GMS, 2022.

14.5.1 Compositing

In order to standardize the sample lengths used in grade estimation, the capped gold assays of the drillhole data for both deposits were composited. Composites of 1.5 m (downhole) were chosen for all mineralized domains, with composites residuals less than 0.5 m retained and included in the previous composite interval. The composite length was chosen considering the dominant sample length (1.0 m or 1.5 m) and the proposed block size (5 m). Histograms of sample lengths are shown in Figure 14-8.

Figure 14-8: Histogram of Assay Lengths



Note: Left: Marban Resource Area; Right: Kierens-Norlartic Resource Area. Source: GMS, 2022.

A grade of 0.00 g/t Au was assigned to missing sample intervals during compositing; however, unsampled intervals within voids were removed from the composite dataset before grade estimation. Table 14-4 summarizes the basic statistics of the gold composites used for the 2019 MRE.

Statistics of capped gold assays and capped composites by domain are presented in Table 14-5 and Table 14-6. The weighted mean grades for each domain are very comparable, and the CV shows a significant reduction as expected.

Table 14-5: Assay and Composite Statistics of Capped Au – Marban

Domain	Assays				Composites				Difference (%)	
	Num. of Samples	Max Au (g/t)	Wtd. Mean Au (g/t)	CV	Num of Samples	Max Au (g/t)	Wtd. Mean Au (g/t)	CV	Wtd. Mean Au	CV
Fold Shell	5034	40.00	1.02	2.36	3717	24.46	1.02	1.88	-0.08	-20.34
Zone A	3096	40.00	0.97	2.45	2303	26.70	0.97	1.98	-0.15	-19.18
Zone B	2490	40.00	0.92	2.37	1844	39.91	0.92	1.99	-0.23	-15.87
Zone C	2794	40.00	0.96	2.50	2070	29.58	0.96	2.07	-0.07	-17.28
Zone D	2102	40.00	1.06	2.72	1543	39.83	1.06	2.29	-0.10	-15.89
Zone D1	879	10.00	0.60	1.89	641	9.97	0.60	1.52	-0.08	-19.50
Zone E	1542	40.00	1.22	2.99	1149	32.96	1.21	2.35	-0.27	-21.27
Zone F	1748	20.00	0.62	2.28	1293	10.96	0.62	1.83	-0.09	-19.86
LG Shell	2416	25.00	0.64	3.16	1804	20.01	0.64	2.41	-0.88	-23.87
Dyke A	1409	14.33	0.43	2.48	1026	10.09	0.43	2.05	0.29	-17.28
Dyke B	898	19.20	0.66	2.53	668	15.35	0.66	2.15	-0.22	-15.08
Dyke C	423	2.45	0.16	2.29	322	2.22	0.16	1.88	0.09	-17.88
Dyke L1	561	7.32	0.28	2.42	416	6.82	0.28	2.13	-0.32	-11.91
Dyke L2	177	1.92	0.12	2.50	132	1.37	0.12	2.13	-0.35	-14.83%
Dyke L3	24	4.36	0.46	1.92	20	2.52	0.46	1.46	-0.74	-24.2
Norbenite	1477	10.00	0.27	3.14	1068	9.15	0.27	2.79	0.26	-11.20
NW A	472	20.00	0.59	3.01	362	13.85	0.59	2.44	-0.36	-18.86
NW B	222	35.27	0.63	3.62	165	19.53	0.62	2.66	-0.52	-26.51
NW C	169	6.14	0.50	1.99	133	4.85	0.51	1.70	1.84	-14.43
UM1	145	12.45	0.92	2.07	103	10.98	0.92	1.80	-0.04	-13.04
UM 2	231	20.00	1.42	1.96	167	12.97	1.42	1.47	-0.01	-24.77

Note: Wtd. = weighted mean by length; CV = coefficient of variation

Table 14-6: Assay and Composite Statistics of Capped Au – Kierens-Norlartic

Domain	Assays				Composites				Difference (%)	
	Num of Samples	Max Au (g/t)	Wtd. Mean Au (g/t)	CV	Num of Samples	Max Au (g/t)	Wtd. Mean Au (g/t)	CV	Wtd. Mean Au	CV
Kierens A	1,367	100.00	1.81	5.06	869	58.03	1.81	3.26	0.00	-35.57
Kierens B	3,177	70.00	1.75	2.94	1,792	56.79	1.75	1.93	0.00	-34.35
Kierens C	1,578	27.00	1.38	2.18	1,012	21.95	1.38	1.57	0.00	-27.98
Kierens D	617	22.86	0.64	3.11	442	14.55	0.64	2.20	0.00	-29.26
Kierens NE	676	50.00	1.83	3.28	368	24.62	1.83	1.86	0.00	-43.29
Norbenite LG	14,028	21.00	0.28	3.15	9,923	20.22	0.28	2.21	0.00	-29.84
Norlartic A	6,122	34.00	1.39	1.97	3,896	25.16	1.41	1.46	1.44	-25.89
Norlartic B	3,965	50.00	1.13	2.98	2,566	30.86	1.13	2.07	0.00	-30.54
Norlartic C	1514	20.00	0.60	2.62	1,191	11.67	0.60	1.95	0.00	-25.57
Norlartic NE	455	18.56	1.35	1.72	270	12.23	1.35	1.34	0.00	-22.09
North-North A	566	25.00	1.27	2.31	365	10.07	1.29	1.41	1.57	-38.96
North-North B	575	25.00	0.92	2.93	377	18.39	0.94	1.98	2.17	-32.42
North-North LG	2,486	10.00	0.21	3.84	2,026	6.72	0.21	2.41	0.00	-37.24
North-Shear LG	4,875	30.00	0.19	6.27	4,341	18.32	0.19	3.22	0.00	-48.64
North-Shear NE	70	3.94	0.24	2.96	49	3.90	0.25	2.84	4.17	-4.05
North-Shear SE A	276	20.00	0.58	3.41	226	7.79	0.58	1.91	0.00	-43.99
North-Shear SE B	637	40.00	1.22	4.25	467	34.10	1.22	2.81	0.00	-33.88
North-Shear SE C	498	16.46	0.36	3.22	399	11.40	0.36	2.64	0.00	-18.01

Note: Wtd. = weighted mean by length; CV = coefficient of variation

14.6 Density Assignment

Blocks have been assigned a density based on the lithology as summarized in Table 14-7. A density of 2.2 g/cm^3 was assigned to the overburden and 0.0 g/cm^3 to the historical underground workings and stopes (voids).

Table 14-7: Summary of Bulk Density by Area and Lithology

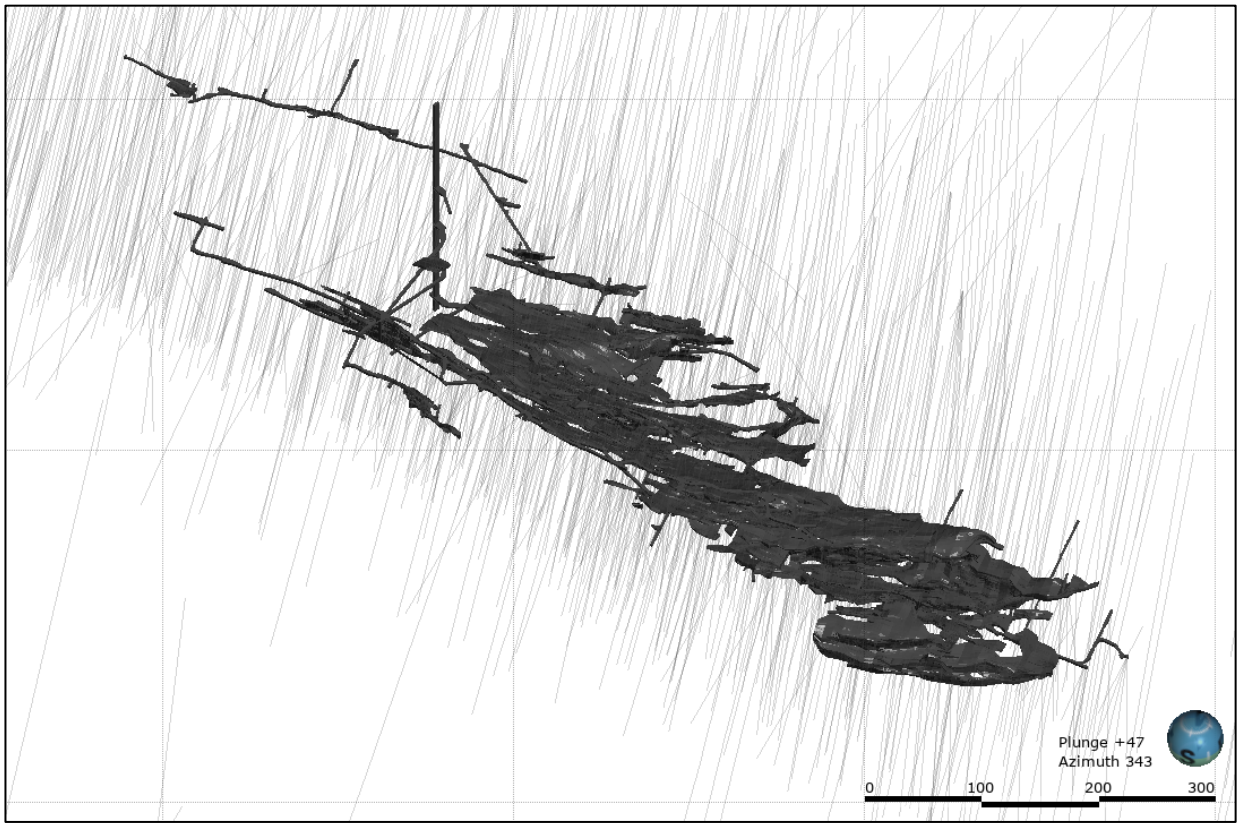
Marban		Kierens-Norlartic	
Lithology	SG	Lithology	SG
Felsic Intrusion	2.71	Diorite	2.74
Basalt	2.80	Felsic Intrusion	2.69
Ultramafic	2.87	Basalt	2.80
Marban Shear	2.78	Ultramafic-Komatiite	2.72
		Mafic-Diorite	2.73

Note: SG = specific gravity.

14.7 Historic Underground

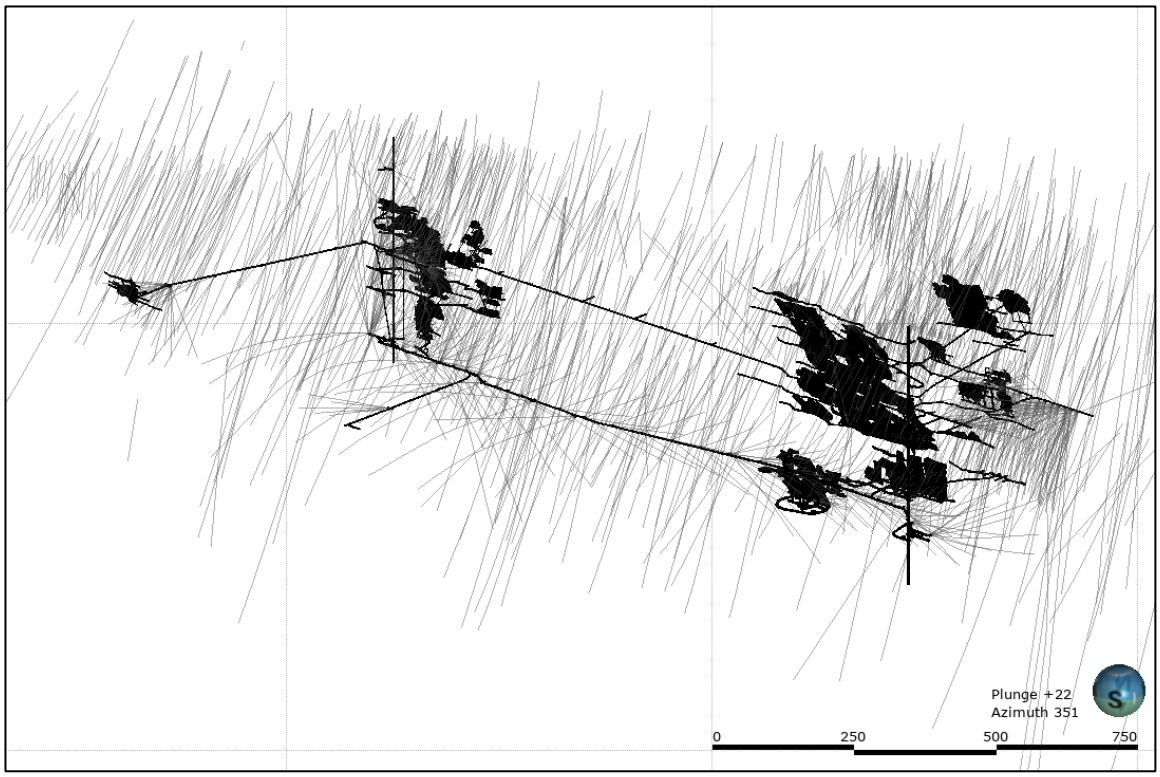
Historic underground shapes were provided by O3 Mining as three-dimensional solids and remain unchanged since the PEA. The voids for Marban and Kierens-Norlartic are illustrated in Figure 14-9 and Figure 14-10. GMS verified the stopes against the drilling, and found the breakthroughs identified in drilling to correlate well with the void model. Material mined historically has not been included in the MRE.

Figure 14-9: 3D View Looking Northwest of Void Model (black) – Marban Deposit



Source: GMS, 2022.

Figure 14-10: 3D View Looking Northwest of the Void Model – Kierens-Norlartic Deposit



Source: GMS, 2022.

14.8 Block Modelling

Separate rotated sub-block models were created for Marban and Kierens-Norlartic with a parent block size of 5 m x 5 m x 5 m and a sub-block size of 2.5 m x 2.5 m x 2.5 m. The sub-block triggers are the mineralization wireframes and the void model. GMS validated the volumes between the void wireframes and their respective block volumes to ensure that the historical UG mined-out material is well represented in each block model.

Block model extents are presented in Table 14-8 and Table 14-9 for each deposit.

Table 14-8: Marban Model Extents

Block Model Name	Description	Easting (m)	Northing (m)	Elevation (m)
BMod_Mar_08032022	Origin coordinates	277,450	5,335,200	374
	Parent/ Sub-block size	5/2.5	5/2.5	5/2.5
	Number of parent blocks	374	280	160
	Rotation	003º		

Table 14-9: Kierens-Norlartic Model Extents

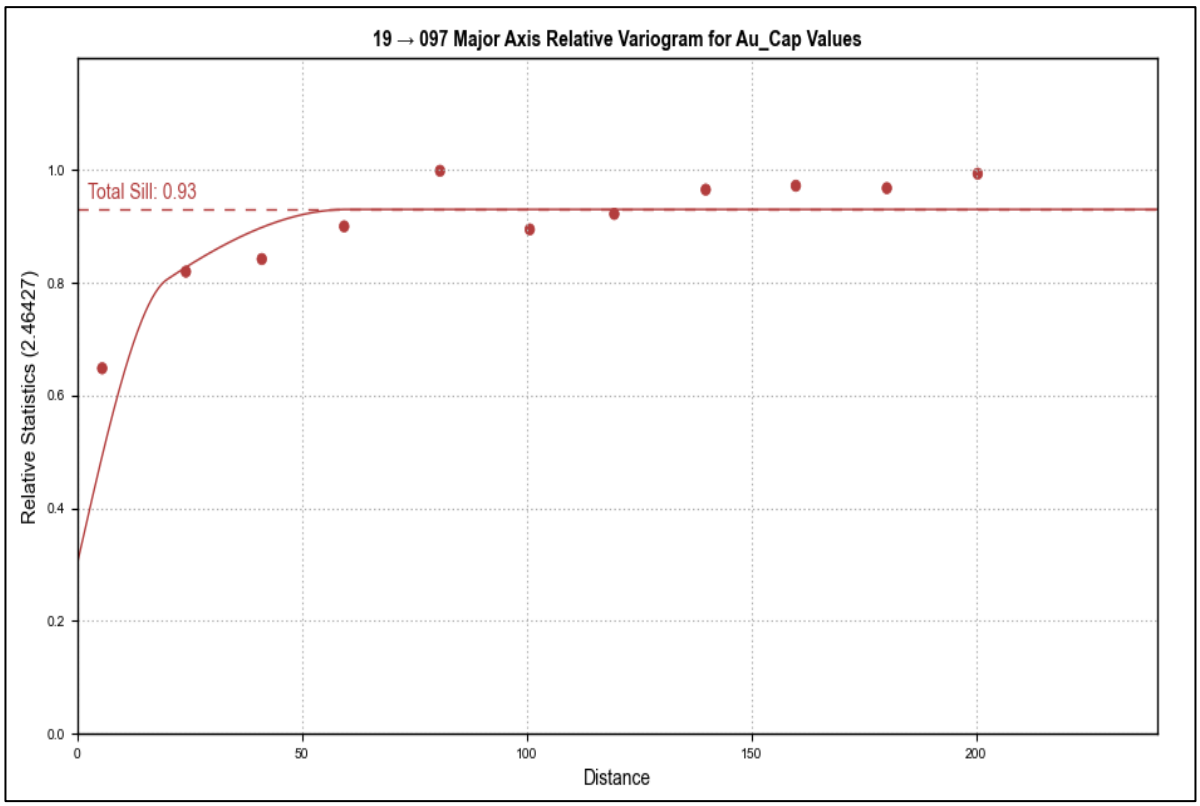
Block Model Name	Description	Easting (m)	Northing (m)	Elevation (m)
BMod_KN_08032022	Origin coordinates	276,000	5,337,800	350
	Parent/ Sub-block size	5/2.5	5/2.5	5/2.5
	Number of parent blocks	440	230	180
	Rotation	033º		

14.8.1 Variography

Variographic analysis was conducted using domain groupings based on orientation of mineralization and type. At Marban, the variograms were for the most part easy to interpret, with the exception of the fold shell where internal dilution and various orientations of mineralization made it difficult to interpret the structures. The reinterpretation of the domains has resulted in a reduction of internal dilution and has improved the structure of the variograms accordingly. Figure 14-11 and Figure 14-12 illustrate examples of relative and pair-wise variograms interpreted for the grouped domain of Zone B at Marban, and the Kierens B domain.

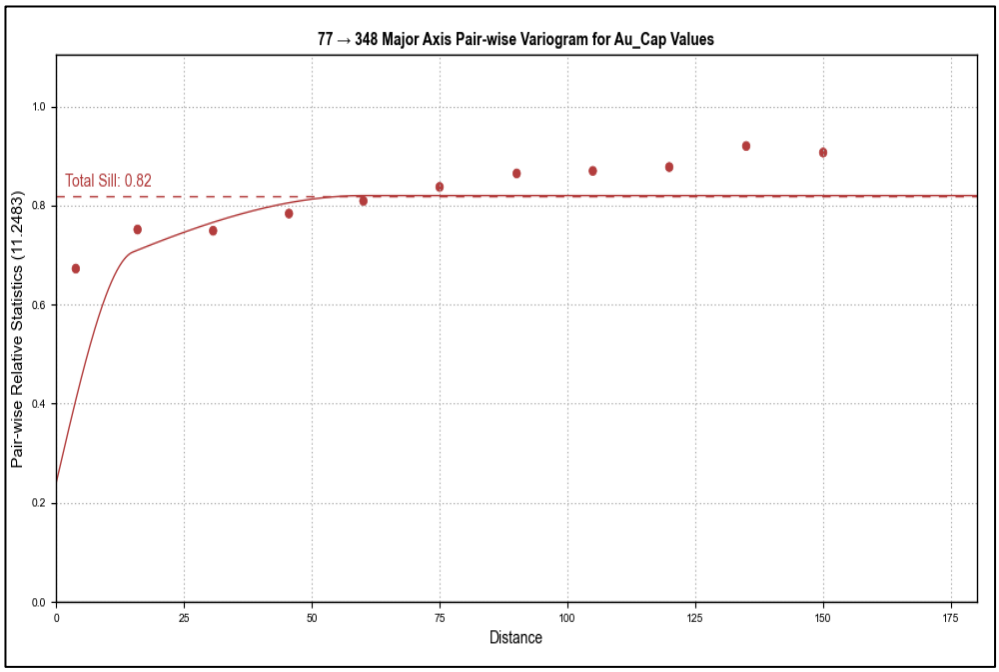
A summary of variogram model parameters for each deposit are presented in Table 14-10 and Table 14-11.

Figure 14-11: Major Axis Relative Variogram for Zone B – Marban



Source: GMS, 2022.

Figure 14-12: Major Axis Pair-wise Variograms for Kierens B – Kierens-Norlartic



Source: GMS, 2022.

Table 14-10: Variogram Model Parameters for Grouped Domains at Marban

	Dip	Dip Azimuth	Pitch	Nugget		Sill 1	Range 1	Sill 2	Range 2
Zone A	51	11	155	0.3	Major	0.34	15	0.32	50
					Semi		15		45
					Minor		3		7
Zone B	51	23	155	0.3	Major	0.39	20	0.24	60
					Semi		20		45
					Minor		4		7
Zone C	40	11	165	0.3	Major	0.45	20	0.31	60
					Semi		20		45
					Minor		4		9
Zone D	37	11	165	0.3	Major	0.41	20	0.27	70
					Semi		25		60
					Minor		3		8
Zone E	40	355	165	0.3	Major	0.41	20	0.33	65
					Semi		25		57
					Minor		3		8
Zone F	40	5	165	0.3	Major	0.41	30	0.25	65
					Semi		25		57
					Minor		3		8

Table 14-11: Variogram Parameters for Domains – Kierens-Norlartic

Domain	Dip	Dip Azimuth	Pitch	Nugget		Sill 1	Range 1	Sill 2	Range 2
Kierens A	80	38	72	0.4	Major	0.51	14	0.32	40
					Semi		17		41
					Minor		2		8
Kierens B	81	33	81	0.24	Major	0.4	15	0.18	60
					Semi		18		45
					Minor		8		7
Kierens C	81	35	113	0.2	Major	0.57	21	0.23	60
					Semi		20		60
					Minor		3		8
Kierens D	79	37	66	0.2	Major	0.59	21	0.35	65
					Semi		18		35
					Minor		3		7
Kierens NE	75	21	85	0.26	Major	0.44	27	0.4	50
					Semi		15		27
					Minor		3		6
Norlartic A	67	33	88	0.25	Major	0.28	20	0.27	42
					Semi		18		30
					Minor		3		10
Norlartic B	68	33	70	0.3	Major	0.4	24	0.24	40
					Semi		28		44
					Minor		3		7
Norlartic C	67	33	68	0.25	Major	0.37	18	0.49	34
					Semi		16		36
					Minor		3		8
Norlartic NE	71	344	72	0.24	Major	0.43	23	0.19	52
					Semi		18		50
					Minor		3		8
North-North A	53	26	118	0.26	Major	0.37	23	0.35	40
					Semi		20		38
					Minor		2		15
North-North B	52	21	64	0.5	Major	0.4	25	0.4	40
					Semi		20		30
					Minor		4		8
North-Shear NE	64	355	101	0.2	Major	0.32	42	0.4	62
					Semi		30		30
					Minor		3		8
North-Shear SE A	56	31	56	0.3	Major	0.48	36	0.32	45
					Semi		32		50
					Minor		7		10
North-Shear SE B	60	34	71	0.25	Major	0.39	15	0.45	33
					Semi		25		74
					Minor		4		9
North-Shear SE C	50	30	76	0.2	Major	0.4	23	0.28	55
					Semi		39		55
					Minor		4		11

Source: GMS, 2022.

14.8.2 Grade Estimation - Marban

GMS investigated numerous methods of interpolating capped gold grades into the block model, assessing the statistics of each domain and the observed grade continuity derived from the variogram models.

At Marban, Inverse Distance Cubed (ID3) was used for the fold shell due to the importance of preserving internal dilution in the domain and to avoid grade smearing. Ordinary Kriging (OK) was used for Zones A–F, and Inverse Distance Squared (ID2) for all other periphery zones where variograms were difficult to interpret. The fold shell domain was estimated using a hard boundary. However, when estimating Zones A–F, data was allowed to be shared with the fold shell. All other zones were estimated using hard boundaries.

At Kierens-Norlartic, Ordinary Kriging (OK) was used for all mineralized zones, and ID2 was used to estimate low-grade zones and waste domains.

Search ellipse dimensions were determined considering the observed variogram ranges and drill spacing. As mineralization at Marban is tightly folded, Dynamic Anisotropy was used to guide the orientation of the search ellipse during interpolation. At Kierens-Norlartic, Dynamic Anisotropy was also applied. Other interpolation parameters are summarized below:

First Pass

- 40m x 40m x 10m;
- Minimum samples = 7;
- Maximum samples = 12;
- Maximum samples from drillhole = 3;

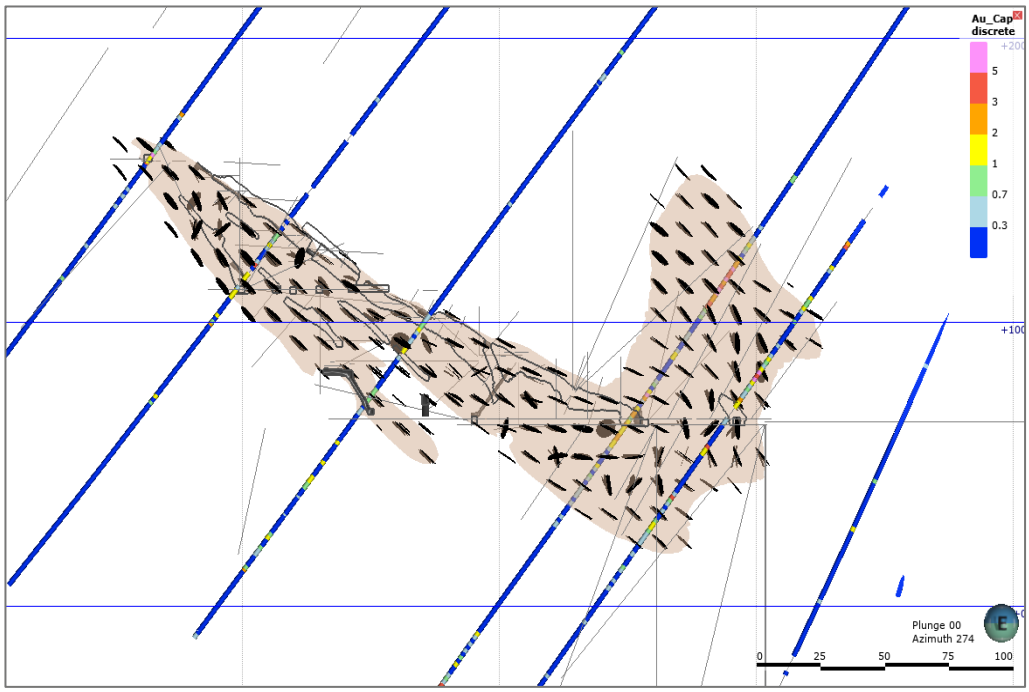
Second Pass

- 60m x 60m x 10m;
- Minimum samples = 4;
- Maximum samples = 12;
- Maximum samples from drillhole = 3;

Third Pass

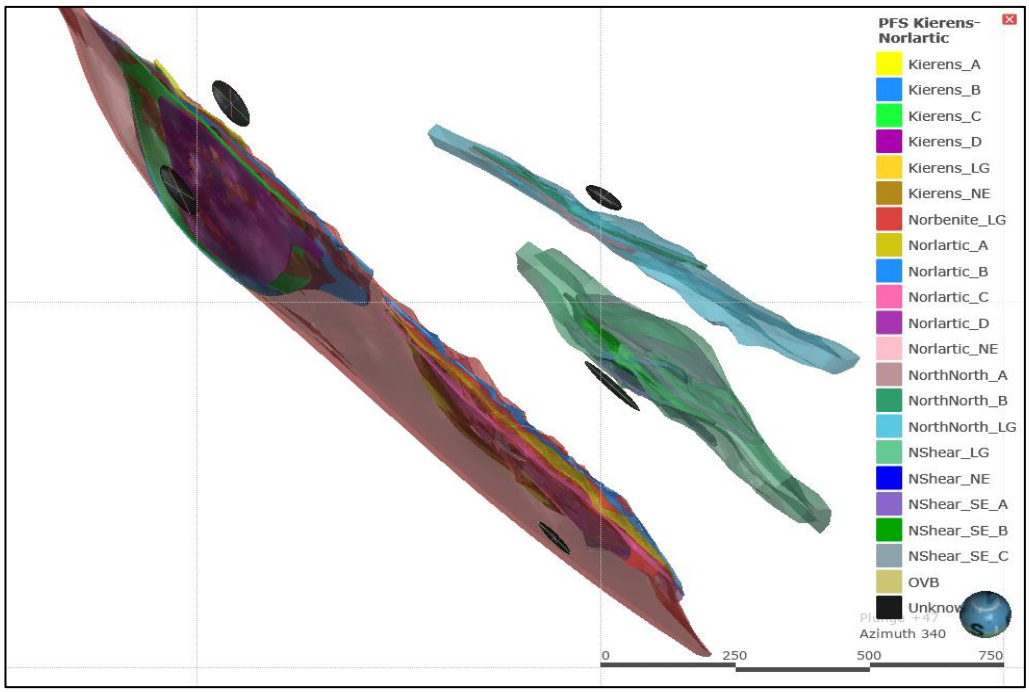
- 80m x 80m x 15m;
- Minimum samples = 3;
- Maximum samples = 12;
- Maximum samples from drillhole = 3;

Figure 14-13: 3D View Looking West of the Fold Shell and Variable Orientation Search – Marban



Source: GMS, 2022.

Figure 14-14: 3D View Looking North of the Mineralized Domains and Search Ellipses – Kierens-Norlartic



Source: GMS, 2022.

14.9 Block Model Validation

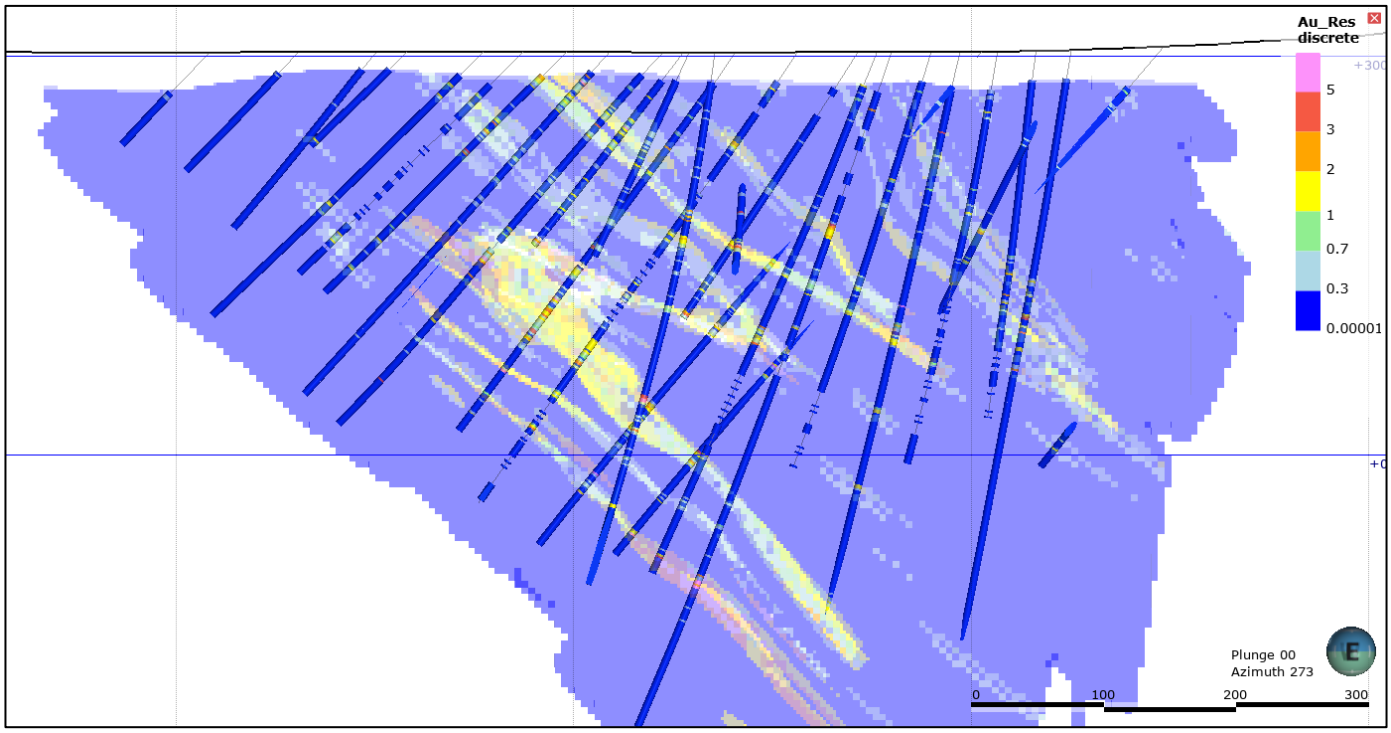
Various block model validation steps were taken to ensure that the block model is a robust representation of the composites. The following validations were undertaken:

- Visual checks on-section comparing composite gold grades against block gold grades;
- Global statistical checks comparing the gold grades of the block model against the de-clustered composite data;
- Local statistical checks to identify any over-smoothing or areas of grade over-extrapolation;
- Confirmation that the volumes of the blocks in the estimated domains and voids were representative of the mineralized and mined-out wireframes; and
- Sensitivity estimates to determine the impact of the capping and interpolation approach (uncapped, nearest neighbour).

14.9.1 Visual Validation - Composites Grades vs. Block Grades

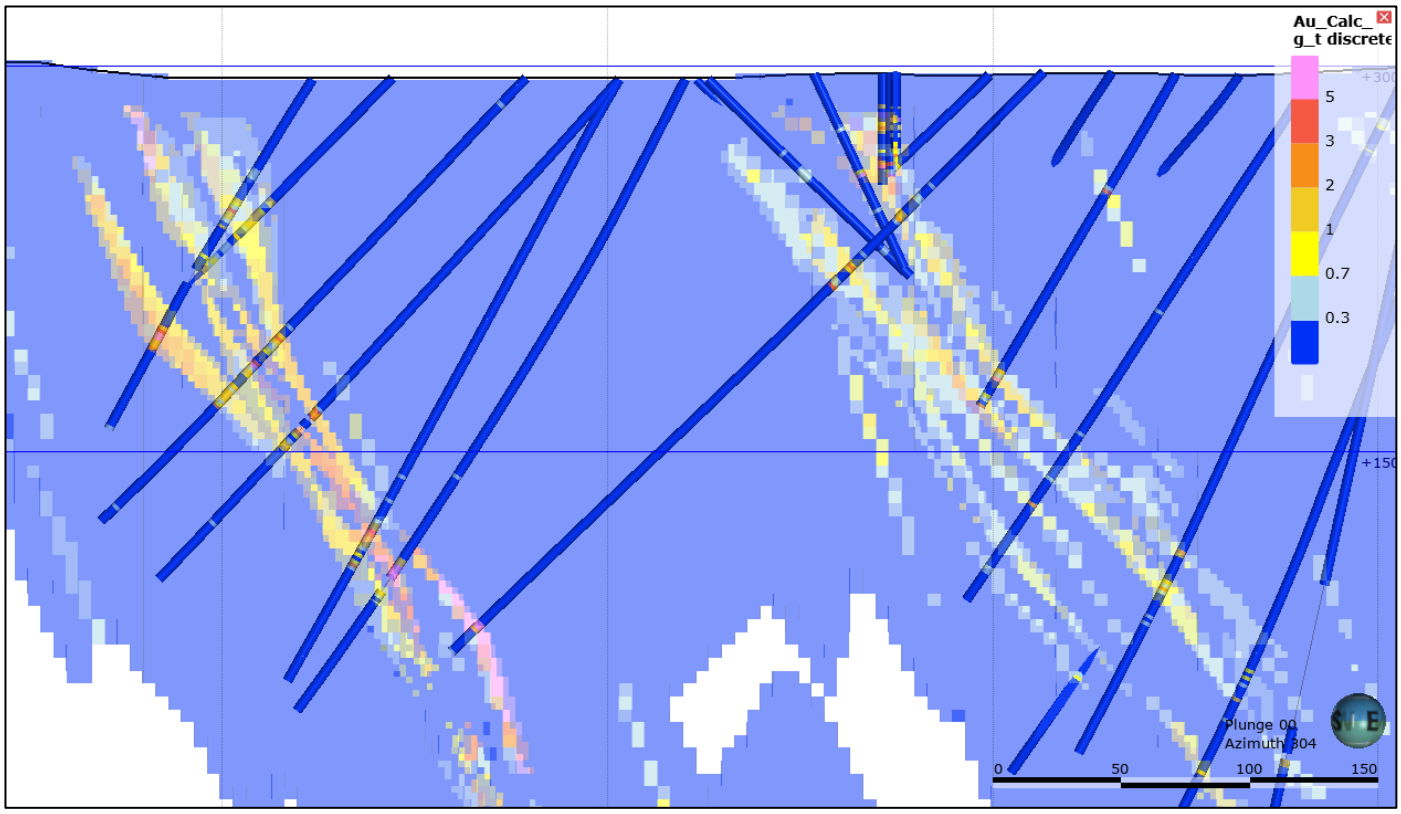
GMS performed a visual verification of the block grades and composite grades in cross-sections for both deposits (see Figure 14-15 and Figure 14-16). The visual grade smearing appears minimal in the block models, and the block grades are good representation of the composite grades.

Figure 14-15: Block Grades vs. Capped Composites Grades – Section 278,635 mE (Looking West) – Marban



Source: GMS, 2022.

Figure 14-16: Block Grades vs. Capped Composites Grades – Section 277,540 mE (Looking West) – Norlartic, North Shear



Source: GMS, 2022.

14.9.2 Global Statistical Validation

A comparison of block mean values (at zero cut-off) with the de-clustered composite data is provided in Table 14-12 and Table 14-13 for Marban and Kierens-Norlartic, respectively. The table illustrates no significant bias within the principal domains, with block grade mean falling within 10% of the de-clustered composite mean. Domains with higher differences in grade are generally domains in which either the grade is low or the drill spacing is wide. In these cases, the block grades are lower than the composite grades.

Table 14-12: Comparison of De-clustered Composites to Model Grades – Marban

Domain	Composites					Blocks				Mean Difference (%)
	Count	Length	Mean	Decl Mean	Max	Count	Volume	Mean	Max	
Fold Shell	3717	5525	1.02	1.03	24.46	165764	6504484	1.04	17.16	1.0
Zone A	1679	2456	0.94	0.96	26.70	89985	3227547	0.93	9.65	-3.1
Zone A1	128	183	0.44	0.45	3.19	8523	169922	0.45	1.50	-0.8
Zone A2	159	234	0.93	0.87	20.42	9489	268359	0.82	6.78	-4.7
Zone A3	337	489	1.36	1.40	25.72	34435	1082078	1.49	9.94	7.0
Zone B	774	1139	0.76	0.81	16.59	31768	1187844	0.81	9.62	-0.1
Zone B1	743	1090	1.17	1.18	39.91	49081	1812844	1.17	14.21	-1.0
Zone B2	327	479	0.83	0.73	26.90	15460	438547	0.75	8.99	1.9
Zone C	1764	2560	0.99	0.99	29.58	89801	2641703	0.96	11.16	-2.3
Zone D	1115	1633	1.14	1.19	28.78	52873	1674344	1.13	10.94	-4.4
Zone D1	641	940	0.61	0.62	9.97	27228	958203	0.58	3.67	-6.0
Zone D2	428	603	0.95	0.88	39.83	30156	541953	0.88	17.20	-0.4
Zone E	829	1206	1.32	1.26	32.10	46868	1297016	1.21	12.82	-4.1
Zone E1	320	468	0.91	0.94	32.96	9508	392031	0.93	14.76	-0.5
Zone F	1293	1896	0.62	0.64	10.96	74337	2376125	0.63	4.28	-0.9
LG Shell	1804	2619	0.62	0.61	20.01	104054	3479859	0.65	14.40	5.7
Dyke A	1026	1514	0.44	0.46	10.00	65826	2700984	0.46	7.40	0.1
Dyke B	668	976	0.65	0.70	12.00	52818	1910719	0.67	11.26	-4.0
Dyke C	322	475	0.17	0.17	2.22	18731	820844	0.15	1.31	-9.5
Dyke L1	416	618	0.29	0.29	6.82	79867	5196250	0.24	4.60	-16.5
Dyke L2	132	195	0.12	0.11	1.37	41554	1372578	0.09	1.17	-12.1
Dyke L3	20	30	0.46	0.44	2.52	12779	707063	0.35	1.90	-19.5
Norbenite	1068	1585	0.27	0.28	9.15	184749	10467813	0.25	8.38	-10.8
NW A	362	530	0.59	0.56	13.85	19365	632563	0.49	9.13	-13.1
NW B	165	242	0.62	0.69	19.53	7330	192406	0.62	8.50	-10.0
NW C	133	192	0.53	0.54	4.85	5683	144141	0.50	3.82	-6.8
UM1	167	150	1.39	0.90	12.97	15454	534375	0.87	9.70	-4.0
UM 2	103	245	0.90	1.40	10.98	26606	1058844	1.15	8.63	-17.7

Note: Max = maximum; Decl = Declustered

Table 14-13: Comparison of De-clustered Composites to Model Grades – Kierens-Norlartic

Domain	Composites					Blocks				Mean Difference (%)
	Count	Length	Mean	Decl Mean	Max	Count	Volume	Mean	Max	
Kierens_A	869	1243	1.80	1.18	58.03	33980	742141	0.88	21.69	-25.4
Kierens_B	1792	2590	1.73	1.15	56.79	70272	1687969	1.15	18.17	0.5
Kierens_C	1012	1440	1.38	1.26	21.95	49628	1037063	1.13	9.38	-10.1
Kierens_D	442	630	0.64	0.60	14.55	32758	614547	0.54	7.00	-9.7
Kierens_NE	368	536	1.83	1.71	24.62	8096	154500	1.64	7.88	-4.2
Norbenite_LG	9922	14403	0.28	0.30	20.22	382297	11803625	0.27	13.38	-8.0
Norlartic_A	3896	5713	1.40	1.35	25.16	130949	3860172	1.21	12.49	-10.5
Norlartic_B	2566	3745	1.13	0.90	30.86	104332	2834516	0.79	12.95	-12.0
Norlartic_C	1191	1718	0.59	0.59	11.67	74537	1685047	0.52	5.26	-12.2
Norlartic_NE	270	395	1.38	1.44	12.23	12240	331141	1.56	8.21	8.6
NorthNorth_A	365	535	1.29	1.16	10.07	19594	520531	1.10	4.79	-4.6
NorthNorth_B	377	545	0.94	0.81	18.39	27106	678703	0.76	8.23	-5.7
NorthNorth_LG	2026	2967	0.21	0.24	6.72	100237	4508609	0.20	4.94	-17.0
NShear_LG	4341	6450	0.20	0.20	18.32	216255	13389422	0.21	8.34	4.9
NShear_NE	49	70	0.26	0.13	3.90	4501	112547	0.16	2.96	24.2
NShear_SE_A	226	325	0.57	0.47	7.79	25392	653563	0.45	5.13	-5.1
NShear_SE_B	467	685	1.26	0.87	34.10	35339	1057266	0.88	16.10	1.0
NShear_SE_C	399	582	0.36	0.44	11.40	37618	885172	0.39	4.56	-12.4

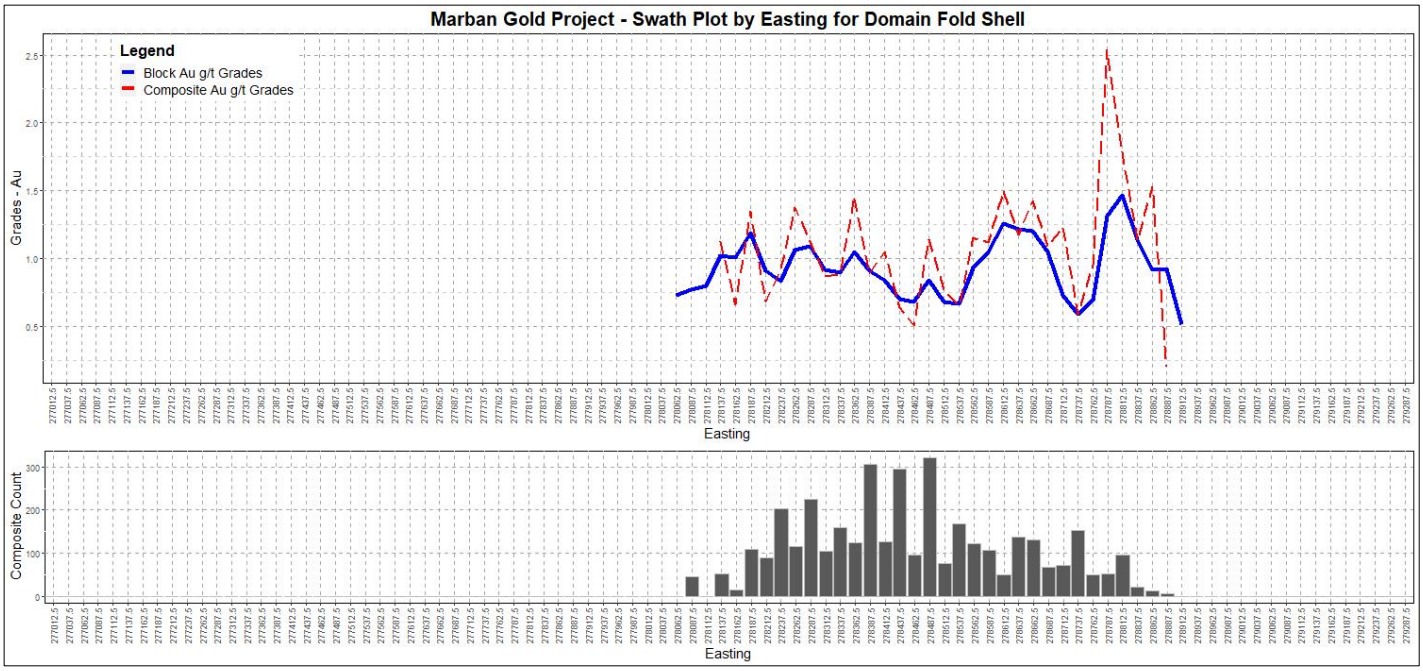
Note: Max = maximum; Decl = Declustered

14.9.3 Local Statistical Validation - Swath Plots

The swath plot method is a local validation tool which compares the block mean grades versus composites mean grades within a 3D moving window. The block models were rotated to E–W and swath plots were built by easting, northing and elevation.

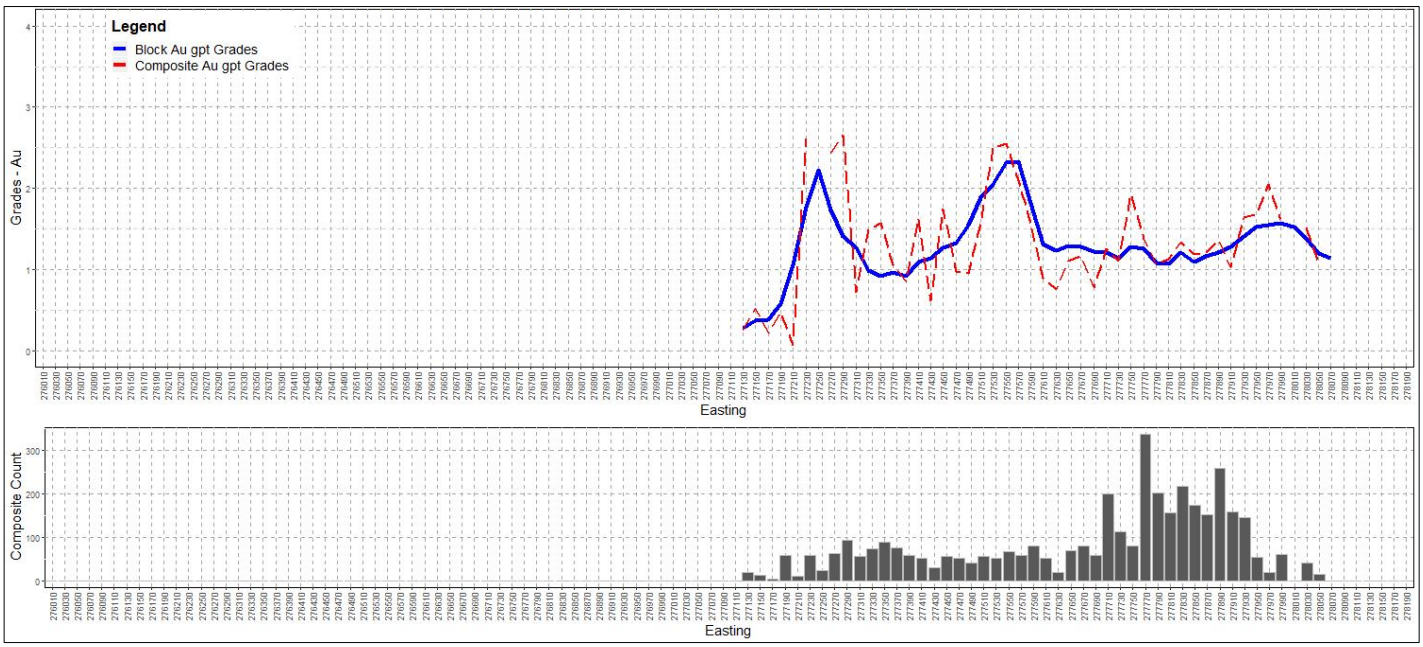
Figure 14-17 and Figure 14-18 illustrate swath plots by easting of gold grades for the fold shell domain at Marban, and the Norlartic A domain of Kierens-Norlartic.

Figure 14-17: Swath Plot by Northing for Domain Fold Shell – Marban



Source: GMS, 2022.

Figure 14-18: Swath Plot by Easting for Norlartic A – Kierens-Norlartic



Source: GMS, 2022.

14.9.4 Discussion on Block Model Validation

Overall, validation checks confirm that the block estimates are a reasonable representation of the drill composites. Visual inspection shows minor smearing and no significant over/under-estimation of gold grades. Local statistical validations from swath plots illustrate good local correlation between the estimated blocks as compared with the composites of mineralized domains.

14.10 Classification of Mineral Resources

Block model grades estimated for the Marban Engineering Project were classified according to the CIM's "Definition Standards for Mineral Resources and Mineral Reserves" (2014) and adhere to the CIM's "Estimation of Mineral Resources and Mineral Reserves Best Practice Guidelines" (2019). As defined by the CIM, all classified material must be within a potentially mineralized wireframe and within the "reasonable prospects of eventual economic extraction" shapes.

Due to the folded nature of the deposit, assigning a Resource category based solely on estimation pass resulted in a discontinuous category not suitable for mine planning purposes. GMS considered variographic ranges, drill spacing, quality of drilling and sampling data and confidence in mineralisation wireframes to determine parameters to define the Resource category. GMS chose to base the Resource category on distance to drillholes rather than the estimation pass approach, as it produced a more cohesive Resource category that is better suited for the maiden reserve.

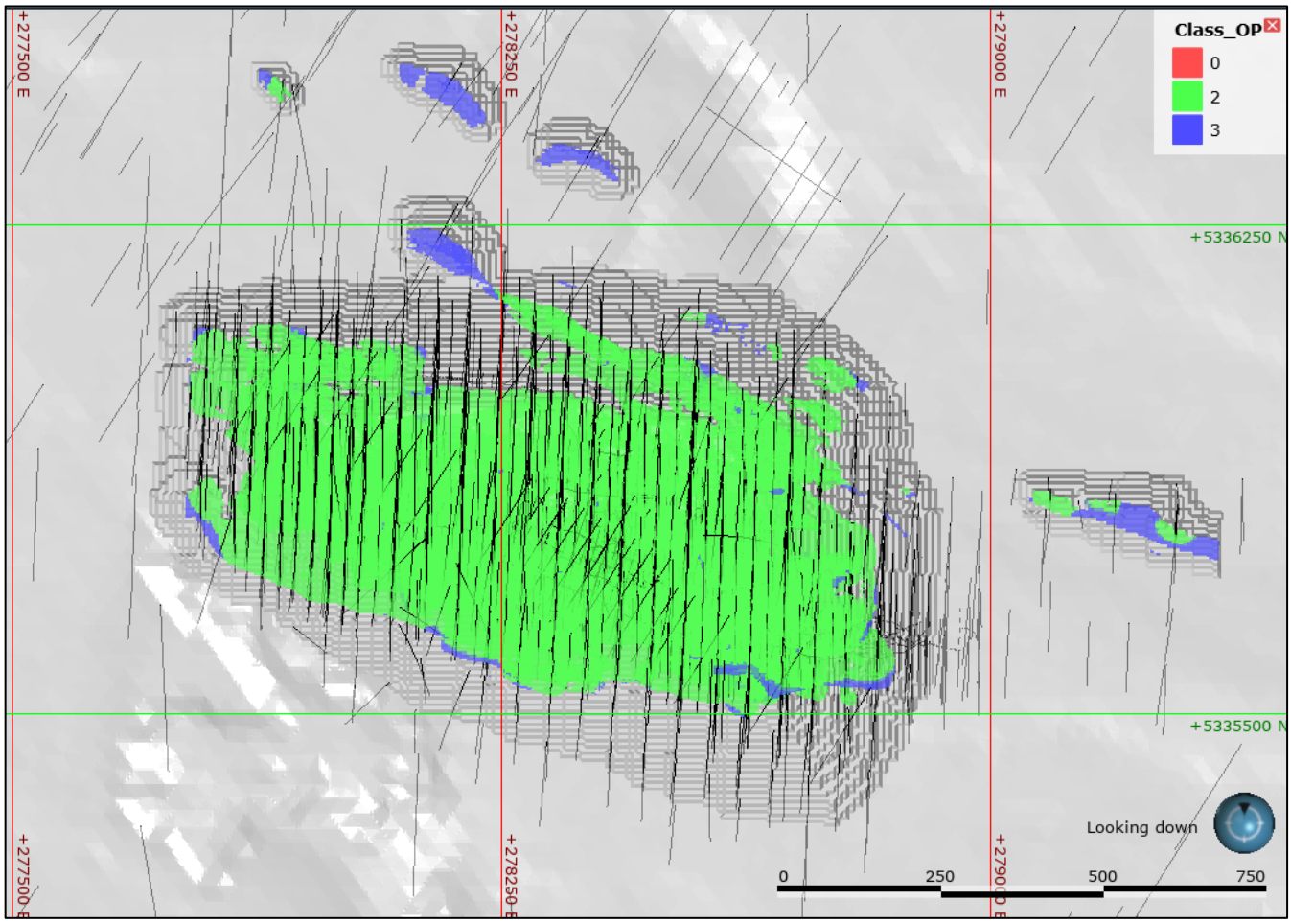
The mineral resource classification of measured, indicated, and inferred category undertaken by GMS is summarized as follows:

- OP classification was defined by the average distance to the nearest three drillholes approach:
  - Indicated mineral resources are defined as blocks estimated where the average distance to the three nearest drillholes is less than 35 m.
  - Inferred mineral resources are defined as blocks estimated where located to an average distance of greater than 35 m and estimated in Pass 1, 2, or 3.
- Underground resource classification generally based on an average distance to the nearest three drillholes of 25 m or less and was then edited manually to remove isolated blocks.
- No measured mineral resources were defined by both block models.

The resource category is based on improved and longer variogram ranges than those obtained in the previous study (due to refinement of mineralisation wireframes and reduction of internal waste), and improved geological understanding associated with the additional infill program conducted in 2021. Previous gaps in vicinity of historical UG stopes were successfully filled with drilling in 2021

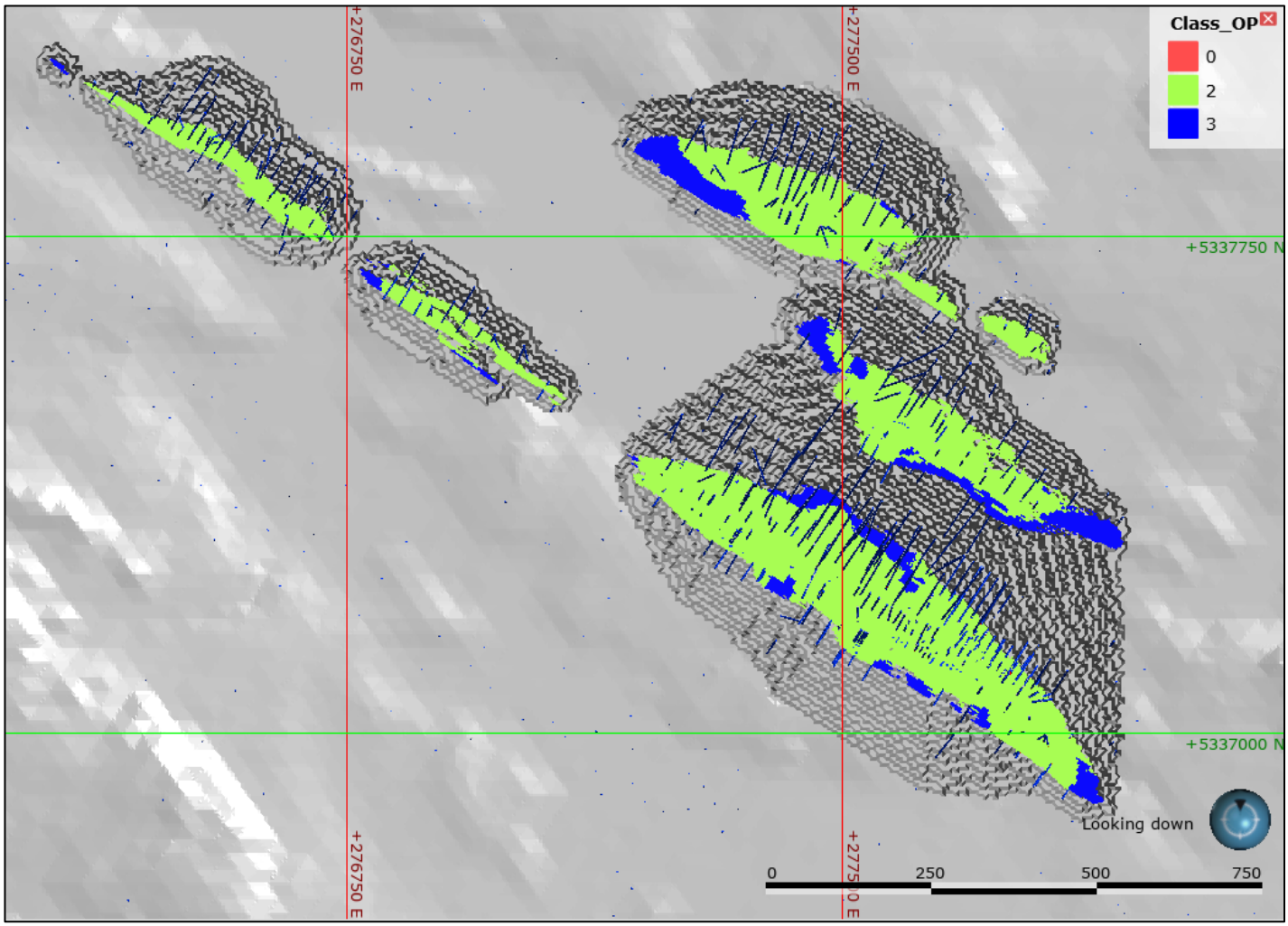
Figure 14-19 and Figure 14-20 show the current mineral resource classification with the 2022 whittle optimized pit shell delimiting the open pit mineral resources.

Figure 14-19: Block Classification, Blocks > 0.3 g/t Au – Indicated Resources (green) and Inferred Resources (blue) – Marban



Source: GMS, 2022.

Figure 14-20: Block Classification – Indicated Resources (green) and Inferred Resources (blue) – Kierens-Norlartic



Source: GMS, 2022.

14.11 Reasonable Prospects of Eventual Economic Extraction (RPEEE)

As illustrated above, the Marban and Kierens-Norlartic mineral resources have been constrained by a whittle pit shell using updated economic parameters. Underground resources have been visually constrained within areas showing sufficient continuity and that respect RPEEE, and isolated blocks have been removed from the classification.

To define the resource pits, economic parameters have been selected and are presented in Table 14-14. Geotechnical parameters for the PFS were not yet finalized at the effective date of this mineral resource; therefore, values from the PEA have been assumed.

Table 14-14: Whittle Parameter Inputs

Optimization Parameters	Unit	Value
Economic Parameters		
Discount Rate	%	5.00%
Gold Price	US$/oz	1,900
Transport & Refining Cost	US$/oz	4.3
Royalty Rate	%	1.50%
Royalty Cost – Gold	US$/oz	28.42
Net Gold Value	US$/oz	1,867
Recovery & Dilution Factors		
Average Gold Recovery	%	93.70%
Mining Dilution (in Block Model)	%	5.00%
Mining Loss (in Block Model)	%	0.00%
Other Ore-Based Costs		
Total Processing Cost incl. Power	C$/t milled	14.5
General & Administration Costs	C$/t milled	3.7
Total Ore Based Cost & Cut-Off Grade		
Total Ore Based Cost	C$/t milled	18.2
Gold Cut-off Grade	g/t Au	0.3
Mining Costs		
Total Mining Reference Cost	C$/t mined	2.4
Incremental Bench Cost	C$/5 m bench	0.003

14.12 Mineral Resource Sensitivity to Cut-off Grade

The sensitivity of the open pit resource to cut-off grade is summarized in Table 14-15. Sensitivity of the underground to cut-off grade was not undertaken because, by definition, all material within an underground shape is reported as a resource. The tonnages and grade at differing cut-offs shown below are for comparison purposes only, and do not constitute an official mineral resource.

Table 14-15: Sensitivity of the Open Pit Resource to Cut-off Grade – PFS Block Model

Deposit	Cut-off	Indicated			Inferred		
		Mass (Kt)	Average Grade Au (g/t)	Ounces (000's)	Mass (Kt)	Average Grade Au (g/t)	Ounces (000's)
Marban	0.25	55,433	0.99	1,763	1,150	0.90	33
	0.3	52,437	1.03	1,736	1,038	0.97	32
	0.4	45,965	1.13	1,664	910	1.06	31
	0.5	39,839	1.23	1,575	783	1.16	29
Kierens-Norlartic	0.25	15,678	1.17	590	1,159	1.33	49
	0.3	14,795	1.22	582	1,068	1.42	49
	0.4	13,362	1.32	566	969	1.53	48
	0.5	11,966	1.42	546	857	1.67	46
Total	0.25	71,111	1.03	2,353	2,308	1.12	83
	0.3	67,232	1.07	2,318	2,106	1.2	81
	0.4	59,327	1.17	2,229	1,879	1.3	78
	0.5	51,805	1.27	2,121	1,641	1.42	75

Mineral resources that are not mineral reserves do no have demonstrated economic viability

14.13 Comparison to the Previous Resource Estimate

The previous resource estimate was completed in 2020 by MMTS (PEA, 2020). The resource is compared below using a cut-off grade for the current open pit resource of 0.30 g/t Au. The differences between the two MRE's are determined to be:

- The current open-pit indicated resources represent an increase of 517,000 ounces (+29%) including an average gold grade increase by 2% and 26% of tonnage compared to the previous PEA Resource estimate.
- Increase in tonnage and gold content is mainly due to:
  - Better geological understanding supported by the additional infill drilling program conducted in 2021.
  - New interpretation on mineralized domains (wireframes) which reduces internal dilution and demonstrates improved continuity of gold grades.
  - Modified pit shells used to report the mineral resources.

Table 14-16: Comparison between Open-pit Mineral Resources – 2000 vs. 2022

Deposit	Classification	2020 MRE			2022 MRE		
		Mass (Kt)	Average Grade Au (g/t)	Ounces (000's)	Mass (Kt)	Average Grade Au (g/t)	Ounces (000's)
Marban	Measured	65	1.32	3	0	0	0
	Indicated	46,260	1.03	1,537	52,437	1.03	1,736
	M + I	46,325	1.03	1,540	52,437	1.03	1,736
	Inferred	6,465	1.09	227	1,038	0.97	32
Kierens - Norlartic	Measured	450	1.03	15	0	0	0
	Indicated	6,646	1.15	246	14,795	1.22	582
	M + I	7,096	1.14	261	14,795	1.22	582
	Inferred	6,299	1.42	287	1,068	1.42	49
All	Measured	515	1.08	17.9	0	0	0
	Indicated	52,906	1.05	1,783	67,232	1.07	2,318
	M + I	53,421	1.05	1,801	67,232	1.07	2,318
	Inferred	12,764	1.25	514	2,106	1.2	81

14.14 QP Comment

Mr. James Purchase, P.Geo., is not aware of any factors or issues that materially affect the mineral resource estimate other than normal risks faced by mining projects in the province in terms of environmental, permitting, taxation, socio-economic, marketing, and political factors and additional risk factors regarding indicated and inferred resources.

15 MINERAL RESERVE ESTIMATES

15.1 Summary

The proven and probable ore reserve for the Marban project is estimated at 56.4 Mt at an average grade of 0.91 g/t Au for 1,647 koz of contained gold as summarized in Table 15-1. There is no reserve within the proven category.

Table 15-1: Marban Engineering Project Ore Reserve Estimate (August 17, 2022)

	Tonnage (kt)	Grade (g/t Au)	Contained Gold (koz)
Proven	-	-	-
Probable	56,437	0.91	1,647
Proven and Probable	56,437	0.91	1,647

Notes: **1.** The mineral reserve is estimated using the Canadian Institute of Mining, Metallurgy and Petroleum (CIM), Estimation of Mineral Resources & Mineral Reserves Best Practice Guidelines (November 29, 2019) and CIM Definition Standards for Mineral Resources & Mineral Reserves, May 19th, 2014. **2.** The Qualified Person for the estimate is Mr. Carl Michaud, Eng. M.B.A., Vice President of Mining Engineering for GMS. Effective date of the estimate is August 17, 2022. **3.** Mineral reserves are estimated for a long-term gold price of US$ 1,600/oz. **4.** Mineral reserve cut-off grade is 0.3 g/t Au for all materials. **5.** A dilution skin width of 1 m was considered resulting in an average mining dilution of 5.4%. **6.** The average strip ratio is 5.07:1. **7.** Numbers may not add due to rounding.

The open pit mine design and ore reserve estimate have been prepared by GMS to a level appropriate for a Prefeasibility Study. The mineral reserve stated herein is consistent with the CIM definitions and is suitable for public reporting. As such, the mineral reserves are based solely on measured and indicated mineral resources with applicable modifying factors and therefore exclude any inferred mineral resources. The inferred mineral resources contained within the mine design are classified as waste for reporting purposes.

The factors that may affect the mineral reserve include the following:

- geological complexity, geological interpretation, and mineral resource block modelling;
- cut-off grade estimations;
- commodity prices, market conditions and foreign exchange rate assumptions;
- operating cost assumptions;
- sustaining capital costs to develop;
- rock quality and geotechnical constraints, dilution, and mining recovery factors;
- hydrogeological assumptions; and
- metallurgical process recoveries.

There are no other environmental, legal, title, taxation, socioeconomic, marketing, political or other relevant factors known to the QP that would materially affect the estimation of mineral reserves that are not discussed in this report. It is reasonably expected that all necessary government approvals will be issued for the project to proceed.

15.2 Resource Block Model

The resource block model was compiled by GMS. It was imported to Deswik CAD™ software as multiple block models covering the two primary pit groups, Marban and Kierens. The original model contained detailed sub-blocking down to a

1 m x 1 m x 1 m sub celling along the ore contact zones. The model was reblocked to a SMU block size of 5 m x 5 m x 5 m to better represent the open mining methodology applied.

15.3 Pit Optimization

Open pit optimization was conducted in GEOVIA Whittle™ to determine the optimal economic shape of the open pit to guide the pit design process. This task was undertaken using Whittle™ software, which utilizes the Lerchs-Grossmann algorithm. The method works on a block model of the ore body, and progressively constructs lists of related blocks that should, or should not, be mined. The method uses the values of the blocks to define a pit outline that has the highest possible total economic value, subject to the required pit slopes defined as structure arcs in the software. This section describes all the parameters used to calculate block values in Whittle™.

The pit optimizations performed to generate optimal pit limits to guide the ultimate pit design were based only on measured and indicated resource category blocks and excluded inferred blocks.

15.4 Geomechanical Recommendation

15.4.1 Geomechanical Pit Slope Field and Laboratory Program

Ausenco developed the pit slope field and laboratory program for the development of pit slope design criteria for this study, O3 Mining engaged WSP in 2021 to carry out field geomechanical investigation, which included drilling of three (03) of the pits. This provided information to characterize the rock mass geomechanical conditions, estimate the geotechnical parameters and identify geotechnical units.

WSP completed an eighteen (18) geomechanical borehole program in 2021, followed by laboratory testing of representative rock samples. Representative samples were obtained from the in-situ investigations carried out by WSP in 2021. Samples were tested by WSP (using the "Geomechanical" laboratory) to estimate the geotechnical parameters of the rock mass.

The televiewer logging of drillholes was completed with the main objective of measuring the orientation and geotechnical characteristics of discontinuities in the rock mass. Data processing was completed in April 2022 by WSP. Recording used optical and acoustic methods in thirteen (13) of boreholes (2 in Kierens Pit, 5 in Norlartic Pit, and 6 in Marban Pit) intended to characterize the rock masses of the walls of each of these open pits.

15.4.2 Pit Slope Recommendations

Pit designs are configured on 20 m bench heights, with 8.5 m wide berms. The maximum inter-ramp vertical distance is 160 m otherwise design 25 m minimum safety bench (crest to toe) at 160 m vertical intervals. Bench face angles, and subsequent inter-ramp angles, are varied based on prescribed geotechnical zones.

The pit slope criteria are based on a 2022 geotechnical report by Ausenco Engineering Canada. (Ausenco, 2022). Based on background information, the geotechnical units identified in the study area are Komatite, Granodiorite, Diorite, Basalt, and Iron Basalt, of which Komatite is predominant in Marban Pit, while Diorite, Granodiorite and Komatite are in the same proportion in Kierens and Norlartic Pits. Foliation and schists planes have been identified near Komatite formation. The Norbernite fault is an important northwest-trending deformation zone that dips to the northeast at moderate to steep angles (40° to 75°). The shear zone ranges from 45 to 90 metres in width. It is composed by ultramafic rock units and felsic intrusions, altered, and deformed. This fault cuts through the Kierens and Norlartic Pits. The Marbenite fault is an important northwest-southeast-trending deformation zone that dips to northwest at moderate to steep angles (40° to 70°). The shear zone ranges to 10 to 70 metres in width. It is composed of units of ultramafic rocks and basalts rich in Fe, altered and highly deformed. This fault cuts through the Marban Pit.

The majority of the rock mass at the Marban Engineering Project is strong and can be described as typical Abitibi belt hard rock. Overall rock mass failure is not a concern for the proposed 39 m to 365 m rock slopes that will form the ultimate pit

walls based on review of selected drill core. No known major geological structures (faults or shears) are expected to control slope design. Inter-ramp and overall slope stability will be controlled by the achievable geometry and stability of individual benches.

Structural fabric is anticipated to be the primary control on bench stability. Zones of lower quality rock seem to match Schists zones, which appear to be mostly located near basalt/ultramafic contacts. The schists altered in talc and chlorite are the weakest. A geotechnical definition of the Talc-Chlorite-Schist (TCS) should be carried-out in the next stage of the study. The TCS weak layers in the Malartic region can be from a few metres to 10's of metres thick and relatively continuous along the schistosity/foliation orientation.

The pits were divided into two to three sectors based on the identified geology and faults within the pits. Bench face and inter-ramp slopes in defined zones are listed in Table 15-2 for the Marban Pit, Table 15-3 for the Norlartic and North zones pits, and Table 15-4 for the Kierens.

Table 15-2: Marban Pit Bench Face & Inter-Ramp Angle Inputs

Pit	Sector	Azimuth	Bench Height (m)	Cross Bench Width (m)	Bench Face Angle (°)	Inter-Ramp Angle (°)
Overburden		0° – 360°	10	8.5	37	27
Marban	I	240° – 0° – 150°	20	8.5	69	51
	II	150° – 240°	20	8.5	53	40

*Overall slope angles are inputs for pit optimisations only

Table 15-3: Norlartic Pit Bench Face Inter-Ramp Angle Inputs

Pit	Sector	Azimuth	Bench Height (m)	Cross Bench Width (m)	Bench Face Angle (°)	Inter-Ramp Angle (°)
Overburden		0° – 360°	10	8.5	37	27
Norlartic	I	10° – 50°	20	8.5	65	48
	II	170° – 240°	20	8.5	65	48
	III	50° – 170° 240° – 10°	20	8.5	69	51

Table 15-4: Kierens Pit Bench Face Inter-Ramp Angle Inputs

Pit	Sector	Azimuth	Bench Height (m)	Cross Bench Width (m)	Bench Face Angle (°)	Inter-Ramp Angle (°)
Overburden		0° – 360°	10	8.5	37	27
Kierens	I	240° – 0° – 150°	20	8.5	69	51
	II	150° – 240°	20	8.5	64	48

15.5 Mining Dilution and Ore Loss

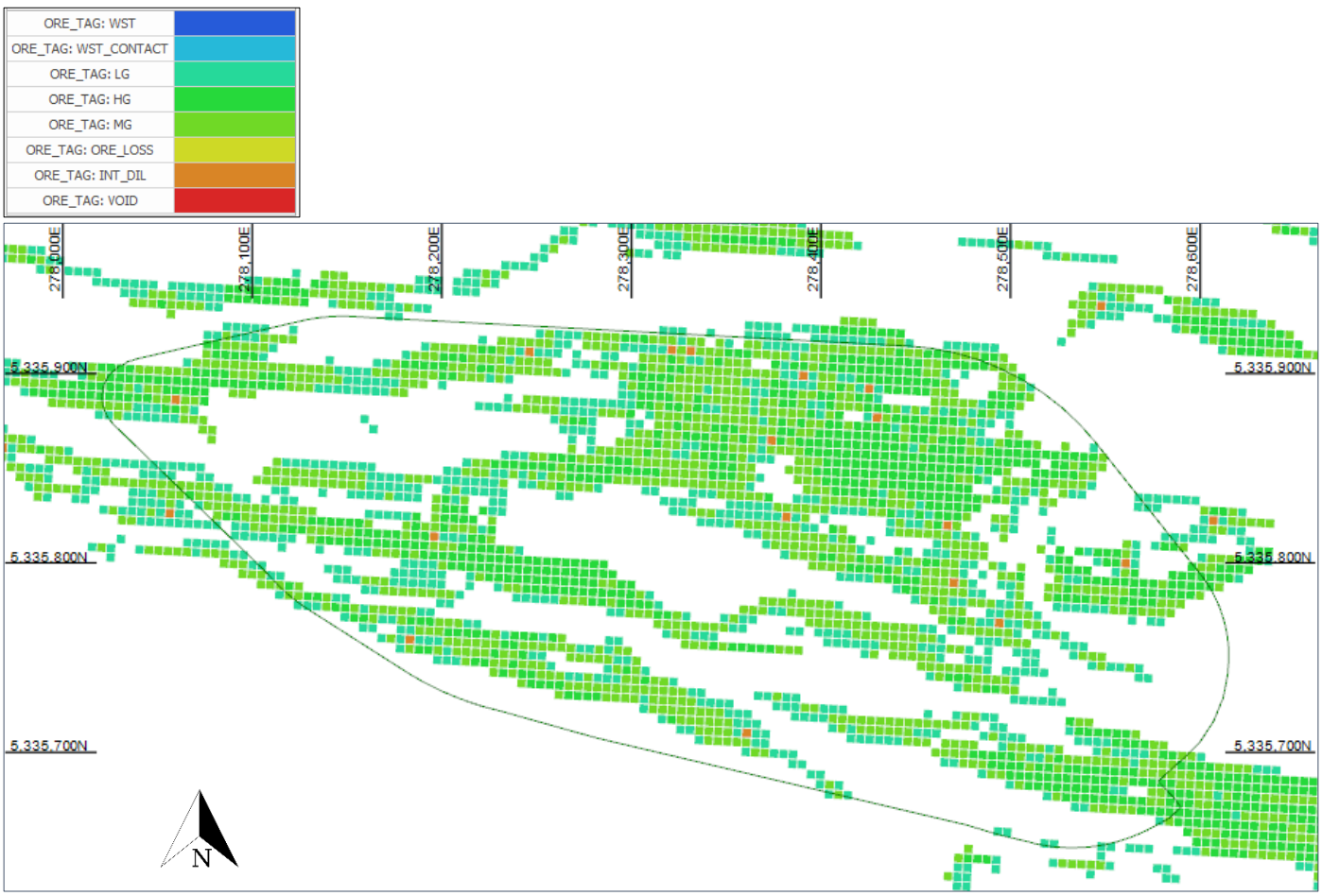
A mining dilution assessment was made by evaluating the number of contacts for blocks above an economic cut-off grade (CoG). The block contacts are then used to estimate a dilution skin around ore blocks to estimate an expected dilution during mining.

The dilution skin consists of 1.0 m of material in a north-south direction and 1.0 m in an east-west direction. For each mineralized block in the resource model diluted grades and a new density are calculated by considering the in-situ grades and in-situ density of the surrounding blocks. Dilution skin is calculated via half the width of the predominant ore shovel used in mining.

Blocks that are completely surrounded by waste are re-classified as ore loss and waste blocks surrounded by ore are classified as internal dilution. Figure 15-1 depicts Level 65 of the Marban Pit and the various block classifications.

Due to mined out underground stopes, special consideration was made to ensure that the block model reflected material that, when blasted, would move into and fill the voids. This material would be difficult to track and would be labelled as void loss. A 3 m skin was made on the hanging wall of the voids and any ore in this skin would be designated as void loss. There is a potential that this material will be captured in lower levels at unknown dilution levels.

Figure 15-1: Ore Body Classification (RL 65)



Source: GMS, 2022.

15.6 Pit Optimization Parameters and Cut-Off Grade

A summary of the open pit optimization parameters is presented in Table 15-5. The parameters used for optimization were updated from previous work done on the Marban Engineering Project as well as benchmarking on similar projects. A long-term metal price assumption of US$1,600/oz was used. The mining reference cost (i.e., for a block near surface) is C$2.40/t with an incremental cost of C$0.003/t per 1 m added to account for the additional haulage cycle time. The total ore-based cost is C$16.55/t. The ore-based cost is based on a nominal throughput of 6.0 Mt/a. A CoG of 0.3 g/t was set for the project.

Table 15-5: Economic Optimization Parameters

Optimization Parameters	Units	Marban
Economic Parameters		
Discount Rate	%	5.0%
Gold Price	US$/oz	1,600
Transport & Refining Cost	C$/oz	3.00
Royalty Rate	%	1.50%
Royalty Cost – Gold	C$/oz	29.94
Net Gold Value	C$/oz	1,967
Recovery & Dilution Factors		
Average Gold Recovery	%	Variable
Mining Dilution (in Block Model)	%	5.0%
Mining Loss (in Block Model)	%	95.0%
Other Ore Based Costs		
Total Processing Cost incl. Power	C$/t milled	13.55
General & Administration Costs	C$/t milled	3.00
Total Ore Based Cost & Cut-Off Grade		
Total Ore Based Cost	C$/t milled	16.55
Gold Cut-Off Grade	g/t Au	0.30
Mining		
Total Mining Reference Cost	C$/t mined	2.40
Incremental Bench Cost	C$/5 m-bench	0.003

The variable recovery is dependent on pit group and grade. Equation 15-1 presents the formula used to determine recovery.

Equation 15-1: Recovery Formula by Pit Group

$$Norlartic\ Rec\ \% = 12.084 * Au\ Grade + 80.2 - 0.5: Max\ of\ 95.4\%$$
$$Kierens\ Rec\ \% = 94.9\%$$
$$Marban\ Rec\ \% = 94.9\%$$

15.7 Pit Optimization Results

The Whittle™ nested shell results are presented in Table 15-6 and Table 15-7, for the two primary pit groupings, using only the measured and indicated (M&I) mineral resource, applying a 5% dilution and ore loss within Whittle™. The nested shells are generated using revenue factors (RF) to scale the selling price up and down from the base case.

15.7.1 Marban Pit Group

Table 15-6: M&I Whittle™ Shell Results at US$1,600/oz (Marban)

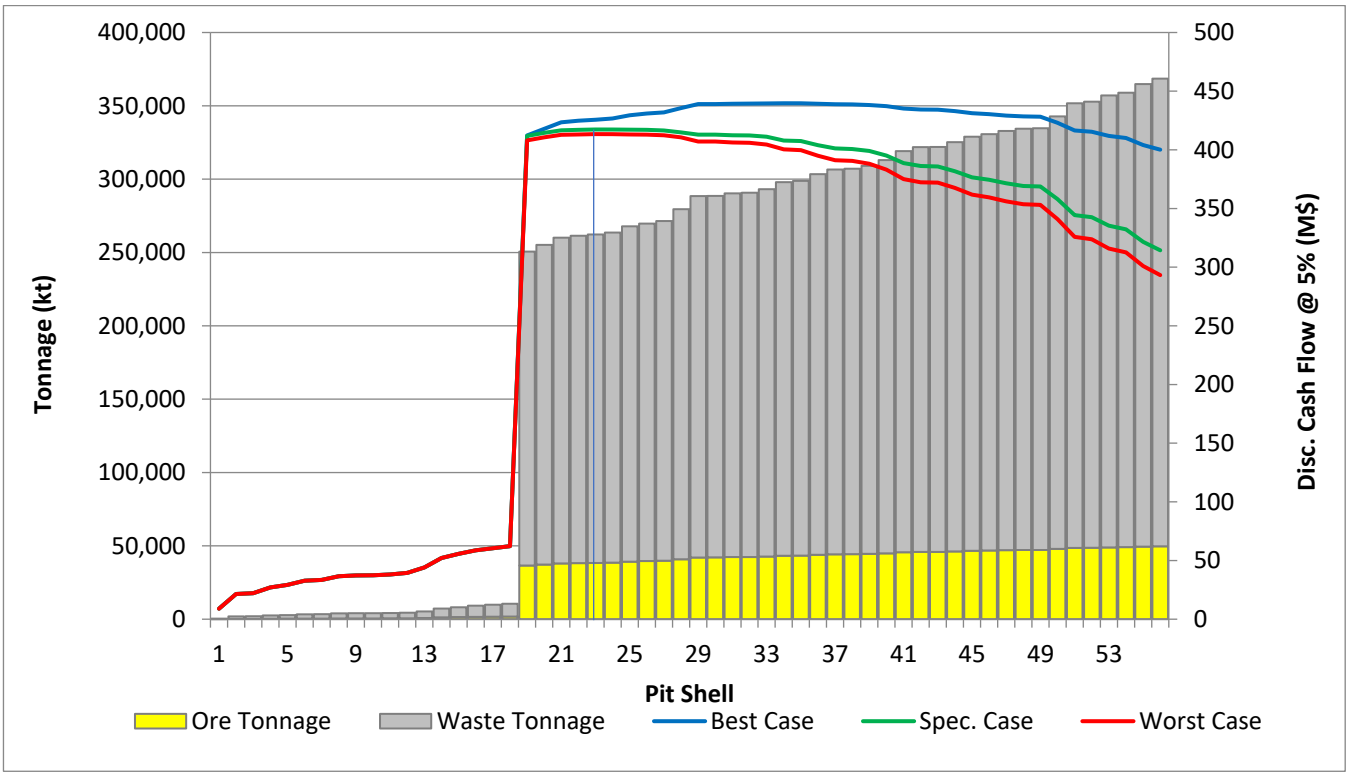
Pit Shell	Best Case Disc. @ 5% (C$M)	Specified Disc. @ 5% (C$M)	Worst Case Disc. @ 5% (C$M)	Total Tonnage (kt)	Waste Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Au Grade (g/t)
1	9	9	9	384	330	54	6.15	3.63
2	22	22	22	1,923	1,733	190	9.14	2.86
3	22	22	22	1,989	1,790	199	8.99	2.81
4	27	27	27	2,571	2,284	287	7.95	2.45
5	29	29	29	2,831	2,501	330	7.57	2.33
6	33	33	33	3,320	2,916	404	7.22	2.19
7	33	33	33	3,386	2,959	426	6.95	2.12
8	37	37	37	4,003	3,513	490	7.17	2.07
9	37	37	37	4,088	3,570	519	6.88	2.00
10	37	37	37	4,106	3,583	523	6.85	2.00
11	38	38	38	4,213	3,661	553	6.63	1.94
12	39	39	39	4,450	3,840	610	6.29	1.84
13	44	44	44	5,305	4,446	859	5.18	1.55
14	52	52	52	7,211	5,960	1,250	4.77	1.36
15	56	56	56	8,162	6,725	1,436	4.68	1.30
16	59	59	59	9,258	7,675	1,584	4.85	1.28
17	60	60	60	9,906	8,244	1,663	4.96	1.28
18	62	62	62	10,564	8,780	1,784	4.92	1.25
19	412	412	408	250,576	214,031	36,545	5.86	0.99
20	418	414	411	255,195	217,967	37,228	5.85	0.99
21	423	417	413	260,103	222,126	37,978	5.85	0.99
22	425	417	413	261,395	223,189	38,206	5.84	0.99
23	426	417	413	262,331	223,961	38,370	5.84	0.99
24	427	417	413	263,638	225,052	38,586	5.83	0.99
25	429	417	413	267,863	228,631	39,232	5.83	0.98
26	431	417	413	269,695	230,067	39,628	5.81	0.98
27	432	416	412	271,492	231,640	39,852	5.81	0.98
28	436	415	411	279,571	238,718	40,853	5.8	0.98
29	439	413	407	288,484	246,388	42,096	5.9	0.97
30	439	413	407	288,637	246,514	42,123	5.9	0.97
31	439	412	406	290,350	247,991	42,360	5.9	0.97
32	439	412	406	290,736	248,329	42,408	5.9	0.97
33	440	411	405	293,193	250,479	42,714	5.9	0.97
34	440	408	400	298,006	254,750	43,256	5.9	0.97
35	440	407	400	298,807	255,469	43,338	5.9	0.97
36	439	404	395	303,408	259,518	43,890	5.9	0.96

The shell selection is presented in Table 15-7 and Figure 15-2. Pit shell 23 was selected as the optimum final pit shell which corresponds to a US$1,248/oz pit shell (i.e., revenue factor 0.78). This shell has a total tonnage of 262.3 Mt including 38.4 Mt of ore. This is the smallest shell that achieves close to maximum value using a practical phasing approach. Typically, internal phases are chosen from smaller revenue factor Whittle™ shells. In this case, no such practical shells exist and instead internal Marban phases will be designed via 100 m push backs to ensure mining productivity is not lost during phasing.

Table 15-7: M&I Pit Shell Selection (Marban)

Shell Selection	Best	Spec.	Worst	Selection
Shell Number	34	23	11	23
Shell Revenue Factor	1.00	0.78	0.54	0.78
Shell Price (US$/oz)	1,600	1,248	864	1,248
Total Tonnage (kt)	298,006	262,331	4,213	262,331
Waste Tonnage (kt)	254,750	223,961	3,661	223,961
Strip Ratio (W:O)	5.89	5.84	6.63	5.84
Ore Tonnage (kt)	43,256	38,370	553	38,370
Au Grade (g/t)	0.97	0.99	1.94	0.99

Figure 15-2: M&I Pit Shell Selection @ US$1,600/oz (Marban)



Source: GMS, 2022.

15.7.2 Norlartic Pit Group

Table 15-8: M&I Whittle Shell Results @ US$1,600/oz (Norlartic)

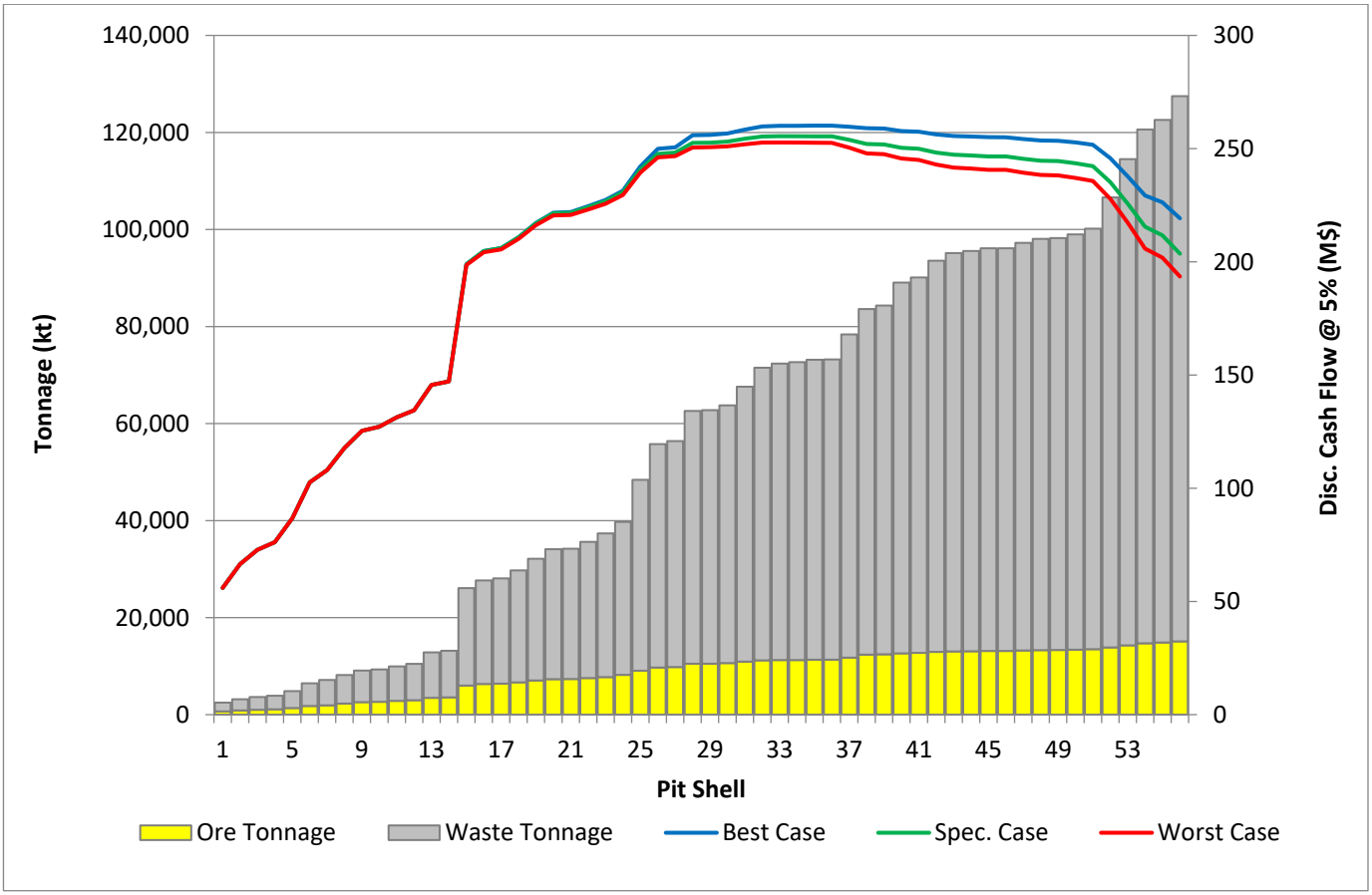
Pit Shell	Best Case Disc. @ 5% (C$M)	Specified Disc. @ 5% (C$M)	Worst Case Disc. @ 5% (C$M)	Total Tonnage (kt)	Waste Tonnage (kt)	Ore Tonnage (kt)	Strip Ratio (W:O)	Au Grade (g/t)
1	56	56	56	2,472	1,797	675	2.66	1.99
2	66	66	66	3,162	2,276	886	2.57	1.85
3	73	73	73	3,644	2,626	1,018	2.58	1.80
4	76	76	76	3,911	2,821	1,091	2.59	1.77
5	87	87	87	4,848	3,494	1,354	2.58	1.67
6	103	103	103	6,467	4,658	1,809	2.57	1.55
7	108	108	108	7,138	5,197	1,942	2.68	1.53
8	118	118	118	8,171	5,885	2,287	2.57	1.46
9	125	125	125	9,093	6,517	2,576	2.53	1.41
10	127	127	127	9,321	6,663	2,658	2.51	1.39
11	131	131	131	9,949	7,109	2,840	2.50	1.37
12	134	134	134	10,486	7,503	2,983	2.52	1.35
13	146	146	146	12,829	9,349	3,481	2.69	1.30
14	147	147	147	13,171	9,612	3,559	2.70	1.30
15	199	199	199	26,098	20,081	6,017	3.34	1.19
16	205	205	204	27,669	21,348	6,321	3.38	1.18
17	206	206	206	28,083	21,689	6,394	3.39	1.18
18	211	211	210	29,723	23,067	6,656	3.47	1.18
19	217	217	216	32,120	25,081	7,039	3.56	1.17
20	222	221	220	34,110	26,783	7,327	3.66	1.16
21	222	222	221	34,190	26,833	7,357	3.65	1.16
22	225	224	223	35,618	28,065	7,553	3.72	1.16
23	227	227	226	37,357	29,599	7,758	3.82	1.16
24	231	231	230	39,711	31,521	8,190	3.85	1.14
25	242	241	239	48,395	39,332	9,064	4.34	1.14
26	250	248	246	55,764	46,040	9,724	4.73	1.14
27	251	248	247	56,373	46,564	9,810	4.75	1.14
28	256	253	251	62,611	52,119	10,493	5.0	1.13
29	256	253	251	62,772	52,255	10,517	5.0	1.13
30	257	253	251	63,762	53,136	10,626	5.0	1.13
31	258	254	252	67,587	56,643	10,944	5.2	1.13
32	260	255	253	71,496	60,312	11,184	5.4	1.13
33	260	255	253	72,377	61,129	11,248	5.4	1.14
34	260	255	253	72,648	61,379	11,268	5.4	1.14
35	260	255	253	73,162	61,839	11,323	5.5	1.13
36	260	255	253	73,216	61,883	11,333	5.5	1.13

The shell selection is presented in Table 15-9 and Figure 15-3. Pit shell 34 was selected as the optimum final pit shell which corresponds to a C$ 1,920/oz pit shell (i.e., revenue factor 0.96). This shell has a total tonnage of 72.6 Mt including 11.3 Mt of ore. This is the smallest shell that achieves close to maximum value using a practical phasing approach. This shell profile gives ample choice for internal phases. Shell 24 was selected to guide internal phases of the Norlartic pit.

Table 15-9: M&I Pit by Pit Graph (Norlartic)

Shell Selection	Best	Spec.	Worst	Selection
Shell Number	35	34	11	34
Shell RF	0.98	0.96	0.50	0.96
Shell Price (US$/oz)	1,568	1,536	800	1,536
Total Tonnage (kt)	73,162	72,648	9,949	72,648
Waste Tonnage (kt)	61,839	61,379	7,109	61,379
Strip Ratio (W:O)	5.46	5.45	2.50	5.45
Ore Tonnage (kt)	11,323	11,268	2,840	11,268
Au Grade (g/t)	1.13	1.14	1.37	1.14

Figure 15-3: M&I Pit by Pit Graph @ US$1,600/oz (Norlartic)



Source: GMS, 2022.

15.8 Mine Design

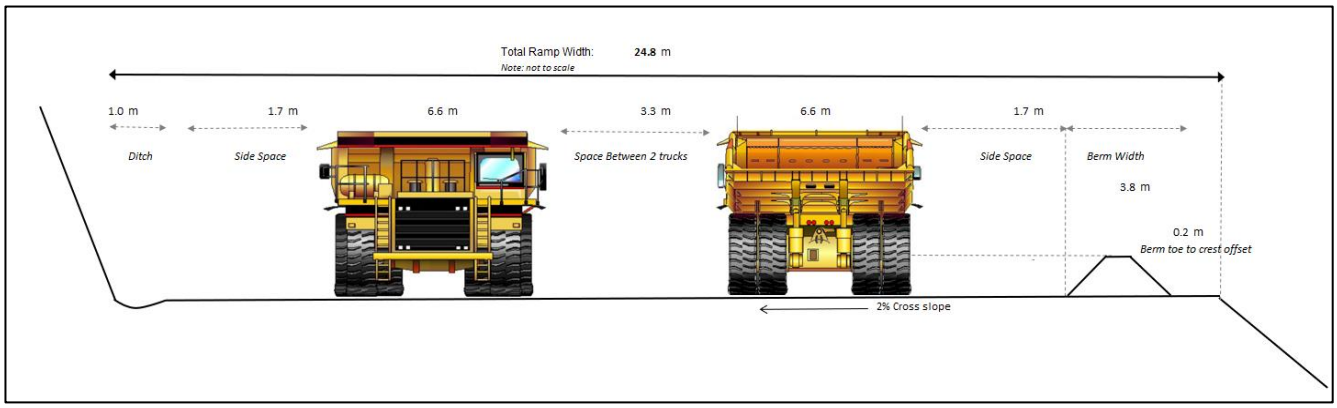
15.8.1 Ramp Design Criteria

The ramps and haul roads are designed for the largest equipment being a 150-t haul truck with a canopy width of 6.2 m. For double lane traffic, industry best-practice recommends designing a travelling surface of at least three times the width of the largest vehicle. Ramp gradients are established at 10%.

A shoulder barrier or safety berm on the outside edge will be constructed of crushed rock to a height equal to the rolling radius of the largest tire using the ramp. The rolling radius of the truck tire is 1.5 m. These shoulder barriers are required wherever a drop-off greater than 3 m exists and will be designed at 1.1H:1V. A ditch planned on the highwall will capture run-off from the pit wall surface and assure proper drainage of the running surface. The ditch will be 1.0 m wide. To facilitate drainage of the roadway, a 2% cross slope on the ramp is planned.

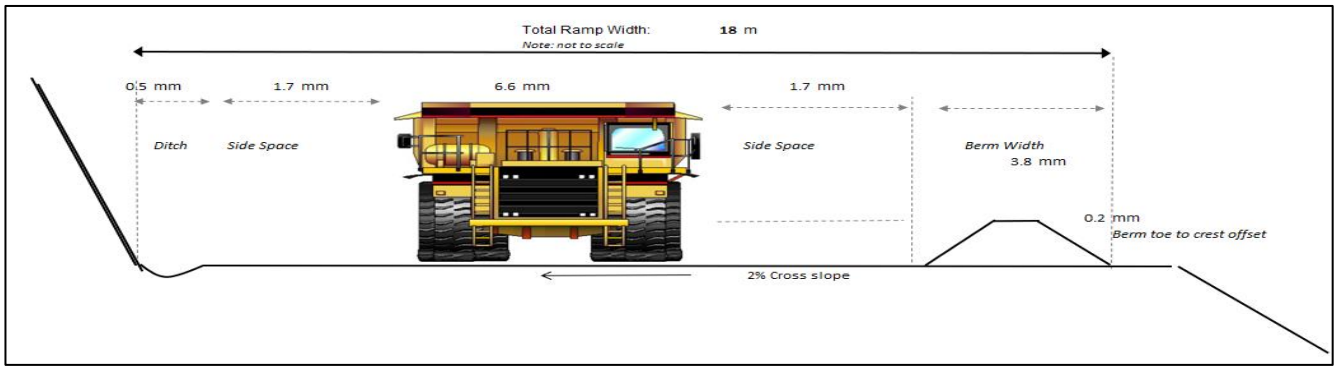
The double lane ramp width is 24.8 m wide, and the single lane ramp is 17.7 m wide. Single-lane ramps are introduced in the pit bottom when the benches start narrowing and when the mining rates will be significantly reduced. Double and single-lane ramp configurations are shown in Figure 15-4 and Figure 15-5.

Figure 15-4: Double-Lane Design Criteria



Source: GMS, 2022.

Figure 15-5: Single-Lane Design Criteria



Source: GMS, 2022.

15.8.2 Other Design Considerations

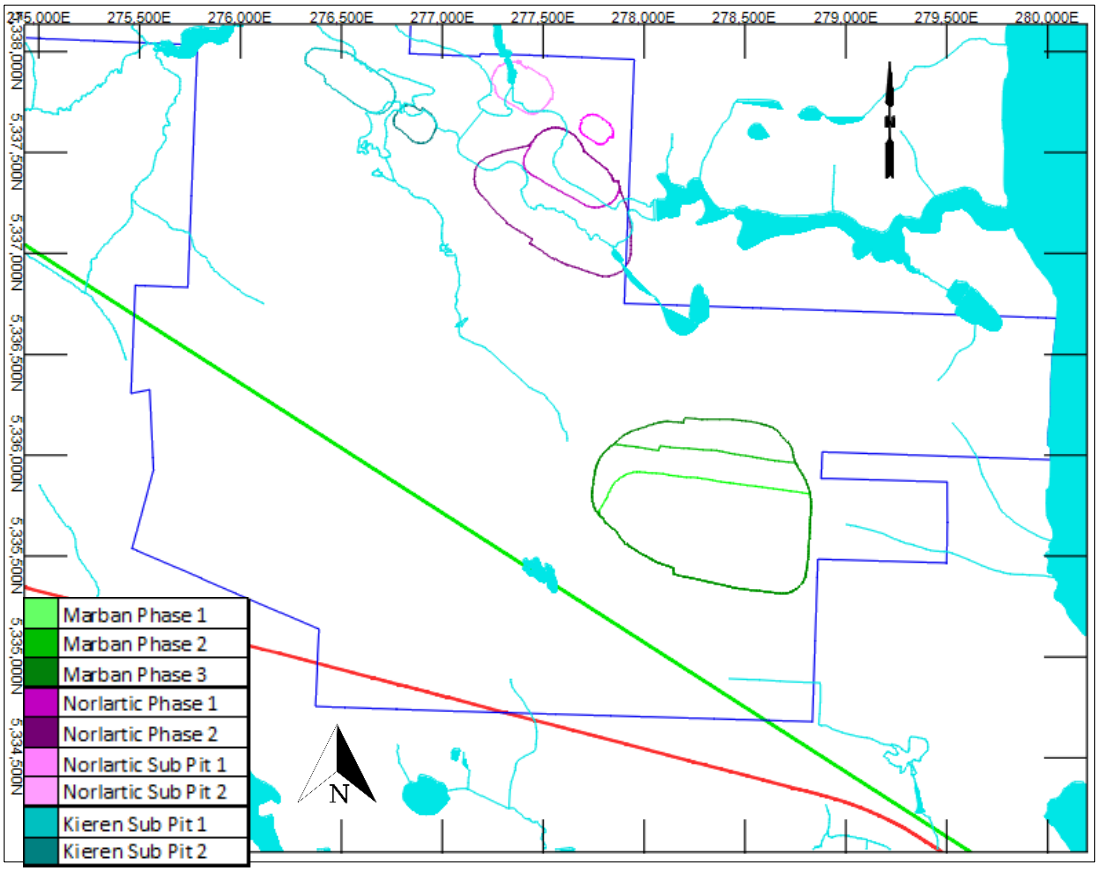
The Marban Engineering Project posed some site-specific restrictions for mining and access that must be respected for permitting and proper practice. Figure 15-6 depicts the layout and several design considerations to be accounted for in the designs.

The mining claims must be respected in terms of blasting and mining of material. It is possible to exceed the boundaries for overburden (OVB) and access. This limit is depicted in blue.

Power lines (green) and the highway (red) will be considered when creating access to the pit and when raising powerlines to allow haul trucks to access areas south of the powerline. Blasting parameters are altered when blasting near sensitive and high traffic areas.

Water and lake areas are accounted for in the construction sections of the report; this is critical for the Kierens and Norlartic zones.

Figure 15-6: Site Layout



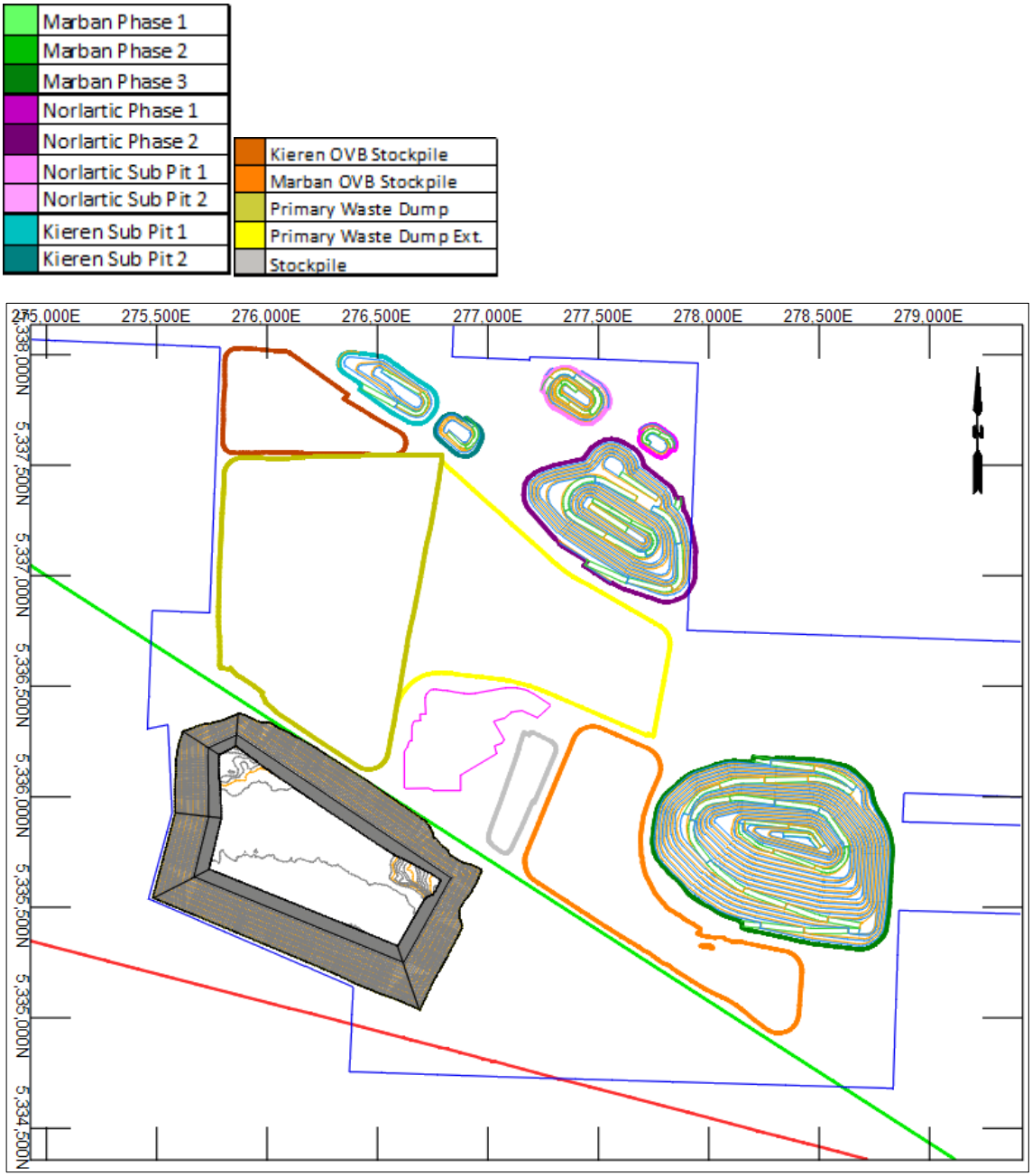
Source: GMS, 2022.

15.9 Open Pit Mine Design Results

The Marban pit is mined as three separate pit groups. The Marban pit group is the largest and makes up the majority of tonnage mined in the project. The Marban pit group consists of a three-phase nested pit located to the east of the project.

The Norlartic group is the next largest pit located to the north. The Norlartic main pit has two phases. There are also two sub pits located directly adjacent to it. The smallest pits are the Kierens group. This group consists of two small sub pits west of the Norlartic pits. Refer to Figure 15-7 below for a plan view of the final pit design.

Figure 15-7: Final Pit Designs End LOM (Plan View)



Source: GMS, 2022.

15.10 Mineral Reserve Statement

The ore reserve and stripping estimates are based on the final pit design presented in Section 15.8. The proven and probable ore reserves are inclusive of mining dilution and ore loss. The total ore tonnage before external mining dilution, ore loss, void loss, and reclassification is estimated at 54.7 Mt at an average grade of 0.96 g/t Au.

The external mining dilution around the ore blocks results in a dilution tonnage of 8.6 Mt at 0.15 g/t. The dilution tonnage represents % of the ore tonnage before dilution and the dilution grade is estimated from the block model and corresponds to the average grade of the dilution skin. Table 15-10 presents a mineral resource to ore reserve reconciliation. A reclassification of the blocks is done after the dilution to re-evaluate ore blocks that are below cut-off after dilution. This reduces the effects of dilution with the loss of some ounces.

Table 15-10: Mineral Resource to Ore Reserve Reconciliation

	Unit	Tonnage (kt)	Grade (g/t)	Gold (koz)
Ore before Loss & Dilution	unit	54,672	0.96	1,696
Less: Ore Loss	unit	146	0.39	2
Less: Void Loss	unit	1,230	1.08	43
Ore before Dilution	unit	53,296	0.96	1,651
Add: Ore Dilution	unit	8,571	0.15	42
Diluted Reserve	unit	61,867	0.85	1,693
Reclassify Resource	unit	56,329	0.91	1,645

16 MINING METHODS

16.1 Summary

The Marban Engineering Project is a conventional open-pit mine using drilling and haul trucks coupled with a hydraulic shovel. The project is split into three mining pit groups: Marban, Kierens, and Norlartic, which are further split into nine sub-pits and phases. The peak mining rate is 52.3 million tonnes (Mt) per year over a mine life of 9.6 years. A total of 56.4 million tonnes (Mt) of ore will be mined at an average grade of 0.91 grams/tonne (g/t), for a total of 343.8 Mt of mined waste, resulting in a stripping ratio of 5.07 tonnes waste per tonne of ore. Primary production equipment includes 16 m^3-electric production shovels and 150-tonne off-highway mining trucks, plus a smaller secondary fleet focused on overburden of 100-tonne trucks and 90-tonne excavators. An owner-operated mine is planned and blasting activities are outsourced to contractors.

Pre-production (PP) mining will take place just over one year to provide materials for construction and to remove overburden to allow access to pits. A total of 16 Mt of rock and overburden will be mined in preproduction.

The milling rate is planned at 6.0 Mt/a with a ramp-up of one year at a rate of 4.8 Mt/a. Stockpiling is minimal with a peak stockpiled inventory of 0.6 Mt taking place in Year 7, predominately made up of low-grade ore.

16.2 Open Pit Designs

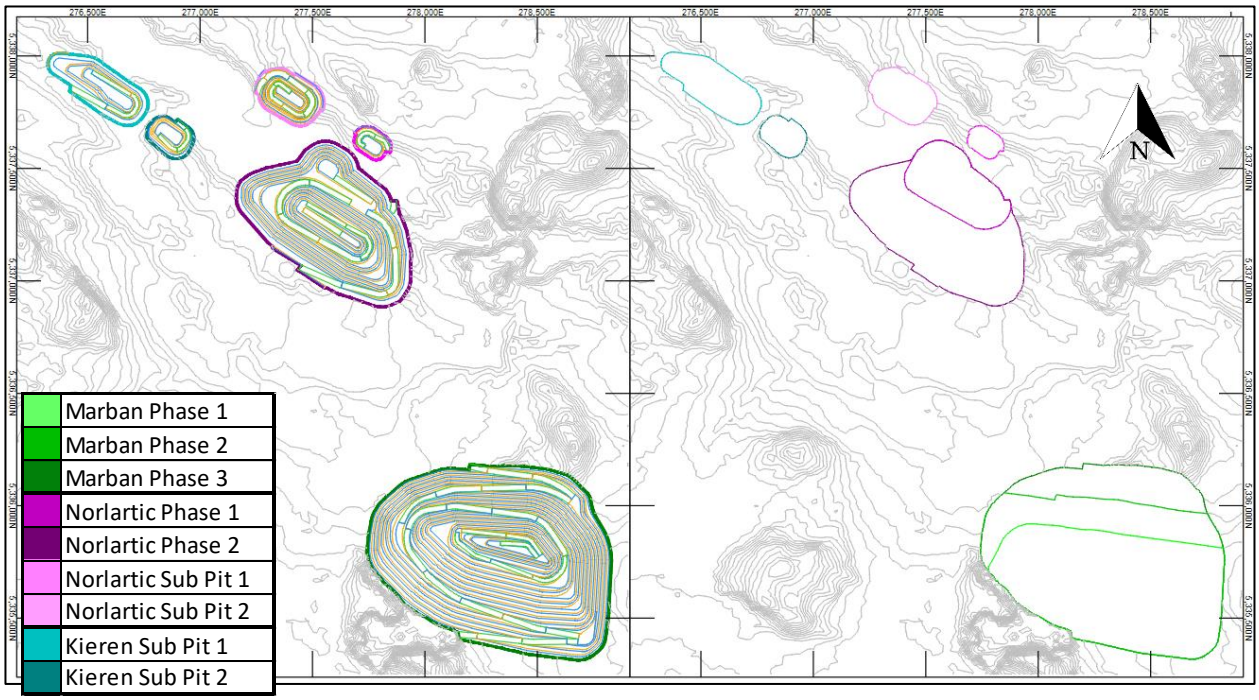
16.2.1 Mine Design Parameters

The open pit mine designs were guided using optimal Whittle™ shells, pit slope, ramp design criteria, and dilution parameters are outlined in Section 15. Mining of the Marban open pit Project is planned with nine phases / sub-pits. The mining physicals of each of the mining phases are summarized in Table 16-1 and final configuration of the pit is presented in Figure 16-1.

Table 16-1: Mining Reserve by Phase and Pit Group

	Unit	Grand Total	Marban			Norlartic				Kierens	
			Phase 1	Phase 2	Phase 3	Sub Pit 1	Sub Pit 2	Phase 1	Phase 2	Sub Pit 1	Sub Pit 2
Total Tonnage	kt	342,584	109,586	61,288	85,037	4,297	529	11,057	64,481	5,063	1,245
Waste	kt	286,147	95,380	51,772	65,830	3,495	472	9,524	54,144	4,403	1,126
Stripping Ratio	O:W	5.07	6.71	5.44	3.43	4.36	8.29	6.21	5.24	6.67	9.51
Ore	kt	56,437	14,206	9,516	19,207	802	57	1,533	10,337	660	118
Gold Grade	g/t	0.91	0.82	0.88	0.92	1.10	0.66	0.85	1.02	1.13	0.88

Figure 16-1: End of LOM Pit Layout and Phase Limits



Source: GMS, 2022.

16.2.2 Pit Phases

16.2.2.1 Marban

The Marban is the largest pit group and produces 74% of all ounces for the project. The Marban Pit Group is located southeast of the other pit groups along the edge of the mining claims. The Marban pit is composed of three nested phases, each using the south footwall as a base and expanding via 100 m pushbacks. Special considerations of pit designs are made to avoid ramping along underground voids left from historical underground mining. Those voids will pose a structural weakness and will require backfilling if ramping or critical walls fall along these sections and are avoided. Geotechnical berms are required for every 160 m of unsupported pit walls, which are not required as ramping will break up any long vertical wall sections and prevent the requirement of geotechnical berms throughout at all phases. In all phases the haulage ramp transitions to a single lane from double lane along the last three benches to better capture ore pockets at the bottom of the phase. Pit exits are planned to exit to the west to allow the shortest haul to all potential destinations.

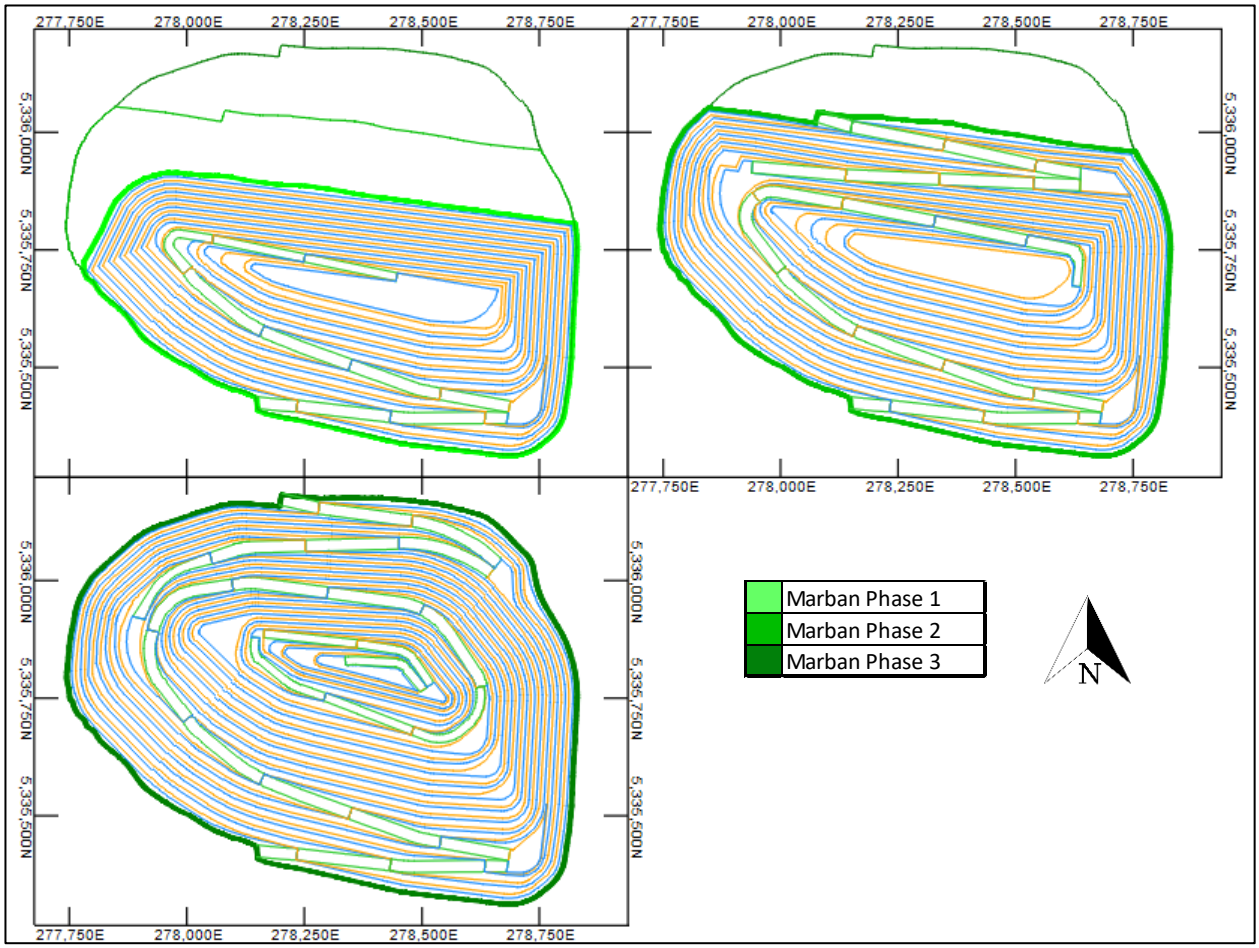
Marban Phase 1 is the smallest nested pit. This phase starts from the final wall and ramp along the footwall which will be expanded in later phases. Marban Phase 1 is 1 km long, 600 m wide, and has a maximum depth of 200 m.

Marban Phase 2 expands from the ramp created in Marban Phase 1. A new temporary ramp is driven along the pushback on the north wall. Once the depth of Marban Phase 2 exceeds that of Phase 1, it will switch its access ramp to the primary ramp along the footwall and continue it down. Marban Phase 2 is 1.1 km long, 500 m wide, and has a maximum depth of 240 m.

Marban Phase 3 is the final pit of the Marban Pit Group. Marban Phase 3 contains a 100 m pushback from Marban Phase 2 and drives a ramp along the north wall. The north ramp will only be used until the depth of Marban Phase 3 exceeds that of Marban Phase 2, then the south primary ramp will be driven to the bottom of the pit. Marban Phase 3 is 900 m long, 540 m wide, and has a maximum depth of 360 m.

The different Marban phases are presented in Figure 16-2.

Figure 16-2: Marban Phase Designs (Clockwise: Phase 1, Phase 2, Phase 3)



Source: GMS, 2022.

16.2.2.2 Norlartic

The Norlartic Pit group is the second largest pit group and produces 24% of all recovered ounces for the project. Located in the north-central area, the Norlartic Pit group is constrained by the mining limits to the east and north. The Norlartic Pit Group is composed of a large pit split into two phases (Norlartic Phase 1, Norlartic Phase 2) and two small satellite pits surrounding it (Norlartic Sub Pit 1, Norlartic Sub Pit 2). The Norlartic pit contains voids from historical underground mining which are avoided along final walls and ramps to prevent structural weakness in the pit. Norlartic Phase 1 is guided via pit shell 29 and subsequent pits and phases by the final selected Whittle™ shell. Single lanes are used for the two lowest levels of the pit. Efforts are being made to ensure that pits exit south to create the shortest haul route to all destinations. The Norlartic Pit group lies underneath Keriens Creek, which will require relocation before mining commences. When the Kierens and Norlartic Pit Groups are completed, they will be used as tailing depositories to eliminate the requirement to build large tailing dams elsewhere. These groups will need to be finished by Year 4 to accommodate the tailings deposition plan.

Norlartic Phase 1 is the northernmost ore pocket of the main Norlartic pit group. The pit contains two switchbacks with plans to join with Norlartic Phase 2 along the first switchback. Norlartic Phase 1 is 530 m long, 350 m wide, and has a maximum depth of 100 m.

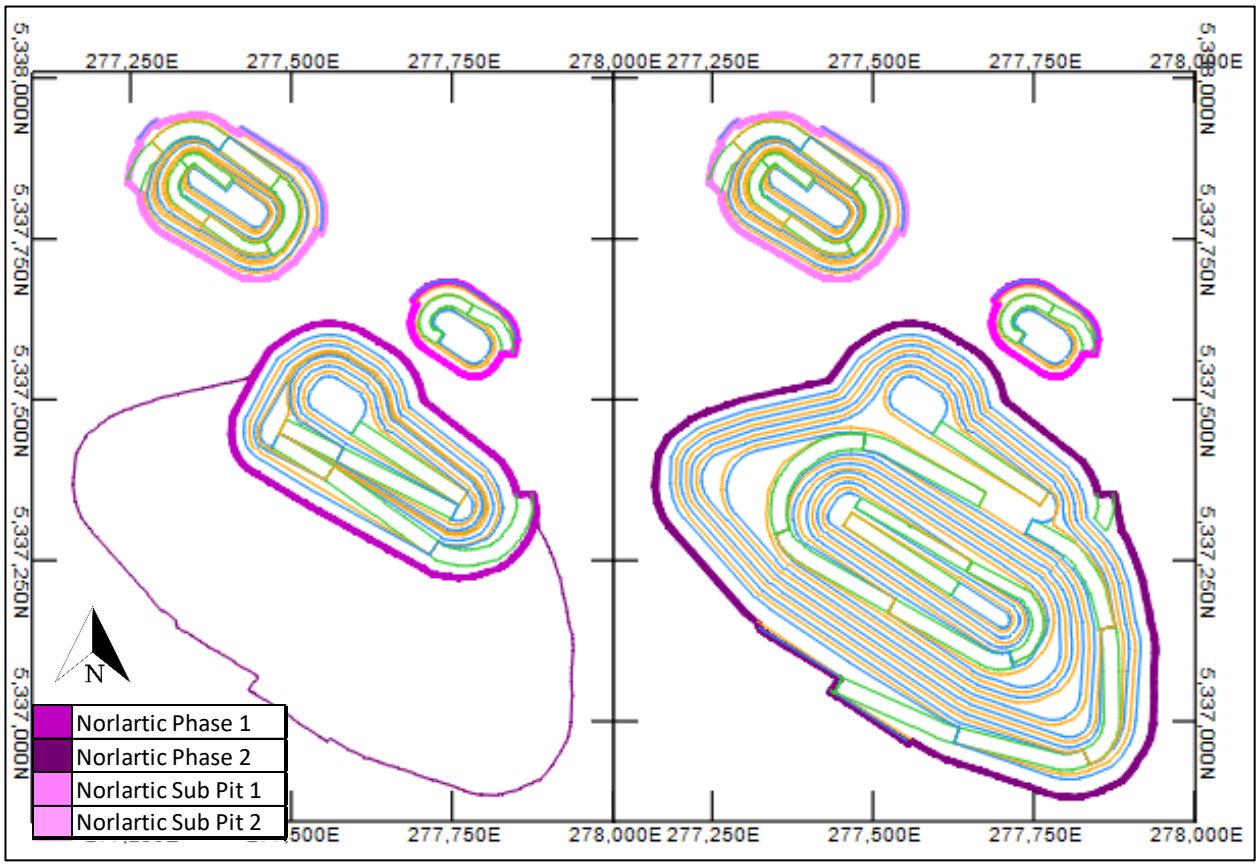
Norlartic Phase 2 is the second phase and expansion to the Norlartic Phase 1. A new ramp is driven from the south that combines with the switch back from Norlartic Phase 1 joining the two pits together before continuing to the pit bottom. Norlartic Phase 2 is 900 m long, 430 m wide, and has a maximum depth of 220 m.

Norlartic Sub Pit 1 is 300 m long, 200 m wide, and has a maximum depth of 80 m.

Norlartic Sub Pit 2 is 180 m long, 120 m wide, and has a maximum depth of 30 m.

The different Norlartic phases are presented in Figure 16-3.

Figure 16-3: Norlartic Designs



Source: GMS, 2022.

16.2.2.3 Kierens

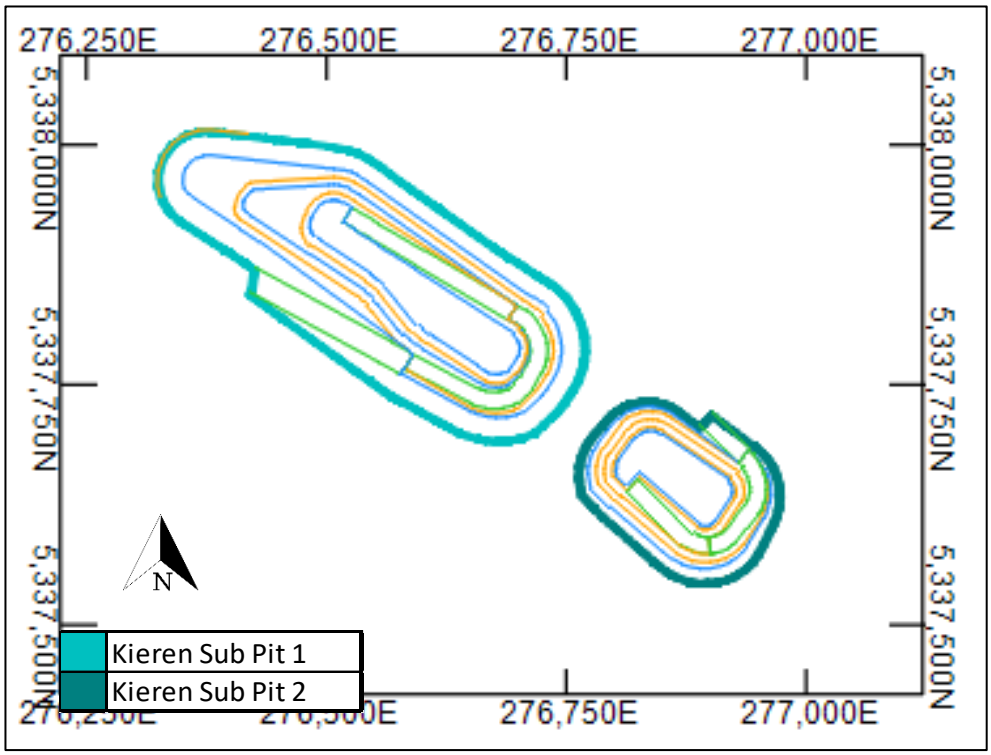
The Kierens Pit Group is the smallest of all pit groups and produces 2% of all gold produced. Located northwest, the Kierens Pit Group is constrained by mine limits only along the northernmost edge of Kierens Sub Pit 1. Composed only of two sub-pits (Kierens Sub Pit 1 and Kierens Sub Pit 2), due to their small size and limited depth, only single lanes are required.

Kierens Sub Pit 1 is 500 m long, 200 m wide, and has a maximum depth of 60 m.

Kierens Sub Pit 2 is 200 m long, 160 m wide, and has a maximum depth of 40 m.

The different Kierens phases are presented Figure 16-4.

Figure 16-4: Kierens Design



Source: GMS, 2022.

16.3 Waste & Stockpile Storage Facilities

A total of 286.1 Mt of waste rock and overburden will be produced over the mine life. Rock material will be sent to the Primary Dump and the Primary Dump Extension. Overburden material will be sent either to the Marban OVB Stockpile if sourced from the Marban Pit Group and to the Kierens OVB Stockpile if sourced from the Norlartic or Kierens Pit groups. The Primary dump will be done into two phases with the extension to allow the shortest possible cycle times from the corresponding pits. The Primary Dump will be predominantly filled from Norlartic and Kierens Pit Groups and the extension from the Marban Pit Group. The two phases of primary dump will allow for two-year deferral of grubbing and site preparation for the dumps. An estimated 17% of the waste will be overburden and so it was optimal to have to overburden stockpiles near the pit groups to significantly reduce long distance haulage. Dumps are planned in order to minimize the haulage distance from their respective sources. Dump accesses face the pit exits and main haul routes.

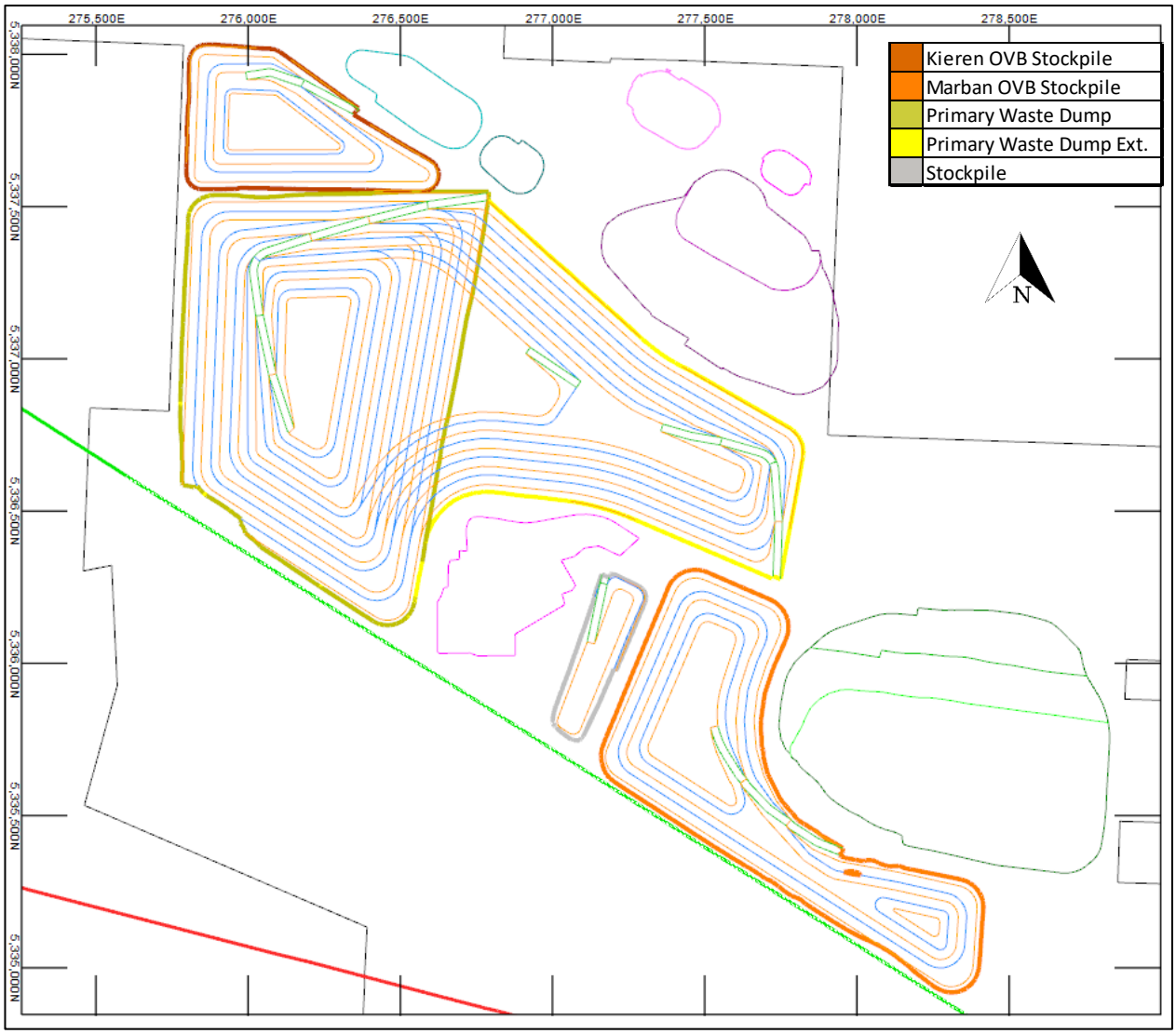
A total of 1.4 Mt of waste rock and 1.9 Mt of overburden are planned for construction. These values include the building of the initial tailings dam, site laydowns, processing plant, and haulage roads.

Figure 16-5 depicts the layout of the dumps and stockpiles. Table 162 depicts the design parameters and inventories of the dumps. All dumps are created using an Overall Slope Angle (OSA) of 36 degrees. The ore stockpile is designed to have additional capacity to allow for variations in the mine plan and to compensate for potential stockpile spikes within periods.

The ore pad is also expected to facilitate ore of different grades. An oversized pad will allow for variable sizes of stockpiled material as required. Ore bins are as follows:

- Low grade: 0.3 to 0.5 g/t;
- Medium grade: 0.5 to 1.0 g/t; and
- High grade: +1.0 g/t.

Figure 16-5: Waste Storage Facilities



Source: GMS, 2022.

Table 16-2: Dump Capacities and Parameters

	Capacity (Mm^3)	Capacity Used (Mm^3)	% Filled	OSA (degrees)
Primary	61.9	60.7	98%	36
Primary Ext.	50.3	50.3	100%	36
Marban OVB Dump	17.2	15.2	88%	36
Norlartic OVB Dump	7.7	7.3	95%	36
Ore Stockpile	1.00	0.2	24%	36

16.4 Mine Haul Roads

Mine haul roads are designed to efficiently connect all pit exits to all possible haulage destinations, such as the fuelling bays, truck shop, and construction locations. There are 7.8 km of off-highway haulage roads. Figure 16-6 depicts the haulage roads in red. The southernmost road goes to the initial tailings dam as crush material and will be sourced from the pits for construction purposes. This road goes under the powerlines and will require the powerlines to be raised to allow clearance for the 150-t trucks. Figure 16-7 depicts the design criteria for the haulage roads.

Figure 16-6: Mine Haul Road Layout

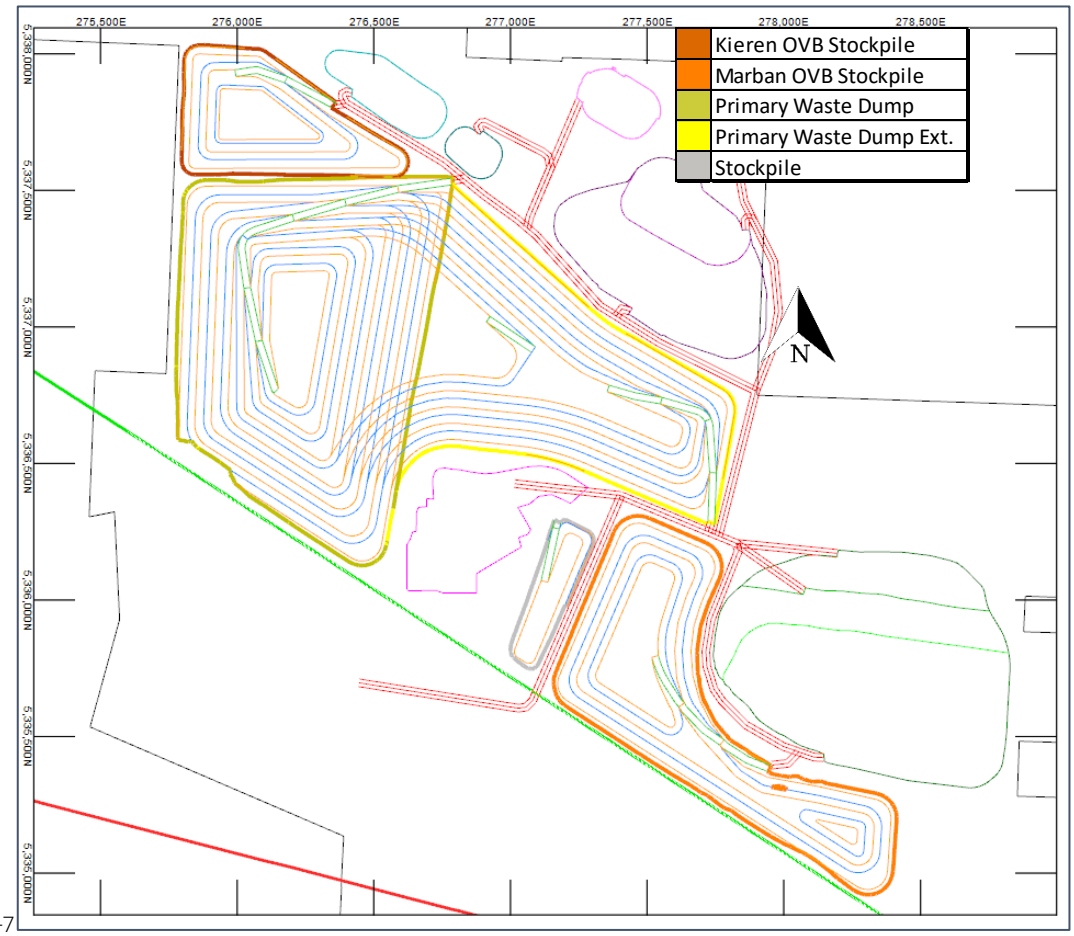
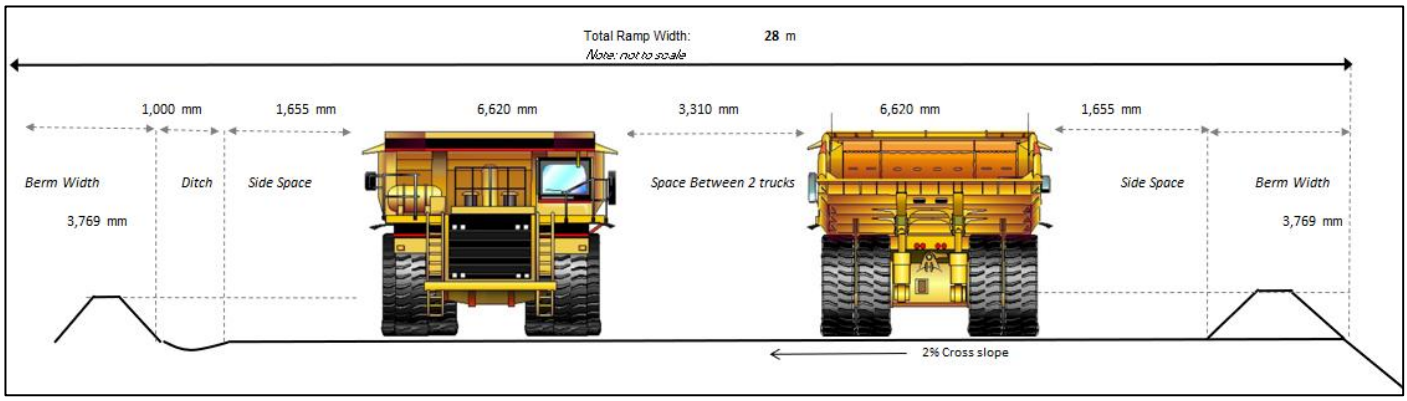


Figure 16-7

Source: GMS, 2022.

Figure 16-7: Ex-Pit Haul Road Design Criteria



Source: GMS, 2022.

16.5 Production Schedule

The life-of-mine production schedule was optimized using Minemax^{tm} Scheduler, which is an industry leading schedule optimizer using best in class CPLEX technology. Minemax^{tm} Scheduler is an automated mine scheduling tool which leverages multi-period optimization to determine maximum net present value (NPV) while imposing various physical constraints and targets. The optimization includes mine sequencing and mining rate, stockpile usage and rehandling, and fleet usage. The strategic optimal plan from Minemax^{tm} on an annual basis was then further detailed by month using Deswik^{tm} to track material movements, stockpile inventory, mill blending, waste movements and equipment usage / movements.

16.5.1 Mining Schedule

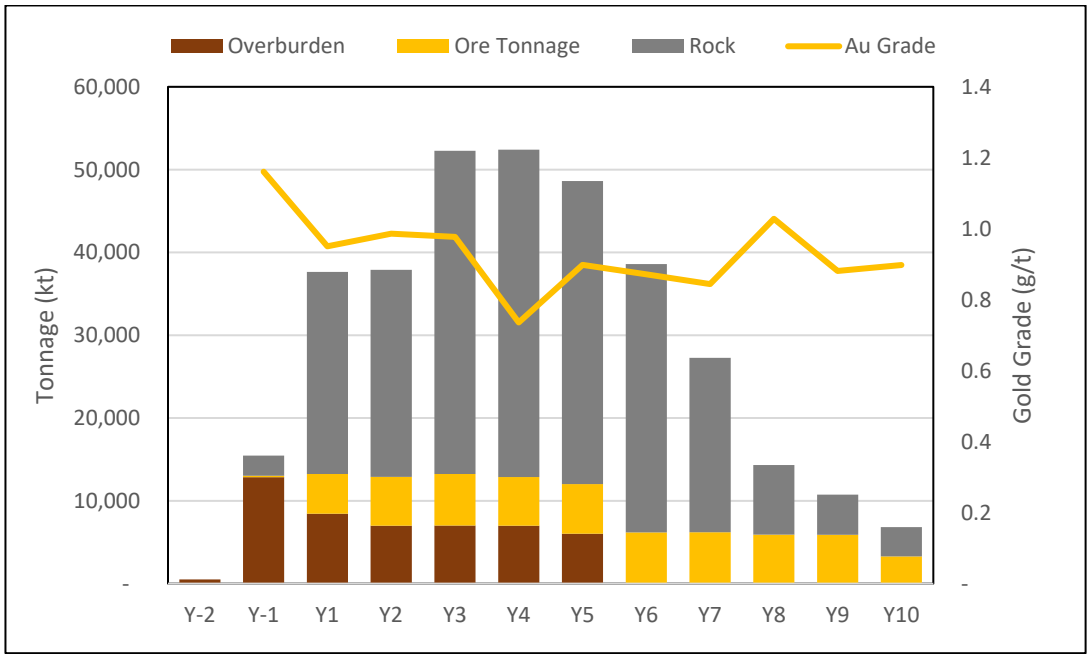
Mining activities are planned over a duration of 12 years which includes 2 years of pre-production mining. Once the open pit is depleted and mining activities are stopped, stockpile reclaim continues for another 1 month to continue feeding the mill. The mining rate reaches a peak of 52.3 Mt/y in Year 3 of commercial production. Figure 16-8 presents the mining schedule by material type and gold grade.

Figure 16-9 presents the tonnage mined from the various pit groups over the mine life. In any given year there are up to two active mining phases at once, where generally one phase is the primary source of ore and the other is being stripped. The mine plan kept a maximum sinking rate of 60 m. Special consideration is made to ensure that Norlartic and Kierens pits are finished by Year 4 to allow depositions of tailings in the pit.

Mine production details showing mined grades and material movement are presented in Table 16-3.

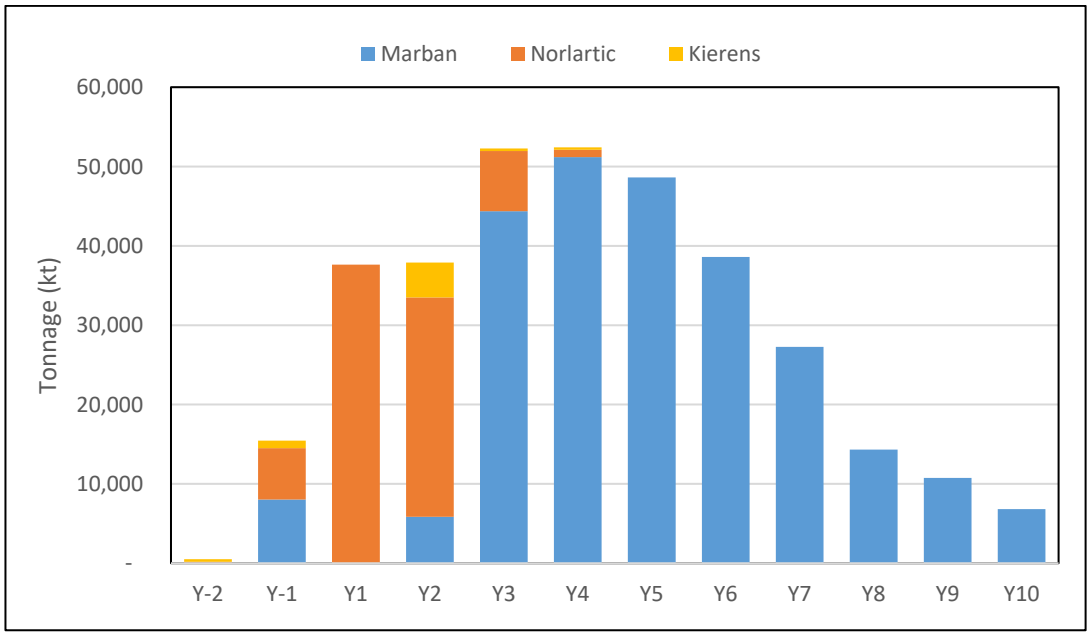
Figure 16-10 through Figure 16-13 depict the mine progression for the selected years as well as the end of activity at Year 10.

Figure 16-8: Mine Production by Material Type



Source: GMS, 2022.

Figure 16-9: Mine Production Schedule by Pit Group



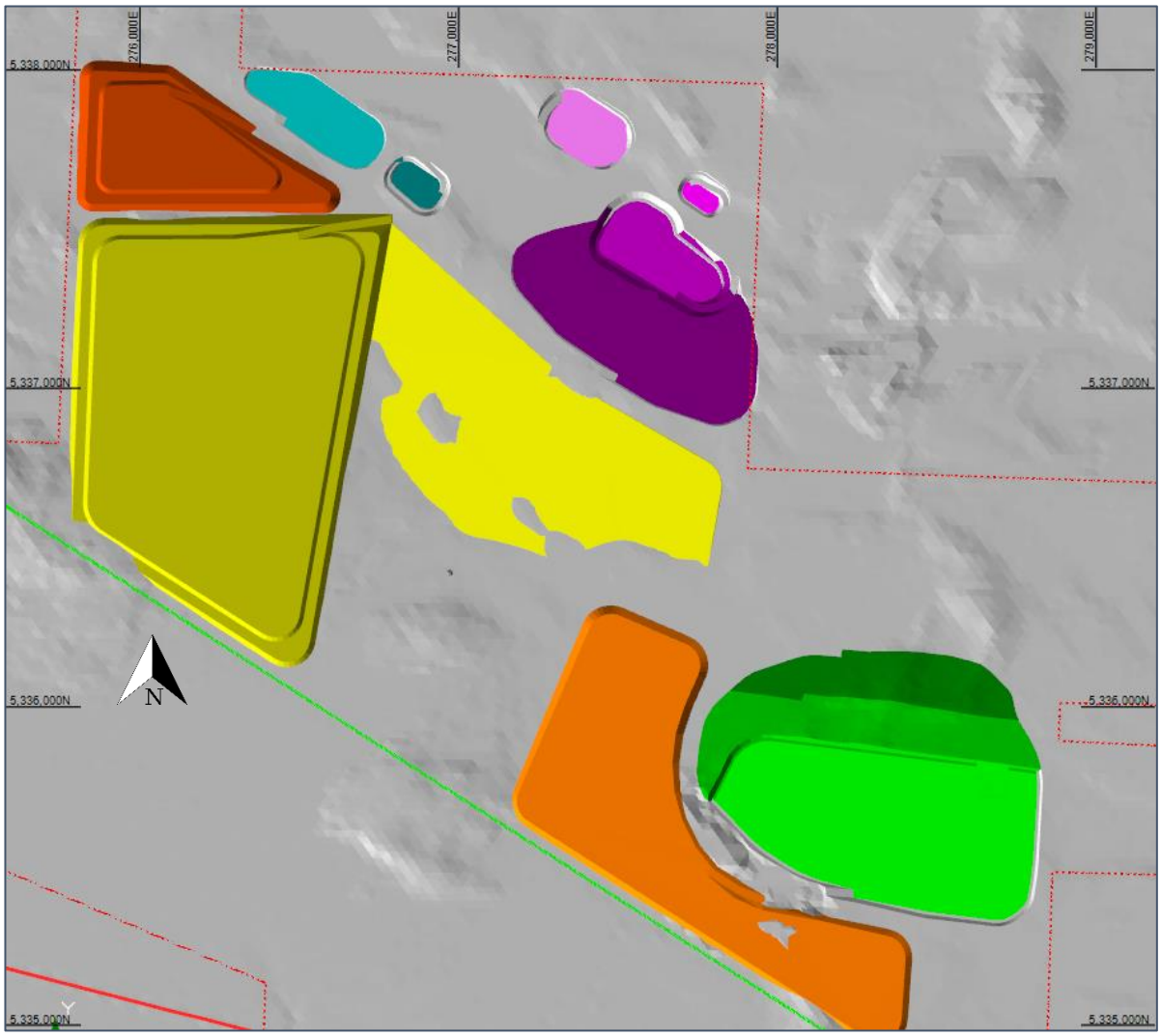
Source: GMS, 2022.

Figure 16-10: Mine Development – Y1

	Marban Phase 1
	Marban Phase 2
	Marban Phase 3
	Norlartic Phase 1
	Norlartic Phase 2
	Norlartic Sub Pit 1
	Norlartic Sub Pit 2
	Kieren Sub Pit 1
	Kieren Sub Pit 2

	Kieren OVB Stockpile
	Marban OVB Stockpile
	Primary Waste Dump
	Primary Waste Dump Ext.
	Stockpile

Figure 16-7



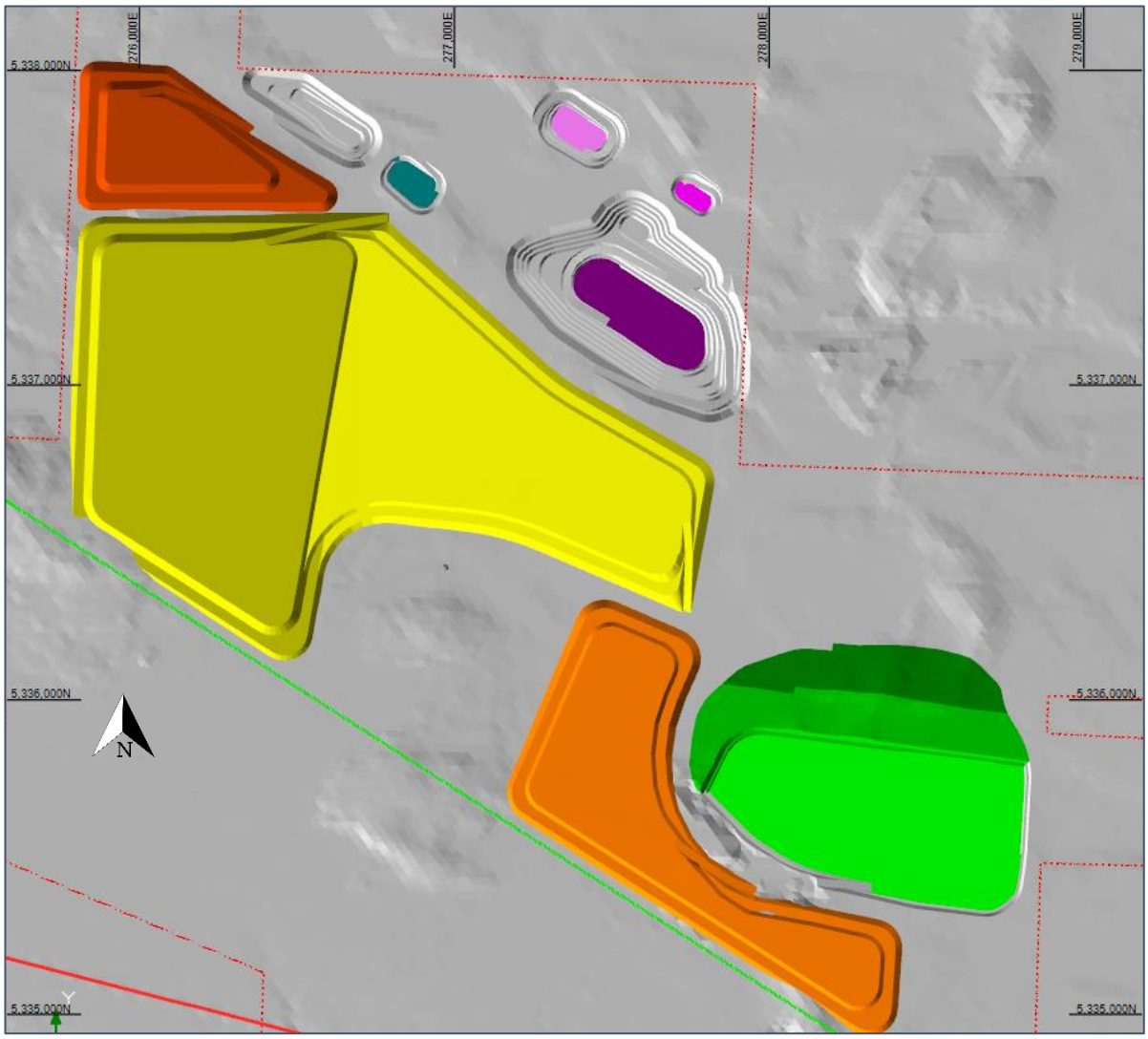
Source: GMS, 2022.

Figure 16-11: Mine Development – Y3

	Marban Phase 1
	Marban Phase 2
	Marban Phase 3
	Norlartic Phase 1
	Norlartic Phase 2
	Norlartic Sub Pit 1
	Norlartic Sub Pit 2
	Kieren Sub Pit 1
	Kieren Sub Pit 2

	Kieren OVB Stockpile
	Marban OVB Stockpile
	Primary Waste Dump
	Primary Waste Dump Ext.
	Stockpile

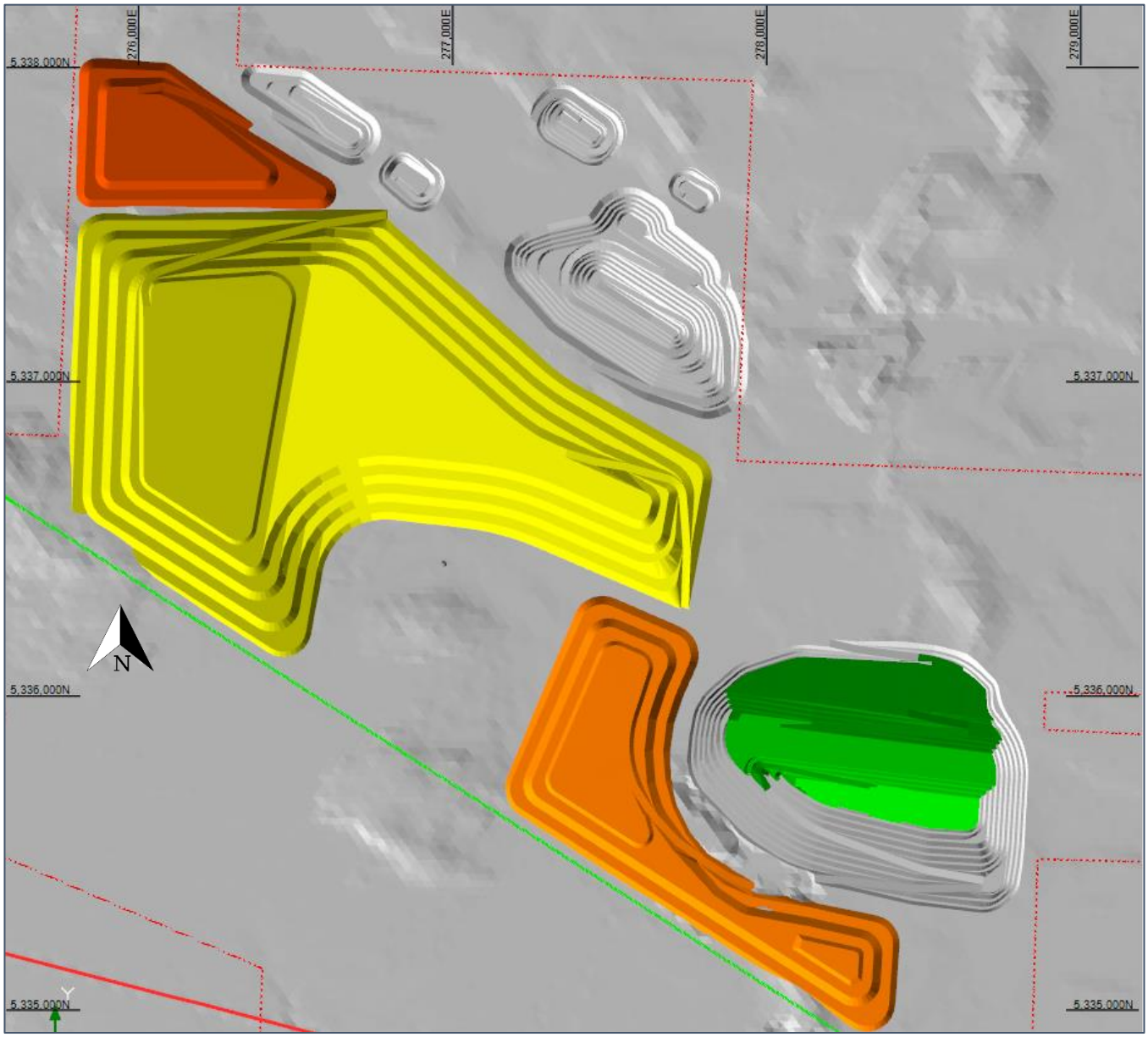
Figure 16-7



Source: GMS, 2022.

Figure 16-12: Mine Development – Y6

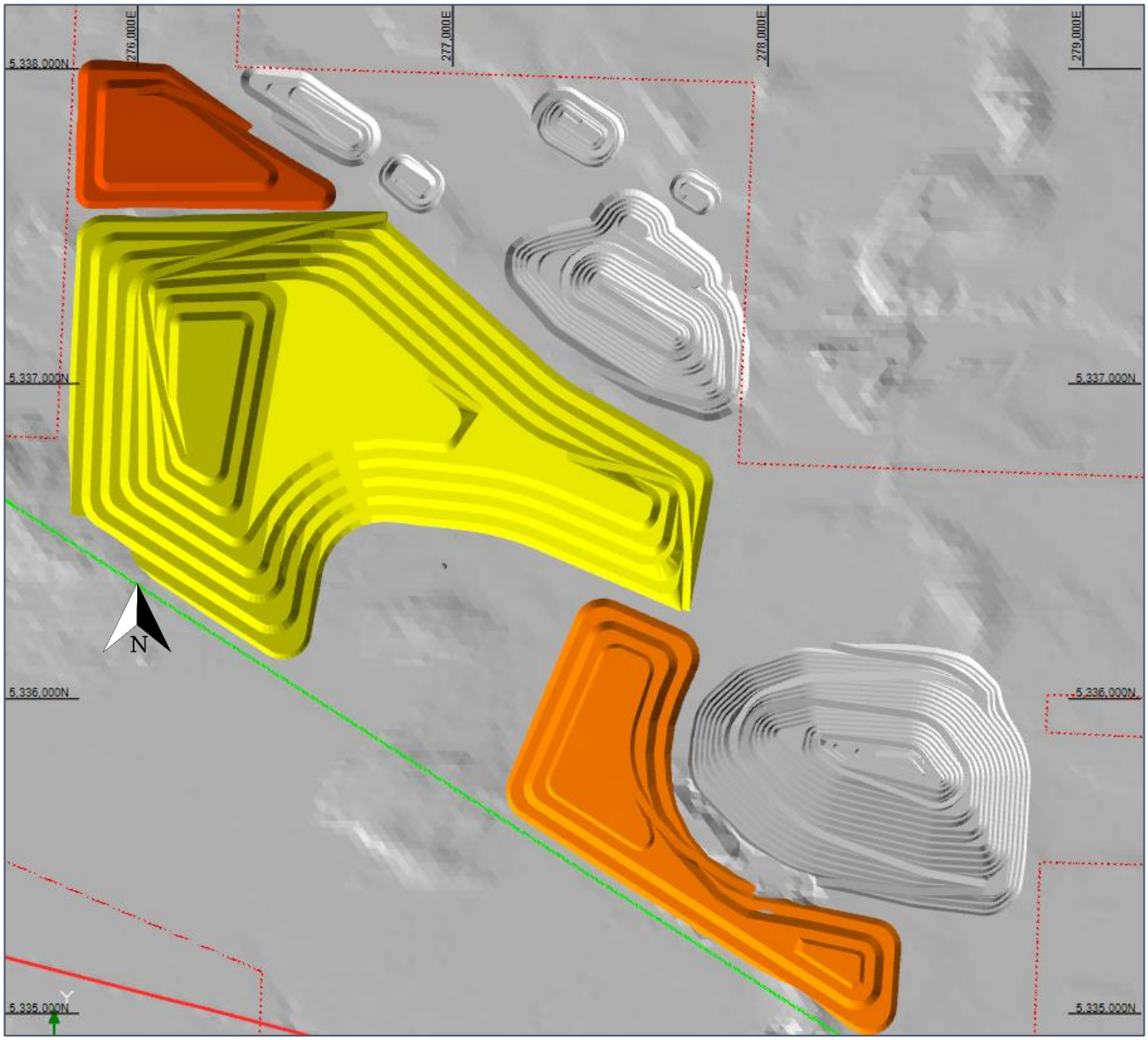
	Kieren OVB Stockpile
	Marban OVB Stockpile
	Primary Waste Dump
	Primary Waste Dump Ext.
	Stockpile



Source: GMS, 2022.

Figure 16-13: Mine Development – Y10

	Kieren OVB Stockpile
	Marban OVB Stockpile
	Primary Waste Dump
	Primary Waste Dump Ext.
	Stockpile



Source: GMS, 2022.

Table 16-3: Mining Production Schedule Summary

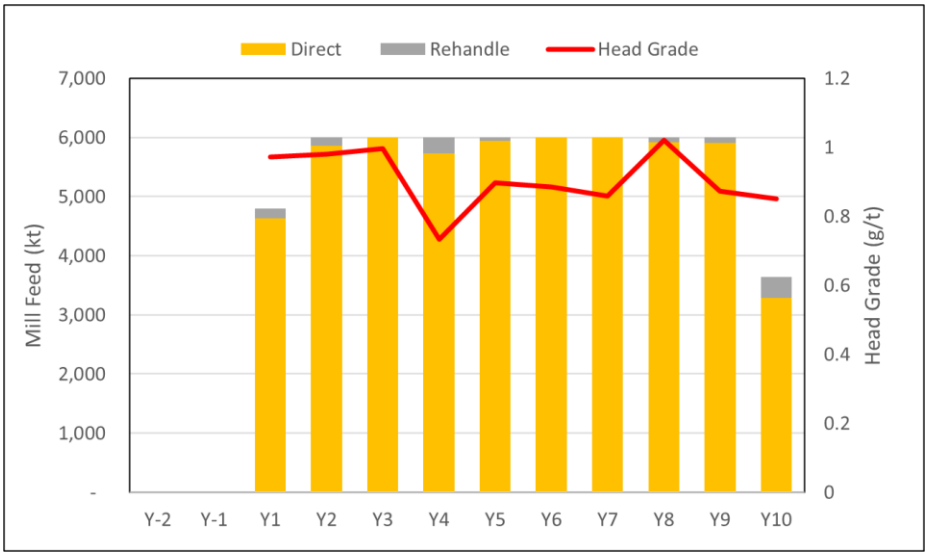
	Unit	LOM	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Total Tonnage	000 t	342,580	500	15,460	37,642	37,892	52,281	52,407	48,618	38,604	27,273	14,330	10,748	6,828
Total Moved	000 t	343,762	500	15,460	37,809	38,035	52,281	52,678	48,680	38,604	27,273	14,414	10,846	7,182
Waste Tonnage	000 t	286,144	500	15,287	32,844	31,986	46,055	46,538	42,629	32,436	21,071	8,412	4,841	3,545
Overburden	000 t	48,864	500	12,857	8,440	6,997	7,014	7,003	6,039	13	-	-	-	-
Rock	000 t	237,280	0	2,430	24,404	24,989	39,040	39,535	36,590	32,422	21,071	8,412	4,841	3,545
Strip Ratio	W:O	5.07	-	88.71	6.84	5.42	7.40	7.93	7.12	5.26	3.40	1.42	0.82	1.08
Ore Tonnage	000 t	56,436	-	172	4,798	5,906	6,226	5,869	5,989	6,168	6,202	5,918	5,907	3,282
Au Grade	g/t	0.907	-	1.16	0.95	0.99	0.98	0.74	0.90	0.87	0.84	1.03	0.88	0.90
Ore Milled Total	000 t	56,436	-	-	4,800	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,636
Head Grade	g/t	0.91	-	-	0.97	0.98	1.00	0.73	0.90	0.88	0.86	1.02	0.87	0.85

16.5.2 Processing Schedule

The mill schedule includes a ramp-up year at 80% nameplate throughput after which commercial throughput of 6.0 Mt/a is achieved for 8.5 years. Mill feed is maximized with direct feed from the pit and rehandled stockpiled material. Figure 1614 presents the mill feed by resource and the average head grade per period.

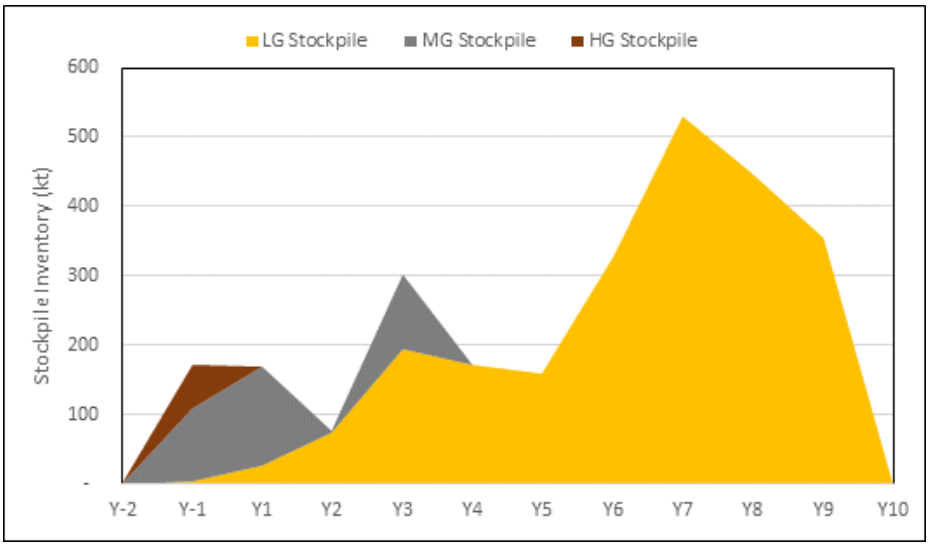
Figure 16-15 depicts the end of year (EOY) stockpile inventories. High and medium grade material is prioritized to mill and rehandle to increase early gold production. Stockpile inventories peak in Year 7 with 0.5 Mt of low-grade material that will be milled at the end of mine life.

Figure 16-14: Mill Feed



Source: GMS, 2022.

Figure 16-15: End of Year Stockpile Inventories



Source: GMS, 2022.

Table 16-4: Milling Production Schedule

	Unit	LOM	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Ore Milled Total	000 t	56,436	-	-	4,800	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,636
Head Grade	g/t	0.91	-	-	0.97	0.98	1.00	0.73	0.90	0.88	0.86	1.02	0.87	0.85
Recovered Gold	koz	1,551.4	-	-	138	175	181	134	164	162	157	187	160	94
Avg. Rec.	%	0.94	-	-	0.92	0.92	0.94	0.95	0.95	0.95	0.95	0.95	0.95	0.95
Direct to Mill	000 t	55,254	-	-	4,632	5,857	6,000	5,729	5,937	6,000	6,000	5,916	5,901	3,282
Grade Au	g/t	0.92	-	-	0.96	0.99	1.00	0.74	0.90	0.88	0.86	1.03	0.88	0.90
From Stockpile	000 t	1,182	-	-	168	143	-	271	63	-	-	84	99	354
Grade	g/t	0.52	-	-	1.09	0.50	-	0.45	0.38	-	-	0.39	0.39	0.39
To Stockpile	000 t	1,182	-	172	166	49	226	140	51	168	202	2	6	-
Grade	g/t	0.52	-	1.07	0.49	0.40	0.46	0.38	0.35	0.39	0.40	0.33	0.38	-
Stockpile Inventory	000 t	529.2	-	172	171	76	302	171	159	327	529	447	354	0
Grade	g/t	0.44	-	1.07	0.48	0.39	0.45	0.38	0.37	0.38	0.39	0.39	0.39	-

16.6 Mine Operations and Equipment Selection

16.6.1 Drilling and Blasting

Production drilling is planned on 10 m benches using 6.5-inch (165.1 mm) diameter holes. Production drills capable of single-pass drilling specifically designed for 10 m benches with a hole range of 152 mm to 270 mm were selected, which are capable of rotary drilling or down-the-hole (DTH) drilling.

Drill and blast specifications were established according to material type and whether ore or waste. A uniform drill pattern is proposed with a 5.1 m burden and 5.1 m spacing with 1.5 m of sub-drill in fresh rock. These drill parameters combined with a high energy bulk emulsion with a density of 1.2 kg/m^3 result in a powder factor of 0.31 kg/t for ore and 0.29 kg/t for waste. Blast holes are planned to be initiated with non-electric detonators and primed with boosters.

Table 16-5 presents the drill patterns and production drill parameters.

Table 16-5: Drill and Blast Parameters

Drill & Blast Parameters	Unit	Ore	Waste
Drill Pattern			
Diameter (D)	m	0.165	0.165
Burden (B)	m	5.10	5.20
Spacing (S)	m	5.10	5.20
Subdrill (J)	m	1.50	1.50
Stemming (T)	m	3.00	3.00
Pattern Yield			
Rock density	t/BCM	2.70	2.77
BCM/hole	BCM/hole	260	270
Yield per hole	t/hole	702	749
Yield per metre drilled	t/m drilled	61	65
Powder factor	kg/t	0.31	0.29
Drill Productivity			
Re-drills	%	5.0%	5.0%
Pure penetration rate	m/hr	35.0	35.0
Overall drilling factor (%)	%	0.50	0.50
Overall penetration rate	m/hr	17.5	17.5
Drilling efficiency	t/hr	1,069	1,140
Drilling efficiency	holes/hr	1.52	1.52

Controlled blasting techniques will be used including buffer blasts and pre-splits. The pre-split consists of closely spaced holes along the design excavation limit. The holes are loaded with a light charge and detonated simultaneously or in groups separated by short delays. Firing the pre-split row creates a crack that forms the excavation limit and helps to prevent wall rock damage by venting explosive gases and reflecting shock waves.

A pre-split drill rig (4.5–8 inch) was selected for this application; additionally, it will be used for pioneering work due to its mobility and drilling range. Pre-split drill will also be responsible for topo drilling and small-scale blasts.

Blasting activities will be outsourced to an explosives provider who will be responsible for supplying and delivering explosives in the hole through a shot service contract. The mine engineering department will be responsible for designing blast patterns, relaying hole information to the drilling team, and supervising all blasting activities.

16.6.2 Loading

The primary loading fleet consists of three 16 m^3 electric hydraulic shovels, one 12 m^3 diesel electric shovel, and a 10.7 m^3 diesel front end loader. The loading fleet is augmented by the overburden (secondary) fleet consisting of four 60 t shovels to handle overburden material, scale pit walls, and augment the primary fleet. The 60t shovels will be predominately loading 100-t trucks while the primary production loading units will be servicing the 150-t fleet.

The secondary OVB loading shovels were chosen due to the reduced production of the larger equipment when loading overburden material. A smaller set of shovels and trucks were required to better access the topo and strip the bedrock and prep the surfaces for large scale mining.

The primary loading fleet is capable of loading the 100-t trucks at a reduced production rate and will be used to cover gaps in the fleet requirements. The OVB fleet is not capable of loading the 150-t trucks. The OVB fleet will operate up to Year 5, moving on average 8 Mt of OVB per year. The schedule was optimized to ensure that the secondary fleet will have a consistent OVB feed to best use the equipment to its potential.

To reduce the dilution of the ore, it is planned to excavate the ore in two 5-metre passes.

Table 16-6: Primary Fleet Loading Productivity Assumptions

		Ore / Waste		
Loading Unit		16 m^3 Electric Hydraulic Shovel	12 m^3 Diesel Electric Shovel	10.7 m^3 Diesel Front End Loader
Haulage Unit		150t Truck	150t Truck	150t Truck
Rated Truck Payload	t	144	144	144
Heaped Tray Volume	m^3	78	78	78
Bucket Capacity	m^3	16	9.1	11.4
Bucket Fill Factor	%	90%	90%	90%
Passes (whole)	#	5.0	9.0	7.0
Production / Productivity				
Productivity Dry Tonnes	t/hr	1869	1067	1125
Production / Productivity				
Dry Annual Production Capacity	kt/yr/unit	11,215,385	6,404,012	6,752,363
Number of Units	#	4	1	1
Productivity	t/yr	44,861,538	6,404,012	6,752,363

16.6.3 Hauling

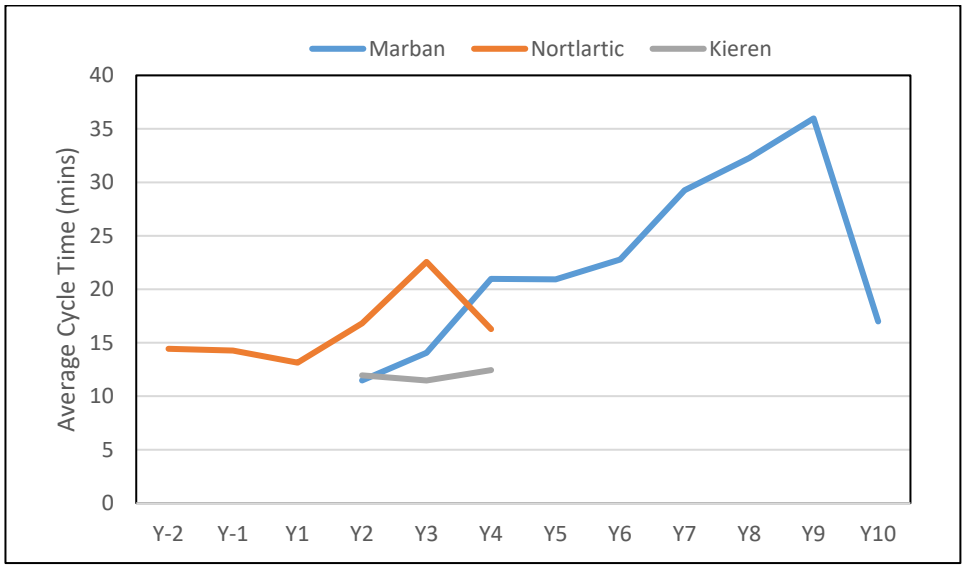
Haulage will be preformed by two sets of off-highway mining trucks. The 150-t fleet will be responsible for the majority of rock material and will be loaded via the primary production loading fleet. The 100-t will be responsible for the overburden material and will be loading predominantly by the secondary overburden fleet with the option to augment the main hauling fleet if required.

The truck requirements haven been calculated in Deswik LHS (Landform and Haulage) software. This software links the mining schedule to the waste movements to determine optimal haulage routes and simulates them using Rimpull data from the fleet. The following assumptions were used when running the simulations.

- Max site speed limit of 50 km/hr;
- Max speed loaded and downhill of 30 km/hr;
- Average rolling resistance of 3%; and
- A fixed time of loading and spotting time is variable depending on the loader, truck, and material. This value ranges from 4–7 mins.

Multiple waste and overburden stockpiles are used to reduce distance and cycle time with the goal to reduce the peak requirement of trucks. Material is always sent to the closest available location. Figure 16-16 depicts average cycle time of rock from each source. Note that cycle time increases as pits get deeper due to increased uphill required haulage. Plateaus or dips in the cycle time represent transitions to new pushbacks starting from the surface, temporarily reducing cycle time. Marban is the deepest pit and has the longest cycle time maxing at 36 mins at the end of mine life; note that this does not include fixed times. The sudden drop off in the final year represents the minimal mining done at the pit bottom, which goes directly to the mill or the stockpile as near zero waste is mined in that period and no haulage is done up the waste pile.

Figure 16-16: Rock Cycle Times by Source Pit Group



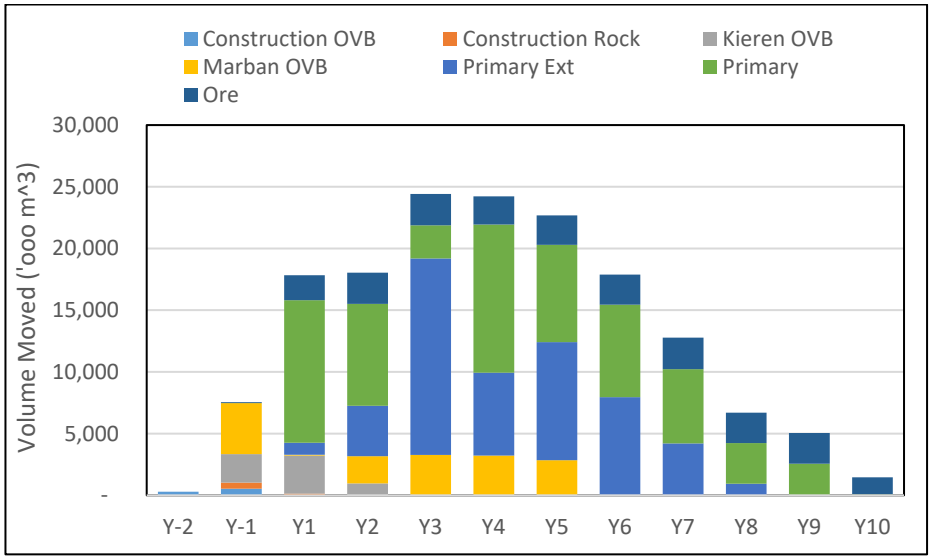
Source: GMS, 2022.

Figure 16-17 depicts the mass balance of the project. Ore material is any material going to the plant or the stockpiles. Construction requirements by material type is included as a destination. A total of 0.88 Mm^3 of overburden and 0.65 Mm^3

of waste rock is planned for construction use in pre-production and Year 1. This includes the tailing dam, site laydowns, pads, and road construction.

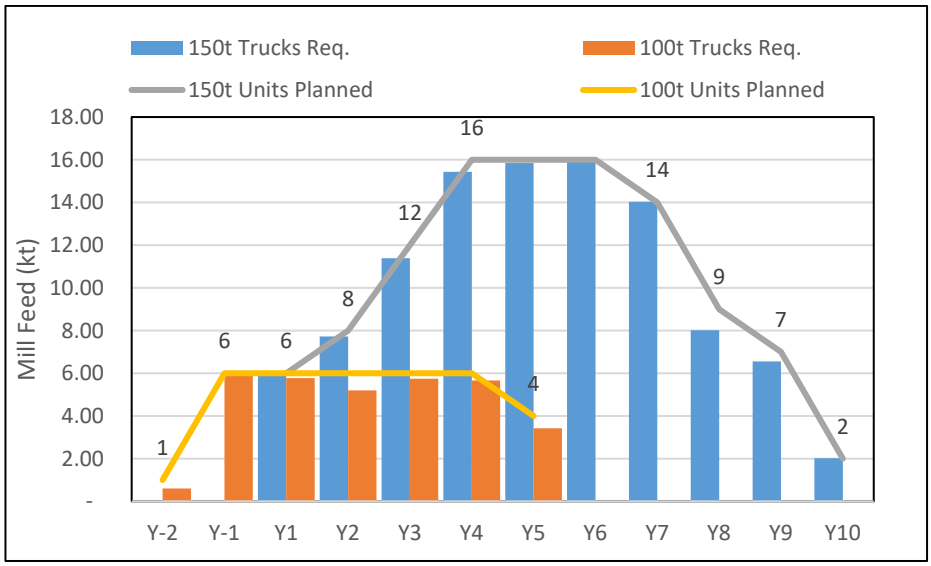
Figure 16-18 depicts the total fleet requirements by truck. Sixteen 150-tonne trucks are required to maintain production at peak mining rate; this is held through Year 4–Year 6 until it starts to wind down as mining rate decreases. The 100-t trucks peak at 6 trucks at Year 1 in pre-production and are used until Year 5 when all overburden has been mined out. There are no repurchasing trucks due to the short life of mine and the trucks bought are used until end of life.

Figure 16-17: Material Movements



Source: GMS, 2022.

Figure 16-18: Truck Requirements



Source: GMS, 2022.

16.6.4 Support Operations

Support equipment requirements are based on typical open pit mine operation and maintenance requirements to safely support the loading, hauling, and drilling fleets.

Support equipment is planned for maintaining dump areas, stockpiles, pit floors and mine roads. The fleet of support equipment consist of the following:

- 7 x 436 HP dozers for face clean-up and dump maintenance;
- 3 x 16-ft blade motor graders for road upkeep;
- 1 x water trucks for dust suppression; and
- 2 x 496 PH Wheel Dozers.

16.6.5 Dewatering

Dewatering is planned to be serviced via four electrical submersible pumps that will be moved as required as only four phases / pits are mined at once. Pump head is calculated via the pump performance curves, piping specifications and pit depth. All pits require a single pump to successfully pump the water and allow mining except for Marban which will require a booster pump in Year 7 to overcome the additional head from pit depth.

Water inflows are estimated from yearly values. Additional factor of safety is placed on the pumps to account for larger than normal weather events and to overcome flooded mine voids located in the pits.

16.6.6 Mine Fleet Requirements

Table 16-7 summarizes the gross operating hours used for subsequent equipment fleet requirement calculations. The mine is expected to operate 22 hours per day, 355 days per year. This accounts for shift changes and weather delays from heavy rain events. Additional delays and applied factors are described in productivity calculations for each fleet.

Table 16-7: Equipment Usage Assumptions

	Unit	Shovels	Loaders	Trucks	Drills	Ancillary	Support
Days in period	days	365	365	365	365	365	365
Schedule Outages	days	10.0	10.0	10.0	10.0	10.0	10.0
Shifts per day	shift/day	2.0	2.0	2.0	2.0	2.0	2.0
Hours per shift	hrs/shift	12.0	12.0	12.0	12.0	12.0	12.0
Availability	%	82.0	80.0	85.0	80.0	85.0	85.0
Use of Availability	%	90.0	90.0	90.0	90.0	85.0	80.0
Utilization	%	73.8	72	76.5	72	72.25	68
Effectiveness	%	87.0	85.0	87.0	85.0	80.0	80.0
Overall Equipment Efficiency	%	64.2	61.2	66.6	61.2	57.8	54.4
Total Hours	hrs	8,760	8,760	8,760	8,760	8,760	8,760
Scheduled Hours	hrs	8,520	8,520	8,520	8,520	8,520	8,520
Down Hours	hrs	1,534	1,704	1,278	1,704	1,278	1,278
Delay Hours	hrs	817	920	847	920	1,231	1,159
Standby Hours	hrs	699	682	724	682	1,086	1,448
Operating Hours	hrs	6,288	6,134	6,518	6,134	6,156	5,794
Ready Hours	hrs	5,470	5,214	5,670	5,214	4,925	4,635

Table 16-8: Major Equipment Requirement Schedule

Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Production Drill (6–10 inch)	4	-	-	3	3	4	4	4	4	3	2	2	1
Auxiliary Pre-split Drill (4.5–8 inch)	1	-	1	1	1	1	1	1	1	1	1	1	1
Electric Hydraulic Shovel (16 m^3)	3	-	-	2	2	3	3	3	3	2	1	1	1
Diesel Hydraulic Excavator (12 m^3)	1	-	-	1	1	1	1	1	1	1	1	1	1
Wheel Loader (10.7 m^3)	1	-	1	1	1	1	1	1	1	1	1	1	1
Mining Haul Truck (150 t)	16	-	-	6	8	12	16	16	16	14	9	7	3
Mining Haul Truck (100 t)	6	1	6	6	6	6	6	4	-	-	-	-	-
Track Dozer (436 HP)	8	-	4	7	7	8	8	7	5	4	3	3	3
Motor Grader (16 ft)	2	-	1	2	2	2	2	2	2	2	1	1	1
Water / Sand Truck (76 kL tank)	1	-	1	1	1	1	1	1	1	1	1	1	1
Wheel Dozer (496 HP)	1	-	1	1	1	1	1	1	1	1	1	1	1

Table 16-9: Support Equipment Requirement Schedule

Equipment	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Excavator (49 t)	1	-	1	1	1	1	1	1	1	1	1	1	1
Excavator (90 t)	4	1	4	4	4	4	4	3	1	1	1	1	1
Hydraulic Hammers for Excavator 49 t	1	-	-	1	1	1	1	1	1	1	1	1	1
Wheel Loader 271 HP	1	-	1	1	1	1	1	1	1	1	1	1	1
Cable Handling Wheel Loader 271 HP	1	-	-	1	1	1	1	1	1	1	1	1	1
Boom Truck 28 t	1	-	1	1	1	1	1	1	1	1	1	1	1
Telehandler	1	-	1	1	1	1	1	1	1	1	1	1	1
Forklift Diesel 4 t	1	-	1	1	1	1	1	1	1	1	1	1	1
Mechanic Service Truck	3	-	2	3	3	3	3	3	3	3	2	2	2
Fuel & Lube Truck 10 Wheel	2	-	2	2	2	2	2	2	2	2	2	2	2
Truck Tractor for trailers	1	-	1	1	1	1	1	1	1	1	1	1	1
Trailer Lowboy	1	-	-	1	1	1	1	1	1	1	1	1	1
Pick-up Truck	15	-	10	15	15	15	15	15	15	10	10	10	10
Pit Bus	1	-	1	1	1	1	1	1	1	1	1	1	1
Mobile Air Compressor 185CFM	1	-	1	1	1	1	1	1	1	1	1	1	1
Welding Machine Electric	2	-	1	2	2	2	2	2	2	2	1	1	1
Welding Machine Diesel 400A	2	-	2	2	2	2	2	2	2	2	2	2	2
Light Plant	4	-	4	4	4	4	4	4	4	4	4	4	4
Genset 6 kW	2	-	2	2	2	2	2	2	2	2	2	2	2
Genset 60 kW	3	-	3	3	3	3	3	3	1	1	1	1	1
Water Pump 3" - Gasoline	4	-	4	4	4	4	4	4	4	4	4	4	4
Water Pump 10” - Diesel	4	-	-	2	3	3	3	4	4	2	2	2	2
Diesel Powered Air Heaters	3	-	3	3	3	3	3	2	2	2	2	2	2
Snow Blower	1	-	1	1	1	1	1	1	1	1	1	1	1

16.7 Mine Manpower Requirements

Mine personnel were divided into hourly and staff positions and were divided between mine operations, mine maintenance, mine engineering, and geology. Hourly positions were all associated with a shift roster of 7 days on and 7 days off and as such, each unit of equipment requires four operators hired in hourly positions.

Staff positions in management, supervision, or technical services roles will also be on 5/2 roster schedule. In some case where 24-hour support in the staff role was necessary, the staff position was planned to be on the same 7 days on / 7 days off schedule as the hourly staff.

Table 16-10 on the following page shows the estimated mine workforce requirements over the life of mine. The mine workforce peaks at 250 individuals in Year 4.

16.8 Mine Management and Technical Services

The operations team is responsible for achieving production targets in a safe manner. The engineering and geology team will provide support to the operations team by providing short- and long-term planning, grade control, surveying, mining reserves estimation, and all other technical functions.

16.9 Crushing Plant

The production of crushed material will be necessary for blasthole stemming purposes, for road maintenance, or for spreading road abrasive on ramps during winter. It is assumed that the required aggregate material production will occur during summertime, with the mobilization of a contracted mobile crusher to site. Waste rock to feed the small crushing plant will come from the pit, and the material produced will be stockpiled for use throughout the year. Cost of such contract services have been accounted for in the cost/tonne of aggregate used in the model.

16.10 Pit Slope Monitoring

Pit slope monitoring systems are used to gather any information on micro and macro movements of the pit walls. It usually consists of strategically placed prisms that are surveyed under a controlled environment (windless, rainless and stationary). No monitoring system has been developed during this phase of the prefeasibility study.

16.11 Mine Maintenance

The project has not included a maintenance and repair contract (MARC) for its mobile equipment fleet. The maintenance department and personnel requirement has been structured to fully manage this function, performing maintenance planning and training of employees. However, reliance on dealer and manufacturer support will be key for the initial years of the project, and major component rebuilds will be supported by the original equipment manufacturer's (OEM) dealer throughout the LOM. An evaluation of a MARC will be considered with the Feasibility study process. Tire monitoring, rotation and/or replacement will be carried out by a specialized contractor.

Some other equipment will also be purchased to facilitate the maintenance activities and support the operation such as fuel and lube trucks, forklift, telehandler, low-boy trailer, and tractor for moving the tracked equipment. Other small equipment such as mechanic service truck, generators, compressors, light towers, welding machines, water pumps, air heaters.

Table 16-10: Workforce Forecast

Mine Operations	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mine Operations - Mine Manager	1	-	1	1	1	1	1	1	1	1	1	1	1
Mine Operations - Mine Superintendent	1	-	1	1	1	1	1	1	1	1	1	1	1
Mine Operations - Mine Ops. General Foreman	2	-	-	2	2	2	2	2	2	2	2	2	2
Mine Operations - Supervisor	6	-	1	6	6	6	6	4	4	4	4	4	4
Mine Operations - Mine D&B Supervisor	2	-	-	2	2	2	2	2	2	2	2	2	2
Mine Operations - Clerk	1	-	1	1	1	1	1	1	1	1	1	1	1
Mine Operations - Trainer	1	-	1	1	1	1	1	1	1	1	1	1	-
Mine Operations - Laborer	8	-	4	8	8	8	8	8	4	4	4	4	4
Mine Operations - Shovel / Excavator Operator	16	-	-	12	12	16	16	16	16	12	8	8	8
Mine Operations - Loader Operator	4	-	4	4	4	4	4	4	4	4	4	4	4
Mine Operations - Haul Truck Operator	56	-	-	24	28	44	56	56	56	52	32	28	12
Mine Operations - Drill & Blast Operator	16	-	-	12	12	16	16	16	16	12	8	8	4
Mine Operations - Drill & Blast Operator	4	-	-	4	4	4	4	4	4	4	4	4	4
Geology - Grade Control Laborers / Samplers	8	-	1	8	8	8	8	6	4	4	4	4	4
Mine Operations - Dozer Operator	28	-	16	24	24	28	28	24	20	16	12	12	12
Mine Operations - Grade Operator	8	-	2	8	8	8	8	8	8	8	4	4	4
Mine Operations - Haul Truck Operator	4	-	2	4	4	4	4	4	4	4	4	4	4
Mine Operations - Dozer Operator	4	-	2	4	4	4	4	4	4	4	4	4	4
Mine Operations – Shovel / Excavator Operator	4	-	2	4	4	4	4	4	4	4	4	4	4
Mine Operations - Ancillary Equipment Operator	4	-	2	4	4	4	4	4	4	4	4	4	4
Subtotal Mine Operation	178	-	40	134	138	166	178	170	160	144	108	104	83
Mine Maintenance	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mine Maintenance - Superintendent	1	-	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance - General Foreman	1	-	1	1	1	1	1	1	1	1	1	1	1
Mine Maintenance - Supervisor	4	-	2	4	4	4	4	2	2	2	2	2	2
Mine Maintenance - Planner	2	-	-	2	2	2	2	2	2	2	2	2	2
Mine Maintenance - Trainer	1	-	1	1	1	1	1	-	-	-	-	-	-
Mine Maintenance	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Mine Maintenance - Mechanical Engineer	1	-	1	1	1	1	1	-	-	-	-	-	-
Mine Maintenance - Clerk	1	-	1	1	1	1	1	1	1	1	1	1	1
Mechanic	28	-	10	28	28	28	24	21	17	14	10	10	10
Electrician	4	-	4	4	4	4	4	4	2	2	2	2	2

Welder / Machinist	4	-	2	4	4	4	4	3	3	2	2	2	2
Fuel & Lube Technician	4	-	4	4	4	4	4	4	4	4	4	4	4
Toolcrib Attendant	2	-	1	2	2	2	2	2	2	2	2	2	2
Maint. Helper	4	-	2	4	4	4	4	4	2	2	2	2	2
Subtotal Mine Maintenance	57	-	30	57	57	57	53	45	37	33	29	29	29
Mine Geology	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Geology - Chief Geologist	1	-	-	1	1	1	1	1	1	1	1	1	1
Geology - Senior Geologist	1	-	-	1	1	1	1	1	-	-	-	-	-
Geology - Resource Geologist	1	-	-	1	1	1	1	1	-	-	-	-	-
Geology - Geologist	2	-	-	1	1	1	1	1	2	2	2	2	2
Geology - Junior Geologist	1	-	-	1	1	1	1	1	-	-	-	-	-
Geology - Geology Technician	4	-	-	4	4	4	4	4	4	4	4	4	4
Geology - Grade Control Laborers / Samplers	8	-	1	8	8	8	8	6	4	4	4	4	4
Subtotal Mine Geology	9	-	-	9	9	9	9	9	7	7	7	7	7
Mine Engineering	Total	Y-2	Y-1	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10
Engineering - Chief Mine Engineer	1	-	1	1	1	1	1	1	1	1	1	1	1
Engineering - Long-Term Planning Engineer	1	-	1	1	1	1	1	1	1	1	-	-	-
Engineering - Short-Term Planning Engineer	2	-	1	2	2	2	2	2	2	2	2	2	2
Engineering - Junior Mine Engineer	1	-	-	1	1	1	1	1	1	1	1	1	1
Engineering - Technician	4	-	2	4	4	4	4	4	4	4	4	4	4
Engineering - Senior Surveyor	1	-	1	1	1	1	1	1	-	-	-	-	-
Engineering - Clerk	1	-	1	1	1	1	1	1	1	1	1	1	1
Subtotal Mine Engineering	11	-	7	11	11	11	11	11	10	10	9	9	9
Total Workforce	250	-	76	210	214	242	250	234	213	193	152	148	127

17 RECOVERY METHODS

17.1 Overall Process Design

The testwork provided was analysed and several options for process routes were reviewed in the initial stages of the prefeasibility study. Based on the analysis, a conventional leach and carbon-in-pulp process route was chosen as the most suitable for the deposit and project economics. The unit operations selected are all typical for gold recovery and the proposed flowsheet uses standard processes and technologies.

Key operating criteria for the process plant are listed below:

- Nominal throughput of 16,438 t/d or 6.0 Mt/a;
- Crushing plant availability of 70%; and
- Plant availability of 92% for grinding, leach plant, and gold recovery operations.

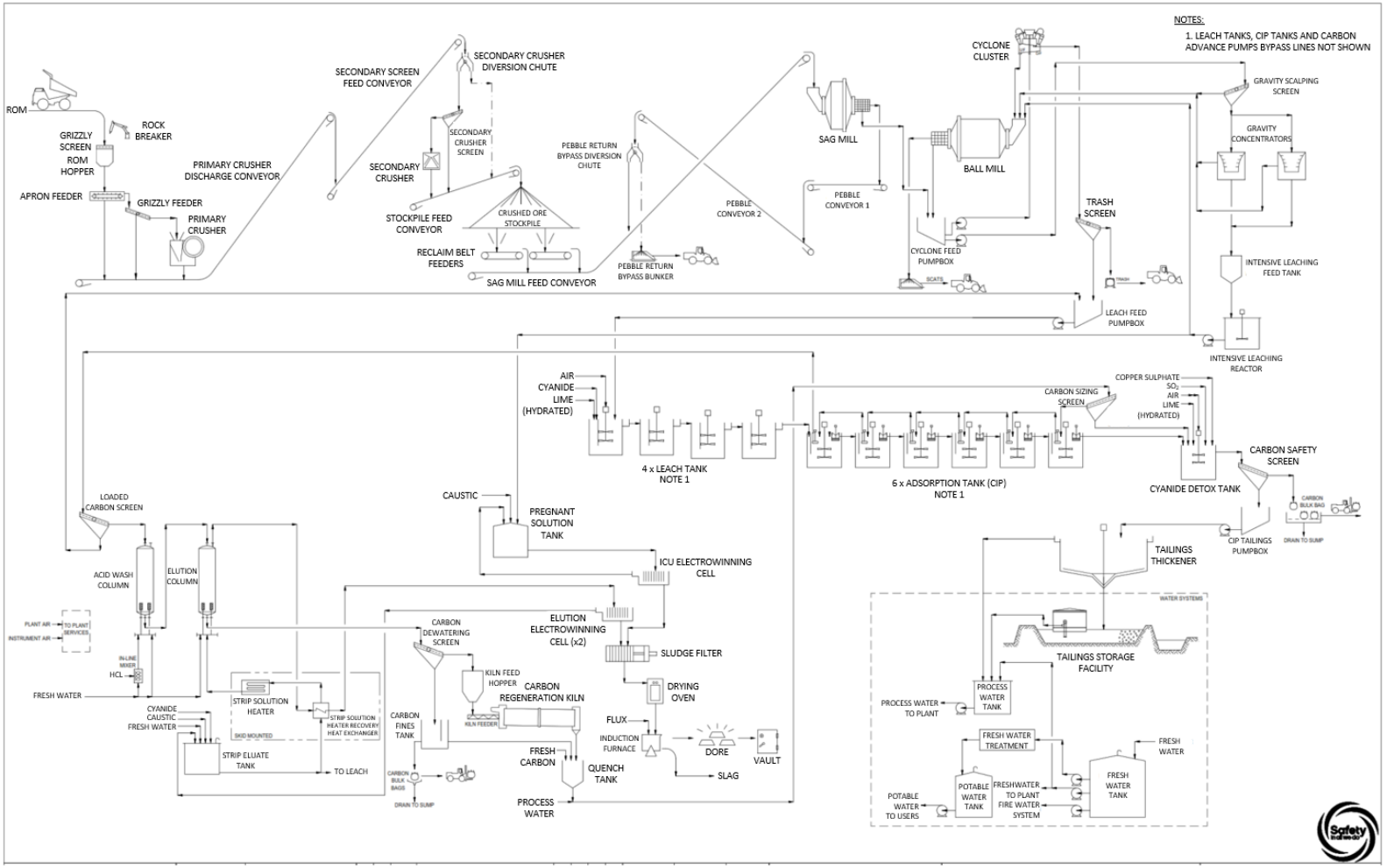
17.2 Process Plant Description

The process design is comprised of the following circuits:

- Two-stage crushing of run-of-mine (ROM) material;
- A covered, crushed ore stockpile to provide buffer capacity ahead of the grinding circuit;
- SAG mill with trommel screen followed by a ball mill with cyclone classification;
- Gravity concentration and intensive leaching of the concentrate;
- Trash screening;
- Leach + carbon adsorption (L/CIP);
- Acid washing of loaded carbon and Pressure Zadra type elution followed by electrowinning and smelting to produce doré;
- Carbon regeneration by rotary kiln;
- Cyanide destruction of tailings using the SO_2/air process; and
- Tailings thickening.

The overall process flow diagram is presented in Figure 17-1.

Figure 17-1: Overall Process Flow Diagram



Source: Ausenco, 2022.

17.2.1 Plant Design Criteria

Key process design criteria for the mill are listed in Table 17-1.

Table 17-1: Key Design Criteria

Design Parameter	Units	Value
Plant Throughput, yearly	Mt/y	6.0
Plant Throughput, daily average	t/d	16,438
Gold Grade – Design Mill Head	g/t	1.02
Crushing Plant Availability	%	70
Crushing Plant Throughput, hourly	t/h	978
Mill Availability	%	92
Mill Throughput, hourly	t/h	744
Bond Crusher Work Index (CWi), 75th percentile	kWh/t	22.0
Bond Rod Mill Work Index (RWi), 75th percentile	kWh/t	15.5
Bond Ball Mill Work Index (BWi), 75th percentile	kWh/t	14.0
SMC Axb, 25th percentile	-	31.8
Bond Abrasion Index (Ai), design	g	0.194
Material Specific Gravity	t/m³	2.78
Primary Crusher	-	C150 or equivalent
Secondary Crusher	-	HP900 or equivalent
SAG Mill Dimensions	m diam. X m EGL	9.14 x 4.88
SAG Mill Installed Power	MW	8.0
SAG Mill Discharge Density	% w/w solids	70
SAG Mill Ball Charge, design	% v/v	18
SAG Mill Ball Charge, maximum for mechanical design	% v/v	20
Ball Mill Dimensions	m diam. x m EGL	6.71 x 10.2
Ball Mill Installed Power	MW	8.7
Ball Mill Discharge Density	% w/w solids	70
Ball Mill Ball Charge, design	% v/v	32
Ball Mill Ball Charge, maximum for mechanical design	% v/v	40
Primary Grind size (P_{80})	µm	85
Gravity Circuit Feed Source	-	Cyclone feed slurry
Gravity Circuit Feed Rate	% mill feed	100
Gravity Gold Recovery (design)	% Au	24.6
Leach & CIP Tanks	#	4 + 6
Leach Residence Time	h	18
Leach-CIP Operating Density	% w/w solids	42.5
Leach pH Target	-	10.5
Leach DO Target	ppm	20
Leach Sodium Cyanide Addition, design	kg/t	0.50
Leach Hydrated Lime Addition, design	kg/t	0.83
CIP Residence Time	h	6
CIP Carbon Concentration	g/L	22
Tonnes of Carbon per Elution Column	t	11

Design Parameter	Units	Value
Elution Strips per Day	#	1
Acid Wash and Elution Operating Days per Week	days/week	7
Cyanide Detoxification Method	-	SO_2/air
Cyanide Detoxification Residence Time, design	min	60
Cyanide Detoxification Tanks	#	1
Cyanide Detoxification Copper Sulphate Addition, design	mg/LCu^{+2}	25
Cyanide Detoxification SO_2 Addition (as liquid SO_2), design	SO_2:CN_{WAD} ratio (w/w)	5.0
Cyanide Detoxification Lime Addition, design	$Ca(OH)_2$:SO_2 (w/w)	0.37
Cyanide Detoxification Discharge CN_{WAD}, design	mg/L	< 8
Tailings Thickener Underflow Density	% w/w solids	60
Total Gold Recovery, design	% Au	94

17.2.2 Crushing and Stockpiling

The crushing circuit is designed for an annual operating time of 6,132 hours or 70% availability at a capacity of 16,438 t/d from the outset.

Ore is hauled from the mine and direct tipped into to the run-of-mine (ROM) bin, which has a static grizzly screen on top to remove oversize ore. A fixed rock breaker is utilized to break oversize rocks at the feed to the ROM bin. Ore is withdrawn from the ROM bin by an apron feeder and is passed through a vibrating grizzly feeder at 978 t/h to feed the jaw crusher. The primary crusher discharge is combined with the vibrating grizzly undersize and conveyed to a vibrating double-deck secondary screen. The secondary screen oversize from both decks is fed to the secondary cone crusher. The cone crusher discharge is combined with the secondary screen undersize and is conveyed to a covered stockpile that provides approximately 9,000 t of live storage. The stockpile disconnects crushing from the mill to allow for crusher maintenance.

The mill feed stockpile is equipped with two belt feeders to regulate feed at 744 t/h into the SAG mill via the SAG mill feed conveyor. Pebbles from the SAG mill are recirculated by a pebble conveyor, which discharges on the SAG mill feed conveyor to recycle to the SAG mill.

The material handling and crushing circuit includes the following key equipment:

- ROM hopper;
- Fixed rock breaker;
- Primary crusher apron feeder;
- Vibrating grizzly;
- Primary jaw crusher;
- Secondary cone crusher;
- Secondary screen;
- Mill feed belt feeders (equipped with VSDs); and
- Material handling equipment.

17.2.3 Grinding

The grinding circuit consists of a SAG mill followed by a ball mill in closed circuit with hydrocyclones. The circuit is sized based on SAG mill feed size of 80% passing 46 mm and a ball mill product of 80% passing 85 µm. The SAG mill slurry

discharges through a trommel screen, where oversize pebbles are recycled to the SAG mill via conveyors. Trommel screen undersize discharges into the cyclone feed pumpbox. The SAG mill is powered by a variable speed drive (VSD) to allow for changes in SAG mill motor speeds in the event of changes in ore hardness.

The ball mill is fed by cyclone underflow. The ball mill discharges through a trommel, and the oversize is screened out and discharged to a scats bunker, whereas the trommel undersize discharges into the cyclone feed pumpbox.

Based on an evaluation of the available ore hardness testing and the selected process flowsheet, no significant pebble generation in the SAG mill circuit is expected. For this reason, no pebble crusher is included in the circuit.

Water is added to the cyclone feed pumpbox to obtain the appropriate density prior to pumping to the cyclones. The cyclone feed pumpbox is equipped with two pumps, one to feed the cyclone cluster, and another to feed the gravity circuit. Cyclone overflow at 43.0% w/w solids is sent to a trash screen prior to the leach circuit.

The grinding circuit includes the following key equipment:

- 9.14 m diameter x 4.88 m Effective Grinding Length (EGL) 8,000 kW SAG mill (equipped with VSD);
- 6.71 m diameter x 10.2 m EGL 8,700 kW ball mill;
- Cyclone feed pumpbox;
- Cyclone feed pump;
- Gravity circuit feed pump;
- Classification cyclones; and
- Trash screen.

17.2.4 Gravity Recovery Circuit

The gravity circuit comprises one scalping screen and two centrifugal batch concentrators in parallel. Feed to the circuit is directed from the cyclone feed pumpbox via a dedicated pump to the scalping screen. Gravity scalping screen oversize at +2 mm reports back to the ball mill.

Scalping screen undersize is fed to the centrifugal concentrator. Operation of the gravity concentrator is semi-batch and the gravity concentrate is collected in the concentrate storage cone and subsequently leached by the intensive cyanidation reactor circuit. The tailings from the gravity concentrator also report to the ball mill.

The gravity recovery circuit includes the following key equipment:

- Gravity feed scalping screen; and
- Gravity concentrator.

17.2.5 Intensive Leaching Reactor

Concentrate from the gravity circuit reports to the intensive leaching reactor (ILR) to extract the contained gold. The concentrate from the gravity concentrator is directed to the ILR gravity concentrate storage cone and de-slimed before transfer to the ILR.

ILR leach solution (mixture of NaCN, NaOH, and an oxidant) is made up within the heated ILR reactor vessel feed tank. From the feed tank, the leach solution is circulated though the reaction vessel, then drained back into the feed tank. The leached residue within the reaction vessel is washed with wash water, and then the solid gravity leach tailings are pumped to the cyclone feed pumpbox.

The ILR pregnant leach solution is pumped from the reaction vessel feed tank to the ILR pregnant solution tank located in the gold room.

ILR pregnant solution is treated in the gold room for gold recovery as gold sludge using a dedicated electrowinning cell. The sludge is combined with the sludge from the carbon elution electrowinning cells and smelted. It can also be smelted separately for metallurgical accounting purposes.

The ILR circuit includes the following key equipment:

- Intensive leaching feed tank;
- Intensive cyanidation reactor; and
- ILR pregnant solution tank.

17.2.6 Leach and Adsorption Circuit

The leach-adsorption circuit consists of four leach tanks and six carbon in pulp (CIP) tanks. The circuit is fed by the trash screen undersize. Barren solution from electrowinning cells is periodically transferred to the leach circuit. The leach and CIP tanks have a total circuit residence time of 24 hours at 42.5% w/w solids.

Air is sparged into alternating tanks to maintain adequate dissolved oxygen levels for leaching at 8 mg/L. Hydrated lime is added to further adjust the operating pH to the desired set point of 10.5 and cyanide solution is added to the first leach tank. 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 by pumping slurry and carbon. Slurry from the last CIP tank flows to the cyanide detoxification tanks.

The intertank screen in each 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 CIP tank. Recessed impeller pumps are used to transfer slurry between the CIP tanks and from the lead tank to the loaded carbon screen mounted above the acid wash column in the elution circuit.

The leach and carbon adsorption circuit includes the following key equipment:

- Leach/CIP tanks and agitators;
- Loaded carbon screen;
- Intertank carbon screens; and
- Carbon sizing screen.

17.2.7 Cyanide Destruction

CIP tailings are discharged to the cyanide detoxification tank at 42.5% w/w solids. Cyanide destruction is accomplished using the SO_2/air method. The reagents required are air, lime, copper sulphate, and SO_2. The cyanide destruction tank is equipped with an agitator with downshaft addition to ensure that the reagents are thoroughly mixed with the tailings slurry. Sulphur dioxide is added through a cone at the bottom of the tank. The water used for acid rinse and carbon transfer is also included in the feed to detoxification circuit. The detoxification tank discharge feeds the carbon safety screen, which recovers carbon and carbon fines remaining in the tailings.

The tank operates with a total residence time of 60 minutes to reduce weak acid dissociable cyanide (CN_{WAD}) concentration from 145 mg/L to less than 8 mg/L.

The carbon safety screen oversize (recovered carbon) is collected in carbon bulk bags for potential return to the CIP circuit or processing with other carbon fines. The screen undersize is pumped to the tailings thickener.

The main equipment in this area includes:

- Carbon safety screen;
- Cyanide destruction tank and agitator; and
- Tailings pumpbox.

17.2.8 Tailings Thickening

Detoxified tailings from the carbon safety screen undersize are thickened before discharge to the Tailings Storage Facility (TSF). The overflow of the thickener is reused as process water in the plant. Flocculant is combined with the feed to the thickener to improve the solids settling rate. The underflow is pumped to the TSF.

The main equipment in this area includes:

- High-rate thickener, 38 m diameter; and
- Underflow / final tailings pumps in duty-standby.

17.2.9 Carbon Acid Wash, Elution, and Regeneration Circuit

17.2.9.1 Carbon Acid Wash

Prior to gold elution, loaded carbon from the first CIP tank is treated with a weak hydrochloric acid solution to remove calcium, magnesium, and other salt deposits that could render the elution less efficient or become baked on in subsequent steps and ultimately foul the carbon.

Loaded carbon from the loaded carbon recovery screen flows by gravity to the acid wash column. Entrained water is drained from the column and the column is refilled from the bottom up with the hydrochloric acid solution. Once the column is filled with the acid, it is left to soak, after which the spent acid is rinsed from the carbon and discarded to the cyanide destruction tank.

The acid-washed carbon is then hydraulically transferred to the elution column for gold stripping.

The main equipment in this area includes:

- Acid wash carbon column – 11-tonne capacity;
- Hydrochloric acid feed pump; and
- Spent solution discharge sump pump.

17.2.9.2 Carbon Stripping (Elution) and Electrowinning

The gold stripping (elution) circuit uses the Pressure Zadra process.

A high cyanide, caustic solution is recirculated through a pressure elution column at 140°C to strip the precious metals from the carbon. The precious metal-rich solution from the column exchanges heat with barren solution going to the column. Cooled solution then flows through electrowinning cells to deposit the gold and silver on the cathodes before the solution is recycled back to the elution column.

The stripping circuit includes the following key equipment:

- Carbon elution column – 11-tonne capacity;
- Strip solution heater (electric) with heat exchangers; and

- Strip eluate solution tank.

17.2.9.3 Gold Room

Gold sludge is recovered from the electrowinning cells and smelted to produce doré bars.

The gold-rich sludge is washed off the steel cathodes in the electrowinning cells using high-pressure spray water and transferred by gravity 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 equipped with access control, intruder detection, and closed-circuit television equipment.

The electrowinning circuit and gold room include the following key equipment:

- Electrowinning cells with rectifiers;
- Sludge pressure filter;
- Drying oven;
- Flux mixer;
- Induction smelting furnace with bullion moulds and slag handling system;
- Bullion vault and safe;
- Dust and fume collection system; and
- Gold room security system.

17.2.9.4 Carbon Regeneration

Carbon is regenerated in an electric rotary kiln. Dewatered barren carbon from the stripping circuit is held in an 11-tonne kiln feed hopper. A screw feeder metres the carbon into the reactivation kiln, where it is heated to 650° to 750°C in an atmosphere of superheated steam to restore the activity of the carbon.

Carbon discharging from the kiln is quenched in water and screened on a carbon sizing screen located on top of the CIP tanks to remove undersized carbon fragments. The undersize fine carbon gravitates to the carbon safety screen, while carbon screen oversize is directed to the CIP circuit.

As carbon is lost by attrition, new carbon is added to the circuit using the carbon quench tank. The new carbon is then transferred along with the regenerated carbon to feed the carbon sizing screen.

The carbon reactivation circuit includes the following key equipment:

- Carbon dewatering screen;
- Regeneration kiln (electric) including feed hopper and screw feeder; and
- Carbon quench tank.

17.3 Reagent Handling and Storage

Each set of compatible reagent mixing and storage systems are located within curbed containment areas to prevent incompatible reagents from mixing. Storage tanks are equipped with level indicators, instrumentation, and alarms to ensure spills do not occur during normal operation. Appropriate ventilation, fire and safety protection, eyewash stations, and Material Safety Data Sheet (MSDS) stations are located throughout the facilities. Sumps and sump pumps are provided for spillage control.

The following reagent systems are required for the process:

- Hydrated lime;
- Sodium cyanide;
- Hydrochloric acid;
- Copper sulphate pentahydrate;
- Liquid sulphur dioxide;
- Sodium hydroxide;
- Flocculant;
- Activated carbon; and
- Smelting fluxes.

17.3.1 Hydrated Lime

Hydrated lime is delivered in bulk and is pneumatically conveyed from the truck to the hydrated lime silo. The hydrated lime is extracted from the lime silo and fed to the lime mixing/storage tank. The solid reagent discharges into the tank and is slurried in process water to achieve the required dosing concentration. The slurried hydrated lime is pumped through a ring main with distribution points in leaching and cyanide destruction. An extraction fan is provided over the lime silo to remove reagent dust that may be generated.

17.3.2 Sodium Cyanide (NaCN)

Sodium cyanide is delivered to site in an isotainer containing about 18.5 t as briquettes. Water is recycled through the isotainer to dissolve the briquettes. Once the dissolution cycle is completed, the contents are transferred to the cyanide storage tank. Compressed air is used to remove any residual solution in the isotainer into the sodium cyanide mixing tank.

Sodium cyanide is delivered to the leach circuit and elution circuit with dedicated dosing pumps.

17.3.3 Copper Sulphate

Copper sulphate pentahydrate is delivered in solid crystal form in bulk bags and stored in the warehouse. Process water is added to the agitated copper sulphate mixing tank. A pallet of bags is lifted using a frame and hoist, and periodically a single bag is placed on the copper sulphate bag breaker on top of the tank. The solid reagent falls into the tank and is dissolved in water to achieve the required dosing concentration.

Copper sulphate solution is transferred by gravity to the copper sulphate storage tank, which has a stacked arrangement with the mixing tank. Copper sulphate is delivered to the cyanide detoxification circuit using the copper sulphate dosing pump. An extraction fan is provided over the copper sulphate bag breaker/mixing tank to remove reagent dust that may be generated during reagent addition/mixing.

17.3.4 Liquid Sulphur Dioxide (SO_2)

Liquid SO_2 is delivered in 24 t bulk tanker trucks and transferred to a 38 t pressure vessel storage tank with a padding air system to maintain the sulphur dioxide in liquid form.

17.3.5 Sodium Hydroxide (NaOH)

Sodium hydroxide (NaOH) is delivered in intermediate bulk containers (IBCs) as a solution at 50% w/v. NaOH flows from the IBCs by gravity to a standing pipe to allow for the continuous dosage of NaOH from two IBCs simultaneously. Dosing pumps automatically deliver the reagent to the required locations—elution circuit, electrowinning, and cyanide mixing—to ensure the dosing requirements are met.

17.3.6 Hydrochloric Acid (HCl)

Hydrochloric acid is delivered in IBCs as a solution at 33% w/v concentration and flows from the IBCs by gravity to a standing pipe. The standing pipe allows for the continuous dosage of two IBCs simultaneously. A dosing pump automatically delivers the reagent to the elution circuit when required. Hydrochloric acid is mixed with raw water (inline) to achieve the required 3% w/v concentration.

17.3.7 Flocculant

Powdered flocculant is delivered to site in bulk bags and stored in the warehouse. A self-contained mixing and dosing system is installed, including a flocculant storage hopper, flocculant blower, flocculant wetting head, flocculant mixing tank, and flocculant transfer pump. Powdered flocculant is loaded into the flocculant storage hopper using the flocculant hoist. Dry flocculant is pneumatically transferred into the wetting head, where it is contacted with water.

Flocculant solution, at 0.50% w/v, is agitated in the flocculant mixing tank for a pre-set period. After a pre-set time, the flocculant is transferred to the flocculant storage tank using the flocculant transfer pump. Flocculant is dosed to the tailings high-rate thickener using a variable speed helical rotor style pump. Flocculant is further diluted to 0.05% w/v concentration prior to the addition point using an in-line mixer.

17.3.8 Activated Carbon

Six by 12 mesh size (US sieve) coconut shell activated carbon is delivered in solid granular form in bulk bags. When required, the fresh carbon is introduced to the carbon quench tank.

17.3.9 Gold Room Smelting Fluxes

Borax, silica sand, sodium nitrate, and soda ash are delivered as solid crystals/pellets in bags or plastic containers and stored in the warehouse until required.

17.4 Plant/Instrument Air

High-pressure air at 700 kPag is produced by compressors to meet plant requirements. The high-pressure air supply is dried and used to satisfy both plant air and instrument air demand. Dried air is distributed via the air receivers located throughout the plant.

17.5 Water Supply

17.5.1 Fresh Water Supply

Fresh water is supplied to a raw water storage tank from the Marban pit. Raw water is used for all purposes requiring clean water with low dissolved solids and low salt content, primarily as follows:

- Reagent make-up;
- Elution circuit make-up;

- Fresh water is treated and stored in the potable water storage tank for use in safety showers and other similar applications;
- Fire water for use in the sprinkler and hydrant system; and
- Cooling water for mill motors and mill lubrication systems (closed loop).
- A total of 12.1 m^3/h of fresh water is required for the process plant.

17.5.2 Process Water Supply System

Overflow from the final tailings thickener and decant water from the TSF are returned to the process water tank to be recirculated to required areas of the process plant. Two (one duty, one standby) horizontal centrifugal slurry pumps supply process water to the various consumers throughout the plant site, but predominantly to the grinding circuit. The process water tank is constructed from mild steel and has a live volume ensuring 30 minutes of residence time.

The overflow from the final tailings thickener and decant water do not meet the main process water volume requirements. Mine wastewater provides any additional make-up water requirements.

17.5.3 Gland Water

Two gland water pumps arranged in duty/standby configuration are fed from the process water tank to supply gland water to all slurry pumps in the main plant.

17.6 Reagent and Consumable Requirements

Reagent consumptions are based on testwork results and standard industry practices. A summary of the nominal estimated reagent and consumables rates is shown in Table 17-2.

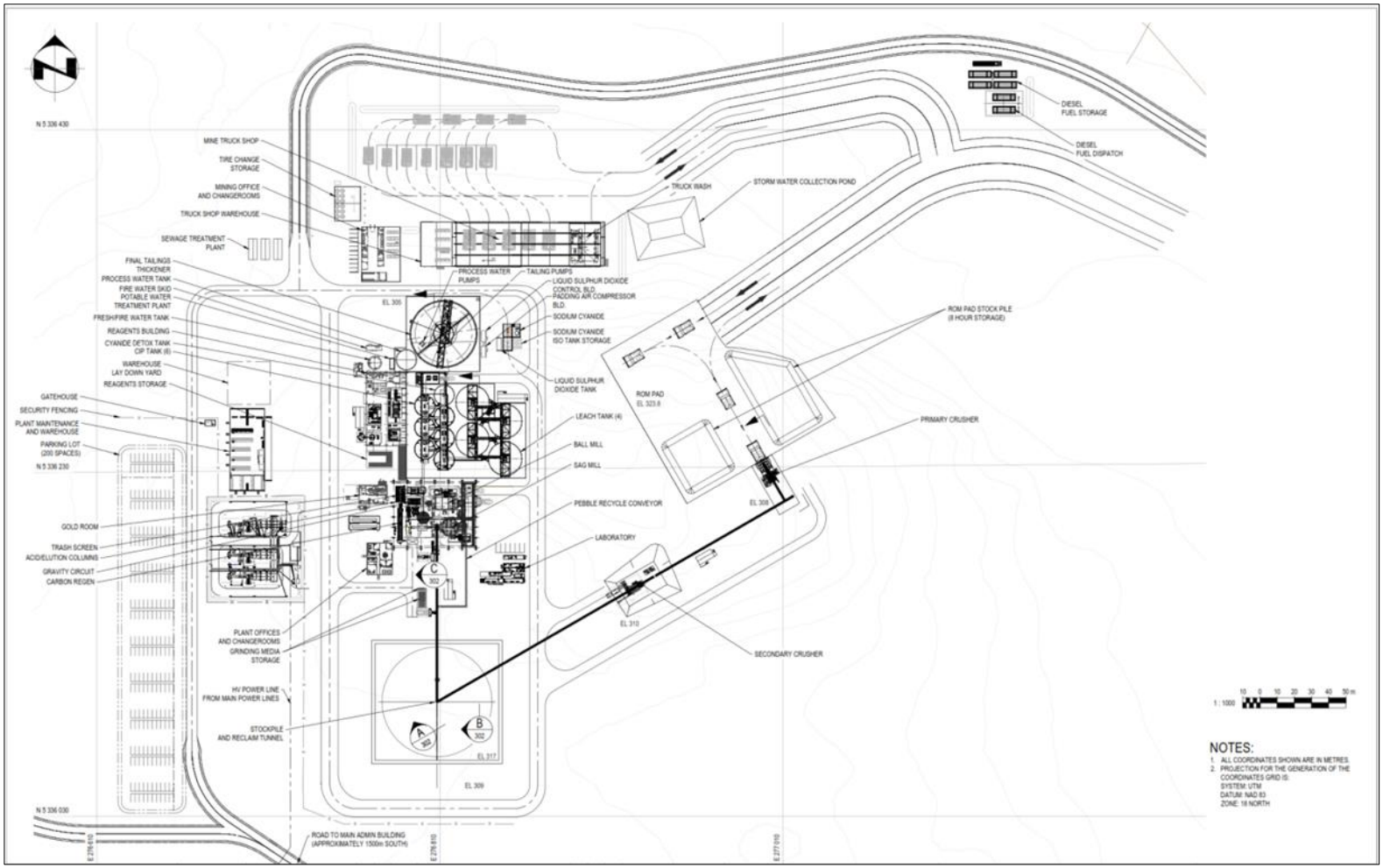
Table 17-2: Nominal Reagent Consumption Rates

Reagent	Consumption Amount	Consumption Rate	Packaging Unit
Sodium Cyanide	3.34	t/d	18.5 isotainer
Hydrated Lime	11.0	t/d	12 t truck
Activated Carbon	240	t/y	0.5 t Bulk Bag
Copper Sulphate	1.99	t/d	1 t Bulk Bag
Flocculant	0.33	t/d	800 kg bag
Hydrochloric Acid	1.01	m^3/day	Liquid, IBC
Sodium Hydroxide	1.60	m^3/day	Liquid, IBC
Liquid SO_2	7.30	t/d	Tanker
SAG Mill Media	5.70	t/d	Bulk
Ball Mill Media	9.12	t/d	Bulk

17.7 Process Plant Layout

The overall process plant layout is shown in Figure 17-2.

Figure 17-2: Process Plant Layout



Source: Ausenco, 2022.

18 PROJECT INFRASTRUCTURE

18.1 Introduction

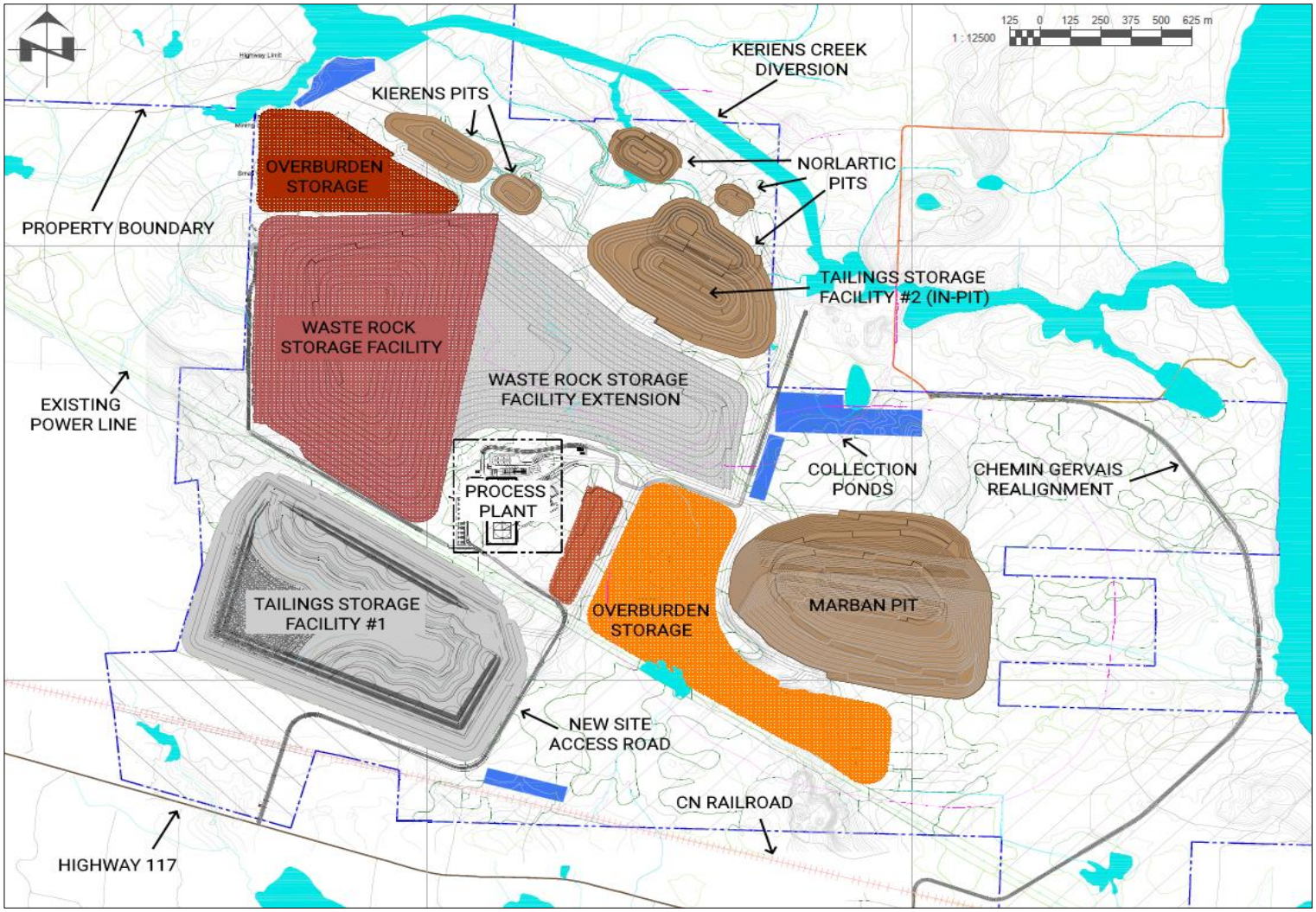
The Marban Engineering Project is located in Abititi-Témiscamingue, between Val-d'Or and Malartic, west of de Montigny Lake in the province of Quebec. The site is accessible by a short well-maintained gravel road from Highway 117.

The overall site plan (Figure 18-1) outlines the mining pits and storage facilities, water management facilities (including the creek diversion), the Process Plant, Tailings Storage Facility (TSF), the existing Hydro Quebec high voltage line, existing roads and railways, and the new access roads for the site.

The location of the site facilities was based on the following criteria:

- Facilities located within the claim boundaries;
- Minimize building on wetlands;
- Eliminate placement of infrastructure on fish-bearing habitats;
- Locate the primary waste rock stockpiles close to the mine pits to reduce haul distance;
- Locate the ROM and Process Plant in between the pits to minimize hauling and on solid bedrock (based on the limited geotechnical data available);
- Provide a dedicated new site access road to eliminate project vehicles travelling on the new Chemin Gervais alignment;
- Proximity of the Process Plant to the existing power line (Hydro-Québec); and
- Maintain distance from existing railway.

Figure 18-1: Overall Site Layout



Source: Ausenco, 2022.

18.2 Roads and Logistics

The Property is accessed from Provincial Highway 117 (the Trans Canada Highway) that links Rouyn-Noranda, Val d'Or and Malartic.

An existing public secondary road (Chemin Gervais) currently crosses over the footprint of the Marban Pit. A new alignment of this road, approximately 4 km long, will be constructed east of the project site, to maintain access to the existing bridge over Keriens Creek and the properties north of the creek. The construction of the new alignment will be completed before the existing alignment is decommissioned.

Access to the Process Plant & Truck Shop area will be from a new 2.5 km gravel road, from a new turnoff at the existing Highway 117.

The roads within the project site that connect the Process Plant, crushing area, mining operation structures, and all process buildings will be around 10 m in width. These will also be connected to the process pad and will be designed to include drainage and safety berms/bunds where appropriate. Haul roads connecting the TSF, mine pits, ROM pad, and rock stockpiles will be 30 m wide.

18.2.1 Rail

The Canadian National Railway runs through the Marban property, parallel to Provincial Highway 117. A new railway crossing will be necessary on the new main access road to the Process Plant. The new alignment of the existing Chemin Gervais will use the existing railway crossing so it will not require a new crossing.

18.2.2 Air

There is no airport at project site; nearby airport facilities are listed in Table 18-1.

Table 18-1: Nearby Airports

Airport	Distance to Site (Road Travel) (km)
Val D'Or Airport (YVO) (Regional)	24
Rouyn-Noranda Airport (YUY) (Regional)	80
Amos Airport (CYEY) (Regional)	80
Timmins Airport (YTS) (Regional)	312
Montréal-Pierre Elliot Trudeau Airport (YUL) (International)	545

18.2.3 Port Facilities

Nearby port facilities are listed in Table 18-2.

Table 18-2: Nearby Port Facilities

Port	Distance to Site (Road Travel) (km)
Old Port of Montreal	537
Port de Trois-Rivieres	601
Port of Toronto	712
Port Saguenay	741

18.2.4 Security

The site will be accessible year-round via the main access road off Highway 117.

Access to the Process Plant and mining areas is controlled by an entrance gate located 85 m away from Route 117.

Access to the Process Plant & Truck Shop area is further controlled by a security gate and perimeter fencing, which will prevent interactions between personal vehicles and site vehicles. A site peripheral fence will be installed to prevent site access to unauthorized people and wildlife.

18.2.5 Tailings Storage Access and Pipeline Service Road

Except for a small portion at the Processing Plant and TSF area, the tailings and reclaim water pipes will run above-ground and will sit on a granular pad (pipe bench). The pipe bench is designed with two low points allowing the tailings pipe to be completely drained for maintenance needs.

18.3 Electrical Power System

18.3.1 Electrical System Demand

Total installed power for the project is 33.0 MW, while the maximum demand for the Marban site is estimated at 26.2 MW. Power will be distributed at 13.8 kV to the various electrical rooms, which are summarized in Table 18-3.

Table 18-3: E-Room Descriptions

E-Room	E-Room Description	Installed kW	Maximum Demand kW
3200-ER-001	Process Plant Main Electrical Room	13,319	10,722
2100-ER-002	Primary Crushing Electrical Room	1,363	1,211
2200-ER-003	Stockpile & Reclaim Electrical Room	479	412
2300-ER-004	Grinding Electrical Room	14,057	11,064
2400-ER-005	Leaching / Reagents / Utilities Electrical Room	3,737	2,790
	Total	32,954	26,199

18.3.2 Facility Power Supply

Power will be supplied by Hydro-Québec and will be delivered to the site via a 120 kV overhead transmission line which terminates at the plant's 120 kV Outdoor Substation. The existing Hydro-Québec 120 kV transmission line traverses the Marban property and runs adjacent to the Process Plant, and will be raised to allow sufficient clearance for passage of haul trucks beneath.

Emergency power for Process Plant critical loads will be generated onsite by diesel-powered standby generators, which are located within proximity of the critical loads.

18.3.3 Electrical Outdoor Substation

The 120 kV Outdoor Substation will be a radial power distribution system, and will be located west of the Process Plant. The substation comprises two 25/33.3 MVA, 120 kV/13.8 kV oil-filled, power transformers, each sized to provide the Process Plant's maximum demand. The secondary transformer will be connected to a double-ended 13.8 kV switchgear with a normally open bus tie. When one transformer is out of service, the power system configuration will allow the other

transformer to support the total process load, thus enhancing system reliability. The substation will also include 4 MVAR of power factor correction equipment.

18.3.4 Site Power Reticulation

Power will be distributed across the site via 13.8 kV overhead lines originating from the plant's 13.8 kV switchgear housed within the primary Electrical Room at the Outdoor Substation.

Overhead distribution lines will be constructed using aluminium conductor steel-reinforced cable (ACSR) and supported by wooden poles.

The overhead powerlines will provide power to the following facilities:

- Process Plant and Truck Shop area;
- Security Gatehouse;
- TSF reclaim pumps;
- Freshwater intake pumps; and
- Marban, Norlartic, and Kierens open pits to power the electric shovel loaders and mine dewatering pumps.

A low voltage diesel generator will supply power to the explosives storage facility, due to the remoteness of its location.

18.3.5 Plant Power Distribution

The largest electrical loads at the Process Plant include the SAG and Ball mills, and the electric elution heater, and electrically heated regeneration kiln. The SAG and Ball Mill motors are induction motors equipped with a combination variable frequency drive (VFD) and bypass switchgear to minimize voltage disturbance throughout the power distribution system during motor start-up. These mill motors will be supplied via buried cable circuits from the plant's primary 13.8 kV switchgear. All other process and non-Process Plant loads will be powered via 4160 V and 600 V motor control centres (MCC) housed within electrical rooms strategically located throughout the Process Plant.

Power to the electrical rooms will be supplied by resistance grounded, secondary substation type, oil filled distribution transformers located adjacent to the respective electrical room. All electrical rooms will be adequately rated for the environment and outfitted with heating and ventilation, lighting and small power transformers, distribution boards, and uninterrupted power supply (UPS) systems. To reduce installation time, the electrical rooms will be prefabricated modular buildings, installed on structural framework above ground level for bottom entry of cables. Additionally, electrical rooms will be located as close as practical to the electrical loads, to optimize conductor sizes and minimize cable lengths.

Solidly grounded pad-mounted and pole-mounted transformers will be used to step down the voltages at the Fuel Storage area, Security Gatehouse, Mine Office, Laboratory, Warehouse, and Truck Shop areas and will terminate at the building's local 600 V distribution boards.

18.4 Fuel Storage

Four 60,000-L tanks of diesel will be used to fuel equipment onsite during operations. This is approximately 3 days of fuel consumption for mining and site equipment.

These tanks will be located north-east of the Truck Shop area, out of any main vehicle travel path, and on a concrete pad.

18.5 Support Buildings

18.5.1 Main Plant Site Area Buildings

As shown in the Process Plant layout in Figure 18-2, the main plant site area consists of several buildings. The Process Plant buildings are listed in Table 18-4. The main buildings are detailed further in the following sections.

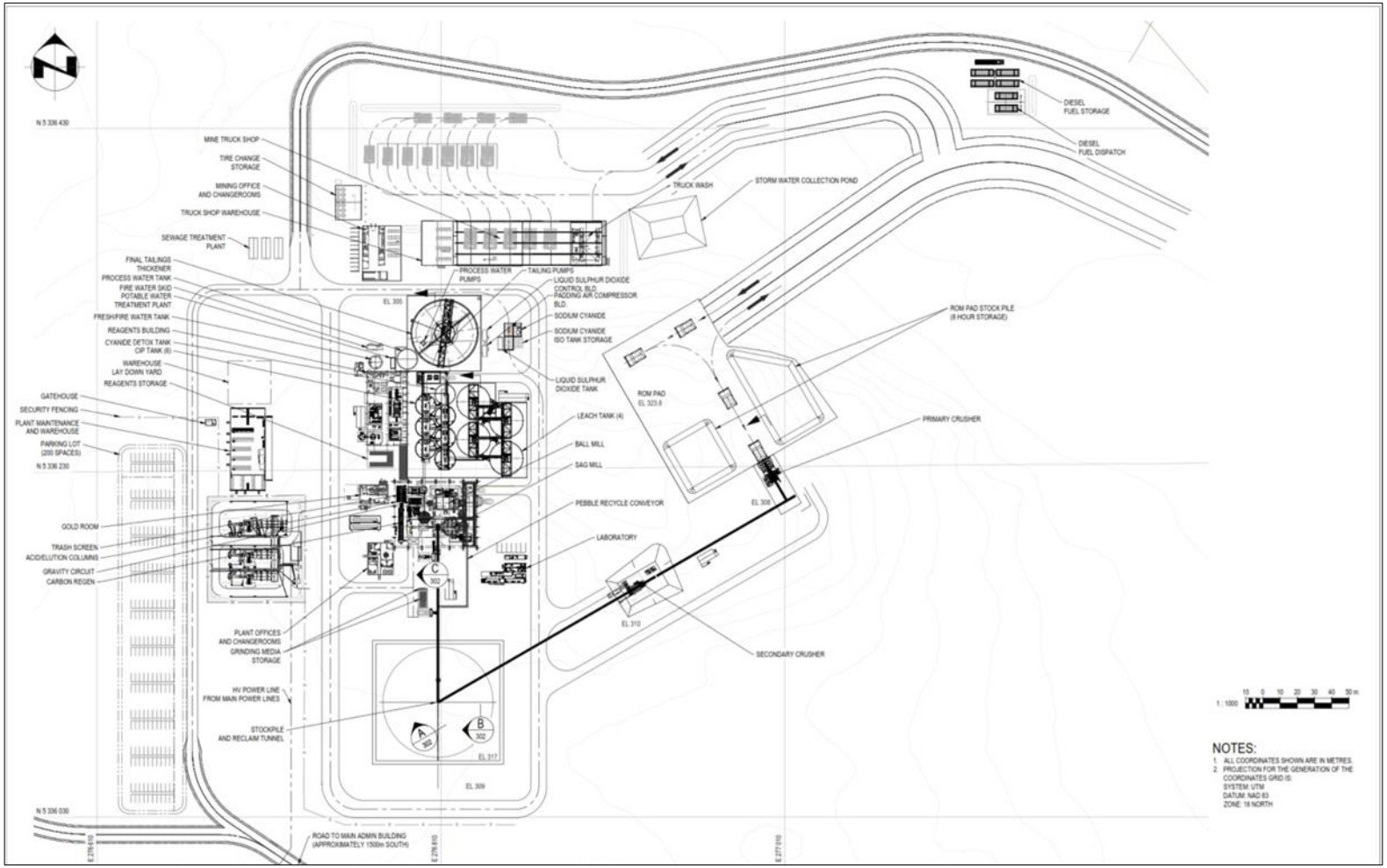
Table 18-4: Process Plant Buildings

Description	Building Construction	Length (m)	Width (m)	Height (m)	Area (m^2)
Secondary Crushing Building	Fabric	18.0	12.0	21.3	216.0
Stockpile Cover	Fabric	76.0	72.0	26.0	5,472.0
Mill Building	Pre-Engineered	48.0	38.0	26.6	1,824.0
Gold Room Building	Pre-Engineered	16.6	11.1	13.0	184.3
Cyanide Storage and Mixing Shed	Fabric	14.1	7.6	6.0	107.2
Reagents Building	Fabric	41.0	22.5	16.2	922.5
Reagents Storage Building	Fabric	15.2	9.8	6.0	149.0
Assay Laboratory	Modular	Dimensions by Contractor			
Plant Warehouse and Maintenance Building	Fabric	48.0	20.0	6.0	960.0
Truck Wash Building	Fabric	25.9	18.3	14.0	474.0
Truck Shop Building	Fabric	65.8	25.9	14.0	1,704.2
Truck Shop Warehouse	Fabric	25.9	18.3	6.0	474.0
Mill Office	Modular	12.2	14.6	3.4	178.4
Mine Dry & Mining Office	Modular	12.2	32.9	3.4	401.3
Security Building	Modular	12.2	3.7	3.4	44.6

18.5.1.1 Secondary Crushing Building

The Secondary Crushing Building will be of fabric design and will house the secondary screen and secondary cone crusher, as well as all associated chutes, platework, and material handling equipment.

Figure 18-2: Process Plant Layout



Source: Ausenco, 2022.

18.5.1.2 Stockpile Cover

The Stockpile Cover is a fabric building measuring 76.0 m in length, 72.0 m in width, and 26.0 m in height to house the 8,833 live tonne ore stockpile. There are two 4.0 m wide x 5.0 m high overhead doors with curtains and two 0.914 m wide x 2.13 m high metal man-doors. The structure material is to be provided by the contractor. The structure has a concrete foundation, and no heating is required. Electrical lighting and small electrical receptacles are as per code.

18.5.1.3 Mill Building

The Mill Building is a pre-engineered rigid frame metal building measuring 48.0 m long x 38.0 m wide x 26.6 m high. Areas included in this building are:

- SAG mill;
- Ball mill;
- Hydrocyclones;
- Gravity concentration and intensive leaching; and
- Acid wash & elution area.

The roof is built with cladding 24 Ga galvalume-finished standing seam roof with R-28 fiberglass blanket insulation and vapour barrier. Wall cladding is 24 Ga with exterior coating, R-20 fiberglass blanket insulation, and vapor barrier facing. The building has a 65-tonne overhead crane with a 5-tonne auxiliary hook that runs along the length of the building. There are four (4) metal man-doors measuring 0.914 m wide x 2.13 m high, one (1) 7.0 m wide x 8.0 m high overhead door, one (1) 3.5 m wide x 5.0 m high overhead door, and one (1) 5.5 m wide x 6.5 m high overhead door. All three overhead doors are complete with curtains. The building includes canopies overhead doors. The flooring is concrete slab-on-grade. The HVAC system consists of electric unit heaters and local ventilation for areas as per code requirements (min. 5° C operating temperature). The electrical system consists of LED lighting including the emergency lights as per code. Small power receptacles are included as per code. No provision was included for a washroom.

18.5.1.4 Gold Room Building

The Gold Room Building is a pre-engineered rigid frame metal building measuring 16.6 m long x 11 m wide x 13 m high with a gable roof (1:12). Areas included in this building are:

- Electrowinning & Gold Room

The roof is built with cladding 24 Ga galvalume-finished standing seam roof with R-28 fiberglass blanket insulation and vapour barrier. The wall cladding is 26 Ga with exterior coating, R-20 fiberglass blanket insulation, and vapor barrier facing. The building has a 1-tonne monorail that runs along the building length. The building includes one (1) metal man-door measuring 0.914 m wide x 2.13 m high and one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains. The HVAC system is Propane Fired Heating with local ventilation for areas as per code requirements (min. 5° C operating temperature). The electrical system is LED lighting including emergency lights as per code. Small power receptacles are placed as per code requirements. No provision was made for a washroom.

18.5.1.5 Cyanide Storage and Mixing Shed

The Cyanide Storage and Mixing Shed is a 107 m^2 fabric building housing the sodium cyanide mixing and storage tanks.

18.5.1.6 Assay Laboratory

The Assay Laboratory is a single-storey modular building housing the typical equipment for the Process Plant assays.

Mining assays will be completed at an external laboratory. Process plant assays to be analysed in the Assay Laboratory include plant solid samples, plant solution samples, carbon samples, bullion samples, samples from the cyclone underflow, and environmental samples.

18.5.1.7 Reagent Building

The Reagent Building is a 41.0 m long x 22.5 m wide x 16.2 m high fabric building, complete with a 10-tonne crane. The building includes two (2) metal personnel access-doors measuring 0.914 m wide x 2.13 m high and two (2) 4.0 m wide x 5.0 m high overhead doors complete with curtains. The building houses the reagent mixing and storage tanks, as well as four days worth of reagent storage. The reagents consist of lime, sodium hydroxide, hydrochloric acid, carbon, copper sulphate, and flocculant. Additional reagent storage will be available in the Reagent Storage Building.

18.5.1.8 Reagent Storage Building

The Reagents Storage Warehouse is a fabric building, 15.2 m long x 9.8 m wide x 6.0 m high, providing space for longer-term storage of reagents for use in the Process Plant. The building includes one (1) metal personnel access-door measuring 0.914 m wide x 2.13 m high and one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains. The Reagent Storage Building is located in close proximity to the main reagents building.

18.5.1.9 Truck Wash Building

The Truck Wash Building is a 25.9 m long x 18.3 m wide x 14.0 m high fabric building located northwest of the ROM pad. The building includes one (1) metal personnel access-door measuring 0.914 m wide x 2.13 m high, one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains, and five (5) 7.0 m wide x 7.0 m high haul truck door. Areas included in this building are:

- One (1) wash bay and bays for major and minor equipment.

18.5.1.10 Truck Shop Building

The Truck Wash Building is a 65.8 m long x 25.9 m wide x 14.0 m high fabric building located northwest of the ROM pad. The building includes two (2) metal personnel access-doors measuring 0.914 m wide x 2.13 m high, one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains, and one (1) 7.0 m wide x 7.0 m high haul truck door. Areas included in this building include:

- Five (5) bays for truck service; and
- One (1) bay for equipment.

The Truck Shop is equipped with air compressors, air filters, and pumps for gear oil, engine oil, transmission oil, grease, coolant, and windshield washer distribution.

18.5.1.11 Truck Shop Warehouse

The Truck Shop Warehouse is a 25.9 m long x 18.3 m wide x 6.0 m high fabric building including one (1) metal personnel access door measuring 0.914 m wide x 2.13 m high and one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains.

18.5.1.12 Plant Warehouse and Maintenance Building

The Plant Warehouse and Maintenance Building is a 48.0 m long x 20.0 m wide x 6.0 m high fabric building including two (2) metal man-doors measuring 0.914 m wide x 2.13 m high and one (1) 4.0 m wide x 5.0 m high overhead door complete with curtains.

18.5.1.13 Mine Dry & Mining Office

The Mine Dry & Mining Office is a single-storey 401 m^2 modular building consisting of nine (9) 12.2 m long x 3.65 m wide containers. The foundation is to be built by the contactor, as per code requirements. The HVAC system is a propane heating system, as per code requirements. The electrical system includes lighting and small electrical receptacles as per code requirements. The required rooms for the Mine Dry & Mining Office include:

- Changing room for men with showers;
- Changing room for women with showers;
- Six (6) offices; and
- Two (2) lunchrooms.

18.5.1.14 Mill/Plant Office

The Mill/Plant Office is a single-storey 178.4 m^2 modular building consisting of four (4) 12.2 m long x 3.65 m wide containers. The required rooms for the Mill/Plant Office include:

- Four (4) offices; and
- Lunchroom.

18.5.2 Site Entrance

The Security Building is located at the main site entrance, where the new site access road meets Highway 117. The building is a 44.6 m^2 single-storey modular building consisting of a 12.2 m long x 3.65 m wide container with one boom gate for vehicle access and another for personnel. There will be a shack where the gate security personnel will be allocated, with a section where induction training can be performed for visitors and new employees, as well as first aid, and it will also be the parking location for the onsite ambulance.

18.5.3 Explosives Storage

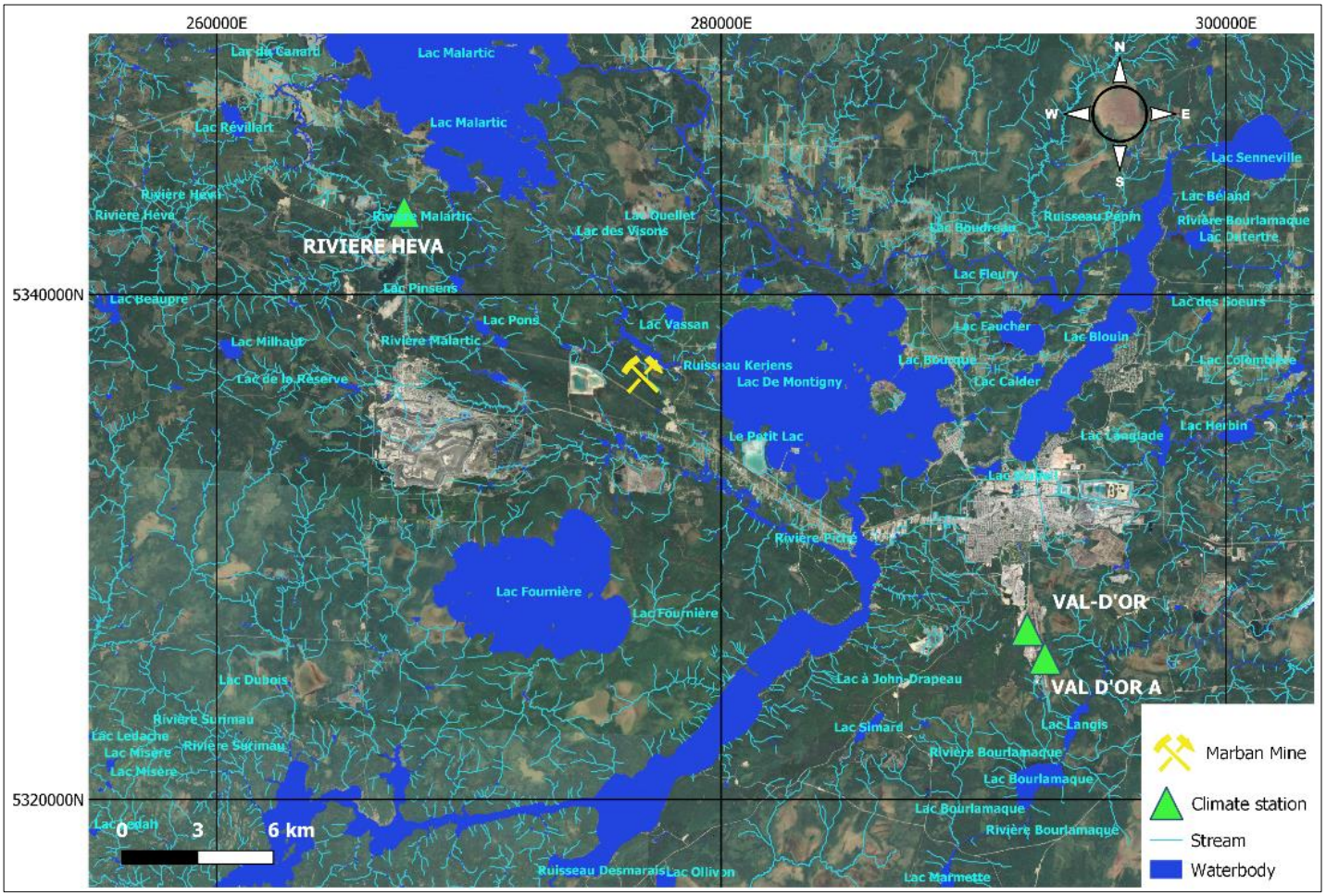
A 10 m wide access road provides access to the Explosives Storage Facility from the Process Plant. The storage area is located as per provincial quantity-distance guidelines: 380 m radius from buildings and roads, and 760 m radius from any residence. The gated pad will be approximately 100 m x 28 m, ensuring the two storage magazines are 82 m apart and separated by a berm. The magazines will store 32,000 kg of powdered explosives and 600 cases of explosives caps. The magazines are steel structures, similarly sized as shipping containers, measuring 12.19 m long x 2.44 m wide x 3.66 m high.

18.6 Water Management Facilities

18.6.1 Site Water Balance

A sitewide water balance analysis was completed to account for various water inflows, containments, losses, reclaims, make-up water and discharges from and into the project site. Figure 18-3 shows the approximate location of the Marban Engineering site, water bodies, streams, and climate stations nearby.

Figure 18-3: Location of Marban Engineering Project, Waterbodies, Streams, and Climate Stations



Source: Ausenco, 2022.

To develop a site-wide water balance, different water components and losses were considered as follows:

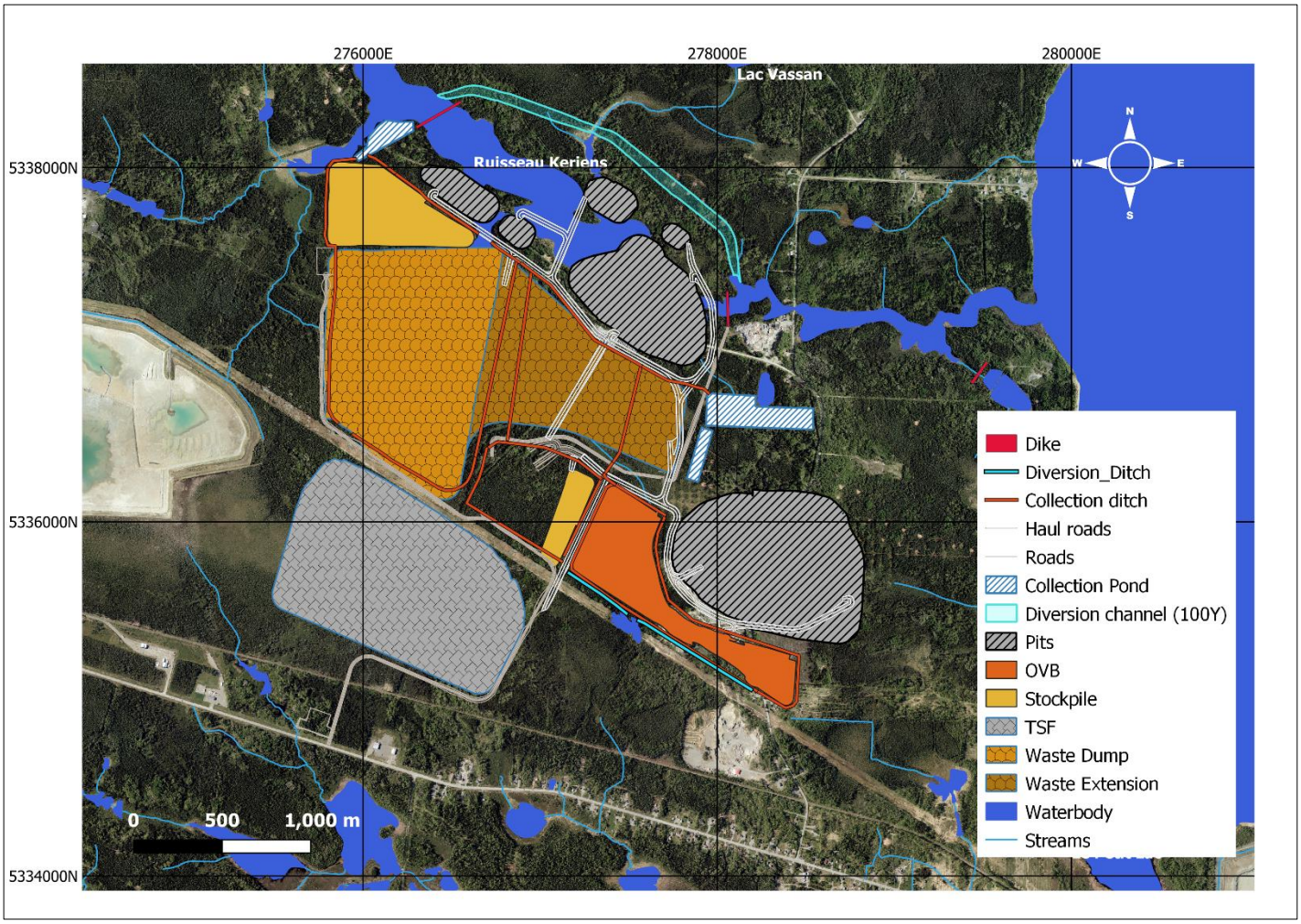
- Runoff as a result of precipitation falling on different facilities, pits, roads, and ponds;
- Groundwater inflow into pits which will need to be pumped out during operation;
- Make-up water requirement of process facilities and the amount of water leaving the site with the final products; and
- Water losses due to evaporation and seepage.

The water balance of the TSF and water exchange between the Process Plant and the TSFs were used as input for the analysis and modelling.

18.6.2 Water Management

Contact water across the mine site was designed to be collected using a collection system. Additionally, clean runoff is designed to be diverted by diversion ditches. Figure 18-4 shows the location of mine facilities and the water management system.

Figure 18-4: Mine and Water Management Facilities



Source: Ausenco, 2022.

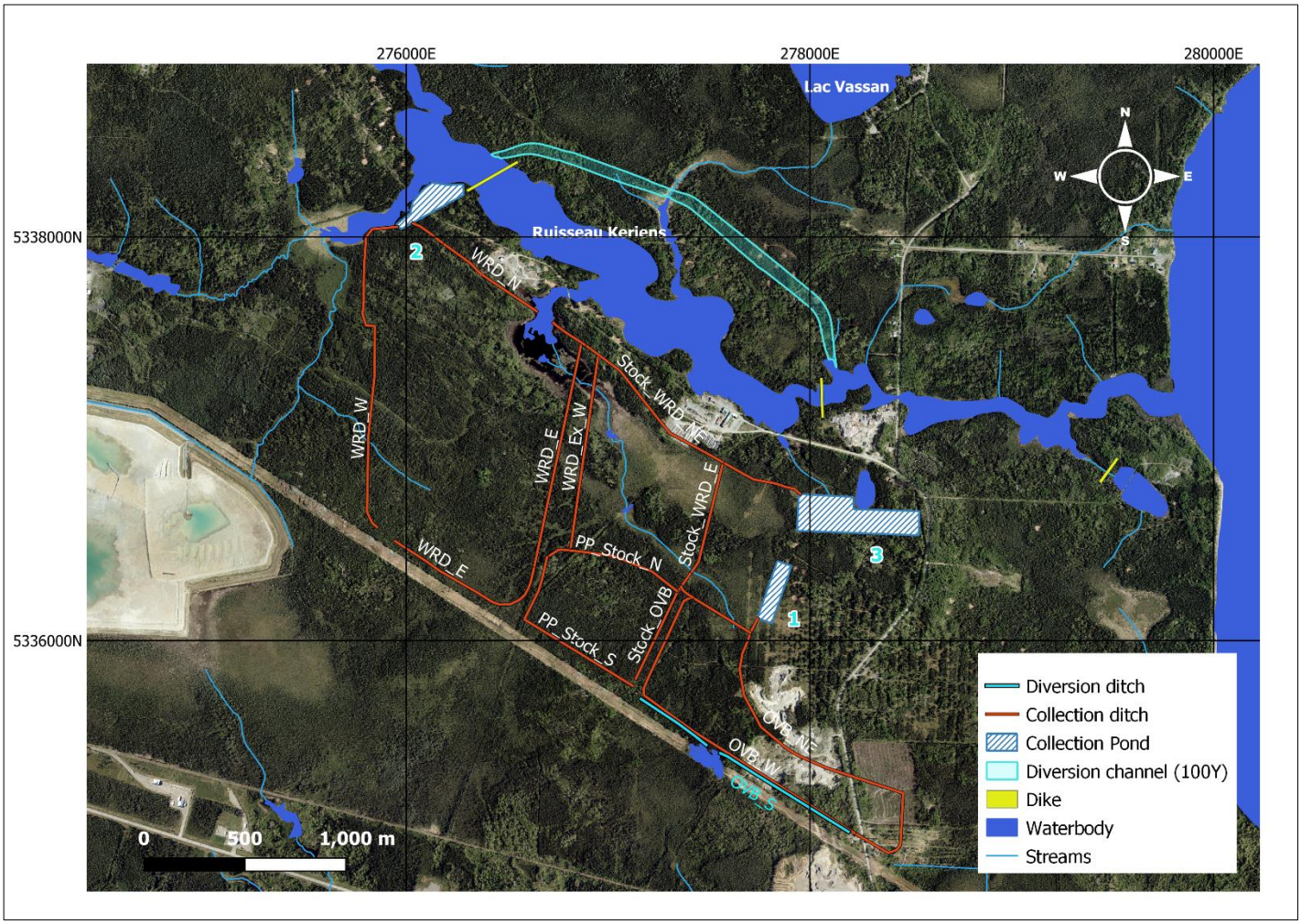
Runoff from the majority of mine facilities is considered contact water and will be collected by a collection system and contained in ponds before discharge to the environment. The runoff is conveyed throughout the site using collection ditches.

Ditches have been sized to convey the design storm event (24-hour, 50-year return period). Ponds were sized to contain the 24-hour, 10-year storm event.

Design peak flow rates and flood volumes were estimated using the Hydrologic Engineering Center – Hydrologic Modelling System (HEC-HMS) model, assuming a Soil Conservation Service (SCS) Type II storm with precipitation depths at the Riviere Heva station. Extreme storm events were derived from Intensity-Duration-Frequency curves and tables developed by Environment and Climate Change Canada (ECCC).

The dimensions of the sized ditch segments and ponds (shown in Figure 18-5) are presented in Table 18-5 to Table 18-7. Note that ditches for the tailings facilities are not shown, as they are included in the design of the facilities themselves

Figure 18-5: Collection Ditches, Diversion Ditches, and Ponds



Source: Ausenco, 2022.

Table 18-5: Dimensions of Sized Ditch Segments

Ditch Segment	Bottom Width (m)	Side Slope (H:V)	Design Depth (m)
WRD_Sat_W	1.0	2:1	1.0
WRD_Sat_N1	1.0	2:1	1.0
WRD_Sat_N2	1.2	2:1	1.0
WRD_Sat_N3	1.5	2:1	1.0
WRD_Sat_S2	1.5	2:1	1.3
WRD_Sat_S1	1.5	2:1	1.4
WRD_W	1.5	2:1	1.0
WRD_N	1.0	2:1	0.9
WRD_NW	0.7	2:1	0.6
OVB_W	1.0	2:1	0.9
OVB_NE	1.0	2:1	1.0
Stock_OVB	0.7	2:1	0.6
PP_Stock_N	1.5	2:1	1.0
Stock_WRD_E	1.5	2:1	1.5
Stock_WRD_NE	2.0	2:1	1.7
WRD_Ex_W	1.2	2:1	0.9
WRD_E_Outlet	2.0	2:1	1.5
WRD_E	1.5	2:1	1.2
PP_Stock_S	0.7	2:1	0.7
Outflow_PondN	5.0	2:1	1.9

Table 18-6: Dimensions of Sized Diversion Ditches

Ditch Segment	Bottom Width (m)	Side Slope (H:V)	Design Depth (m)
Div_PondN	1.5	2:1	1.2
DivOVB_WRD	0.7	2:1	0.7

Table 18-7: Dimensions of Sized Collection Ponds

Pond #	Maximum Depth (m)	Length (m)	Width (m)
1	3.2	340	70
2*	3.2	300	65
3	3.2	405	85
4	3.2	595	125

Note: *Shape of Pond#2 is not rectangular due to limitations of nearby facilities

It should be noted that 50 cm of the pond depth is reserved for the settled sediments as well as 50 cm for the freeboard.

18.6.2.1 Water Components

Each water component of the water balance is described below.

18.6.2.1.1 Runoff

Monthly runoff was estimated as a combination of rainfall and snowmelt for three scenarios: average conditions, wet year, and dry year. Runoff coefficients were determined with a weighted SCS Curve Number (CN). Monthly estimates are presented in Table 18-8.

Table 18-8: Monthly Runoff Estimates Across the Site for Different Climate Scenarios

Month	Runoff (m^3/day)		
	Average	Wet	Dry
January	5,304	269	6,632
February	2,063	2,563	964
March	4,981	9,279	5,061
April	9,465	11,202	11,526
May	12,229	12,802	4,797
June	9,133	11,545	5,861
July	7,744	5,698	3,616
August	7,563	12,212	9,609
September	8,624	10,794	3,153
October	6,388	14,852	3,949
November	5,141	9,282	2,448
December	907	2,656	4,864

18.6.2.1.2 Groundwater Inflow

Groundwater inflow into the pits depends on the excavation depth in the course of mine operation. Ausenco received estimates of groundwater inflow from WSP as constant daily values as shown in Table 18-9.

Table 18-9: Estimates of Groundwater Inflow for Different Pits, Provided by WSP

Pit	Groundwater Inflow (m^3/day)	
	Year 1 to Year 7	Year 7 to Year 11
Marban	3,892	4,225
Norlartic	3,125	3,154
Norlartic Sub-pit 1	196	192
Norlartic Sub-pit 2	475	478
Kierens Pit	450	449
Kierens Sub-pit	101	102

Estimates in Table 18-8 do not include precipitation or evaporation from the pits. Additionally, most of the Norlartic pit will be used to store tailings material throughout mine operations (TSF2 is activated from the fifth year onward). Consequently, Norlartic groundwater inflow was assumed to decrease linearly as tailings deposits gradually fill the pit.

18.6.2.1.3 Process Plant and TSF

Water requirements of the Process Plant were compared to the recoverable water at the TSF and monthly estimates of make-up water demand (to be sourced from other sources) were calculated for two phases of the project. Phase 1 is the first 4.5 years of operation and Phase 2 extends from Year 4.5 to the end of mine life (Year 11). Table 18-10 presents estimates of hourly water requirements.

Table 18-10: Estimates of Process Water Needs for Different Project Phases

Month	Water Requirement (m^3/h)	
	Phase 1 – TSF1	Phase 2 – TSF2
January	273.3	250.1
February	273.3	263.5
March	273.3	240.6
April	176.2	219.1
May	211.3	238.6
June	221.2	261.0
July	249.7	261.0
August	278.8	261.0
September	229.4	261.0
October	183.1	215.9
November	225.1	232.2
December	273.3	236.8

Note that phases in the above table are differentiated based on TSF locations.

18.6.2.1.4 Evaporation Losses

Water loss due to evaporation has been estimated using long-term monthly evaporation rates and ponded surface areas. Long-term monthly evaporation rates were extracted from ECCC climate normal between 1981–2010 at Amos station, 48 km north of the Marban mine site. Evaporation losses are summarized in Table 18-11.

These values of water inflows and outflows, along with containment capacities of the ponds were incorporated into the water balance model to estimate water discharges from the site.

Table 18-11: Evaporation Loss from Ponded Areas

Month	Evaporation Loss (m^3/day)
January	0.0
February	0.0
March	0.0
April	0.0
May	517.1
June	623.6
July	623.6
August	486.7
September	304.2
October	0.0
November	0.0
December	0.0

Note: Ponded areas are based on Amos station climate normal and ponded surface areas.

18.6.3 Water Balance Model

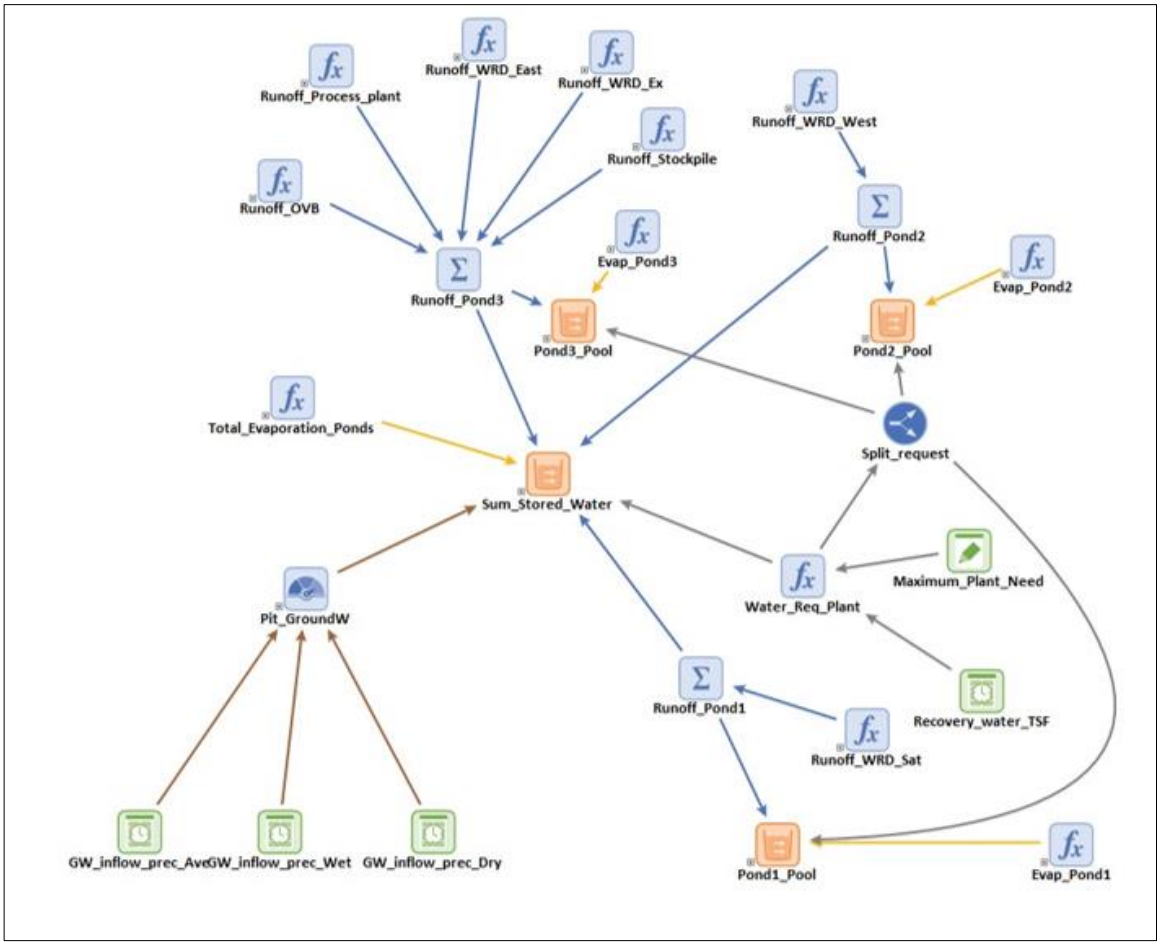
Different water components of the site were incorporated into the GoldSim model. GoldSim is a probabilistic simulation software suitable for system dynamics problems and discrete event simulations. The model uses Monte Carlo framework to randomly sample variables and simulate mathematical problems.

- The model was structured to include 3 main components (inputs, computations, and outputs) at the top level, with 6 sub-levels for inputs, and 73 elements in total.
- Simulations were launched for three different weather scenarios: 1- Average, 2- Wet, and 3- Dry.
- The model was set up to run for 11 years including different phases for TSF location and groundwater (GW) inflow rates at daily time steps.

In general, the Marban site has a water excess, and discharge rates are seasonally variant. Almost half of the water inflow to the site is of GW origins (GW inflow) and the remaining is the surface component. The estimates of required water for dust suppression are not included in this analysis.

Figure 18-6 shows the schematic water exchange in the GoldSim model.

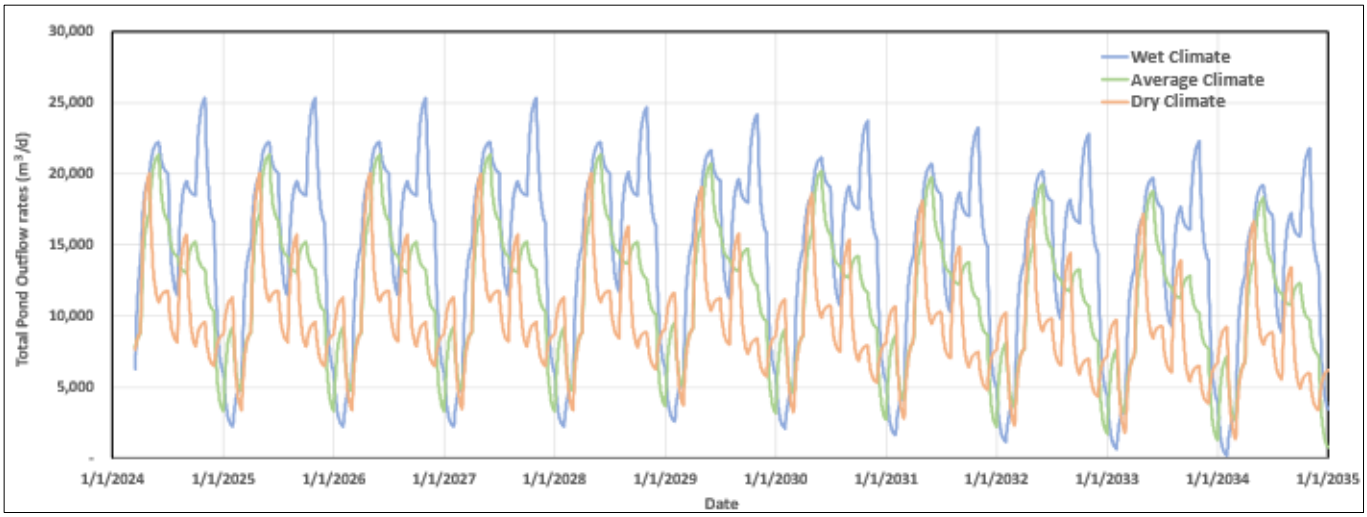
Figure 18-6: Schematic Representation of Water Components in GoldSim Model



Source: Ausenco, 2022.

Figure 18-7 shows the variation of excess water during the LOM for different weather scenarios. In an average climate, daily excess water at the mine site ranges between 9,600 to 13,700 m^3/d. Based on provided rates, pit GW inflow could meet Process Plant water needs during the mine operation. Depending on the quality of the GW (whether or not water treatment is needed), the remaining GW inflow may be directed to a treatment plant, contained in a temporary pond, or released to environment.

Figure 18-7: Modelled Estimations of Excess Water during the LOM Period



Note: The date axis (horizontal) shows the simulation time period (2024–2033) and does not necessarily show the actual LOM period. Source: Ausenco, 2022.

As shown in Figure 18-7, monthly excess water decreases particularly during the operation period of TSF 2, as the Norlartic Pit would be filled with tailings material (reduction of GW inflow). The reduction in GW inflow has been assumed to be linear with time. Table 18-12 shows the data from Figure 18-7 in tabular form.

Table 18-12: Mean, Minimum, and Maximum Excess Water from the Mine Site for Different Climate Scenarios

Year	Excess Water from the Mine Site (m^3/day)								
	Average Climate			Wet Climate			Dry Climate		
	Mean	Min	Max	Mean	Min	Max	Mean	Min	Max
2024	13,691	3,321	21,349	17,541	6,043	25,363	10,875	6,488	20,035
2025	12,319	3,290	21,347	15,175	2,241	25,350	10,267	3,422	20,035
2026	12,319	3,303	21,347	15,184	2,242	25,350	10,278	3,422	20,035
2027	12,315	3,303	21,346	15,182	2,225	25,350	10,275	3,437	20,007
2028	12,312	3,304	21,350	15,173	2,230	24,697	10,270	3,393	20,035
2029	12,017	3,215	20,720	14,873	2,590	24,206	9,966	3,732	19,075
2030	11,537	2,735	20,239	14,402	2,110	23,725	9,496	3,251	18,594
2031	11,069	2,266	19,773	13,936	1,628	23,258	9,029	2,802	18,107
2032	10,562	1,768	19,292	13,423	1,149	22,782	8,520	2,272	17,643
2033	10,100	1,296	18,803	12,956	674	22,287	8,048	1,816	17,158
2034	9,615	810	18,318	12,480	190	21,802	7,574	1,331	16,673

To meet the water requirements of the Process Plant, the following points should be considered:

- The Process Plant requires 14 m^3/h of fresh water for its users. This requirement could be supplied from collection ponds. However, if the collected contact water does not meet water quality criteria for these users, a separate water supply point should be determined. Nearby natural ponds could be used for this purpose.
- When pit excavation reaches galleries and GW inflow increases, pumped water from the pits is estimated to be sufficient for process requirements.
- During the cold season when surface runoff is limited and pumping capacity may be affected, water from collection ponds could be used meet the process requirements.

18.6.4 Keriens Creek Diversion Channel

The project requires relocation of a section of Keriens Creek, upstream of the outlet to De Montigny Lake, in order to provide access to the Kierens and Norlartic Pits, located underneath the existing creek alignment.

18.6.4.1 Hydrological Analysis

The closest climate stations to the site are the Rivière-Héva, Val D'Or, and Amos stations.

Intensity Duration Frequency (IDF) curves and tables for these three stations were consulted for 24-hr storms (Table 18-13). The precipitation spatial pattern appears to be moderately heterogeneous across these three stations, with the lowest precipitations at the Val d'Or station, and highest precipitation at the Amos station, approximately 45 km north of the mine site.

Table 18-13: Extreme Precipitation Depths for the Nearby Stations during 24-hr Storms

Return Period (Years)	Rivière-Héva (mm)	Val d'Or (mm)	Amos (mm)
2	43.1	41.6	48.6
5	54.5	50.3	68.0
10	62	56.1	80.9
50	78.6	68.9	109.2
100	85.6	74.3	121.1

The diversion channel is classified as an Extreme Consequence Facility (Hydraulic structure), as per the Canadian Dam Association (CDA). This selection is based on the proposed mining operations along the existing creek bed (potential for over 100 fatalities) and its use for tailings storage during the second phase of the mine life (potential major loss of habitat). For an Extreme Consequence Facility, the Canadian Dam Association Safety Guidelines specify that the Probable Maximum Flood (PMF) should be the design target level.

Extreme storm events and Probable Maximum Precipitation (PMP) estimates previously reported by Golder (2020) are shown in Table 18-14, for the purposes of the PFS design.

Note that a creek overflow situation would not directly result in flooding of Lac Vassan, but could result in localized flooding indirectly as Lac Vassan might not be able to drain into Keriens Creek.

Table 18-14: Extreme Precipitation Variables for the Marban Engineering Project

Parameter	Value (mm)	Source
1:100-year snow cover water equivalent	350	SNC-Lavalin, 2004
24-h 1:100-year rainfall	91	Golder analysis based on historical precipitation data (1961–2017)
24-h 1:1000-year rainfall	121	
24-h 1:2000-year rainfall	130	
24-h Probable Maximum Precipitation (PMP)	355	SNC-Lavalin, 2004

Source: Golder, 2020.

18.6.4.2 Diversion Channel Design

Based on the combination of LiDAR elevation datasets (MFFP, 2020) and bathymetric data (Niogold, 2013), grade and alignment of the diversion channel were optimized (to minimize excavation volumes). Designs and quantity estimates were completed for the 100-year 24-hour storm event (1:100Y) and Probable Maximum Flood (PMF). The 100 year flood was selected as the design event for the diversion channel per O3 Mining request. This included setting up a hydrologic model for this weather scenario to determine peak flow rates and designing the optimum channel dimensions to safely convey the storm runoff.

A diversion channel to safely convey the 100-year 24-hour event, should be at least 25 m wide, and 4 m deep (assuming a 2H:1V side slope). Inlet and outlet invert elevations were set to +294.5 m and +294.0 m respectively, to provide a longitudinal slope similar to that of natural channel. Table 18-15 shows estimates of excavation volume for the diversion channel system, including the dikes.

Table 18-15: Estimates of Excavation Volumes for the Diversion System

Type	Cut (m^3)	Fill (m^3)
Diversion Creek	776,900	570
Dikes	300	68,300

Source: Ausenco, 2022.

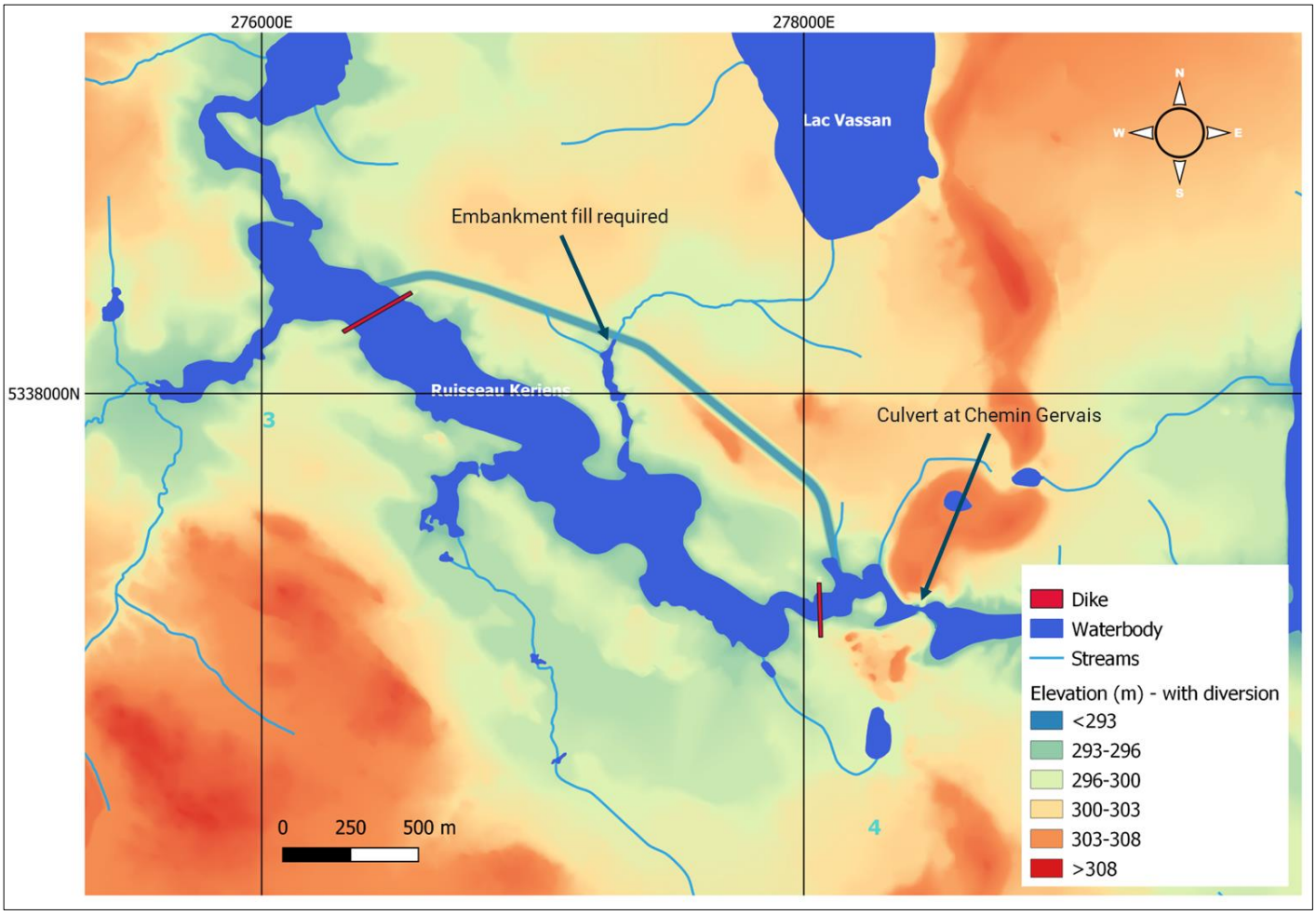
18.6.4.3 Hydraulics Model

The flood resulting from the 100-year 24-hour storm event was routed along the diversion system (dike and channel) downstream using diffusion wave equations for the proposed terrain. Results showed that the channel is capable of conveying the design peak flow without overtopping at a maximum flow velocity of 0.6 m/s. It has also been shown that the contraction at the existing Chemin Gervais bridge over Keriens Creek results in a ponding effect upstream during the storm event. The culvert under the bridge is a circular pipe of 2.6 m diameter.

The conveyance capacity of the culvert is limited during the 100-year 24-hour storm event and would temporarily result in elevated water levels upstream. Proposed channel excavations provide enough freeboard for the diversion channel along most of its length, however, embankment fill would be required at the middle section. This is the intersection of a stream segment connecting Lac Vassan to the Keriens Creek, as shown in Figure 18-8.

Flooding from the diversion would not reach Lac Vassan (northeast of the diversion channel), but high water levels of the diversion could cause backflow of runoff from the lake and cause localized flooding adjacent to the lake.

Figure 18-8: Plan View of the Developed Elevation with the 100-year 24-hour Diversion Channel



Source: Ausenco, 2022.

18.6.5 Process Water Supply & Distribution

Process water from the tailings thickener overflow and decant TSF water will report to the process water tank. From there it will be distributed to required users in the plant, such as grinding mills, reagent mixing, and vibrating screen spray bars.

18.6.6 Fresh Water Supply & Distribution

Fresh water will be sourced from the Marban Pit. It will be directed to the fresh/fire water tank, where it will be distributed to required points in the plant, and feed the potable water treatment system, elution circuit, and reagent systems. The bottom section of the fresh/fire water tank will be dedicated for the fire water system.

18.6.7 Potable Water Supply

The quality requirement for the potable water treatment plant will match the local drinking water guidelines. Fresh water will be sourced from the freshwater intake pump and processed through the potable water treatment skid before being stored in the potable water tank.

Prior to further use, the potable water will be heated by the tepid water heating skid before being distributed to safety showers and other points in the Process Plant facilities. The distribution piping will be heat traced and insulated wherever it is not inside a heated building. Where necessary, manual drain points will be included.

18.6.8 Fire Water Supply

All facilities will have a fire suppression system in accordance with the structure's function. For the most part, fire water will be used with an underground ring main network around the facilities. All buildings will have hose cabinets and handheld fire extinguishers. Electrical and control rooms will be equipped with dry-type fire extinguishers. Ancillary buildings will be provided with automatic sprinkler systems. For the reagents, appropriate fire suppression systems will be included according to their material safety datasheets.

18.6.9 Domestic Effluent and Sanitary System

A domestic effluent and sanitary system package will be supplied at the Process plant area to treat all domestic waste collected within the site. The collection network will be underground. Office and domestic waste will be collected and disposed of off site in accordance with applicable regulations.

18.7 Site Geotechnical Conditions

WSP conducted a geotechnical-hydrogeological field program in 2021 from May 28 to December 13. The associated lab programs were completed in Spring 2022.

During the program, the following infrastructure facilities were investigated:

- Process Plant (PP)
- Tailings Storage Facility (TSF)
- Waste Rock Stockpile (WRS)
- Overburden Stockpile (OS)
- Stockpile (LGS)

18.7.1 Drilling Program

WSP completed 20 geotechnical boreholes and 27 hydrogeological boreholes between May 28 and December 13, 2021, for the 2021 geotechnical-hydrogeological field program.

According to the WSP Marban Project Factual Report (WSP, 2022), all the geotechnical boreholes and some of the hydrogeological boreholes that were installed as observation wells in the bedrock were photographed and logged for geology, rock quality designation (RQD), and total core recovery (TCR). Disturbed and undisturbed soil samples were collected for laboratory tests. In-situ tests such as standard penetration tests (SPT) and Shear Vane tests were conducted. Hydrogeological boreholes installed as observation wells in the soils were advanced in a destructive mode. Permeability tests and well development were completed in the hydrogeological boreholes.

Open standpipes and/or Casagrande piezometers were installed in all geotechnical boreholes, and observation wells were installed in all hydrogeological boreholes. Installation details of the open standpipes and/or Casagrande piezometers, and observation wells are described in the WSP Marban Project Factual Report (WSP, 2022).

The numbers of boreholes drilled at each proposed mine facility are listed below:

- Low-Grade Stockpiles (LGS): two boreholes;

- Waste Rock Stockpiles (WRS): seven boreholes;
- Tailings Storage Facility 1(TSF 1): two boreholes near the TSF;
- Process Plant (PP): one borehole.

In 2021, ten (10) geotechnical/hydrogeological boreholes (173.26 m total drilling) were completed at or near the proposed mine infrastructure facilities locations to evaluate the deeper foundation conditions, assess the constructability of the different facilities, and aid the design. It should be noted that some of the boreholes were not in the locations of the final PFS designs due to layout improvements during the PFS engineering.

18.7.2 Test Pit Program

According to WSP, the 2021 test pit program was conducted from September 4 to December 8, 2021. A total of 52 test pits were excavated to provide a detailed visual examination of near-surface soil, groundwater, and bedrock level. Soil stratigraphy was logged, and soil samples were collected continuously at an interval of 0.5 m at each test pit. Soil samples in 36 test pits were collected at regular depths for chemical analysis to determine the reference state of the soils on the site, in accordance with MELCC's recommendations (WSP, 2022).

The number of test pits at each proposed mine facility are listed below:

- Low-Grade Stockpiles (LGS): one test pit;
- Waste Rock Stockpiles (WRS): five test pits;
- Tailings Storage Facility (TSF): one test pit;
- Process Plant (PP): one test pit.

Out of the total 52 test pit completed in 2021, eight (8) test pits were completed within the current footprint of the project's infrastructure. It should be noted that some of the test pits were not in the location of the final PFS designs due to layout improvements during the PFS engineering.

18.7.3 Laboratory Program

Representative samples were collected from test pits and the boreholes during the 2021 field program for laboratory testing programs. The laboratory testing programs included a variety of tests to classify soil and rock samples to characterize the ground conditions in the project area and provide data for future engineering studies.

The laboratory testing program on selected borehole samples included the following tests:

- Grain size distribution analysis (BNQ 2501-025);
- Atterberg Limits (BNQ 2501-090);
- Water content determination (BNQ 2501-170);
- Volumetric weight (ASTM D7263);
- Consolidation test (ASTM D2435);
- Uniaxial Compression Strength (ASTM D2216);
- Consolidated Undrained (CU) Triaxial Test (ASTM D4767); and
- Hydraulic Conductivity Using a Flexible Wall Permeameter (ASTM D5084).

18.7.4 Stratigraphy of the Project Area

The project site consists of overburden overlying bedrock. Overburden material consists of backfill material and natural soil. The 2021 field investigation indicated the overburden thickness varies from 0.7 m to 32.6 m across the project site. Native soil across the site generally contains five stratigraphic units deposited in the sequence of the oldest to the youngest from the bottom to the top, including organic soil near the ground surface underlain by oxidized glaciolacustrine sediments. Beneath the oxidized glaciolacustrine sediments is glaciolacustrine sediments overlying glaciofluvial sediments that underlain by till sediments.

However, due to the complexity of the geological history, sediment transport, and deposition processes, the thickness of stratigraphic units varies across the site and some stratigraphic units may be thinner or not be present in some areas of the site. For example, glaciolacustrine sediments thickness varies as ground altitude changes. Places at low altitudes are more likely to have thicker glaciolacustrine sediment deposits, whereas places at high altitudes are more likely to have thinner or no glaciolacustrine sediment deposits. Because glaciolacustrine sediment was deposited in pale-glacial lakes in low altitude areas, an area with low elevation is more likely to contain thicker glaciolacustrine deposits. As elevation increases, the glaciolacustrine sediment becomes thinner. Overburden thickness is generally thinner in high altitude areas.

The bedrock lithology is consistent across the project site consisting of dark grey, chloritized basalt with quartz and/or carbonate veins. The upper 2 to 3 m of bedrock is often highly fractured. Below the fracture zone, the bedrock becomes good to excellent quality.

18.7.5 Groundwater

According to WSP (2022), groundwater level data were collected from geotechnical and hydrogeological boreholes in December 2021 when most of the fieldwork was completed. The groundwater level in the boreholes ranged from 0 to 8.89 m below ground surface (mbgs) across the site.

18.8 Tailings Storage Facilities

The Marban project has multiple open pits; the northern pits (Kierens and Norlartic) are exhausted prior to the end of the project mine life and are therefore available for in-pit tailings storage during operations. This reduces the need to place 100 percent of the tailings in a traditional TSF where the tailings are stored behind an embankment.

The primary design objectives for the TSFs are secure confinement of tailings and protection of the regional groundwater and surface water during both mine operations and in the long-term (after closure). The design of the TSFs and associated water management facilities has accounted for the following:

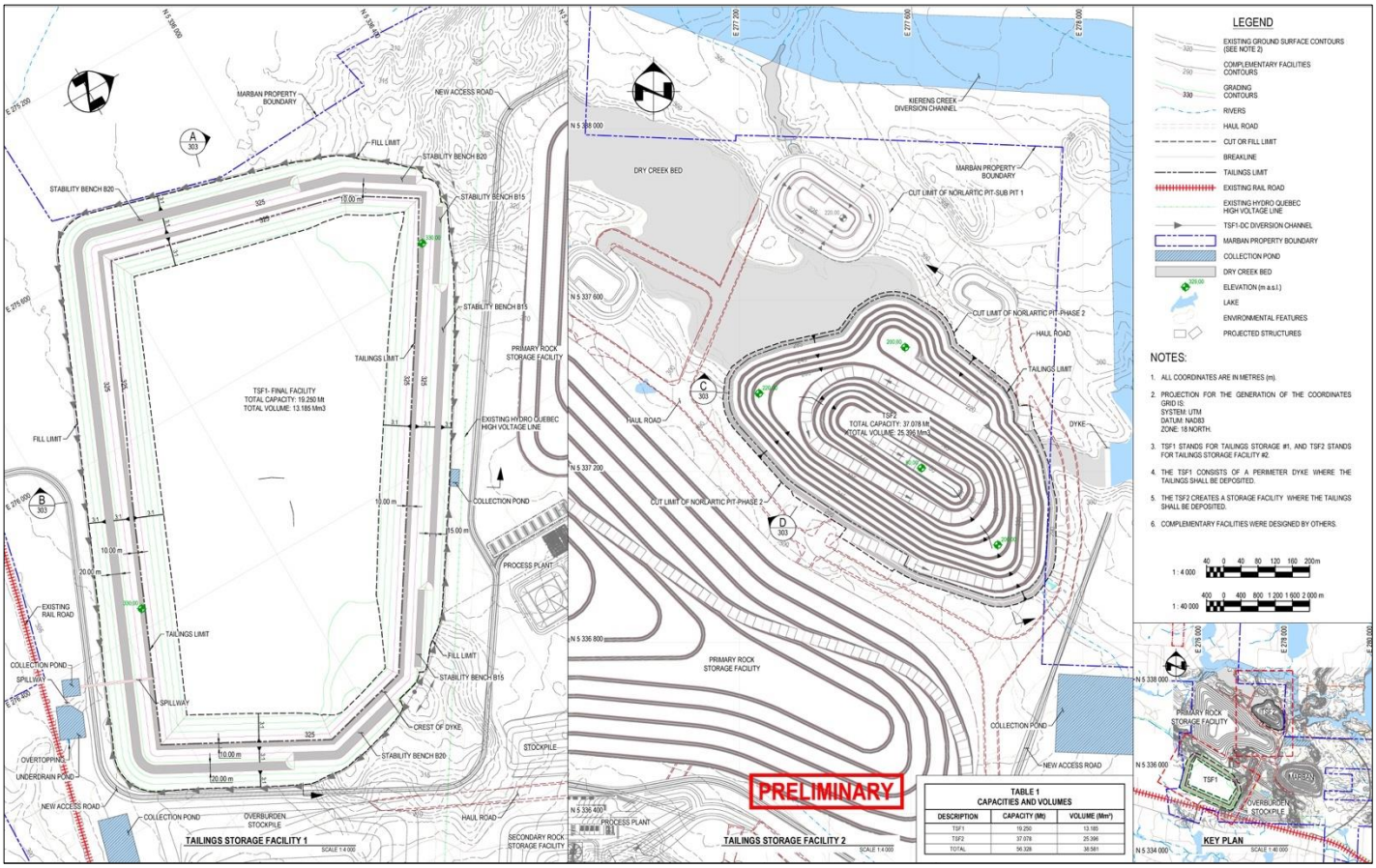
- No geosynthetic lining of the bottom of the facility to limit seepage (based on the seepage analysis by WSP);
- Staged development of the facility over the life of the project;
- Flexibility to accommodate additional tailings if the resource increases; and
- Control, collection, and removal of water from the facility during operations for recycle as process water to the maximum practical extent.

Approximately 56.4 Mt of tailings will be stored within the two tailings facilities, including 19.3 Mt in a conventional tailings storage facility (TSF 1) and 37.1 Mt in in-pit storage (TSF 2).

18.8.1 Tailings Storage Facilities Design Criteria

The proposed process plant site location in relation to the TSF 1 and TSF 2 is shown in Table 18-9. Design and operating criteria have been developed for the current design.

Figure 18-9: TFS 1 and TSF 2 General Layouts



Source: Ausenco, 2022.

The design criteria reflect the mine plan and operating strategy. Considerations used in the development of the TSF design are summarized as follows:

- TSF and water management facilities are located on Marban Property.
- Integration of mine rock release schedules from the GMining group.
- Required storage of 56.4 Mt of tailings:
  - TSF 1 has a capacity of 19.3 Mt.
  - TSF 2 has a capacity of 37.1 Mt.
- Nominal tailings discharge of 6.0 Mt/a (16,438 t/d).
- Tailings discharge solids content = 60% w/w.
- Dry tailings density of 1.45 t/m^3.
- Annual staging of the TSF 1 dam lifts allows for storage of subsequent years of tailings disposal, and in-pit TSF 2 a minimum operational pond volume of 0.3 Mm^3 and storage of the inflow design flood (IDF) during operations with sufficient freeboard for wave run-up and embankment settlement.

- TSF filling schedule based on the detailed mine schedule including material movements for dam construction and mine rock disposal.
- Water for the process plant sourced from the TSF supernatant pond with additional make-up water from Marban Pit as required.
- Limiting impacts to wildlife and fisheries resources.
- Designing cover system for closure.
- Meeting or exceeding applicable regulatory requirements and industry guidelines for stability events.

TSF 1 is a ring dike that is classified as "VERY HIGH" consequence category under the Canadian Dam Association's (CDA's) "Dam Safety Guidelines" (2014). According to CDA, the consequences of failure include potential loss of life of 100 or fewer and very high critical economic losses. Losses may include, but are not limited to, significant fish or wildlife habitat, infrastructure damage, loss of mining equipment, ore sterilization, and loss of tailings containment.

TSF 2 is in-pit disposal, therefore there is no dam. However, TSF 2 has been classified as "LOW" consequence category. According to CDA, the consequences of failure include potential loss of life of 0 and low economic losses. Losses may include, but are not limited to, low fish or wildlife habitat, infrastructure damage, loss of mining equipment, ore sterilization, and loss of tailings containment.

Despite the differences in TSF dam consequence classifications, the TSF 1 and TSF 2 for the Marban Engineering Project were designed for the most extreme events (i.e., "EXTREME" consequence category), which is the highest design standard defined by CDA Dam Safety Guidelines:

- IDF – Probable Maximum Flood Event (PMF) = 355 mm.
- EDGM –1:10,000-year event (or MCE) = 0.14 g.

18.8.2 Construction and Operations

Prior to construction of the TSF 1 Embankment, the embankment foundation and select areas of the TSF basin will be prepared for the earthwork dam construction. The following activities will be required for the preparation of the TSF 1 Embankment foundation and TSF 1 basin:

- Clearing and grubbing of the TSF Embankment footprint and select areas of the TSF basin;
- Stripping of topsoil and organics to stockpile;
- Removal of unsuitable soil and preparation of foundation; and
- Installation of the TSF 1 Embankment underdrain system to the underdrain pond, including supply and installation of dual wall perforated pipes, drainage rock and non-woven geotextile.

Seepage through the embankment will primarily be controlled by the geomembrane, low permeability soil, filter zone on the upstream face of the embankment, and the seepage cut-off structures along the upstream toe of TSF 1. The tailings facility water management includes diversion ditches and sediment ponds.

TSF 2 will be placed into operation in Year 4.5, which is in-pit tailings disposal in to Norlartic open pit that has a capacity for 37.1 Mt. A tailings pipeline and water reclaim pipeline will be constructed at the beginning of Year 4 so operations can commence around June of Year 4. The tailings pipeline will be constructed around the pit with multiple spigots to evenly discharge tailings into the facility. There are no berms or diversion channel required since all surface runoff structure were constructed for development of the open pit.

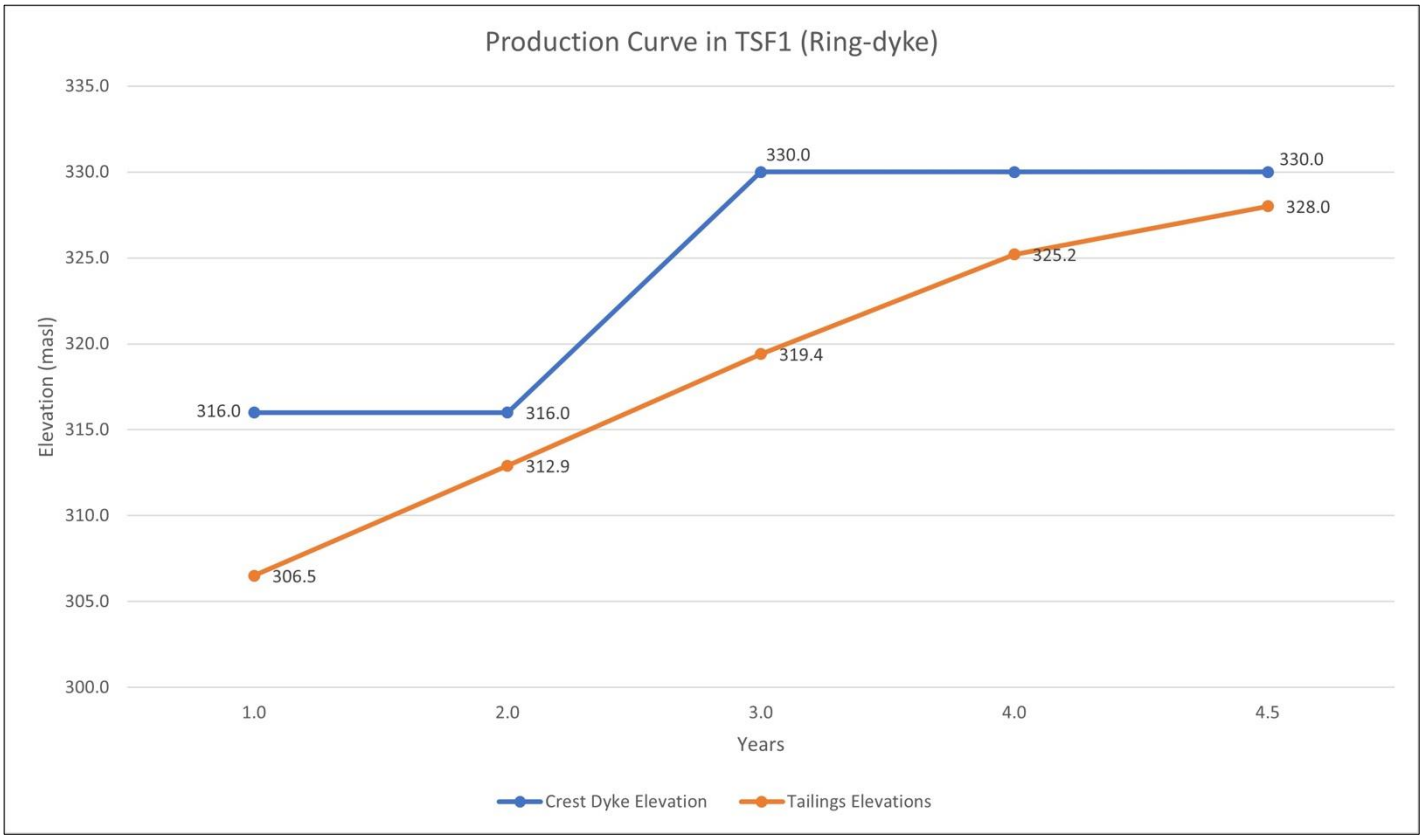
18.8.3 TSF 1 Embankment Construction

The TSF 1 embankments will be constructed to an ultimate elevation of 330.0 masl. The total fill requirement for tailings dam construction is 7.7 Mm^3, of which 1.7 Mm^3 will be used to construct the Starter Embankment (Crest El. 316.0 masl). Initial construction material will be taken from initial waste rock from the development of the pits and local borrow sources, which will be constructed using a contractor fleet. The remainder of the embankment construction stages will be constructed using waste rock by the mine and local borrow sources by a contractor.

The geomembrane, low permeability soils, and filter zone will be constructed on the upstream face from low-permeability glacial till material and filter material from nearby borrow sources and the geomembrane will be placed on top of these material to create a seepage barrier. The geomembrane will be anchor 6 m into the subgrade along the interior toe of the embankment and back filled will local low permeability soils and structural fill to create a seepage cut-off wall. The other embankment zones will include compacted waste rock shell, waste rock external buttress. These zones will be constructed from compacted earth fill and specified waste rock materials, either from a local borrow area or from open pit operations.

A TSF filling schedule was developed that shows the tailings mass balance or cumulative tailings volumetrics, freeboard for PMF, and respective dam crest elevations to contain the water and tailings volumes on an annual basis; is provided in the filling schedule for the TSF in Table 18-10.

Figure 18-10: TFS 1 Filling Schedule

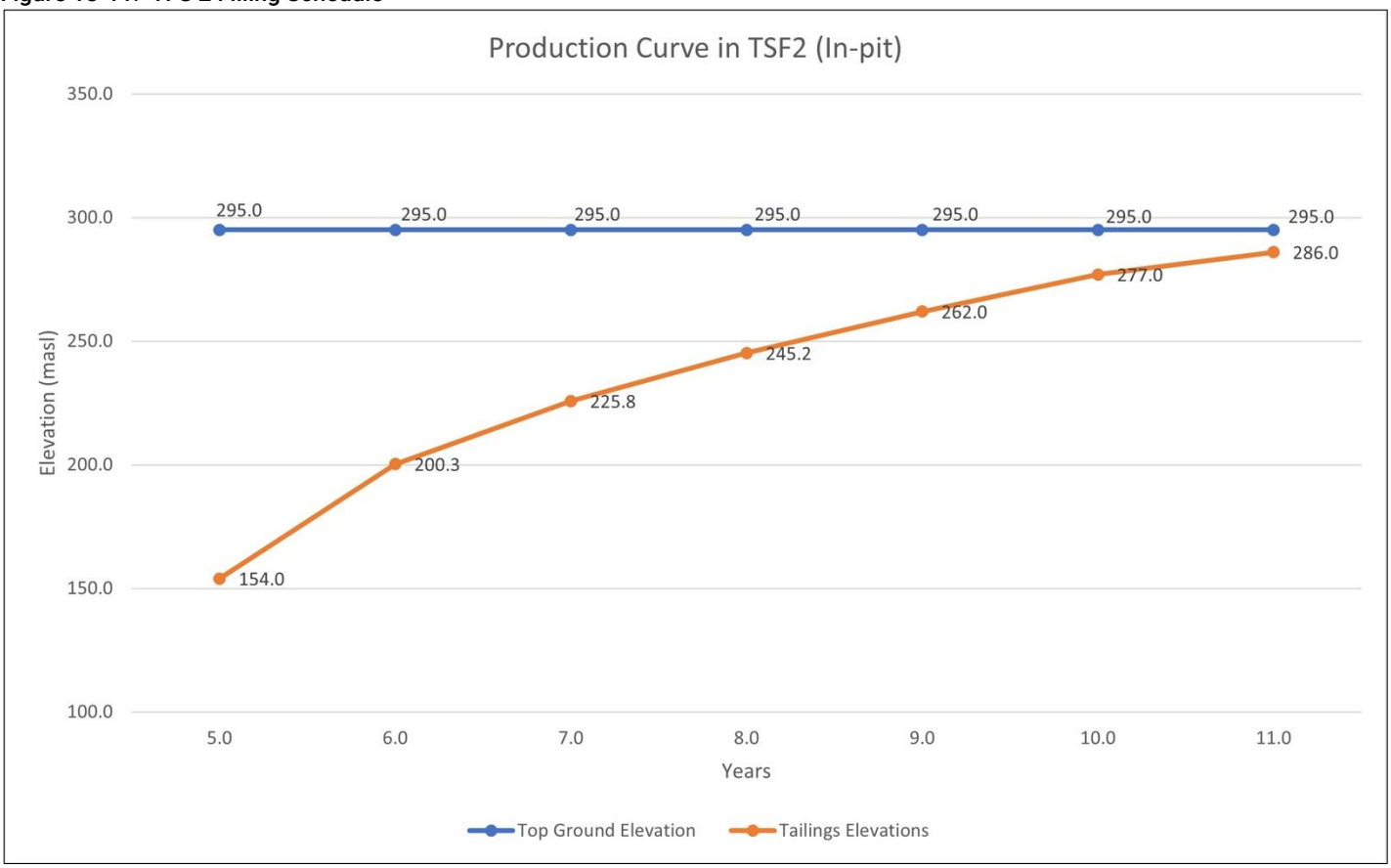


Source: Ausenco, 2022

18.8.4 TSF 2 Embankment Construction

TSF 2 is in-pit tailings disposal and requires not embankments, spillways, diversion channel, access roads to be constructed since these were constructed as part of the development of the Norlartic pit. The only construction required for TSF 2 is the tailings and water reclaim pipelines. A TSF 2 filling schedule was developed that shows the tailings mass balance or cumulative tailings volumetrics, freeboard for PMF, and wave-up and respective dam crest elevations to contain the water and tailings volumes on an annual basis; is provided in the filling schedule for the TSF in Table 18-11.

Figure 18-11: TFS 2 Filling Schedule



Source: Ausenco, 2022

18.8.5 Tailings Distribution & Reclaim System

A start-up water pond volume of approximately 1.6 Mm^3 is expected to be sufficient for the thickened tailings operation based on Ausenco's design criteria of a minimum 180 days of make-up water (Ausenco, 2022). This water will be contained in TSF 1 starter facility. A trenched channel will be excavated within the basin (prior to plant site commissioning) to provide a surface water connection supernatant pond for the barge water recycle system, during the initial production years.

As tailings are deposited into the facility, water will be released from the tailings stream during deposition and subsequent consolidation and will eventually report as supernatant water to the main tailings operations pond. A portion of the supernatant water will be lost due to evaporation, interstitial voids, and from seepage into the foundation. The remaining water will be available as recycle to the plant site. Deposition of tailings will be sequenced such that tailings beaches displace the supernatant pond to the northeast. Over time, the tailings will slowly infill the barge trench. The supernatant

pond level will be maintained at a level that allows for on-going operation of the reclaim water barge. Normal tailings operations will see the lowermost portion of the tailings beach submerged below the supernatant pond. Water will be reclaimed through a floating pump barge and pumped to a reclaim tank near the process plant. The reclaim water line will parallel the same service road being used by the tailings distribution pipeline corridor.

Similarly, TSF 2, in-pit tailings disposal will be sequenced such that tailings beaches displace the supernatant pond to the northeast. Over time, the tailings will slowly infill the barge trench. The supernatant pond level will be maintained at a level that allows for on-going operation of the reclaim water barge. Normal tailings operations will see the lowermost portion of the tailings beach submerged below the supernatant pond. Water will be reclaimed through a floating pump barge and pumped to a reclaim tank near the process plant. The reclaim water line will parallel the haul road being used by the tailings distribution pipeline corridor.

Reclaim and tailing water pipelines will be placed in a HDPE-lined containment trench that will direct potential leaks into emergency pond located at a low point of the corridor.

The tailings distribution and reclaim water systems are designed to convey tailings and reclaim water between the TSF and the plant site.

The tailings distribution system has three key components: a tailings pump station, tailings conveyance pressure pipeline and discharge spigots. Distribution of the tailings around the perimeter of the TSF 1 and TSF 2 is undertaken using two HDPE pressure pipeline alignments, which feed discharge spigots along the embankment crest and around the open pit. The discharge spigots facilitate tailings beach development within the TSF 1 embankment and TSF 2 open pit. The tailings slurry shall be discharged via spigots from various points along the upstream crest of the starter dam and future embankment raises. Whereas for TSF 2, tailings slurry shall be discharged via spigots from various points around the open pit. The preliminary tailings deposition plan for the TSFs are summarized as follows:

- Initial tailings slurry discharge shall be from the southeast embankment of TSF 1 to develop the beaches at this location first to push the supernatant pond to the northwest and tailings slurry discharge shall be from the northwest end of TSF 2 to develop the beaches at this location to push the supernatant pond to the southeast.
- Tailings discharge points shall be moved as required to ensure the apex elevation of the deposition fan does not violate freeboard criteria.
- The method of tailings deposition (spigotting from perimeter) is expected to result in beaches with approximately 1 to 2% overall slopes. The actual tailings beach slope is expected to be steeper at the dam crest (near the deposition point) and flatter downslope. Beach slope angles will vary with solids content, discharge velocity, season, and the total height of tailings. The slurry tailings at 60% w/w solids content will consolidate relatively quickly in the TSF deposit to % solids content with an estimated tailings density of approximately 1.45. Periodic monitoring of beach slope angles and tailings characteristics will allow better prediction of future slope angles and allow for refinement of the tailings discharge plans over time.
- Supernatant pond water will be reclaimed from the TSF 1 and TSF 2 for use as process water for ongoing process plant operations throughout the life of mine.
- Trafficability of tailings will be poor where the tailings are saturated or nearly so. Access to the beach and pond areas shall be restricted.

18.8.6 Instrumentation and Monitoring

Geotechnical instrumentation will be installed along planes through the TSF 1 embankment. The instrumentation will be installed during construction phases and monitored over the life of the project, and into closure. Geotechnical instrumentation is typically comprised of vibrating wire piezometers and slope inclinometers and will be installed in the foundations and embankment fills.

Instrumentation monitoring will be routinely completed during construction, operations, and closure. Measurements during construction will be taken and analyzed on a routine basis to monitor the response of the embankment fill and the foundation from the loading of the embankment fill.

18.8.7 Water Management

The TSF 1 water management consists of diverting non-contact surface water from the surrounding area around the ring dike in trapezoidal diversion channels lined with riprap to existing drainages. The channels are design to convey the 1:100-year storm event. TSF 1 is designed to contain and then pass the Probable Maximum Precipitation (PMP) of 355 mm. The water from large storm events can be used for makeup water or be discharged into the collection pond. A spillway is located at the south end of the facility and discharges into a collection pond which discharges into the underdrain pond. Any excess water flows out of the underdrain pond into a natural drainage located at the south end of the facility.

The TSF 2 water management consists of diversion channel and dikes to convey surface runoff around the Norlartic Pit. These structures are designed to convey the 1:100-year storm event around the pit. TSF 2 is designed to contain and then pass the PMP. The water from large storm events can be used for makeup water or be pumped downstream into Keriens Creek. There is no spillway since the tailings facility is an open pit with no embankments.

19 MARKET STUDIES AND CONTRACTS

19.1 Market Studies

O3 Mining has not completed any formal marketing studies with regards to gold production that will result from the mining and processing of gold ore from the Marban Engineering Project into doré bars. 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 of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with a large number of buyers and sellers active at any given time.

19.2 Commodities Price Projections

The economics analysis for the Marban Engineering Project was calculated at a gold price of US$1,700/oz Au. As of late August 2022, the median consensus price forecast from 30 investment dealers estimated a gold price of US$1,750/oz in 2024 and US$1,700/oz long-term. As of August 30, 2022, the trailing two-year gold price was US$1,830/oz and the trailing three-year gold price was US$1,760/oz. For the purpose of the PFS, a gold price of US$1,700/oz was assumed. The exchange rate used in the study is C$1.00:US$0.77.

19.3 Contracts

O3 Mining plans to contract out the transportation, security, insurance, and refining of doré gold bars. O3 Mining may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in Canada and in countries throughout the world. For the PFS, a cost of C$2.5/oz Au was assumed for transportation, and refining.

20 ENVIRONMENTAL STUDIES, PERMITTING, AND SOCIAL OR COMMUNITY IMPACT

This chapter summarizes the existing environmental and social conditions within the project area based on data available at this stage of the project. It also provides the environmental requirements for ore, waste rock and tailings disposal, site monitoring, and water management. The regulatory context applicable to the project, including the Environmental Impact Assessment (EIA) process and permitting requirements is then overviewed, as well as the social and community requirements. Finally, it outlines mine closure requirements and costs.

20.1 Environmental Baseline Studies

The environmental information provided in this chapter was collected from public databases and from field inventories and environmental surveys (baseline studies) undertaken in 2016, 2021, and 2022 by WSP.

The studied environmental components include the following:

- Hydrological conditions;
- Hydrogeology study and groundwater quality (including hydrogeological modelling);
- Surface water and sediment quality;
- Soil natural background assessment;
- Soil environmental assessment of three old mines sites;
- Biological components (vegetation and wetlands, small mammals, chiropterans, avifauna, herpetofauna, species at risk and aquatic fauna);
- Ambient air quality;
- Ambient noise and vibration;
- Waste rock and ore geochemical assessment (static and kinetic); and
- Tailings geochemical assessment (ongoing).

In 2016, the study area covered the pits and stockpiles area. In 2021, the study area was adjusted to consider a TSF location north and west of Kierens pit (as per the PEA). Tailings storage was subsequently split into two facilities in different locations in the PFS.

In 2022, additional field inventories and environmental assessments were undertaken or will have to be completed in accordance with the general Federal Tailored Impact Statement Guidelines requirements (natural inter-annual and seasonal variability), which include:

- Avifauna (being conducted in 2022);
- Soil environmental assessment on potentially contaminated lands (completed in 2022);
- Species at risk (being conducted in 2022);
- Ambient air quality (to complete over a 1-year sampling period; planned for 2022–2023); and
- Esker monitoring (planned in 2022).

The following environmental studies and assessments will be required to comply with the EIA requirements:

- Predictive hydrology simulation;
- Contaminant dispersion modelling (groundwater);
- Air modelling;
- Greenhouse gas emissions assessment;
- Climate change resilience;
- Noise and vibration modelling; and
- Exhaustive ore, waste rock, and tailings geochemical characterization.

20.1.1 Baseline Conditions

The following subsections summarize the project's current biophysical environmental conditions. Unless indicated otherwise, the information comes from WSP's studies.

20.1.1.1 Hydrological Conditions

The project area is mainly located within Keriens Creek watershed, which flows into the northeast part of the De Montigny Lake. The southern part of the project area is located within the Piché River watershed. Finally, the east part flows directly towards the De Montigny Lake by intermittent streams.

The Keriens Creek and the Piché River are both part of the De Montigny Lake watershed, which is connected to the Harricana River.

The Keriens Creek watershed is covered mainly by forest, lakes, including Vassan Lake, wetlands, and beaver dams. Moreover, a widening of its channel, possibly created by beaver dams, has been observed since 1926 (Corriveau, 2010) and over a distance of 3.2 km upstream of the Gervais Road crossing passage.

At the Gervais Road crossing passage, Keriens Creek has an average slope of 1.7%. From the upstream end of the widening to Gervais Road, the stream has a drop of 0.87 m and an additional drop of 0.48 m from Gervais Road to the De Montigny Lake. During the period from June to October 2016, the average flow was 0.25 m^3/s. Its average annual flow has been estimated at 0.38 m^3/s and its annual low flow rate is 0.062 m^3/s. Supplementary surveys were conducted in 2021 at Keriens Creek and its tributary and at Vassan Lake.

20.1.1.2 Hydrogeological Conditions

The assessment of hydrogeological conditions at the project site was carried out using mainly the data collected in the 2021 investigation campaign but also used data from previous campaigns. Historical data was used to determine the different hydrogeological units and assess hydraulic properties and piezometry as well as groundwater quality.

During the hydrogeological and geotechnical work in 2021, 47 drillings (boreholes and piezometers) were completed. Some of these wells were used as observation wells or for piezometers. Additional stratigraphic surveys (test pits) provided information on the stratigraphy of the study area. A total of 45 permeability tests were performed and packer tests were also completed in some exploration holes to obtain the hydraulic properties of different hydrostratigraphic units.

All collected data and hydraulic properties were used to develop the conceptual model to carry out hydrogeological 3D modelling. The study determined baseline conditions such as groundwater flow direction, hydrogeological formations, permeability of the various units, and groundwater quality. The outcome of the study helped to assess the potential impacts of pit dewatering on groundwater and propose an appropriate monitoring plan. Modelling results show that the drawdown of the water table at the end of the operation is almost zero at about 3 kilometres from the pit in the surface deposits. For

the rock aquifer, the radius of influence of dewatering reaches nearly 4 kilometres. The de Montigny Lake limits the influence of the dewatering in the eastern part. The impacts on rivers would correspond to a base flow reduction between 0 and 88%, or a reduction in average flow between 0 and 6%. The impact on the base flow is maximal (88%) on Keriens Creek and the smaller watercourses -close to the future infrastructure due to Marban pit dewatering. However, the impact will be limited and compensated by deviating the creek.

The percolation rates below the infrastructure have been calculated. They respect the Directive 019 (Ministère du Développement Durable, de L'Environnement et des Parcs du Québec (MDDEP), 2012) criteria for all the mining stockpiles (waste rock, ore, and overburden). Below the tailings storage facility #1 (TSF1) the criteria are respected only if a layer of clay is added to cover the entire storage area.

Given project design modifications and change in the location of the TSF1, additional geotechnical and hydrogeological work will be conducted to determine the stratigraphy and the hydrogeological condition in the area. Some additional boreholes will be installed in the esker to have a better understanding of its extension. The results from these additional investigation campaigns will be used to update the hydrogeological 3D modelling and reassess the potential impacts of the project on the environment. Appropriate mitigation measures will then be proposed, and the monitoring plan will be updated.

20.1.1.3 Ground Water Quality

Results from the sampling campaigns showed that the groundwater in the area has significant concentrations of calcium and magnesium bicarbonates. Among the 40 samples analyzed, 30 samples exceeded the resurgence in surface water (RES) or alert threshold (SA) criteria for one or the other of the following metals: barium, copper, mercury, and zinc. Results for all other metals are below the RES or SA criteria. The drinking water criteria were exceeded for the following metals: aluminum, arsenic, chromium, and manganese.

As a result of these analyses, groundwater natural background levels will be established from the water samples taken from wells distributed in the study area to determine if some parameters may naturally exceed RES or SA criteria.

20.1.1.4 Surface Water & Sediment Quality

Surface water and sediment collection was first conducted in 2012 by Golder at three different stations (RK-12-01, RK-12-02, and RK-12-03). Another sampling was carried out by WSP during the period from July to August 2016, and in May 2017 in order to obtain representativeness of the annual variability at four stations, i.e., ES-01 (Keriens Creek, within the Marban property), ES-02 (Keriens Creek, outside the Marban property), ES-03 (De Montigny Lake) and ES-04 (tributary of De Montigny Lake). The sediments were sampled once in 2016 (same four stations as water samples; SED-01 to SED-04) and in 2021(two additional stations – SED-05 and SED-06).

The water is rather acidic, ranging from 4.94 to 7.53 pH levels, with the ES-04 station being the one with the most acidic pH (4.94 to 5.31). For this station, the pH values were found to exceed the contamination prevention criteria – water and aquatic organisms (CPC (EO)), chronic aquatic life criteria (CVAC) and the criteria and the recommendations of the Canadian Council of Ministers of the Environment (CCME) for the protection of freshwater aquatic life and long-term exposure as well.

Also, a high ammoniacal nitrogen value (5.03 mg L-1) was recorded at station ES-04 in July 2017 which did not meet the CVAC criterion. Total phosphorus also did not meet the CVAC criterion at least once for stations ES-01, ES-03, and ES-04. No exceedance of criteria for major ions was observed.

For trace metals, several criteria exceedances were observed in 2012 and 2016–2017, particularly for stations ES-03 and ES-04. However, the natural characteristics of the environment and old mining activities could explain their presence in large concentrations in surface water (aluminum, arsenic, cadmium, copper, iron, mercury, and lead).

Station ES-02 showed two exceedances for fecal coliforms (CPC(EO) criterion) while stations ES-02, ES-03, and ES-04 showed at least one exceedance for hydrocarbons (C10-C50) for the CVAC criterion.

In 2012, Golder also sampled surface water from the Keriens Creek and found exceedance of the CPC(EO) criteria in one of the three stations for mercury (Golder, 2013).

No particle size analysis was conducted by Golder in 2013. As for the particle size analysis, the sediments from stations SED-01, SED-03, and SED-04 were mainly composed, in order of importance, of silt, sand, and clay. Overall, silt made up half of the total proportion for these three stations. The SED-02 station had a different grain size with sand (close to 2/3), gravel, silt, and clay. Stations SED-05 and SED-06 were mainly composed of clay (more than half of the samples), silt, and sand.

The study conducted by Golder in 2012 showed exceedances of CRE values for all eight metals that were analyzed (arsenic, cadmium, chromium, copper, nickel, mercury, lead, and zinc). Overall, there were 23 exceedances of criteria for three samples. Only one exceedance of the concentration of frequent effects (CFE) was observed for chromium, while copper was the only trace metal that exceeded the concentration of probable effects (CPE). More than 90% of the exceedances observed were for the concentration of rate effects (CRE), threshold concentration producing an effect (TCE), and concentration of occasional effects (COE) criteria. Chromium (n = 2), copper (n = 2), and nickel (n = 3) exceed the COE criterion. At the TCE level, three other metals showed exceedances: arsenic (n = 1), cadmium (n = 3), and zinc (n = 3). Finally, arsenic (n = 1), mercury (n = 3), and lead (n = 3) exceeded the CRE criterion.

Of the parameters analyzed in 2016, values above the CRE, TCE, COE, CPE, and CFE were recorded for seven metals (arsenic, chromium, copper, nickel, mercury, lead, and zinc). In total, 54 exceedances were recorded for all criteria and metals combined in this year for the 20 samples. Cadmium was the only trace metal with no exceedances in this sampling year. Chromium was the parameter with the most exceedances (20) for all criteria combined: Nine samples exceeded the CFE limit, and one sample was above the CPE criterion for this parameter. Finally, seven exceedances were recorded for the COE, while two samples showed exceedances for the TCE, and one for the CRE. For the other metals, sample exceeded the COE for arsenic, while two samples showed exceedances for this criterion for copper, whereas ten samples exceeded this criterion for nickel. Four metals exceeded the TCE criterion: arsenic (n = 3), copper (n = 2), mercury (n = 4), and zinc (n = 2). Finally, these same metals also exceeded the criteria for CRE, in addition to lead.

For the 2021 stations, no sample exceeded the CPE or CFE criterion. A total of 37 exceedances were recorded, with values exceeding the CRE, TCE, or COE criterion for the ten samples analyzed. Chromium had the highest values with four exceedances for the COE and seven exceedances for the TCE. Copper also had one exceedance for COE and three exceedances for TCE for one sample. For cadmium (n = 7), the exceedances recorded were within the limits of the TCE. For the CRE, exceedances were detected for cadmium (n = 2), copper (n = 1), mercury (n = 6), lead (n = 1), and zinc (n = 5), while no exceedances were recorded for arsenic nor nickel.

20.1.1.5 Soil Quality Assessment

20.1.1.5.1 Natural Background of the Soil

The assessment of the natural metal background of the soil was established for each soil unit onsite with soil samples taken from trenches distributed over the study area. Samples from the surface horizon taken outside of the project area were also analysed to have the soil's quality further analysed in case of potential airborne contamination but they won't be used to determine the natural background of the soil on site. The natural background results will be used as reference and comparison purposes with results from an accidental spill or following the restoration and rehabilitation closure works.

The quality of the soils was compared to generic criteria of the MELCC's Intervention Guide – Soil Protection and Rehabilitation of Contaminated Sites (Beaulieu, 2021). Some samples showed results between criteria 'A' and 'B' for chromium, HPC10-C50, and nickel, and one sample showed results between criteria 'B' and 'C' for nickel.

Samples from the surface horizon showed results between criteria 'A' and 'B' for chromium, cadmium, HPC10-C50, total cyanide, and total sulfur. One sample showed results between criteria 'B' and 'C' for manganese. For some parameters such as sulfur or HP C10C50, the maximum concentrations measured at the surface were higher than the maximum

concentrations measured in the deeper samples. Therefore, the surface layer may have been affected by airborne contamination.

20.1.1.5.2 Land Contamination

In 2013, Golder completed an environmental study of the Kierens, Norlartic, and Marban sites (Golder, 2013). Soils/backfill, waste rock, and groundwater were sampled to establish and quantify potential contamination. Drinking water from private wells was also sampled.

Based on the work carried out, petroleum hydrocarbon contamination was confirmed for all sites. The waste rock used as backfill at all sites can be considered contaminated with chromium and nickel and locally with copper. They are not considered as acidogenic, but two out of six samples are leachable for copper with the toxicity characteristic leaching procedure (TCLP) test. However, no metal mobility was detected with synthetic precipitation leaching procedure (SPLP) (acid rain) and equilibrium extraction (neutral pH; CTEU-9) tests.

The groundwater taken from the three wells at Kierens and three wells at Norlartic show exceedances of applicable criteria for manganese, zinc, and five-day biochemical oxygen demand (BOD5). Manganese is ubiquitous in groundwater in the clay regions of Abitibi and is not considered a contaminant for the Marban property. Zinc concentrations exceed the alert threshold (two out of three samples) for both sites and the criteria for a resurgence of groundwater in surface water (one out of three samples) for the Kierens site. The exceedances are small, and the source is unknown. The exceedance for BOD5 could be related to the presence of a septic tank on the site or be caused by the many types of debris on the site.

Some private wells showed exceedance for microbiological parameters. Exceedances of alert thresholds and drinking water criteria were also noted for some metals in several wells. These high values can be related to the geological lithologies as the wells were dug in the rock.

20.1.1.6 Vegetation & Wetlands

Vegetation and wetlands inventories were carried out in 2016 and 2021. Additional inventories are planned in 2022 in the new access road area. Terrestrial environments represent 45% (1,421 ha) of the 2021 study area while an equivalent area 45% (1,431 ha) are wetland habitats. Bodies of water (196 ha) and anthropic lands (112 ha) represent 6% and 4% of the 2021 study area.

The terrestrial environments are mainly represented by mixed stands followed by deciduous and by conifer-dominant stands, represented in similar proportions, and then by non-forested vegetation (shrubs and grasslands).

Wetlands are dominated by both forested bogs and forested swamps, followed by open bogs, shrubby swamps, marshes, and ponds. The overall ecological value of the inventoried natural environments has been assessed as medium. Some have low ecological value because of their level of anthropogenic disturbance (drilling, cutting, and trails) while others, with good ecological integrity, good hydrological links, or a degree of maturity, have a high ecological value.

20.1.1.7 Terrestrial Fauna

Trapping statistics for the fur-bearing animal management unit (UGAF) 03 for the 2017–2018 and 2018–2019 seasons (MFFP, 2020) confirm the presence of 14 species. Inventories carried out in 2016 and 2021 confirmed the presence of 9 species of micromammals in the study area.

Inventories of avian fauna were carried out in 2016 and 2021. Additional inventories are being conducted in 2022. For now, a total of 84 species (28 families) were observed. Of this number, nesting was confirmed for 7 species, and was judged probable for 9 species and possible for 68. By combining the observations from 2016 and the EPOQ database, the list of species using the study area or its surroundings during the spring migration, nesting, fall, and winter migration periods shows the presence of 161 species (42 families). Even though no specific inventory effort was made for birds of prey, 9 species were observed during the 2016, 2021, and 2022 inventories.

Bat inventories were carried out in 2016 and 2021. Additional inventories are being conducted in 2022. The data from 2016 confirmed the presence of 6 bats species. The 2016 and 2022 searches have revealed no anthropogenic or naturel sites of potential hibernaculum for bats within the study area. The 2022 data from the bat maternity inventory have not yet been analysed so no sites have been confirmed.

Extensive herpetofauna have been carried out in 2017 and 2021. An additional turtle inventory is being conducted in 2022. A total of eight species have been confirmed from which five are frogs, two are snakes, and one is a turtle. No salamander species have been detected.

20.1.1.8 Fish & Fish Habitat

Fish habitat description was carried out for De Montigny Lake, Keriens Creek,19 water courses, and 8 ponds. Fishing was carried out in 2016 and 2021 to verify the presence of fish, including the De Montigny Lake. De Montigny Lake and Keriens Creek both present a well-diversified fish population compared to other water bodies in the study area. Overall, 16 water courses and 4 ponds were confirmed to be fish habitat, whereas 2 water courses and 4 ponds were considered as potential habitat (no fish were captured, but the conditions observed on those sites suggested that fish could live there or they had a direct link with a known fish habitat). Only 2 water courses and 2 water course segments were not considered as fish habitat, based on the Oceans and Fisheries Canada (DFO) most recent definition: "Fish habitat is defined to include all waters frequented by fish and any other areas upon which fish depend directly or indirectly to carry out their life processes. The types of areas that can directly or indirectly support life processes include but are not limited to: spawning grounds and nursery, rearing, food supply and migration areas" (DFO, 2019).

20.1.1.9 Species at Risk

A special status floristic species, the ostrich fern (*Matteuccia struthiopteris var. pensylvanica*), was observed within the study area. This species is designated as vulnerable to harvest.

Seven at-risk avian species were observed in the study area: bald eagle (*Haliaeetus leucocephalus*), evening grosbeak (*Coccothraustes vespertinus*), rusty blackbird (*Euphagus carolinus*), bank swallow (*Riparia riparia*), barn swallow (*Hirundo rustica*), common nighthawk (*Chordeiles minor*) and Canada warbler (*Wilsonia canadensis*). The other at-risk species that could potentially be present, as preferential habitats are found in the study area are the whip-poor-will (*Caprimulgus vociferous*), the bobolink (*Dolichonyx oryzivoru*), the short-eared owl (*Asio flammeus*), the olive-sided flycatcher (*Contopus cooperi*), the eastern wood-pewee (*Contopus virens*), the wood trush (*Hylocichla mustelina*), and the eastern meadowlark (*Sturnella magna*).

In 2015, the southern bog lemming (*Synaptomys cooperi*) was captured in the Malartic area (WSP, 2015). This species could potentially be present as preferential habitats are found in the study area (wetlands).

All but one of the bat species confirmed in 2016 have a precarious status, either at the provincial or federal level. The Hoary bat (*Lasiurus cinereus*), the Red bat (*Lasiurus borealis*) and the Silver-haired bat (*Lasionycteris noctivagans*) are susceptible of being designated threatened or vulnerable in Quebec. The Little brown bat (*Myotis lucifugus*) and the Northern bat (*Myotis septentrionalis*) have been listed as endangered in Canada since 2014.

The presence of the Snapping turtle (*Chelydra serpentina*) has been confirmed in the study area. This species has been listed as of special concern in Canada since 2011. Nesting turtle signs have also been reported. The home range of the Smooth Greensnake (*Opheodrys vernalis*) does not cover the study area. However, some sightings have been made in a 25–30 km range from the study area according to the Ministry of Forestry, Wildlife, and Parks (MFFP). This snake has not been observed during field work. It is susceptible to being designated as threatened or vulnerable in Quebec. Two species of salamanders have been targeted because of the habitat availability even though their home range doesn't cover the study area. The Four-toed salamander (*Hemidactylium scutatum*) and the Northern dusky salamander (*Desmognathus fuscus*) are susceptible of being designated threatened or vulnerable in Quebec. As mentioned in the Terrestrial Fauna section, neither of those two species has been found.

The Marban Engineering Project is located in the critical habitat of the woodland caribou of Val-d'Or, as identified in the federal Recovery Strategy for the Woodland Caribou (*Rangifer tarandus caribou*), Boreal Population, in Canada. However, the provincial protected areas for the Woodland caribou do not cover the Marban site.

20.1.1.10 Ambient Air Quality

Air sampling was carried out over a period of 5 months (July to November 2016) at two stations, one located downstream of the prevailing winds (North station) and the second upstream of the prevailing winds (South station). A total of 31 samples at the North station and 29 samples at the South station were analyzed for total particles and metals.

For the North station, the total particle concentration ranged from 2–21 µg/m³ with an average of 10 µg/m³. For the South station, total particle concentration ranged from 4–28 µg/m³ with an average of 13 µg/m³. In both cases, the measured concentrations are lower than the standard limit of 120 µg/m³ and initial concentration of 90 µg/m³ set by the Clean Air Regulation.

For metals, the concentrations measured were below the applicable standards, ranging from 0.12%–45.4% of the standard limits for the North station and ranging from 0.01%–0.7% of the standard limits for the South station.

A new long-term sampling campaign (12 months) will begin in 2022.

20.1.1.11 Ambient Noise & Vibration Levels

20.1.1.11.1 Ambient Noise

Sound measurements were carried out in 2016 near residences located on Gervais and Lac-Vassan roads (three stations), as well as within the study area (one station), as well in 2021 along the Road 117 in order to determine the ambient noise before the mining project and to determine the noise criteria for each sensitive area (residences) according to the applicable regulations.

For most stations (four out of five), the measured residual noise sound levels are lower than those provided for in Instruction Note 98-01 (ranging from 45 to 55 A-weighted decibels (dBA) during the daytime and from 40 to 50 dBA at night). The highest records were 47 dBA during the daytime and 39.9 dBA at night.

20.1.1.11.2 Vibration Levels

Vibration measurements were also carried out in 2016 and located approximately in the same places as the sound measurement stations, as well as for the station of 2021. At all stations, the measured residual vibration levels are low with average values recorded between 0.045 mm/sec. at 0.164 mm/sec.

20.1.2 Environmental Issues

20.1.2.1 Contaminated Land

A residual materials landfill, listed in the La Vallée-de-l'Or regional county municipality (RCM) land-use planning and development plan (MRCVO, revised 2019), is located within the proposed layout. The landfill may present certain environmental and public health risks. Runoff from these sites facilitates the migration of contaminants, while percolation through the soil acts in the same way on groundwater. In addition, a landfill can give off gases and vapours. Initial investigations have begun in collaboration with the Université du Québec en Abitibi-Témiscamingue (UQAT).

Activities designated under the Land Protection and Rehabilitation Regulation were carried out at three old mines sites located within the proposed layout, namely the Kierens site, Norlartic site, and Marban site. According to O3 Mining, the sites were cleaned from the waste materials, but leachable waste rock backfill and contaminated soil might still be present.

Some waste materials and potentially hazardous materials were found on private property located within the proposed layout, specifically garbage, reservoirs, cars, and other types of waste (WSP, 2018c).

As those properties and lands will have to be acquired to carry out the project, O3 Mining will have to evaluate the need for additional soil characterization on potentially contaminated lands to evaluate the environmental liability. Phase 2 environmental studies are underway.

20.1.2.2 Wetlands

A preliminary evaluation estimates that 226 ha of wetlands and 31 ha of shorelines will be directly impacted by the project.

Under the Regulation that respects compensation for adverse effects on wetlands and bodies of water, O3 Mining will have to compensate for the loss of wetlands. In accordance with the second paragraph of Section 46.0.5 of the external quality assessment (EQA), all or part of the payment of the financial contribution can be replaced by work carried out to restore or create wetlands or bodies of water for mining mineral substances work within the meaning of Section 1 of the Mining Act. A compensation plan will have to be submitted to the MELCC for approval.

20.1.2.3 Fish & Fish Habitat

As the diversion of the Keriens Creek will be required to mine the Norlartic and Kierens pits, the project will cause a harmful alteration, disruption, or destruction of fish habitat as defined under the Fisheries Act (i.e., from the placement of watercourse crossings or intake/outfall structures). An application for authorization under Section 35 (2) (b) of the Fisheries Act will have to be submitted to DFO. If required, O3 Mining will work with DFO on the details of the offset requirements.

According to available data and preliminary assumptions, the waste rock storage area is located on one intermittent water course that is a confirmed fish habitat (CE-06). The encroachment of this water course by the waste rock storage area should be considered as a risk because an amendment to the Metal and Diamond Mining Effluent Regulations (MDMER) will be required. Using a natural water body frequented by fish for mine waste disposal (tailings, waste rock and/or overburden) requires an amendment to the MDMER (which is a federal legislative action), addition of the water body to Schedule 2 of the MDMER, and an authorization under Section 36 of the Fisheries Act. As a result, the proponent must also prepare an assessment of mine waste disposal alternatives for consideration (according to federal guidelines) and prepare a fish habitat compensation plan for consideration as part of the EIA.

O3 Mining has confirmed with DFO that in-pit storage of mine waste after the diversion is not considered as disposal on fish habitat.

20.1.2.4 Air Quality

Open mining operations are likely to produce particles (total and fine) and gas emissions, which can cause air quality deterioration.

Moreover, due to habitation proximity to the project's location, dust and gas (nitrogen dioxide) emissions could become health concerns by the local community.

Air modelling will be required to evaluate the potential effects, to identify optimization measures, and if required, to elaborate on mitigation, management, and monitoring measures.

20.1.2.5 Noise & Vibration Levels

Due to habitations proximity to the project's location, noise and vibration from blasting could cause nuisances and damage to infrastructure. Noise and vibration modelling will be required to evaluate the potential effects, to identify optimization measures, and if required, to elaborate on mitigation, management, and monitoring measures.

20.1.2.6 Hydrogeological Conditions

As defined in Section 20.1.1.2, the modelling results show that water table drawdown at the end of the operation is almost zero at about 3 kilometres from the pit in the surface deposits. For the rock aquifer, the radius of influence of dewatering extends nearly 4 kilometres. The drawdown will possibly have an impact on residential wells located to the south. Additional investigation will be completed in order to determine the actual condition of the residential wells which may be impacted by the dewatering. An additional survey of the esker located near Montigny Lake will be completed in autumn 2022. Eskers are usually a good source of water and residential wells may be installed in this aquifer, so it is important to have a better understanding of its extension and the hydrogeological condition.

20.2 Ore, Waste Rock, Tailings & Water Management Requirements

The following sections describe the environmental requirements for storage management facilities for mining materials based on available information. Directive 019 is the main guideline for ore, waste rock, tailings, and environmental management requirements for water. The geochemical assessment results were interpreted according to the new guideline for mine waste and ore characterization (MELCC, 2020).

20.2.1 Geochemical Assessment

A geochemical assessment is required for mining materials in order to define their geochemical characteristics and classify them. This classification serves to determine the design parameters of the storage areas to ensure groundwater protection according to Directive 019, as well as closure requirements to prevent and/or manage acid rock drainage (ARD) or contaminated neutral drainage (CND).

Geochemical characterization of waste material that will be exposed and handled during mining is essential to formulating a mine waste management plan. In addition, the classification of mine waste is required by the MELCC for eventual licensing and to provide environmental design parameters for the construction of mine waste management facilities, including waste rock storage piles, tailing impoundments and sludge ponds, if any.

The geochemical characterization is consistent with the recommendations of the following documents:

- Directive 019 on Mining Industry" (Directive 019 *sur l'industries minière*; Québec, 2012);
- Mine Tailings and Ore Characterization Guide (MELCC, June 2020);
- Prediction Manual for Drainage Chemistry from Sulphidic Geologic Materials (MEND, 2009); and
- "Global Acid Rock Drainage Guide" (GARD, 2009).

The objective of this geochemical characterization program was to: 1) classify mine waste according to Directive 019 and provide environmental design input parameters for the design of mine waste facilities; and 2) determine constituents of environmental interest in the context of future mine contact water quality management.

Short-term static testing methods were used to assess the chemical composition of mine waste, the potential for the generation of acid rock drainage (ARD), the potential for metal leaching (ML) to the receiving environment upon exposure to ambient conditions and thus, the risk associated with the waste material and subsequent mine waste management practices according to Directive 019.

Results of the static testing program are also meant to guide mine contact water management practices for the proposed mining operation. The long-term weathering characteristics of waste will have been evaluated through a kinetic weathering test.

Preliminary testing had been done on ore, waste, and tailings from 2018 to 2021. The following sections discuss the static and kinetic results obtained during the geochemistry program.

Following the results of the geochemical characterization of ore (WSP, 2020), in order to refine the conclusions on the potential for acid generation and leaching of ore and waste rock over the long term, in 2021 kinetic tests in columns were carried out.

Kinetic tests make it possible to assess the propensity of geological material to achieve the geochemical potential defined by static tests. These tests aim to simulate weathering under accelerated conditions in order to verify their respective acidogenic potential and the quality of their drainage in the short, medium, or long term, as well as to assess the potential for the generation of any contaminated neutral drainage.

Kinetic tests were completed by WSP on two composite samples, one representing ore and one representing waste rock. The experimental program developed by the Mineral Technology Research and Service Unit (URSTM) of the University du Quebec en Abitibi-Témiscamingue (UQAT), first includes chemical characterizations, static tests, and leaching tests on the two types of materials (waste rock and ore), and the geochemical quality of the rinsing water from the columns was tracked over time.

According to that reported in the drilling reports that were available and to information collected from the project geologists, two main lithologies were targeted for the geochemical characterization of the ore and waste rock of the Marban project: mafic volcanic rocks (Basalt) and ultramafic volcanic rocks (komatite).

The columns were kept under unsaturated conditions most of the time. The column tests were rinsed weekly with 2 L of deionized water. The test period was optimized based on the results of the tests carried out on the ore and the waste rock, previously in the project. Analytical leachate results were collected systematically for 25 weeks (6 months), between July and December of 2021.

20.2.1.1 Static Testing

20.2.1.1.1 Ore

In 2020, 10 ore samples were selected within the two main geological units (5 samples from each unit) based on recent mineral resources information and available drilling and geochemical data. The material was tested for ARD potential, total metals content, and leachability potential (static tests) (WSP, 2020).

Acid Rock Drainage (ARD) Classification

All 10 ore samples were subjected to the Modified acid-base accounting (MABA) static test. All the samples had total S concentrations greater than 0.04%, but less than 5%.

In addition, all the samples had net neutralizing power (PNN) values greater than 20 kg $CaCO_3$/t, and PN/PA ratios (neutralization potential over acid potential) greater than 2, which means that all the ore samples analyzed are considered not potentially acid generating (NPAG) according to the new Characterization Guide (*Guide de caractérisation des résidus miniers et du minerai* (Guide); MELCC, July 2020).

Chemical Composition: Trace Metal Content and Classification of Risk

For classifying risk and aquifer protection measures in mine waste management facility design, trace element concentrations were compared to Soil Criteria A for the Geological Province "*Secteur Supérieur*" (Beaulieu, 2021).

All 10 ore samples were analyzed to determine their available metal content. All the samples presented results in available metals higher than the "A" criteria of the Intervention Guide for at least one parameter, i.e., chromium (70%), cobalt (20%), copper (40%), manganese (20%) and nickel (100%). Due to these exceedances, the degree of leachability of these samples had to be evaluated using leaching tests.

Metal Leaching Potential: Leachability Classification for the Determination of Environmental Risk (TCLP)

A sample is classified as leachable if for a specific parameter both 1) the solid chemistry concentration exceeds Québec Soil Criteria A, and 2) the TCLP leachate concentration exceeds the RES criteria. Following this guidance, the leachate was analyzed only for metals that exceeded the generic "A" criteria of the Intervention Guide, in addition to certain parameters listed in Table 1 of Annex II of Directive 019.

All of the samples exceeded the RES criteria for copper and manganese. In addition, a sample from the Marban deposit exceeded the RES criterion for nickel. Thus, the samples are considered leachable in the TCLP test for copper, manganese and nickel since the generic "A" criteria were also exceeded for these same parameters in the solid samples. There was no exceedance of the parameters listed in Table 1 of Annex II of Directive 019 and the material is not classified as "high risk."

Constituents of Environmental Interest in Leachate: Synthetic Precipitation Leaching Procedure (SPLP) and Equilibrium Extraction Water Leach Test (CTEU-9)

Results were compared to Directive 019 Effluent Criteria and to Québec RES criteria.

No exceedance of the RES criteria was observed during this test, and the ore would therefore be considered non-leachable in the SPLP test. There was no exceedance of the parameters listed in Table 1 of Annex II of Directive 019 and the material is not classified as "high risk."

Exceedances of the RES criterion for copper were noted in two samples (20%), specifically two samples from the Norlartic deposit. No other exceedances of the RES criteria were observed during this test. Thus, the samples are considered copper leachable in the CTEU-9 test, since generic "A" criteria were also exceeded for this metal in the solid samples. There was no exceedance of the parameters listed in Table 1 of Annex II of Directive 019 and the material is not classified as "high risk."

When mine materials are considered as non-PGA, only SPLP and CTEU-9 tests must be performed (MELCC, 2020). However, since the ARD potential is based on a few samples and wasn't confirmed by kinetic testing, a TCLP test was also performed. About 20% of the ore samples show copper exceedance for the RES criteria under the CTEU-9 tests. All samples show copper and manganese exceedance of the groundwater criteria, as well as 20% for nickel under the TCLP test. Therefore, the ore must be considered as leachable. None of the ore samples are considered as high risk. However, as recommended by the Characterization Guide, kinetic testing must be performed to confirm ore leaching potential.

20.2.1.1.2 Waste Rock

WSP Investigation

In 2018, waste rock from the Marban pit was characterized based on the geological and operation information provided in 2016–2017 (WSP, 2018). Four geological units were tested: V4, I2, V3B, and I1C.

In June 2020, the MELCC released a new guideline for mine waste and ore characterization. This new guideline aims to standardize the characterization process by suggesting guidelines for the testing, interpretation, and classification of mine materials. The waste rock classification was reviewed as per the new guideline (WSP, 2020).

In 2018, 62 samples from the four main geological units that will make up the waste rock were submitted for analysis to briefly assess their geochemical behaviour.

Acid Rock Drainage (ARD) Classification

MABA testing was performed on 22 samples. The results indicated that the waste rock is NPAG, except for one sample from the V3B geological unit. Kinetic tests could confirm the ARD potential, as recommended by the new guideline.

Chemical Composition: Trace Metal Content and Classification of Risk

All of the 62 selected samples were submitted for analysis to assess the available metal concentrations.

The results of the analyzes show that 11 of the 62 samples (18%) present values below the MELCC "A" criteria. Of these 11 samples, 5 (46%) are composed of waste rock from geological unit I1C, 3 (27%) from geological unit I2 and 3 (27%) from geological unit V3B. These 11 samples are therefore considered to be "low risk" and non-leachable waste rock.

All the other samples (including 100% of the samples from geological unit V4) submitted for analysis, specifically 82% of the samples analyzed, showed available metal concentrations above the generic criterion "A" for at least one of the following parameters: Ag, As, Ba, Co, Cr, Cu, Mn, Ni, and Se. The risk level of these waste rock samples (high or low) and their degree of leachability were therefore assessed using the results of TCLP leaching tests carried out on all the samples showing exceedances of the "A" criteria. of the MELCC.

Metal Leaching Potential: Leachability Classification for the Determination of Environmental Risk (TCLP)

The leachate analyzes of the 53 waste rock samples submitted for the TCLP test showed concentrations below the maximum limits in Table 1 of Annex II of Directive 019. Waste rock is therefore not considered high risk according to Directive 019.

Since the results of metals content testing exceed generic criteria 'A' of the MELCC for one or more extractible metals, a TLCP test was performed on 54 out of 62 samples. The results of the TLCP test show that all geological units are leachable except for the I1C geological unit.

Constituents of Environmental Interest in Leachate: Synthetic Precipitation Leaching Procedure (SPLP) and Equilibrium Extraction Water Leach Test (CTEU-9)

A SPLP test was then performed on the 14 samples that showed higher metal mobility under the TLCP test and no exceedance of the groundwater criteria was noted. Based on those results, none of the waste rock sample are considered as high-risk. The 14 samples indicated metal concentrations below the RES criterion for all metals analyzed.

During the 2018 characterization, no CTEU-9 test was carried out, as this analysis was not required for the classification of materials at that time. Waste rock samples should therefore be subjected to this test in order to validate their leaching potential at neutral pH.

Based on static tests performed by WSP and using the Guide criterion, the waste rock from the Marban pit is considered as leachable. However, if it is confirmed in further investigations that the waste rock is NPAG, only SPLP and CTEU9 testing results must be considered. As there was no CTEU-9 testing performed, it may be performed during further investigation. Leaching potential can be assessed by kinetic testing.

2012 Investigation

In 2012, Golder was mandated to carry out an environmental characterization study of the land affected by these former mining operations. The work carried out was aimed to confirm, in a preliminary way, the presence of contamination and/or environmental impacts on the three sectors of the Marban property.

As part of this environmental study of the sites of the former Marban mines, among others, the geochemical characterization of the mining waste rock was carried out. The following analyzes were carried out: metal analyses, determination of acid generating potential (AGP) and leaching tests.

In the Marban site sector, four trenches were excavated in an area containing mining waste rock; a composite sample was taken from the four trenches and was subjected to static analyses and tests.

The results of the analyses of extractable metals in the composite sample of waste rock from the Marban site show that criteria A and B were exceeded for the following parameters: As (>A), Co (>A), Cr (>B), Cu (>B), Mn (>B), Ni (>B).

In this study, to decide on the acid generation potential of waste rock, the D019 criteria were used. According to the results obtained, the tailings are not potentially acid generating since the sulfur concentration measured in all the samples is less than 0.3%.

To conclude, based on the D019 classification in this study, waste rock from the Marban site is NPAG. The tested sample is copper leachable based on the TCLP test. The SPLP and CTEU-9 assays did not demonstrate metal mobility. Given their chromium, copper, manganese and nickel content, Marban waste rock is considered contaminated soil/backfill, since it was used as backfill.

20.2.1.2 Kinetic Test Results

20.2.1.2.1 Acid generation potential (PAG)

According to the results obtained, the ore and waste rock are slightly sulphide, with sulphide contents of 0.434 and 0.106% respectively, which come mainly from pyrite. Regarding total sulfur, the values for both materials do not exceed 5%, but are above 0.04%.

The calculated acidification potentials (AP) are therefore 13.6 kg $CaCO_3$ for the ore and 3.3 kg $CaCO_3$ for the waste rock. The neutralization potentials (PN) measured are relatively high, with values of 114 kg $CaCO_3$ and 99 kg $CaCO_3$, respectively, for the ore and for the waste rock.

The interpretation of the static tests according to D019 as well as according to the Characterization Guide classifies the two materials as non-acidogenic due to their PNN values which are greater than 20 kg $CaCO_3$/tonne of two hammers and the PN/PA ratio, which is also greater than 2 for ore and also for waste rock.

20.2.1.2.2 Metal Leaching Potential (TCLP)

All the leachates produced during the TCLP tests have concentrations of constituents below the limits indicated in Table 1 of Annex 2 of D019 and Annex A (Classification criteria for high-risk mining materials) of the Characterization Guide. Therefore, samples of waste rock and ore in pre- testing are not considered high risk.

The two excesses of the applicable criteria were noted in the ore: for cadmium (exceedance of drinking water criterion [EC]) and for manganese with the value that exceeds the criteria of drinking water (EC) and surface water resurgence (RES).

20.2.1.2.3 Results of kinetic column tests

The results of kinetic tests performed on the composite samples of ore and waste rock indicate that there is no acidification over an accelerated weathering period in the six-month test period.

Thus, for these samples, the reactivity of sulphide minerals is low, and the neutralizing capacity is high and sustained in the short term.

20.2.1.2.4 Tailings

Following the metallurgical tests carried out by Ausenco, the samples of the mining residues were sent in the spring of 2022 to the URSTM in Rouyn-Noranda, to carry out the kinetic tests in columns on the residues.

Some of results from these trials is available. Details of the tailings kinetic tests with full results of these tests will be presented at the end of the tests in 2023.

20.2.2 Ore & Waste Rock Management

20.2.2.1 Ore Storage

When enriched ore or concentrate is leachable, acid generating, or high-risk, the storage, loading, and unloading must be carried out under cover and on a waterproof surface equipped with a water recovery system.

If, for technical reasons, the storage of enriched ore or concentrate cannot be done under cover, adapted measures must be taken according to the characteristics of these materials to ensure adequate protection of surface water or groundwater, in particular through the collection and treatment of contaminated water. Adequate measures to protect the ore storage area, enriched ore, or concentrate against wind erosion must also be put in place.

20.2.2.2 Waste Rock Management

While the majority of samples are non-PAG, since most of them are metal leaching, Level A groundwater protection measures will have to be applied, meaning that for now, without further testing (kinetic tests), the waste rock pile will have to be designed to respect a daily percolation rate < 3.3 L/m^2 at the bottom of the pile.

The hydrogeological model showed that the waste rock stockpile design proposed in Section 18 respects the daily percolation rate criteria of Directive 019 without adding clay or impermeable layer at the bottom of the pile. A model to establish that the measures proposed are sufficient to prevent groundwater quality degradation will also have to be developed.

20.2.3 Tailings Management

A surface tailings storage facility is required to store tailings produced by the ore processing plant. Based on characterization results for the ore samples, it is anticipated that tailings will be non-PAG but will be metal leaching. Therefore, the TSF will have to be designed to respect a daily percolation rate < 3.3 L/m^2 at the bottom of the storage area.

The hydrogeological model showed that tailings storage facility design proposed in Section 18 respects the daily percolation rate criteria of directive 019 only if a clay layer is added at the bottom of the TSF.

20.2.4 Water Management

The waters to be managed are mine water from the pits, contaminated neutral drainage (CND) from the mine waste disposal and ore storage areas, and contaminated runoff water from the ore processing plant area. This contaminated water will have to be collected by ditches and ponds (which may have to be lined to limit water infiltration into the ground) and will be released into the environment when its quality meets compliance with the requirements of Directive 019 and MDMER. Water management infrastructures may have to be lined to limit water infiltration into the ground. Additional environmental discharge objectives (OER) criteria could be added to the previous ones. Those OER criteria would be defined by the MELCC during the permitting process.

20.2.5 Site Monitoring

The objective of the environmental monitoring program is to detect and document any changes in the environment in relation to the baseline (whether or not related to the project), to verify the impact assessment and to evaluate the effectiveness of mitigation or compensation measures proposed in the impact assessment. As part of the project, an environmental monitoring program will be implemented. The main components of the environmental site monitoring program are as follows:

- Effluents Quality Monitoring (Directive 019 and MDMER);
- Groundwater Quality and Piezometric Level (Directive 019);

- Water Quality Monitoring Studies (MDMER);
- Biological Monitoring Studies (MDMER); and#
- Mitigation Measures Monitoring (air quality, noise, vibration, runoff, etc.).

Since residences are located more than 1 km from the project area (see Section 20.4.3.1.2 for information in regard to residences within the immediate vicinity of the project that may be acquired), O3 Mining will have to respect maximum frequency-dependent ground vibration levels (varying between 12,7 mm/s and 50 mm/s for frequencies ranging from <15 Hz to >40 Hz), in addition to respecting a maximum threshold of the air pressures to any residence of 128 linear decibels (Directive 019)..

Additional monitoring programs could be required as condition of a Government authorization.

20.3 Regulatory Context

The regulatory context described in the following sections is based on regulations and acts in force at the time of the preparation of this PFS.

20.3.1 Environmental Impact Assessment Process

20.3.1.1 Provincial Authorities

With its average extraction rate of16,438 t/d and a process plant with a capacity of 16,438 t/d, the proposed mining project is listed in Paragraph 22 and 23 of Part II of Schedule I of the Regulation respecting the environmental impact assessment and review of certain projects:

- 22 (2) the establishment of a mine whose maximum daily capacity for extracting any other metal ore is equal to or greater than 2,000 metric tons.
- 23 (1c) the construction of a treatment plant for any other metal ore whose maximum daily treatment capacity is equal to or greater than 2,000 metric tons.

The projects listed in Schedule I are subject to the EIA and review procedure provided for in Subdivision 4 of Division II of Chapter IV of title I of the environmental quality act (EQA), to the extent provided therein, and must obtain authorization from the Government.

The MELCC oversees the EIA process and relies on the Bureau d'audience publique sur l'environnement (BAPE)(Environmental Public Hearing Office) of Quebec for public hearings. The phases of the provincial EIA process are:

1. Project Notification: The Proponent must file written notice with the Minister, in accordance with Section 31.2 of the EQA.
2. Evaluation: Within 15 days after receiving project notification, the Minister must send a directive to the proponent specifying the nature, scope, and extent of the EIA statement that the proponent must prepare. The EIA statement shall also comply with Directive 019 which includes the guidelines for all mining projects.
3. EIA Statement Preparation: The proponent conducts the studies and assessment according to guideline requirements and submits 12 paper copies as well as 1 electronic version of the EIA statement to the Minister.
4. Compliance Verification: Verification of the EIA's compliance by the MELCC. During this analysis, the MELCC can forward questions or comments to the proponent to obtain additional information and/or clarifications.
5. Ministerial Analysis and Public Participation: The EIA is made public, allowing the population to request public hearings organized by the BAPE. Following the public hearings, the BAPE must send its recommendations to the

Minister. In parallel, the MELCC prepares an environmental analysis report in order to advise the Minister with respect to the environmental acceptability of the project.

6. Decision-making: Based on the conclusions of both the MELCC and BAPE recommendations, the Minister makes his recommendations to the Government. The Government then authorizes the project (with or without modifications and conditions) by decree or rejects it.

The release of the provincial decree does not affect or restrict the application of the EQA. It is the responsibility of the proponent to verify with the MELCC and any other municipal or government entity whether additional authorizations are required to carry out the mining operations (see Section 20.3.2).

To initiate the EIA and review process, the project must be at a feasibility stage. The time frame from project notification to Government decision ranges from 2 to 3 years.

20.3.1.2 Federal Authorities

In August 2019, the new Impact Assessment Act (IAA) came into force, along with a new set of regulations. The IAA replaces the Canadian Environmental Assessment Act, 2012 (CEAA 2012). The IAA continues the approach taken under the 2012 CEAA to designate projects by type and thresholds prescribed by regulation.

The provisions in the schedule for the Physical Activities Regulations describing the project, in whole or in part are the following:

- 18(c) The construction, operation, decommissioning, and abandonment of a new metal mine, other than a rare earth element mine, placer mine or uranium mine, with an ore production capacity of 5,000 tonnes per day or more.
- 18(d) A new metal mill, other than a uranium mill, with an ore input capacity of 5,000 t/day or more.

With its average extraction rate of 16,438 t/d and a process plant with a capacity of 16,438 t/d, it has been confirmed by the Impact Assessment Agency of Canada (Agency) that the project will be subject to the IAA and a federal impact assessment will be required.

The new federal impact assessment process is now divided into five phases:

1. Planning: The Proponent submits an Initial Project Description to the Agency that meets the requirements of the Information and Management of Time Limits Regulations. When the Initial Project Description conforms with the regulations, the Agency posts it on the Agency's Registry and the 180-day time limit starts. The Agency then contacts and consults with federal authorities who may be in possession of specialist or expert information or knowledge and engages with provincial, territorial, and Indigenous jurisdictions that may have responsibilities in relation to the assessment of the designated project. The Agency prepares a Summary of Issues that includes issues raised by provincial, territorial, and Indigenous jurisdictions, Indigenous groups, the public, federal authorities and other participants during consultations and engagement. The proponent prepares a Response to the Summary of Issues that explains how it intends to address the issues raised, and a Detailed Project Description that meets the requirements of the Information and Management of Time Limits Regulations. The Agency reviews the Detailed Project Description and determines if an impact assessment is required. If an impact assessment is required, the Agency posts the draft Tailored Impact Statement Guidelines on the Registry for comment and posts the draft Plans.
2. Impact Statement: The Proponent develops an Impact Statement containing the information and studies outlined in the Tailored Impact Statement Guidelines and submits it to the Agency. The Agency must be satisfied that the proponent has provided the required information or studies outlined in the Tailored Impact Statement Guidelines within three years from the date the Notice of Commencement is posted on the Agency's Registry. Upon request from the Proponent, the Agency may extend the timeline. If the Impact Statement does not conform to the Tailored Impact Statement Guidelines, the Agency requires the Proponent to provide the missing information or revisions.

3. Impact Assessment: Once the Agency is satisfied with the Impact Statement, the time limit of up to 300 days begins. The Agency then continues to engage and consults with federal authorities, provincial, territorial, and Indigenous jurisdictions. The Agency then provides an Impact Assessment Report, a Consultation Report and recommended potential conditions to the Minister.
4. Decision-making: Based on the Impact Assessment Report, the Minister must determine if the adverse effects within federal jurisdiction and the adverse direct or incidental effects are in the public interest, or refer the determination to the Governor in Council. Once the determination is made by the decision-maker (Minister or Governor in Council), the Minister issues a Decision Statement with the reasons for the determination and conditions. When the Minister makes the public interest determination, the Decision Statement must be issued no later than 30 days after the Impact Assessment Report is posted on the Registry. When the Governor in Council makes the public interest determination, the Decision Statement must be issued no later than 90 days after the Impact Assessment Report is posted on the Registry.
5. Post Decision: The Minister issues the Decision Statement including any enforceable conditions with which the Proponent must comply and the final description of the designated project. Conditions must include the implementation of a follow-up program and mitigation measures.

20.3.2 Permitting Requirements

Throughout all stages of the project, activities conducted by O3 Mining will be required to comply with provincial and federal acts and regulations.

Table 20-1 and Table 20-2 present the most significant acts, regulations, directives, and guidelines with which the project could have to comply with. This list is non-exhaustive and is based on information known so far. Their applicability will have to be reviewed as project components are defined.

Following release from the provincial decree and federal authorization (EIA approval), the project will require several approvals, permits, and authorizations to initiate the construction phase up to the closure phase. In addition, O3 Mining will be required to comply with any other terms and conditions associated with the decree and authorization issued by the provincial and federal authorities.

Table 20-3 presents a non-exhaustive list of required approvals, authorizations, permits, or licences based on the known components of the Marban Engineering Project and typical activities related to mining projects.

O3 Mining is preparing to initiate the permitting process for all identified permitting requirements.

Table 20-3

Table 20-1: Main Provincial Acts, Regulations, & Guidelines Applicable for Mining Activities

Provincial Jurisdiction
Mining Act (c. M-13.1)
Regulation respecting mineral substances other than petroleum, natural gas, and brine (M-13.1, r. 2)
Environmental Quality Act (c. Q-2)
Regulation respecting the regulatory scheme applying to activities on the basis of their environmental impact (Q-2, r.17.1) Regulation respecting activities in wetlands, bodies of water and sensitive areas (Q-2, r.01) Clean Air Regulation (Q-2, r. 4.1) Regulation respecting industrial depollution attestations (Q-2, r. 5) Regulation respecting pits and quarries (Q-2, r. 7.1) Regulation respecting compensation for adverse effects on wetlands and bodies of water (Q-2, r. 9.1); 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 the burial of contaminated soils (Q-2, r. 18); Regulation respecting wastewater disposal systems for isolated dwellings (Q-2, r. 22) Regulation respecting halocarbons (Q-2, r. 29) Regulation respecting hazardous materials (Q-2, r. 32) 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 quality of drinking water (Q-2, r. 40) Regulation respecting the charges payable for the use of water (Q-2, r. 42.1)
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)
Watercourses Act (c. R-13)
Regulation respecting the water property in the domain of the State (R-13, r. 1)
Dam Safety Act (c. S-3.1.01)
Dam Safety Regulation (c. S-3.1.01, r. 1)
Sustainable Forest Development Act (c. A-18.1)
Regulation respecting the sustainable development of forests in the domain of the State (A-18.1, r. 0.01)
Conservation and Development of Wildlife Act (c. C-61.1)
Regulation respecting wildlife habitats (C-61.1, r. 18)

Provincial Jurisdiction
Lands in the Domain of the State Act (c. T-8.1)
Building Act (c. B-1.1)
Construction Code (B-1.1, r. 2) Safety Code (B-1.1, r. 3)
Explosives Act (c. E-22)
Regulation under the Act respecting explosives (E-22, r. 1)
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)
Highway Safety Code (c. C-24.2)
Transportation of Dangerous Substances Regulation (C-24.2, r. 43)
Directives and Guidelines
Mining Industry Directive 019 (Directive 019 sur l'industrie minière) (2012) Guidelines for the Valorization of Mine Tailings (Lignes directrices relatives à la valorisation des résidus miniers) (2015) Guidelines for Preparing Mine Closure Plans in Québec (2017) Intervention Guide – Soil Protectin and Rehabilitation of Contaminated Sites (Guide d'intervention – Protection des sols et réhabilitation des terrains contaminés) (2021) Mine Tailings and Ore Characterization Guide (Guide de caractérisation des résidus miniers et du minerai) (2020)

Table 20-2: Main Federal Acts, Regulations, & Guidelines Applicable for Mining Activities

Federal Jurisdiction
Fisheries Act (R.S.C., 1985, c. F-14)
Metal and Diamond Mining Effluent Regulations (SOR/2002-222)
Canadian Environmental Protection Act (S.C. 1999, c. 33)
PCB Regulations (SOR/2008-273) Environmental Emergency Regulations (SOR/2003-307) Federal Halocarbon Regulations (SOR/2003-289) National Pollutant Release Inventory
Species at Risk Act (S.C. 2002, c. 29)
Canada 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)
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)
Directives and Guidelines
Environment Canada Environmental code of practice for metal mines (2009)
Environment Canada Environmental Code of Practice for Metal Mines (2009)
Guidelines for the Assessment of Alternatives for Mine Waste Disposal (2016)
Strategic climate change assessment (2020)

Table 20-3: Preliminary & Non-exhaustive List of Permitting Requirements

Activities	Type of Request	Authority
Rehabilitation and restoration plan	Approval	MERN
Mine waste management facilities and processing plant location	Approval	MERN
Infrastructure implantation on public land	Lease	MERN
Mining operations	Lease	MERN
Construction and operation of an industrial establishment	Authorization	MELCC
Withdrawal of water, including related work and works	Authorization	MELCC
Establishment of potable, wastewater, and mine water management and treatment facilities	Authorization	MELCC
Installation and operation of any other apparatus or equipment designed to treat water, in particular in order to prevent, abate, or stop the release of contaminants into the environment	Authorization	MELCC
Installation and operation of an apparatus or equipment designed to prevent, abate, or stop the release of contaminants into the atmosphere	Authorization	MELCC
Construction on land formerly used as a residual materials elimination site and any work to change the use of such land	Authorization	MELCC
Industrial depollution attestation	Attestation	MELCC
Work, structures, or other interventions carried out in wetlands and bodies of water	Authorization	MELCC
Carry out an activity likely to modify a wildlife habitat	Authorization	MFFP
Forest intervention licence for mining activities	Licence	
Harvest wood on public land where a mining right is exercised	Permits	MFFP
Build or improve a multi-use road	Permits	MFFP
Use of high-risk petroleum equipment	Permits	RBQ
Construction	Permits	City
Powerline relocation	Authorization	HQ
Construct, place, alter, rebuild, remove, or decommission a work in, on, over, under, through or across any navigable water	Notice	Transport Canada
Harmful alteration, disruption, or destruction of fish habitat	Authorization	DFO
High-risk petroleum equipment	Permit	RBQ
Explosives possession, magazine and transportation	Permit	SQ
Explosives manufacturing plant and magazine	Licence	MNR
Explosives transportation	Permit	MNR
Use of nuclear substances and radiation devices	Licence	CNSC
Notice and Environmental Emergency Plan	-	ECCC

20.4 Social or Community Considerations

20.4.1 Consultation Activities

O3 Mining has been proactively engaging with several stakeholders including First Nations. O3 Mining is advocating for open dialogue with concerned parties to enable the inclusion of comments and suggestions in the development of the Marban Engineering Project. O3 Mining's commitments include keeping stakeholders informed on project advancement, transparency and respect for the voicing of opinions; and listening and being receptive to questions and concerns from interested parties. Despite the challenge of reaching out to the community due to the COVID-19 pandemic, O3 Mining set up a few mechanisms to inform stakeholders of the Marban Engineering Project development including:

- A quarterly newsletter that is sent to several stakeholders on project advancement and used for community engagement;
- A specific webpage for the community where specific information can be found such as Q&A, information meeting minutes, etc.;
- Other channels of communication including an email distribution list, physical mailings, phone calls, one-on-one meetings; and
- Nominated a liaison agent who will serve as a direct contact for the community

O3 Mining held a first information meeting with the community surrounding the project to present the company and project. Due to COVID-19 restrictions regarding capacity limits, this meeting was also broadcast on Facebook Live. Specific consultation meetings were held to develop a participatory approach together with the community, and consult with them on the Gervais Road deviation. Early voluntary consultation enables project improvement by taking the host communities concerns into consideration.

The concerns voiced at these initial meetings are typical of any mine development project. As the environmental baseline advances and the technical economical studies are developed, O3 Mining will be able to address the concerns and hold additional consultation sessions and specific working groups. Some of the community's preliminary questions or concerns are as follows:

- Impact on water quantity and quality;
- Blast vibrations;
- Noise;
- Dust/air quality;
- Visual impacts (height of stockpiles, landscape, etc.);
- The distance of the new Gervais Road (will it prolong the travel time);
- Increased traffic;
- Impact on quality of life;
- Depreciation of homes;
- Delocalization of homes/businesses; and
- Impact on hunting activities.

Moreover, O3 Mining has had preliminary meetings with several government and political representatives to discuss the project and concerns.

These representatives include the following:

- City officials of the City of Val-d'Or;
- City officials of the City of Malartic;
- Representatives with the MRC of Vallée-de-l'Or;
- Ministry of the Environment and the Fight Against Climate Change (Ministère de l'Environnement et de la Lutte aux Changements Climatiques);
- Ministry of Natural Resources (Ministère de l'Énergie et des Ressources Naturelles);
- Ministry of Transportation (*Ministère des Transports*);
- Ministry of Wildlife, Forests, and Parks (Ministère de la Faune, des Forêts et des Parcs);
- Ministry of Finance (*Ministère des Finances*);
- Ministry of Labour, Employment, and Social Solidarity (Ministère du Travail, de l'Emploi et de la Solidarité sociale);
- Fisheries and Oceans Canada (DFO);
- Environment and Climate Change Canada (ECCC); and
- Impact Assessment Agency of Canada.

20.4.2 Social Components

20.4.2.1 Land Planning, Development, and Use

The project is located in the administrative region of Abitibi-Témiscamingue (08) in the RCM La Vallée de l'Or. The proposed layout spreads across the territory of two municipalities: Val-D'Or in its eastern portion and Malartic in its western portion.

Land tenure is a mix of public, private, and municipal properties. No federal land is located within the project area. No federal land will be used for to carry out the project.

According to the land use planning and development plan of the RCM La Vallée-de-l'Or (MRCVO, revised 2019), the project area intersects with two land uses: Rural in the east portion and Forest to the west. The exploitation of mineral substances is compatible with both Rural and Forest land uses, and outside the urbanized area. The nearest urban perimeter is located less than 1 km from the southern claims limits (Dubuisson district).

Sixteen permanent residences or businesses are located within 1 km from either the Norlartic pit or the Marban pit. The nearest agglomeration is the Dubuisson district located approximately 600 m from the southern claims limits. Specific residences or business requiring removal/relocation as part of the project will be identified prior to or during the feasibility study.

Forest shelters and resort leases are also located within the proposed layout or its surroundings.

Two non-exclusive leases for surface mineral substances extraction sites (borrow pits) are located within the proposed layout, both belonging to Construction Norascon Inc.

20.4.2.2 Population & Economics

The population of Val-d'Or was estimated at 33,047 people in 2021, while in 2016, it numbered 32,664 (ISQ, 2022), an increase of 1.2%. Malartic experienced a decline in its population, which was 3,381 people in 2016 and is now estimated at 3,289 people in 2021 (-2.7%) (ISQ, 2022).

Val-d'Or is the regional center of the RCM La Vallée-de-l'Or. It constitutes the regional development and services center and thus brings together the main commercial, institutional, industrial, and administrative activities of the RCM.

Malartic's influence is more limited than that of Val-d'Or. Nevertheless, Malartic offers a range of services linked to commercial and institutional functions. In addition, the municipality has both local and regional recreational tourism facilities. The mining industry is the main economic driver and primary sector.

The primary sector is important in the industrial structure in Abitibi-Temiscamingue. In 2016, it monopolized 14.5% of jobs in the region against 2.5% in the province of Québec. A large proportion of the jobs held by workers in the RCM La Vallée-de-l'Or were also linked to the primary sector (16.6%). On the other hand, jobs in the secondary and tertiary sectors were less strongly represented than in the province of Québec.

In Val-d'Or, mining (including quarrying) accounted for 13.9% of employment in 2016. Jobs related to the health care and retail sectors accounted for 13.2% each (Statistics Canada, 2017, updated in 2019). For Malartic, mining represents a high proportion of total jobs (26.4%), followed by health care (14.8%), and retail (11.5%) sectors (Statistics Canada, 2017, updated in 2019).

20.4.2.3 Domestic & Industrial Wells

The Hydrogeological Information System is a public database that presents data from wells/boreholes in Québec. However, not all wells/boreholes are listed in this database. Domestic wells are listed within and near the proposed layout. They are mainly concentrated in the inhabited areas of Dubuisson and Lac Vassan, but others are associated with mining sites, including the former Marban, Norlartic, and Kierens mines.

20.4.2.4 Landscape Components

The project's location is part of the Malartic Lake regional landscape unit and the Val-d'Or regional landscape unit (Robitaille and Saucier, 1998).

The landscape is typical of a gently rolling plain with several lakes and rivers. These elements of the hydrographic network constitute the main natural attractions of the landscape. The esker west of the De Montigny Lake, which stretches from north to south, also represents a specific component of the natural landscape of the area.

The main access roads to the landscape of the proposed layout and its surroundings are National Routes 111 and 117. The Route 117 is identified as a panoramic road corridor. Access to the landscape is also available from several recreational trails, such as the Route Verte, the Trans-Québec snowmobile trail, regional snowmobile trails, and ATV trails.

The visual environment also includes industrial elements such as mining infrastructure, a railroad, and a power line.

20.4.2.5 Archaeology & Heritage

No archaeological sites listed by the Ministry of Culture and Communications, the Land Use Planning and Development Plan of the RCM La Vallée-de-l'Or, and the Inventory of Archaeological Sites of Québec are located within the proposed layout.

An evaluation of the archaeological potential was done in 2016 by Archeo-08 and updated by Asini in 2021. When an area presented a certain potential, it was rated low, medium, or high. According to the 2021 report, some areas have a medium archaeological potential, mainly the shore of the Keriens Creek where the pits are located. High potential has been identified at the mouth of Keriens Creek with Montigny Lake but no infrastructure is planned in this area.

20.4.2.6 First Nations

The project site is located on the ancestral territory of the Algonquin Anishinabeg Nation (Anicinabek). The nearest Indigenous communities are the following:

- Abitibiwinni (Pikogan) First Nation (52 km);
- Anishnabe du Lac-Simon First Nation (50 km);
- Community of Anicinape de Kitcisakik (84 km); and
- Long Point First Nation (Winneway) (76 km).

No land in a reserve is located within the proposed layout. The project area is, however, located on land that is subject to a comprehensive land claims agreement or a self-government agreement.

The Federal Impact Assessment Agency has submitted a broader list of First Nations communities to consult as per the following list. O3 Mining will be communicating with all the communities to verify their interest in being consulted.

- Government of the Crie 1655 Nation (*Gouvernement de la Nation Crie 1655*);
- Nation of Anishnabe du Lac-Simon;
- Community of Anicinape de Kitcisakik;
- Abitibiwinni (Pikogan) First Nation;
- Long Point First Nation (Winneway);
- Wahgoshig First Nation;
- Kebaowek First Nation;
- Kitigan Zibi Anishinabeg;
- Timiskaming First Nation;
- Wolf Lake First Nation; and
- Mitchikanibikok Inik First Nation (*Algonquins de Barriere Lake*).

O3 Mining has initiated dialogue with Pikogan, Lac Simon, and Kitcisakik.

20.4.3 Social Related Requirements

20.4.3.1.1 Engagement Activities Requirements

The Provincial and Federal governments recommend that project initiators engage in good faith, as soon as possible, in a process of information and consultation with locals and indigenous communities, with an approach based on respect, transparency, and collaboration.

The MELCC published the "Guide to Information and Consultation Process carried out with Indigenous Communities by the Intiator of a Project Subject to the Environmental Impact Assessment and Review Process" (*Guide sur la démarche d'information et de consultation réalisée auprès des communautés autochtones par l'initiateur d'un projet assujetti à la procédure d'évaluation et d'examen des impacts sur l'environnement*) for the implementation of an information and consultation process with indigenous communities for projects subject to the EQA assessment and review procedure. The MERN also published a Native Community Consultation Policy specific to the mining sector. Finally, the Agency also published an Interim Guidance: Indigenous Participation in Impact Assessment.

In accordance with the Mining Act, O3 Mining will have to establish a monitoring committee to foster the involvement of the local community. The committee must be established within 30 days after the mining lease is issued and must be maintained until all the work provided for in the rehabilitation and restoration plan has been completed. The lessee determines the number of representatives who are to sit on the committee. However, the committee must include at least one representative of the municipal sector, one representative of the economic sector, one member of the public, and, if applicable, one representative of a Native community consulted by the Government with respect to the project.

20.4.3.1.2 Agreements

Since the project will require lands on which permanent residences, businesses, and public roads are located within the proposed layout, agreements will have to be settled with respective owners. O3 Mining has initiated discussion with some residents and business owners on the footprint of the project, but no agreements have been signed. However, residences and/or lots that have been put on the real estate market have been purchased by O3 Mining as a proactive approach.

20.4.3.1.3 Additional Studies

The following studies should be carried out as part of the EIA process:

- Economic Benefits Assessment;
- Visual Integration (landscape and night light baseline conditions surveys were completed in 2021 by WSP);
- Circulation and Roads Security Assessment; and
- Domestic Wells Inventory (planned in 2022).

20.5 Mine Closure Requirements

Under the Mining Act, a person who performs prescribed exploration or mining work must submit a closure plan for the land affected by their operations, subject to approval by the MERN and is conditional upon receipt of a favourable decision from the MELCC. This approval is required for the release of the mining lease and for mining operations to begin (including the construction phase).

The main objective of a mining closure plan is to return the site to an acceptable condition for the community. Protection, rehabilitation, and closure measures that will be presented will aim to return the site to a satisfactory condition by:

- Eliminating unacceptable health hazards and ensuring public safety;
- Limiting the production and spread of contaminants that could damage the receiving environment and, in the long term, aiming to eliminate all forms of maintenance and monitoring;
- Returning the site to a condition in which it is visually acceptable (reclamation); and
- Returning the infrastructure areas (excluding the tailings impoundment and waste rock piles) to a state that is compatible with future use (rehabilitation).

Also, as the Marban Engineering Project is an open-pit mining operation, a cost-benefit analysis for backfilling the pit must be included in the closure plan.

A proponent whose closure plan has been approved must submit a revised plan every 5 years to the MERN, unless the latter has set a shorter period for approving the closure plan or the revised plan.

Closure work must begin within 3 years of the cessation of operations.

A post-closure monitoring and maintenance program will have to be carried out to ensure the physical stability of infrastructure and the effectiveness of any remedial measures applied at the site. The post-closure monitoring and maintenance program must include:

- A physical stability monitoring and maintenance program;
- An environmental monitoring program; and
- An agronomical monitoring program.

A certificate of release may be issued when:

- The MERN is satisfied that the closure work has been completed in accordance with the closure plan approved by the MERN, and no sum of money is due to the MERN with respect to the performance of the work;
- The MERN is satisfied that the condition of the land affected by the mining operations no longer poses a risk for the environment or for human health and safety; and
- The MERN receives a favourable decision from the MELCC.

The certificate of release relates only to the obligations under the Mining Act and does not release a person from the obligations under the EQA and its regulations.

An amendment to Section 111 of the Regulation respecting Mineral Substances other than Petroleum, Natural Gas, and Brine was made in 2013 (Decree 838-2013). Thus, mining companies must now provide a financial guarantee. This financial guarantee ensures that funds will be available to carry out the work provided for in the closure plan in the event of default by the proponent. It covers the entire cost of land rehabilitation and reclamation work for the entire mine site as provided for in the closure plan.

Moreover, in November 2017, the MERN published the Guidelines for the preparing mine closure plans in Québec. A detailed breakdown of the dismantling cost for all infrastructure built on-site must now be provided and the engineering and supervision fees (indirect costs) have been fixed to a minimum of 30% of the direct cost including the post-restoration monitoring at the conceptual stage of the project. A minimum contingency of 15% must be added to the estimated cost. The proponent who engages in mining operations must pay the financial guarantee according to the following terms:

- The guarantee must be paid in three instalments;
- The first payment must be made within 90 days of receiving plan approval;
- Each subsequent payment must be made on the anniversary of plan approval; and
- The first payment represents 50% of the total amount of the guarantee, and the second and third payments represent 25% each.

Estimated closure cost for the Marban Engineering Project are presented in Section 21. The guarantee must remain in effect until the certificate of release provided for in Section 232.10 of the Mining Act has been issued.

Post-closure scope or activities have not been reviewed as part of the PFS. This will be included in the FS or EIA scope.

21 CAPITAL AND OPERATING COSTS

The capital and operating cost estimates presented in this PFS are for the Marban Engineering Project and are based on open pit mining of the Marban, Norlartic and Kierens deposits and the construction of a process plant, associated tailings storage facility, and infrastructure. The processing plant nameplate capacity is 16,438 t/d (6.0 Mt/y), with a LOM of 10 years. All capital and operating cost estimates are reported in Canadian dollars for this PFS.

The purpose of the capital estimate is to provide substantiated costs which can be used to assess the economics of the project.

21.1 Capital Cost Summary

The capital cost estimate was developed in Q3 2022 from Ausenco's in-house database of projects and studies and experience with similar operations to a level of accuracy of ±25% (Class 4) in accordance with the Association for the Advancement of Cost Engineering International (AACE International). The estimate includes mining, processing, onsite infrastructure, offsite infrastructure, project indirect costs, project delivery, owners' costs, and provisions. The capital cost summary is presented in Table 21-1. The total initial capital cost for the Marban Engineering Project is C$435M and LOM sustaining costs are C$283M. Closure costs are estimated at C$48.9M. Of the initial capital costs, approximately 70% of the project costs were derived from first budgetary quotes.

The following parameters and qualifications were considered:

- For material sourced in US dollars (5.1% of initial capex), an exchange rate of 1.30 Canadian dollar to 1.00 US dollar was assumed.
- No allowance has been made for exchange rate fluctuations.
- There is no escalation added to the estimate.
- A growth allowance was included.
- Data for the estimates have been obtained from numerous sources, including:
    - Mine schedules;
    - Prefeasibility-level engineering design;
    - Topographical information obtained from the site survey;
    - Geotechnical investigations;
    - Budgetary equipment quotes from Canadian and International suppliers;
    - Budgetary unit costs from several local contractors for civil, concrete, steel, electrical, piping, and mechanical works; and
    - 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. Percentage of contingency was allocated to each of these categories on a line-item basis based on the accuracy of the data. An overall contingency amount was derived in this fashion.

Table 21-1: Capital Cost Summary

WBS2	Cost Centre	Initial Capital (C$M)	LOM Sustaining (C$M)	Total Capital (C$M)
1100	Mining General and Administration	7.77	0.0	7.77
1200	Drill and Blast	4.74	0.0	4.74
1300	Material Movement	19.8	0.0	19.8
1400	Mining Civil Infrastructure	11.7	0.0	11.7
1500	Mining Infrastructure & Services	2.71	0.0	2.71
1600	Mine Major Equipment	17.0	211.8	228.8
1700	Mine Support Equipment	8.09	0.00	8.09
Mining Total		71.8	211.8	283.7
2100	Crushing	17.9	0.0	17.9
2200	Stockpile & Reclaim	7.99	3.19	11.2
2300	Grinding	66.2	0.0	66.2
2400	Leaching	29.6	0.0	29.6
2500	Elution, Carbon Regeneration & Gold Room	16.2	0.0	16.2
2600	Cyanide Detoxification & Tailings	9.56	0.0	9.56
2800	Reagent Storage & Distribution	9.19	0.0	9.19
2900	Utilities (Air & Water Services)	4.19	0.0	4.19
Process Plant Total		160.9	3.19	164.1
3100	Bulk Earthworks	7.57	0.0	7.57
3200	Power Supply and Distribution	12.0	0.0	12.0
3300	Fuel Storage	0.40	0.0	0.40
3400	Ancillary Buildings	8.85	1.26	10.11
3500	Site Services	2.10	29.9	32.0
3600	Site Water Services	4.97	0.0	4.97
3700	Site Water Management	12.0	0.0	12.0
3800	Tailings Storage and Management Facilities	45.4	34.3	79.7
On-Site Infrastructure Total		93.3	65.5	158.8
4100	Public Road Diversions and Upgrades	2.33	0.0	2.33
4200	Keriens Creek Diversion	10.1	0.0	10.1
Off-Site Infrastructure Total		12.4	0.0	12.4
5100	Temporary Construction Facilities and Services	11.9	0.50	12.4
5300	Spares (Commissioning, Initial and Insurance)	1.24	0.0	1.24
5400	First Fills & Initial Charges	3.86	0.0	3.86
Project Indirects Total		17.0	0.50	17.5
6100	Engineering & Construction Management Services	25.0	2.00	27.0
Project Delivery Total		25.0	2.00	27.0
7000	Owners' Costs	10.6	0.0	10.6
8000	Contingency	44.0	0.0	44.0
	Grand Total	435.1	283.0	718.0

Source: Ausenco, 2022.

The capital cost estimate was developed in accordance with the responsibility breakdown presented in Table 21-2.

Table 21-2: Estimate Responsibility Summary

WBS	Cost Centre	Responsible Company / Consultant
1000	Mining[A]	GMS
2000	Processing	Ausenco
3000	On Site Infrastructure	Ausenco
4000	Off Site Infrastructure	Ausenco
5000	Project Indirects	Ausenco
6000	Project Delivery	Ausenco
7000	Owners' Cost	O3 Mining
8000	Contingency	Ausenco

Note: [A] Ausenco provided costing for substations for electric-powered mining shovels.

21.2 Basis of Capital Cost Estimate

21.2.1 Exchange Rates

Vendors and contractors were requested to price in native currency. The estimate is prepared in the base currency of Canadian dollars (C$). Pricing has been converted to Canadian dollars using the exchange rates in Table 21-3. The US dollar contributions to the capital estimate are 5.1% of initial capital costs, and 36.1% of sustaining capital costs.

Table 21-2: Estimate Exchange Rates

Code	Currency	Exchange Rate
C$	Canadian	1.00
AU$	Australian Dollar	0.89
EUR	Euro	1.32
US$	United States Dollar	1.30
INR	Indian Rupee	0.016
CNY	Chinese Yuan	0.19

Source: Ausenco, 2022

21.2.2 Direct Costs – Mining

Mine capital costs have been derived from historic data collected by GMS (at other Canadian open pit mining operations along with budgetary quotes received from mining equipment suppliers) combined with the Marban PFS mine production schedule.

Pre-production mine operating costs (i.e., all mine operating costs incurred before mill start-up) are capitalized and included in the capital cost estimate.

Pre-production pit operating costs include drilling and blasting, load and haul, support, and general mining expenses (GME) costs.

All mine operations site development costs — such as clearing and grubbing, topsoil stripping, wetland removal, haul road construction and explosive pad preparation — are capitalized.

It is planned to purchase the mine equipment fleet either through financing or lease agreements with the vendors. Down payments and monthly lease payments are capitalized through the initial and sustaining periods of the project.

Estimated fleet spare and estimated initial fuel, lube, and tire inventories are capitalized.

The following items are also capitalized:

- Explosives magazine;
- Site GPS (global positioning system) and machine guidance systems;
- Mine survey gear and supplies;
- Geology, grade control, and mine planning software licenses; and
- Maintenance tooling and supplies.

Capital costs for the mine fleet maintenance facility, as well as the fuelling station, are accounted for under general project infrastructure.

The PFS mine area capital cost estimates for the Marban Engineering Project are presented in the WBS series 1000 in Table 21-1. It is the QP's opinion that these estimates are reasonable for the location and planned mine development and can be used for a PFS.

21.2.3 Direct Costs – Process Plant & On-Site Infrastructure

The definition of process equipment requirements was based on process flowsheets and process design criteria as defined in Section 17. All major equipment was sized based on the process design criteria in order to derive a mechanical equipment list. Mechanical scopes of work were developed and sent for budgetary pricing to equipment suppliers. For mechanical equipment costs, 72% of the value was sourced from budgetary quotes; the remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

Similarly, the major electrical equipment was sized based on the project's equipment list. Scopes of work were developed in order to receive budgetary pricing from equipment suppliers. For the electrical equipment, 91% of the value was sourced from budgetary quotations. The remainder was sourced by benchmarking against other recent Canadian gold projects and studies.

In support of the major mechanical and electrical equipment packages, the process plant and infrastructure engineering design was completed to a prefeasibility study level of definition, allowing for the bulk material quantities (steel, concrete, earthworks, piping, cables, instruments, etc.) to be derived for the major commodities.

21.2.4 Direct Costs – Off Site Infrastructure

21.2.4.1 Public Road Diversions and Upgrades

The estimate allows for diversions and upgrades to the public roads affected by the Marban Engineering Project. As outlined in Section 18.2, a new 4 km long alignment of an existing public secondary road (Chemin Gervais) will be constructed east of the project site, to maintain access to the existing bridge over Keriens Creek and the properties north of the creek. The costs associated with this estimate include site preparation, bulk excavation stripping, backfilling, trenching, ground improvement, and culvert corrugated steel pipes costs. The total public road diversion and upgrades cost was calculated to be C$2.3 million.

21.2.4.2 Keriens Creek Diversion

The Marban Engineering Project requires relocation of a section of Keriens Creek, upstream of the outlet to De Montigny Lake, in order to provide access to the Kierens and Norlartic pits, located underneath the existing creek alignment. This

estimate includes site preparation, diversion excavation and construction of dikes to block the existing creek route. The total creek diversion cost was calculated to be C$10.9 million.

21.2.5 Indirect costs

21.2.5.1 Temporary Construction Facilities and Services

Contractor indirect costs are related to the contractor's direct costs, and include:

- Mobilization and demobilization;
- Site offices and utilities;
- Construction equipment including mobile equipment, scaffolding, safety supplies, etc.;
- Head office costs/contribution;
- Financing charges;
- Insurances; and
- Profit.

Contractors provided indirect costs as part of their pricing schedules.

21.2.5.2 Commissioning Reps and Assistance

Vendor representative costs during commissioning and construction include vendor representative support during the installation of the purchased equipment.

Vendor representative costs have been based on the engineer's evaluation of recommendations and prices provided by equipment vendors during the pricing enquiry process.

21.2.5.3 Spares

Commissioning spares quantities was recommended and priced by equipment suppliers. Where equipment pricing was not solicited from vendors, factors were applied based on standard estimating practices.

Capital spares prices for mechanical are based on the prices provided by equipment vendors during the enquiry process. If vendors did not provide a cost for capital spares, a factored allowance was included based on the supply price and benchmarked against Ausenco's in-house database of projects. Allowance factors were based on a 6-month period of capital spares.

21.2.5.4 First Fills and Initial Charges

Process first fill quantities (e.g., mill media and reagents) and first fill lubricants (e.g., greases, oils, and hydraulic fluids) were calculated based on the engineering design and priced using quotes that were provided by reagent and media suppliers.

21.2.6 Project Delivery

Engineering, Procurement, and Construction Management costs for the process plant have been estimated based on first principles and are inclusive of:

- Engineering;
- Procurement support (home office based);

- Construction management (site based);
- Project office facilities;
- Field inspection and expediting;
- Corporate overhead and fees;
- Travel expenses;
- Home office expenses;
- Site office expenses; and
- Commissioning support.

21.2.7 Owners' Costs

Owner's costs of 10.6 M have been provided by O3 Mining and are inclusive of the following:

- Owner's project team and expenses;
- Pre-production labour;
- Administration, finance, insurance, and legal fees;
- Environmental consultation and management;
- Human resources, recruiting and training;
- Community relations;
- Site security; and
- Mobile equipment and vehicle leases.

21.2.8 Closure Costs

Ausenco estimated the closure requirements inclusive of all necessary demolition, rehabilitation, revegetation, earth grading/contouring, scrap metal disposal/tipping fees, as well as post-closure monitoring. The total closure cost was calculated to be C$48.9 million.

21.2.8.1 Process Plant

Site closure for the process plant area captures the cost associated with the demolition of equipment, process plant, and mining building infrastructure and remediation works of the site.

21.2.8.2 Tailings Management Facility

Site closure costs for the non-process plant footprint include works to soil cover, revegetate/hydroseed the stockpiles and TSF, and construct a closure spillway.

21.2.9 Contingency

Contingency accounts for the difference in costs between 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 (in terms of the level of engineering definition, basis of the estimate, schedule development, etc.), it is essential that the estimate include a provision to cover the risk from these uncertainties.

The contingency cost is from total installed costs based on the level of uncertainty for each area, using a deterministic approach. Ausenco calculated a contingency of C$44.0 M following the percentage allotments by commodity according to Table 21-4.

Table 21-3: Contingency Applied

Commodity Code	Commodity Description	Contingency Applied
A	Architectural	15%
B	Earthworks	10%
C	Concrete	15%
D	Mining	5%
E	Electrical	15%
F	Platework and Mechanical Bulks	15%
I	Instrumentation	10%
M	Mechanical Equipment	15%
N	Plant & Miscellaneous Equipment	5%
P	Pipework	15%
Q	Electrical Bulks	15%
S	Structural Steel	15%
U	Field Indirects	15%
V	Third-Party Packages/Other	15%
W	EPCM, EPC & EP	15%
X	Provisions	0%
Y	Owner's Costs	0%

Source: Ausenco, 2022.

21.2.10 Salvage

Salvaging costs have been projected by assuming that all mechanical equipment will carry a 10% resale value at the end of the mine life, and that all the projected 1,000 t of steel remaining in the warehousing can be returned to the stock provider for C$258/t. Total salvaging value was estimated at C$10 million.

21.2.11 Growth Allowance

Each line item of the estimate was developed initially at base cost only. A growth allowance has then been allocated to each element of those line item costs to reflect the level of definition of design and pricing strategy.

Estimate growth is:

- Intended to account for items that cannot be quantified based on current engineering status but empirically known to appear;
- Accuracy of quantity take-offs and engineering lists based on the level of engineering and design undertaken at Feasibility Study level; and
- Pricing growth for the likely increase in cost due to development and refinement of specifications as well as re-pricing after initial budget quotations and after finalisation of commercial terms and conditions to be used on the project.

Where an allowance has been used which is the result of factoring, no growth has been applied as the factor has been surmised from a total cost.

Growth has been calculated on a line-item level, by evaluating the status of the engineering scope definition and maturity and the ratio of the various pricing sources for equipment and materials used to compile the estimate. The capital cost growth allowance is presented in Table 21-5.

Table 21-4: Growth Allowance

WBS	Area	Weighted Average Growth Applied
1000	Mining	0%
2000	Process Plant	9%
3000	On Site Infrastructure	10%
4000	Off Site Infrastructure	8%
5000	Project Indirects	10%
6000	Project Delivery	0%
7000	Owner's Costs	0%
8000	Provisions	0%
	Overall	5%

Source: Ausenco, 2022

21.2.12 Exclusions

The following costs and scope will be excluded from the capital cost estimate:

- Land acquisitions;
- Wetlands compensation or relocations;
- Senior finance charges;
- Permitting;
- Royalty buyouts;
- Further testwork and drilling programs;
- Taxes not listed in the financial analysis;
- Environmental approvals;
- This study or any future project studies, including Environmental Impact Studies
- First Nations Impact Benefit Agreement costs
- Sales taxes;
- Operating costs;
- Operational readiness costs;
- Working capital;
- Scope changes and project schedule changes and the associated costs;
- Any facilities/structures not mentioned in the project summary description;

- Geotechnical unknowns/risks;
- Financing charges and interest during the construction period;
- Any costs for demolition/relocation of existing facilities or decontamination for the current site; and
- Third party costs.

21.3 Sustaining Costs

21.3.1 Mining Sustaining

Down payments and monthly lease payments for the mine equipment fleet purchased throughout the life of mine are capitalized through the sustaining periods of the project. Spare major components are also included throughout the sustaining periods, as well as the expansions to the high precision GPS systems, and radio communications systems when each of the additional mobile fleet units are commissioned. A LOM total of C$212M was estimated for mine equipment lease charges and mining major components replacements and repairs.

Table 21-5: Mining Sustaining Costs (C$M)

Description	Year										LOM Total
	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9	Y10	
Mine Equipment Lease Charges	19.8	24.1	27.6	27.5	21.2	11.5	15.1	4.3	1.1	1.0	153.2
Mining Major Repairs	0.4	2.7	15.1	5.0	9.5	9.5	6.5	4.5	3.7	1.7	58.7

21.3.2 Processing Sustaining

Process plant sustaining capital costs include process plant and surface support vehicle leases, stockpile cover repairs, deferred truck shop costs, and the ongoing expansion of the TSF. The process plant sustaining costs are shown in Table 21-6.

Table 21-6: Processing Sustaining Costs (C$M)

Description	Year					LOM Total
	Y1	Y2	Y3	Y4	Y5	
Tailings Facilities	13.1	22.0	-	1.7	-	36.8
Site Mobile Equipment Lease Payments	6.0	6.0	6.0	6.0	6.0	29.9
Buildings	0.04	1.31	0.04	3.06	-	4.5

21.4 Operating Costs

The operating cost estimate is presented in Q3 2022 C$. The estimate was developed to have an accuracy of ±15%. The estimate includes mining, processing, and general and administration (G&A) costs.

The overall life-of-mine operating cost is C$1,419.0 million over 10 years, or an average of C$25.14/t of ore milled in a typical year. Of this total, processing and G&A account for C$520.5 million and mining accounts for C$898.5 million. Table 21-7 provides a summary of the project operating costs.

Table 21-7: Operating Cost Summary

Cost Centre	C$/t Milled (Over LOM)	C$M
Mining		
Drilling	1.24	69.7
Blasting	2.50	141.1
Loading	1.48	83.8
Hauling & Rehandling	5.43	306.2
Overburden Mining	0.05	2.7
Road & Dump Maintenance	2.09	117.9
Misc. Maintenance	0.92	52.1
Mine General & Admin	2.22	125.0
Mining Subtotal	**15.92**	**898.5**
Process Plant		
Reagents	2.13	12.8
Consumables	2.50	15.0
Plant Maintenance	0.42	23.5
Power	1.15	65.2
Laboratory	0.03	1.7
Labour – Process Plant	1.46	87.7
Processing Mobile Equipment	0.06	3.3
Process Plant Subtotal	**7.74**	**442.5**
G&A		
Labour – G&A	0.72	40.8
G&A Expenses	0.58	32.6
Site Maintenance	0.08	4.7
G&A Subtotal	**1.38**	**78.1**
Total Project Operating Costs	**25.14**	**1,419.0**

21.4.1 Overview

Common to all operating cost estimates are the following assumptions:

- Cost estimates are based on Q3 2022 pricing without allowances for inflation.
- For material sourced in US dollars, an exchange rate of 1.30 Canadian dollar to 1.00 US dollar was assumed.
- Estimated costs for diesel and gasoline are C$1.20/L and C$1.045/L, respectively.
- The annual power costs were calculated using a unit price of C$0.048/kWh.
- Labour is assumed to come from the local area of highly skilled workers in Val d'Or.

21.4.2 Operating Costs – Mining

Mine operating costs are built up from first principles and applied to the Marban PFS mine production schedule.

Cost inputs are derived from historical data collected by GMS. This includes cost and consumption rates for such inputs as fuel, lubes, explosives, tires, undercarriage, GET, drill bits/rods/strings, machine parts, machine major components, labour rates, and operating and maintenance labour ratios. Equipment and labour productivity inputs are estimated for the specific equipment fleet and rationalized to existing Canadian open pit mine operations. Simulated hauler cycle times from source pit benches to planned destinations are utilized to inform hauler productivities.

Annual production tonnes are taken from the Marban PFS mine production schedule. Drilling, loading and hauling hours are calculated based on the capacities and parameters of the specified equipment fleet. The production tonnes and primary fleet hours also provide the basis for blasting consumables and support fleet inputs.

Estimated life-of-mine unit mining costs are shown in Table 21-8. It is the QP's opinion that the estimates are reasonable for the location and planned mine operation activities and can be utilized for a PFS.

Table 21-8: Marban Mine Operating Cost Summary

Item	C$/t Mined	C$/t Milled	C$M
Mine Operations	0.10	0.62	35.2
Mine Maintenance Admin.	0.15	0.88	49.7
Mine Geology	0.04	0.26	14.6
Mine Engineering	0.07	0.42	23.5
Grade Control	0.03	0.17	9.7
Electric Equipment Cable Handling	0.02	0.12	6.8
Drilling	0.16	0.98	55.3
Blasting	0.42	2.57	144.8
Pre-Split Drilling and Blasting	0.05	0.27	15.5
Loading	0.26	1.56	87.9
Hauling	0.93	5.66	319.2
Dump Maintenance	0.26	1.52	85.9
Road Maintenance	0.13	0.77	43.7
Dewatering	0.01	0.09	4.8
Overburden Mining Contract	0.01	0.07	4.1
Support Equipment	0.13	0.76	43.1
Rehandling	0.00	0.02	1.3
Total Mining Cost (Pre-Prod)	0.14	0.83	46,730
Total Mining Cost (Ops.)	2.62	15.92	898,504
Total Mine Operating Cost	2.76	16.75	945,234

21.4.3 Process Operating Costs

Unless stated otherwise, all costs presented in this chapter are in C$. The estimate aligns with the principles of a Class 4 feasibility study level estimate with a ±30% accuracy according to the Association for the Advancement of Cost Engineering International (AACE International). The processing operating cost estimate includes costs relating to reagent and consumable consumption, plant maintenance, power use, the laboratory, labour, and processing mobile equipment.

The average yearly operating costs amount to C$46.0 million, or C$7.84/t of ore milled. A breakdown of this value and its unit costs is presented in Table 21-9.

Table 21-9: Annual Process Operating Costs

Cost Centre	C$/t	C$M
Process Plant		
Reagents	2.13	12.5
Consumables	2.50	14.6
Plant Maintenance	0.42	2.4
Power	1.15	6.8
Laboratory	0.03	0.2
Labour – Process Plant	1.55	9.1
Processing Mobile Equipment	0.06	0.3
Process Plant Sub-total	7.84	46.0

21.4.3.1 Basis of Estimate

The following was used to determine the project's LOM process operating costs in agreement with the cost definition and estimate methodologies outlined below. This basis considers the development of a facility capable of processing 16,438 t/d of ore.

Assumptions made in developing the process operating cost estimate are listed below:

- Mill production is set at an average of 6.0 Mt/a.
- Process plant operating costs are calculated based on labour, power consumption, and process and maintenance consumables.
- Off-site gold refining, insurance, and transportation costs are excluded, as they are included elsewhere.
- Labour rates were provided by O3 Mining.
- General and administration (G&A) costs were baselined against previous project experience, defined along with specific inputs from O3 Mining.
- No factor for spare parts has been applied to adjust for consumption of fewer spare parts in early years of operation.
- Grinding media consumption rates have been estimated based on the ore characteristics.
- Reagent consumption rates have been estimated based on the metallurgical testwork results at a nominal basis.
- Mobile equipment cost provides for fuel and maintenance, not for purchase or vehicle lease.

21.4.3.2 Reagents and Consumables

Individual reagent consumption rates were estimated based on the metallurgical testwork results, Ausenco's in-house database and experience, industry practice, and peer-reviewed literature. Major reagent costs were obtained from vendor quotations to Val d'Or, including SAG and ball mill media, sodium hydroxide, hydrated lime, flocculant, activated carbon, and liquid sulphur dioxide (SO_2). Other reagent costs were obtained through benchmarking for similar projects performed by Ausenco. A detailed description of the reagents required for the process is provided in Chapter 17.

Other consumables (e.g., liners for the primary crusher, SAG mill, ball mill, and ball media for the mills) were estimated using:

- Metallurgical testing results (Bond abrasion testing);
- Ausenco's in-house calculation methods, including simulations; and
- Forecast nominal power consumption.

Reagents and consumables represent approximately 60% of the process operating cost at C$4.63/t milled.

21.4.3.3 Maintenance

Annual maintenance consumable costs were calculated based on a total installed mechanical capital cost by area using weighted average factors ranging between 2% and 10%. The factor was applied to mechanical equipment. The total maintenance consumables operating cost is C$0.42/t milled, or approximately 6% of the direct mechanical capital cost, which is equivalent to approximately 5% of the process operating cost.

21.4.3.4 Power

The processing power draw was based on the average power utilisation of each motor on the electrical load list for the process plant and services. Power will be supplied by the Hydro-Québec grid to service the facilities at the site. An estimated 144,000 MWh are nominally required per year, resulting in an annual power cost of C$6.77 million, or C$1.15/t milled. This represents 15% of the total processing operating costs

21.4.3.5 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 plant solid 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 0.5% of the total process operating cost, and the forecasted annual requirement for internal assays will be around 16,200 for the processing plant.

21.4.3.6 Labour

The personnel requirement was estimated by benchmarking against similar projects. The labour costs incorporate personnel requirements for plant operation, such as management, metallurgy, operations, maintenance, site services, assay lab, and contractor allowance. The total process plant labour averages 75 employees.

Individual personnel were divided into their respective positions and classified as either 8-hour or 12-hour shift employees. Salaries were provided by O3 Mining. O3 Mining also confirmed the specific benefits and bonuses to be allocated. Thus, the rates were estimated as overall rates, including all burden costs.

An organizational staffing plan outlining the labour requirement for the process plant is shown in Table 21-10. Process plant labour represents approximately 18% of the total process operating cost at C$1.55/t milled.

Table 21-10: Operations and Maintenance Staffing Plan

Labour / Contractor Summary	#/Shift	# Shifts	Quantity
Process Upper Management			
Sr Metallurgist	1	1	1
Mine Manager	1	1	1
Maintenance Planner	1	2	2
Chief Assayer	1	1	1
Maintenance Superintendent	1	1	1
Mill Operations			
Shift Foreman	1	4	4
Crusher Operator	1	4	4
Crusher Labourer	1	4	4
Grinding Operator	1	4	4
Leach/Detox Operator	1	4	4
Reagents Labourer	1	4	4
Control Room Operator	1	4	4
Gold Room Operator	1	4	4
Surface Crew/Tailings	1	4	4
Elution/Gravity Operator	1	4	4
Technical Services			
Plant Metallurgist	2	1	2
Metallurgical Tech	2	1	2
Assay Lab Tech	2	2	4
Mill Maintenance			
Maintenance Foreman	1	2	2
Electrician	2	2	4
Electrical Apprentice	1	1	1
Welders	1	2	2
Instrument Tech	1	2	2
Millwright/Fitter	4	2	8
Mechanical Apprentice	1	2	2
Total Process Plant	32	63	75

21.4.3.7 Processing Mobile Equipment

Vehicle costs are based on a scheduled number of light vehicles and mobile equipment, including fuel, maintenance, spares and tires, and annual registration and insurance fees. As mentioned, the prices of fuel used in these calculations are C$1.20/L for diesel and C$1.05/L for gasoline.

The cost of operating and maintaining the processing mobile vehicles is estimated as C$0.06/t milled, or 1% of the processing operating costs.

21.4.4 General & Administrative Operating Costs

General and administrative (G&A) costs are expenses not directly related to the production of gold and include expenses not included in mining, processing, external refining, and transportation costs. These costs were developed with input from O3 Mining, as well as Ausenco's in-house data on existing Canadian operations.

The G&A costs were determined for a 10-year mine life with an average cost of C$1.38/t milled. These costs were assembled according to the following departmental cost reporting structure:

- G&A maintenance (includes vehicle maintenance and road maintenance);
- G&A personnel;
- Human resources (includes recruiting, training, and community relations);
- Site administration, maintenance, and security (includes professional memberships and dues, office supplies and equipment, in-town office rental, sewage and garbage disposal, and bank fees);
- Health and safety (includes personal protective equipment and first aid supplies);
- Asset operation (includes non-operation-related vehicles);
- Environmental (includes sampling);
- IT and telecommunications (includes software and microwave link);
- Contract services (includes insurance, sanitation, licenses, and legal fees); and
- Cyanide code fees.

The G&A labour costs were estimated by developing a headcount profile for each department. Labour rates provided by O3 Mining were applied to develop the total G&A labour cost.

G&A labour resources include 38 employees. An organizational staffing plan outlining the G&A labour requirement is shown in Table 21-11.

Table 21-11: General and Administrative Staffing Plan

Labour / Contractor Summary	#/Shift	# Shifts	Quantity
General Manager	1	1	1
H&S and Operations Training Superintendent	1	1	1
Safety Technicians	3	1	3
Nurse	1	1	1
Manager HR	1	1	1
Recruitment and HR	3	1	3
Reception	1	1	1
Manager Environment	1	1	1
Coordinator Environment	1	1	1
Environmental Technician	2	1	2
Security EMT	1	2	2
Security Personnel	2	4	8
Warehouse Foreman	1	1	1
Warehouse Staff	3	2	6
Sr. Accountant	1	1	1
Clerk	3	1	3
IT Techs	1	2	2
Total G&A	27	23	38

A breakdown summary of life-of-mine G&A costs is shown in Table 21-12.

Table 21-12: Annual Average G&A Operating Cost Summary

Cost Centre	C$/t	C$ M
G&A Labour	0.72	4.2
G&A Expenses	0.58	3.4
Site Maintenance (including mobile equipment)	0.08	0.5
Total G&A	1.38	8.1

22 ECONOMIC ANALYSIS

22.1 Cautionary Statements

The results of the economic analyses discussed in this section 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:

- Mineral resource estimates;
- Assumed commodity prices and exchange rates;
- The proposed mine production plan;
- Projected mining and process recovery rates;
- Assumptions as to mining dilution and ability to mine in areas previously exploited using mining methods as envisaged;
- Sustaining costs and proposed operating costs;
- Assumptions as to closure costs and closure requirements; and
- Assumptions as to environmental, permitting, and social risks.

Additional risks to the forward-looking information include:

- Changes to production costs from what is assumed;
- Unrecognized environmental risks;
- Unanticipated reclamation expenses;
- Unexpected variations in quantity of mineralized material, grade, or recovery rates;
- 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;
- Ability to maintain the social licence to operate;
- Accidents, labour disputes and other risks of the mining industry;
- Changes to interest rates; and
- Changes to tax rates.

Calendar years used in the financial analysis are provided for conceptual purposes only. Permits still have to be obtained in support of operations, and approval for development to be provided by O3 Mining's Board.

22.2 Methodology Used

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project based on a 5% discount rate. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess the impact of variations in metal prices, head grades, foreign exchange, operating costs, and capital costs. The capital and operating cost estimates were developed specifically for this project and are summarized in Section 21 of this report (presented in 2022 dollars). The economic analysis has been run with no inflation (constant dollar basis).

22.3 Financial Model Parameters

The economic analysis was performed using the following assumptions:

- Construction period of 18 months;
- Mine life of 9.6 years;
- Base case gold price of US$1,700/oz was based on consensus analyst estimates and recently published economic studies. The forecasts used are meant to reflect the average metal price expectation over the life of the project. No price inflation or escalation factors were taken into account. Commodity prices can be volatile, and there is the potential for deviation from the forecast;
- United States to Canadian dollar exchange rate assumption of 0.77 (US$/C$)
- Cost estimates in constant 2022 C$ with no inflation or escalation factors considered;
- Results are based on 100% ownership with 1.0% Net Smelter Return (NSR;)
- Capital costs funded with 100% equity (i.e., no financing costs assumed);
- Inventory and accounts payable periods of 30 days;
- All cash flows discounted to beginning of construction;
- All metal products are assumed sold in the same year they are produced;
- Project revenue is derived from the sale of gold doré; and
- No contractual arrangements for refining currently 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 tax model was compiled by O3 Mining with assistance from third party taxation professionals. The calculations are based on the tax regime as of the date of the PFS study and include estimates for O3 Mining's expenditures and related impacts to various tax pool balances, between the PFS study and the assumed construction start date. As of the effective date of this report, the project was assumed to be subject to the following tax regime:

- The Canadian corporate income tax system consists of 15% federal income tax and 11.5% provincial income tax.
- Mining tax calculated in accordance with the Quebec Mining Tax Act
- Total undiscounted income and mining tax payments are estimated to be C$454 M over the LOM.
- The tax evaluation was completed by applying the following assumptions:

- All royalty payments are deductible for income tax purposes and non-deductible for Quebec Mining Duties purposes.
- Operating expenses and refining charges are fully deductible for income and mining tax purposes.

- The opening balance of non-capital losses carry forward corresponds to the closing balance as per the T2 and Quebec income tax return filed for fiscal year (FY) 2021.The non-capital losses expire at various years from 2026 onwards. The projections currently indicate the non-capital losses will be fully utilized before expiration.
- The opening balance of Canadian Exploration Expense (CEE) corresponds to the closing balance of regular and successor pools as per the FY 2021 T2 and Quebec income tax return.
- The opening balance of Canadian Development Expense (CDE) corresponds to the closing balance of regular and successor pools as per the FY 2021 T2 and Quebec income tax return.
- The opening balance of Investment Tax Credits (ITC) corresponds to the closing balance as per the FY 2021 T2. The ITCs expire at various years from 2030 onwards. The projections currently indicate the ITCs will be fully utilized before expiration.
- The opening balance of Undepreciated Capital Cost (UCC) corresponds to the closing balance as per the FY 2021 T2.
- For simplicity purposes, it has not been considered accelerated deductions of CDE and UCC under the Accelerated Investment Incentive, as it would result in a timing impact only.
- The Property Plant & Equipment (PP&E) is expected to be disposed of by the end of the life of the mine and salvage proceeds of C$10M are expected to be received. As a result, a terminal loss is being considered for all CCA classes in year 2035. The salvage proceeds were added back in the taxable income calculation.
- The model currently indicated that non-capital losses will be generated on FY 2036 to FY 2038. Such losses can be carried back to the three previous taxation years when taxable income is projected to be generated.
- For simplicity purposes, it has not been considered the Quebec minimum tax in the model. As the minimum tax paid becomes a credit that can be applied against the Quebec regular mining tax on annual profits, based on the current version of the LOM adding the mining tax would result in a timing impact only.
- The FY 2022 opening balance of exploration pools for mining tax purposes considered in the model corresponds to the closing balance as per the FY 2021 Quebec mining tax return.
- The FY 2022 opening balance of tax depreciation for mining tax purposes considered in the model corresponds to the closing balance as per the FY 2021 Quebec mining tax return.
- For simplicity purposes, it has been assumed that all operating expenses would be deductible for purposes of calculating the Mine Mouth Output Value.

22.3.2 Closure Costs & Salvage Value

Closure costs are applied at the end of the LOM. Mine equipment salvage value is applied during operating years, while plant equipment and material salvage value is applied at the end of the LOM. Closure costs were estimated C$48.9M and total salvage value of C$10.0M.

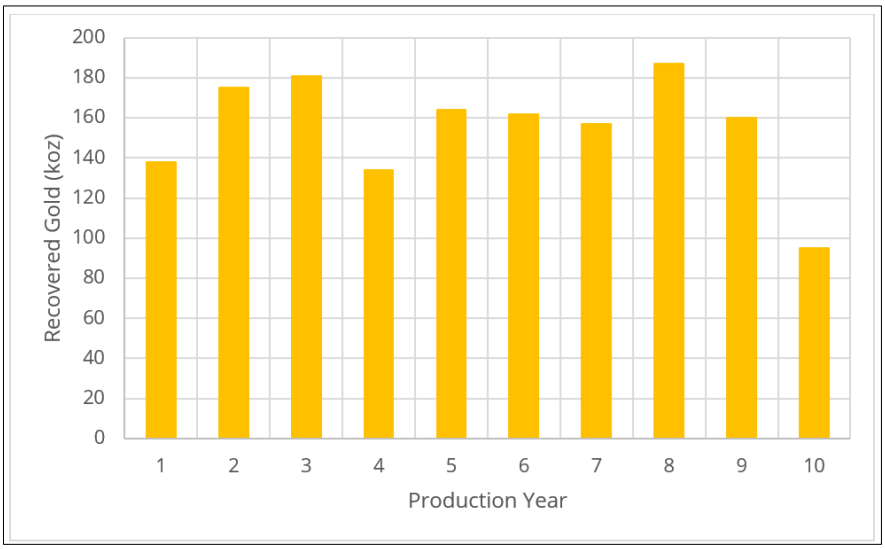
22.4 Royalties

Based on the agreements in place, a 1.0% NSR royalty has been assumed for the project, resulting in approximately C$34.2M in undiscounted royalty payments over LOM.

22.5 Gold Production

Over the LOM, a total of 1,552 koz of gold (average annual: 161 koz) will be produced. Figure 22-1 provides a summary of the annual gold production by year.

Figure 22-1: Annual Gold Production (koz)

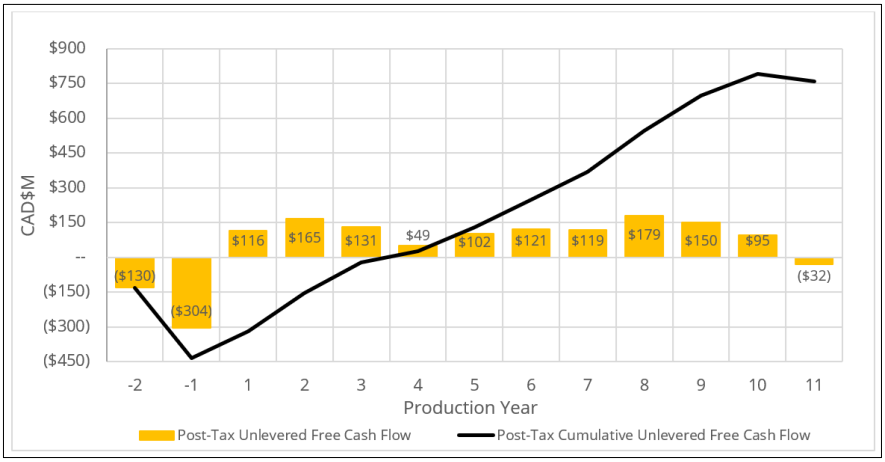


Source: Ausenco, 2022.

22.6 Economic Analysis

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV5%) is C$775M, the internal rate of return IRR is 30.2%, and the payback period is 2.8 years. On an after-tax basis, the NPV5% is C$463M, the IRR is 23.2%, and the payback period is 3.5 years. A summary of the project economics is shown graphically in Figure 22-2 and listed in Table 22-1 and. The cashflow on an annualized basis is provided in Table 22-2.

Figure 22-2: Projected Post-Tax Cash Flow



Source: Ausenco, 2022.

Table 22-1: Summary of Project LOM Cashflow Assumptions & Results

General	LOM Total / Avg.
Gold Price (US$/oz)	$1,700
Exchange Rate ($US:$C)	$0.77
Mine Life (years)	9.6
Total Waste Tonnes Mined (kt)	286,144
Total Mill Feed Tonnes (kt)	56,436
Strip Ratio (waste: mineralization)	5.1
Production	**LOM Total / Avg.**
Mill Head Grade (g/t)	0.91
Mill Recovery Rate (%)	94.2%
Total Mill Ounces Recovered (koz)	1,552
Total Average Annual Production (koz)	161
Operating Costs	**LOM Total / Avg.**
Mining Cost (C$/t Mined)	$2.6
Mining Cost (C$/t Milled)	$15.9
Processing Cost (C$/t Milled)	$7.8
G&A Cost (C$/t Milled)	$1.4
Total Operating Costs (C$/t Milled)	$25.1
Refining & Transport Cost (C$/oz)	$2.5
Royalty NSR	1.0%
Cash Costs (US$/oz Au)*	$723
AISC (US$/oz Au)**	$882
Capital Costs	**LOM Total / Avg.**
Initial Capital (C$M)	$435
Sustaining Capital (C$M)	$283
Closure Costs (C$M)	$49
Salvage Costs (C$M)	$10
Financials – Pre-tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$775
IRR (%)	30.2%
Payback (years)	2.8
Financials – Post-tax	**LOM Total / Avg.**
NPV (5%) (C$M)	$463
IRR (%)	23.2%
Payback (years)	3.5

Notes: * Cash costs consist of mining costs, processing costs, mine-level general & administrative expenses and refining charges and royalties. ** AISC includes cash costs plus sustaining capital, closure cost and salvage value.

Table 22-2: Projected LOM Post-Tax Unlevered Free Cash Flow

	Unit	Total / Avg.	-2	-1	1	2	3	4	5	6	7	8	9	10	11
Production Summary															
Resource Sent to Mill	kt	56,436	--	--	4,800	6,000	6,000	6,000	6,000	6,000	6,000	6,000	6,000	3,636	--
Head Grade (Au Diluted)	g/t	0.91	--	--	0.97	0.98	1.00	0.73	0.90	0.88	0.86	1.02	0.87	0.85	--
Gold Recovered	koz	1,552	--	--	137.7	174.6	181.2	133.8	164.3	161.9	157.2	186.7	160.0	94.6	--
Gold Payable	koz	1,551	--	--	137.7	174.5	181.1	133.7	164.3	161.8	157.1	186.6	159.9	94.6	--
Revenue															
Gold Price	US$/oz	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700	1,700
Exchange Rate	US$:C$	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77	0.77
Gross Revenue	C$M	3,428	--	--	304	386	400	296	363	358	347	412	353	209	--
Operating Costs															
Mine Operating Costs Inc Rehandling Costs	C$M	(898)	--	--	(96)	(99)	(119)	(129)	(119)	(101)	(86)	(58)	(53)	(39)	--
Mill Processing Inc. Water Treatment Costs	C$M	(442)	--	--	(39)	(46)	(46)	(46)	(46)	(46)	(46)	(46)	(46)	(32)	--
G&A Costs	C$M	(78)	--	--	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	(8)	--
Refining and Royalties															
Refining	C$M	(4)	--	--	(0.3)	(0.4)	(0.5)	(0.3)	(0.4)	(0.4)	(0.4)	(0.5)	(0.4)	(0.2)	--
Royalties	C$M	(34)	--	--	(3)	(4)	(4)	(3)	(4)	(4)	(3)	(4)	(4)	(2)	--
Capital Expenditures															
Initial Capital	C$M	(435)	(130)	(304)	--	--	--	--	--	--	--	--	--	--	--
Sustaining Capital	C$M	(283)	--	--	(39)	(56)	(49)	(43)	(37)	(21)	(22)	(9)	(5)	(3)	--
Closure Cost	C$M	(49)	--	--	--	--	--	--	--	--	--	--	--	--	(49)
Salvage Value	C$M	10	--	--	--	--	--	--	1	1	1	1	1	--	6
Pre-Tax Unlevered Free Cash Flow															
Pre-Tax Unlevered Free Cash Flow	C$M	1,214	(130)	(304)	119	172	174	66	151	178	182	287	238	125	(43)
Pre-Tax Cumulative Unlevered Free Cash Flow	C$M		(130)	(435)	(316)	(144)	30	96	247	425	607	894	1,132	1,257	1,214
Taxes															
Unlevered Cash Taxes	C$M	(454)	--	--	(3)	(7)	(43)	(17)	(49)	(57)	(63)	(108)	(88)	(31)	12
Post-Tax Unlevered Free Cash Flow															
Post-Tax Unlevered Free Cash Flow	C$M	760	(130)	(304)	116	165	131	49	102	121	119	179	150	95	(32)
Post-Tax Cumulative Unlevered Free Cash Flow	C$M		(130)	(435)	(319)	(153)	(23)	26	128	249	367	547	697	791	760

22.7 Financial Analysis Summary

A 5% discount rate was applied to the cash flow to derive the NPV for the project on a pre-tax and after-tax basis. Cash flows have been discounted to beginning of construction, June 30, 2024, assuming that the project execution decision will be made and major project financing would be carried out at this time. The project LOM post-tax unlevered free cash flow curve is illustrated in Figure 22-2 above.

22.8 Sensitivity Analysis

A sensitivity analysis was conducted on the base case pre-tax and after-tax 5% NPV and IRR of the project, using the following variables: metal price, foreign exchange rate, discount rate, grade, total capital costs, and operating costs. Table 22-3 summarizes the sensitivity analysis results.

Table 22-3: Pre-Tax Sensitivity Analysis Results

Pre-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.0%	$735	$923	$1,110	$1,298	$1,485
3.0%	$596	$762	$928	$1,094	$1,260
5.0%	$480	$627	$775	$922	$1,070
8.0%	$339	$464	$589	$714	$839
10.0%	$263	$375	$488	$600	$713

Pre-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.0%	21.4%	25.9%	30.2%	34.4%	38.5%
3.0%	21.4%	25.9%	30.2%	34.4%	38.5%
5.0%	21.4%	25.9%	30.2%	34.4%	38.5%
8.0%	21.4%	25.9%	30.2%	34.4%	38.5%
10.0%	21.4%	25.9%	30.2%	34.4%	38.5%

Pre-Tax NPV Sensitivity To FX

FX	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.20	$317	$454	$590	$726	$862
1.25	$399	$540	$682	$824	$966
1.30	$480	$627	$775	$922	$1,070
1.35	$561	$714	$867	$1,021	$1,174
1.40	$642	$801	$960	$1,119	$1,278

Pre-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.20	16.3%	20.7%	24.8%	28.9%	32.8%
1.25	18.9%	23.3%	27.6%	31.7%	35.7%
1.30	21.4%	25.9%	30.2%	34.4%	38.5%
1.35	23.8%	28.4%	32.8%	37.0%	41.2%
1.40	26.2%	30.8%	35.3%	39.6%	43.9%

Pre-Tax NPV Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	$693	$840	$988	$1,136	$1,283
(10.0%)	$586	$734	$881	$1,029	$1,177
--	$480	$627	$775	$922	$1,070
10.0%	$373	$521	$668	$816	$963
20.0%	$267	$414	$562	$709	$857

Pre-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	28.1%	32.4%	36.5%	40.6%	44.5%
(10.0%)	24.8%	29.2%	33.4%	37.5%	41.5%
--	21.4%	25.9%	30.2%	34.4%	38.5%
10.0%	17.9%	22.5%	26.9%	31.2%	35.4%
20.0%	14.4%	19.1%	23.6%	28.0%	32.2%

Pre-Tax NPV Sensitivity To Capex

Total Capex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	$609	$756	$904	$1,051	$1,199
(10.0%)	$544	$692	$839	$987	$1,134
--	$480	$627	$775	$922	$1,070
10.0%	$415	$563	$710	$858	$1,006
20.0%	$351	$499	$646	$794	$941

Pre-Tax IRR Sensitivity To Capex

Total Capex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	29.5%	34.7%	39.8%	44.7%	49.5%
(10.0%)	25.1%	29.9%	34.5%	39.1%	43.5%
--	21.4%	25.9%	30.2%	34.4%	38.5%
10.0%	18.2%	22.5%	26.5%	30.4%	34.2%
20.0%	15.5%	19.5%	23.3%	27.0%	30.6%

Pre-Tax NPV Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(2.0%)	$436	$580	$725	$869	$1,014
(1.0%)	$458	$604	$750	$896	$1,042
--	$480	$627	$775	$922	$1,070
1.0%	$502	$651	$800	$949	$1,098
2.0%	$524	$674	$825	$975	$1,126

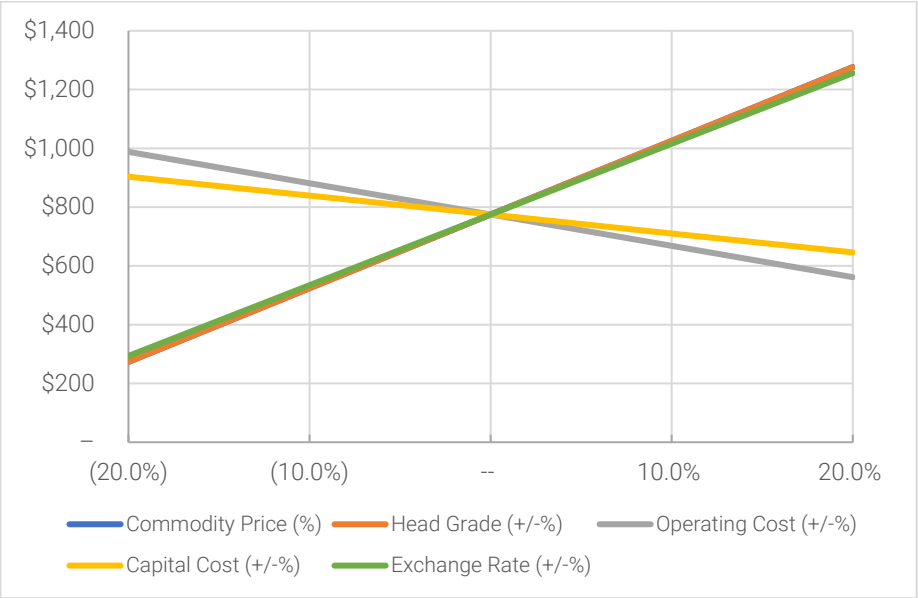
Pre-Tax IRR Sensitivity To Head Grade

Head Grade	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(2.0%)	20.0%	24.5%	28.7%	32.9%	36.9%
(1.0%)	20.7%	25.2%	29.5%	33.6%	37.7%
--	21.4%	25.9%	30.2%	34.4%	38.5%
1.0%	22.1%	26.6%	30.9%	35.1%	39.2%
2.0%	22.7%	27.3%	31.6%	35.9%	40.0%

Figure 22-3, Figure 22-4, and Table 22-4 show the pre-tax sensitivity analysis findings, and Figure 22-5, Figure 22-6, and Table 22-5 show the results post-tax.

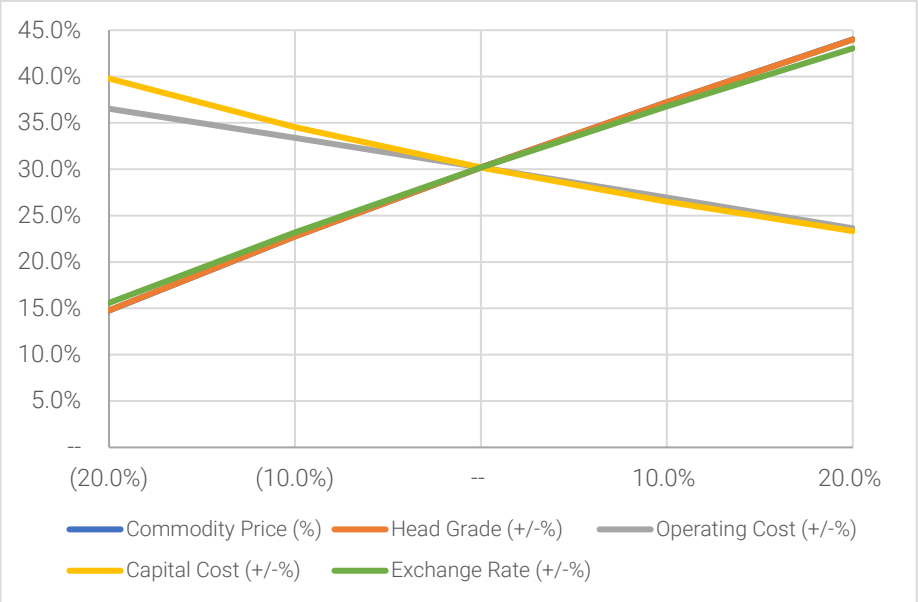
Analysis revealed that the project is most sensitive to changes in metal prices, head grade, and foreign exchange rate, then a lesser extent, to capital costs and operating costs.

Figure 22-3: Pre-Tax NPV 5% Sensitivity Chart



Source: Ausenco, 2022.

Figure 22-4: Pre-Tax IRR Sensitivity Chart



Notes: 1. In Figure 22-3, chart lines for commodity price and head grade overlap. 2. In Figure 22-4, chart lines for commodity price and head grade overlap. Source: Ausenco, 2022.

Table 22-4: Post-Tax Sensitivity Analysis

Post-Tax NPV Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.0%	$462	$577	$690	$802	$912
3.0%	$364	$466	$566	$666	$764
5.0%	$281	$373	$463	$552	$639
8.0%	$181	$260	$336	$412	$487
10.0%	$126	$198	$267	$336	$403

Post-Tax IRR Sensitivity To Discount Rate

Discount Rate	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.0%	16.4%	19.9%	23.2%	26.4%	29.4%
3.0%	16.4%	19.9%	23.2%	26.4%	29.4%
5.0%	16.4%	19.9%	23.2%	26.4%	29.4%
8.0%	16.4%	19.9%	23.2%	26.4%	29.4%
10.0%	16.4%	19.9%	23.2%	26.4%	29.4%

Post-Tax NPV Sensitivity To FX

FX	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.20	$179	$265	$351	$433	$516
1.25	$230	$320	$407	$493	$578
1.30	$281	$373	$463	$552	$639
1.35	$332	$426	$518	$610	$700
1.40	$382	$478	$574	$668	$761

Post-Tax IRR Sensitivity To FX

FX	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
1.20	12.3%	15.8%	19.2%	22.2%	25.3%
1.25	14.4%	18.0%	21.2%	24.4%	27.4%
1.30	16.4%	19.9%	23.2%	26.4%	29.4%
1.35	18.3%	21.8%	25.2%	28.3%	31.4%
1.40	20.2%	23.7%	27.1%	30.2%	33.3%

Post-Tax NPV Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	$412	$501	$588	$676	$762
(10.0%)	$348	$437	$526	$614	$701
--	$281	$373	$463	$552	$639
10.0%	$213	$306	$398	$488	$577
20.0%	$144	$238	$332	$423	$513

Post-Tax IRR Sensitivity To Opex

Opex	Gold Price (US$/oz) $1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	21.6%	24.9%	27.9%	30.9%	33.8%
(10.0%)	19.1%	22.4%	25.6%	28.7%	31.6%
--	16.4%	19.9%	23.2%	26.4%	29.4%
10.0%	13.6%	17.3%	20.8%	24.0%	27.2%
20.0%	10.8%	14.5%	18.1%	21.6%	24.8%

Post-Tax NPV Sensitivity To Capex

Total Capex \ Gold Price (US$/oz)	$1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	$410	$502	$591	$680	$768
(10.0%)	$345	$438	$527	$616	$704
--	$281	$373	$463	$552	$639
10.0%	$217	$309	$398	$487	$575
20.0%	$152	$244	$334	$423	$510

Post-Tax IRR Sensitivity To Capex

Total Capex \ Gold Price (US$/oz)	$1,500	$1,600	$1,700	$1,800	$1,900
(20.0%)	24.8%	28.9%	32.8%	36.6%	40.1%
(10.0%)	20.2%	24.0%	27.5%	31.0%	34.2%
--	16.4%	19.9%	23.2%	26.4%	29.4%
10.0%	13.2%	16.5%	19.6%	22.5%	25.3%
20.0%	10.4%	13.5%	16.4%	19.2%	21.9%

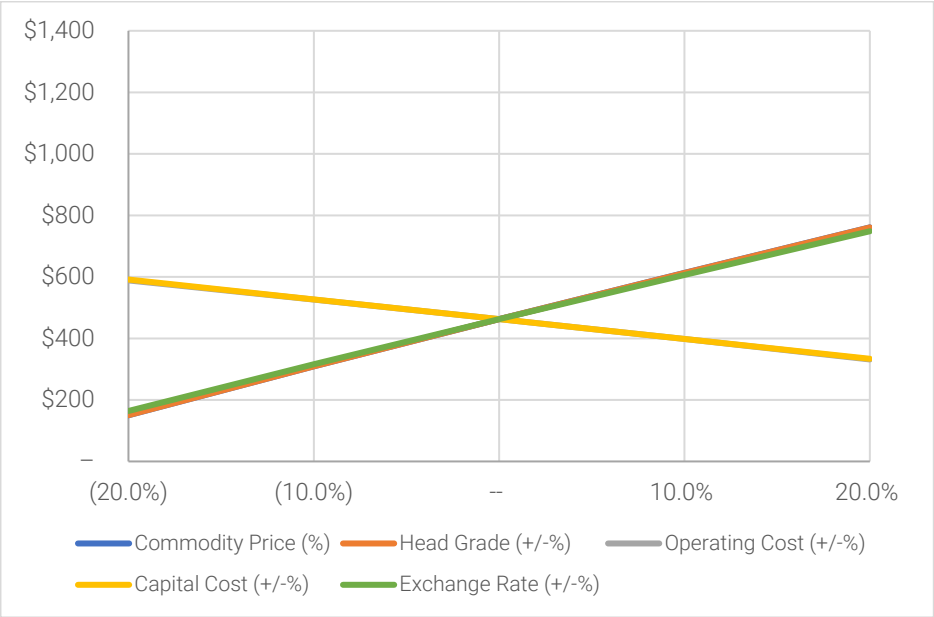
Post-Tax NPV Sensitivity To Head Grade

Head Grade \ Gold Price (US$/oz)	$1,500	$1,600	$1,700	$1,800	$1,900
(2.0%)	$253	$344	$432	$520	$606
(1.0%)	$267	$359	$447	$536	$623
--	$281	$373	$463	$552	$639
1.0%	$295	$387	$478	$567	$656
2.0%	$309	$402	$493	$583	$672

Post-Tax IRR Sensitivity To Head Grade

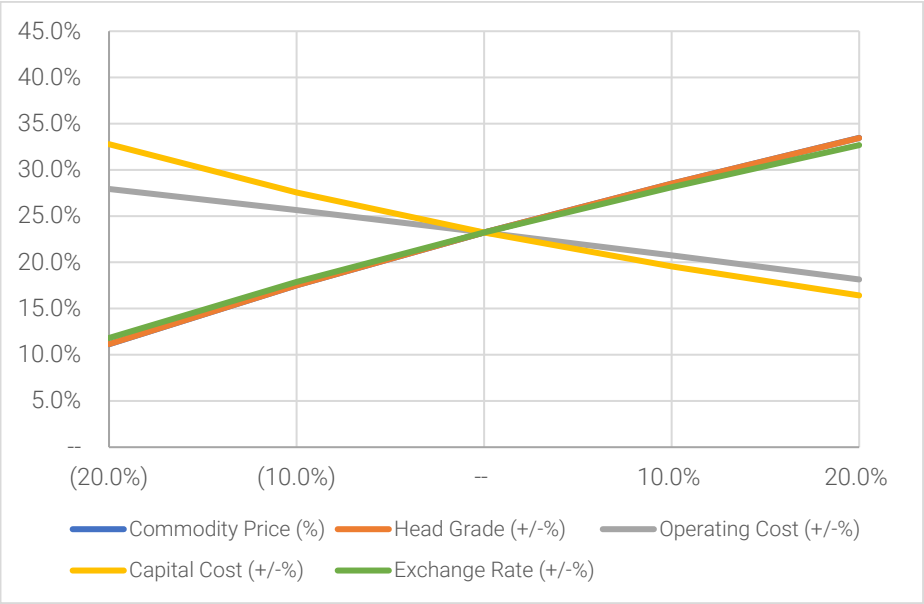
Head Grade \ Gold Price (US$/oz)	$1,500	$1,600	$1,700	$1,800	$1,900
(2.0%)	15.3%	18.8%	22.1%	25.3%	28.3%
(1.0%)	15.9%	19.4%	22.7%	25.9%	28.8%
--	16.4%	19.9%	23.2%	26.4%	29.4%
1.0%	17.0%	20.5%	23.8%	27.0%	30.0%
2.0%	17.5%	21.0%	24.3%	27.5%	30.5%

Figure 22-5: Post-Tax NPV 5% Sensitivity Chart



Source: Ausenco, 2022.

Figure 22-6: Post-Tax IRR Sensitivity Chart

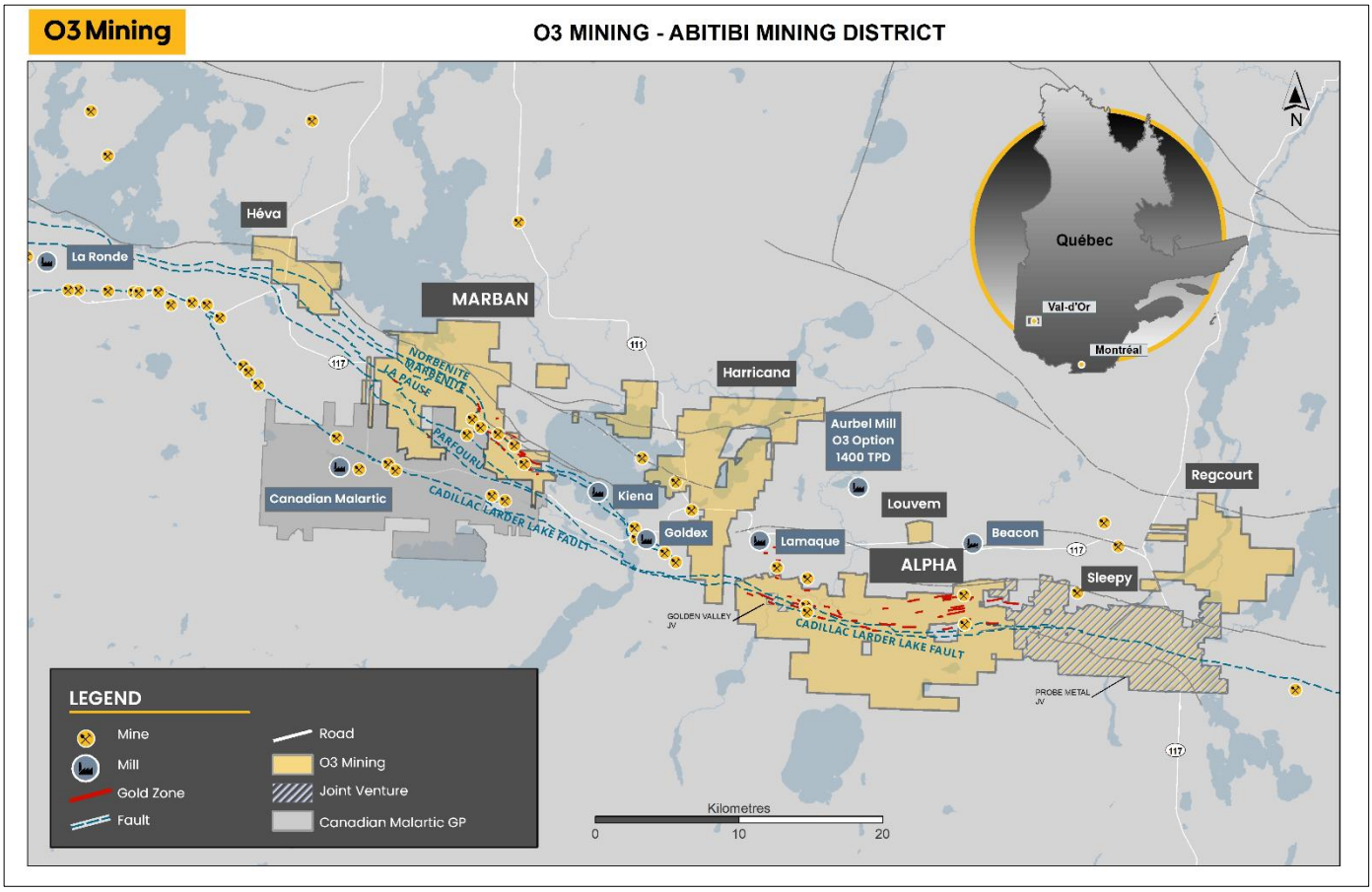


Notes: 1. In Figure 22-5, chart lines for commodity price and head grade overlap, as well as chart lines for capital cost and operating cost. 2. In Figure 22-6, chart lines for commodity price and head grade overlap. Source: Ausenco, 2022.

23 ADJACENT PROPERTIES

O3 Mining controls additional claims adjacent to or near the Marban property. Northwest of the Marban property, the Héva property contains the continuity of the Marbenite and Norbenite shears. To the east, O3 Mining control a stretch of 36 km along the Cadillac Larder Lake Fault with the Harricana property, the Alpha property, a 40% interest in the Sleepy property (Figure 23-1). These properties are not the subject of this report.

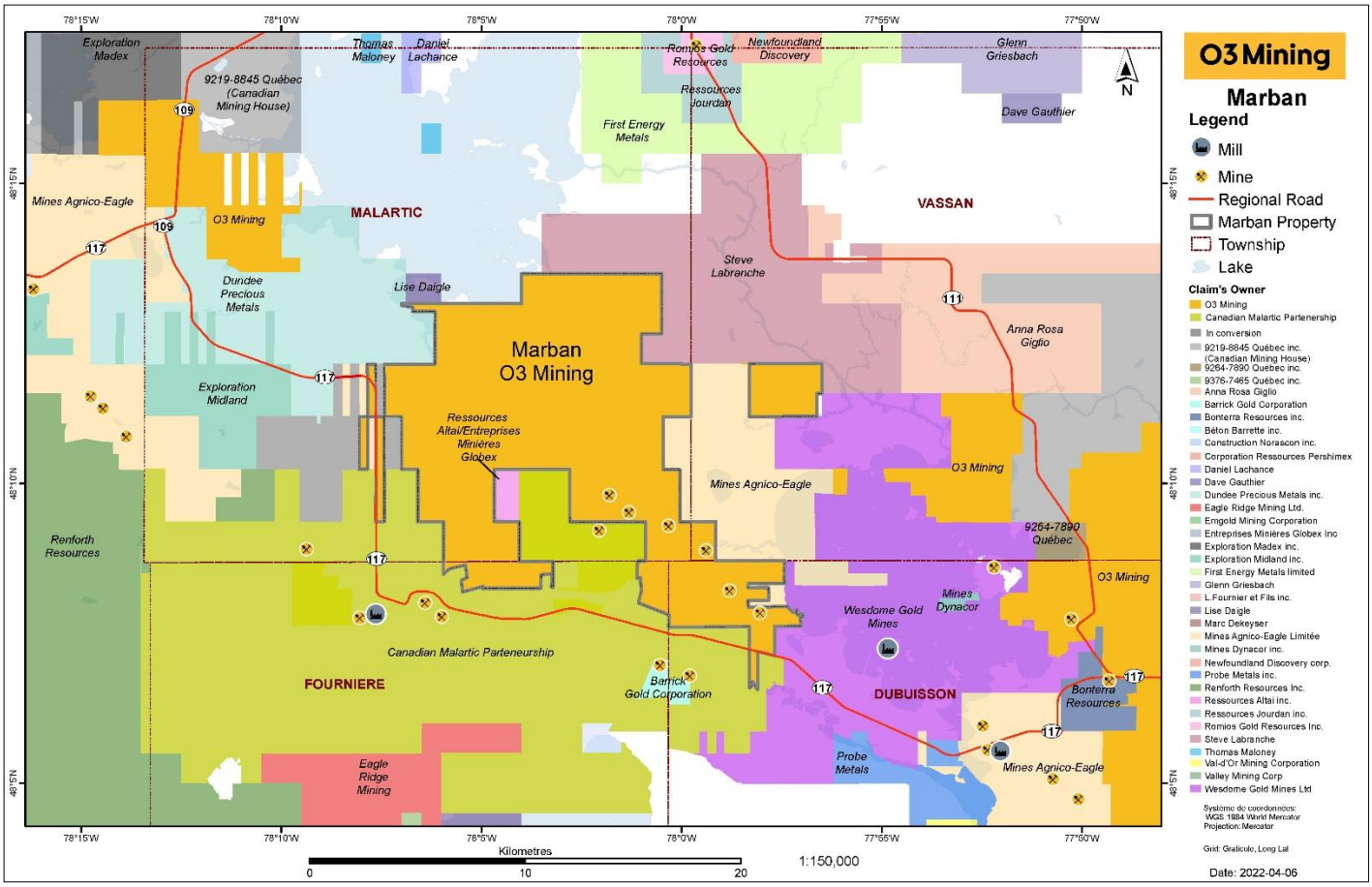
Figure 23-1: O3 Properties



Source: O3 Mining, 2022.

The Marban property is bordered by various exploration properties at different stages of exploration and development (Figure 23-2). Immediately to the south, of the Marban property is the Canadian Malartic mining complex, which is jointly held and operated by the Canadian Malartic partnership, comprising Agnico Eagle Mines Limited and Yamana Gold. The Canadian Malartic Mine produced nearly 800,000 oz Au in 2021 (Agnico Eagle Mines Limited Annual Report for the Fiscal Year December 21, 2021). Approximately 5 km to the east of the Marban Property, Wesdome Gold Mines Ltd ("Wesdome") holds the Kiena gold mine, and for which there is a completed pre-feasibility study filed in May 2021 (Althurion et al., 2021). The QP has been unable to verify the information and that the information is not necessarily indicative of the mineralization on the property that is the subject of the technical report. The northeast and the west are occupied by two Agnico-Eagle's properties. The northwest is bordered by a Dundee Precious Metal property.

Figure 23-2: Adjacent Properties to Marban Property



Source: O3 Mining, 2022.

24 OTHER RELEVANT DATA AND INFORMATION

There is no other relevant data or information.

25 INTERPRETATION AND CONCLUSIONS

25.1 Introduction

The QPs note the following interpretations and conclusions in their respective areas of expertise, based on the review of data available for the technical report.

25.2 Geology and Mineralization

The Marban property is strategically located in the Malartic gold district in the southern Abitibi greenstone belt. The Malartic district is considered to be a highly prospective area with historical production of about 15 million ounces of gold, of which 0.9 Moz has been produced from the Marban Property. The Marban property is located in line with the past producing Kiena and Camflo mines (production of 1.76 and 1.89 million ounces of gold respectively).

The Marban property hosts at least two types of gold mineralization. A first, earlier mineralization, related to the major shears of the property, namely Marbenite, Norbenite, and North shears. Those shears are also related to the Kiena and Goldex deposits. Along those shears, within the Marban property, the mineralization consists of quartz and quartz–carbonate–chlorite veins and veinlets within mafic and intermediate rocks. The thickest quartz veins of this type contain angular fragments of the host rocks. Alteration of the host rock consist of albitization, carbonatization and chloritization, tourmaline is absent. This style of mineralization is common, regardless of the host rocks, to Marban, Norlartic, Kierens, North, and Orion deposits. Sulphide content is generally below 2%, disseminated in the wall rock. At Marban and Orion deposit the veins and veinlets are mainly hosted within an iron-rich basalt and are either transposed along the main schistosity or folded. At Norlartic and Kierens, the veins and veinlets are hosted in intermediate dykes and are less deformed possibility due to a more competent host rock. This mineralization phase can be classified as pre-main deformation event or early orogenic gold deposits.

Another type of gold mineralization is hosted by tonalitic and granodioritic intrusions that cut the early mineralization. The best example is the North-North zones hosted within a tonalitic intrusion. This zone consists of quartz–carbonate–tourmaline veins surrounded by albite alteration halos. They are very similar to the vein systems found at Sigma-Lamaque and Goldex mines. The veins are spatially associated with discrete shear but the veins itself are relatively undeformed. This mineralization event is classified as Sigma-Lamaque, as orogenic gold mineralization.

The Marban Dyke mineralization, located in the northern part of the Marban deposit is a potentially new style of mineralization in the region. It consists of a dissemination of pyrite clusters in strongly sericitized and sheared felsic dykes. The pyrite clusters that correlate with the gold content are relatively undeformed compared to their host rock. The mineralization could be contemporaneous of the quartz–carbonate–tourmaline veins but further studies need to be made to be able to classify this mineralization with a good level of confidence.

25.3 Exploration, Drilling, and Analytical Data Collection in Support of Mineral Resource Estimation

Exploration methods, sampling, sample storage and security and analyses of data are considered appropriate for the resource estimate at the Marban Engineering Project. Historic data that has been used for the resource estimate has been validated through extensive statistical studies, and infill drilling undertaken in 2021 has validated the geological models.

25.4 Metallurgical Testwork

Metallurgical testwork was analysed and several process options were assessed in the initial stages of the prefeasibility study. Analysis included grind size, leach retention time and reagent addition rates. Cyanide detox testing optimized retention time and reagent addition rates. Metallurgical testwork and associated analytical procedures were appropriate to

the mineralisation type, appropriate to investigate the optimal processing routes, and were performed using samples that are typical of the mineralisation styles found within the various mineralized zones.

Samples selected for testing were representative of the various types and styles of mineralisation. Samples were selected from a range of depths within the deposits. Sufficient samples were taken so that tests were performed on sufficient sample mass.

Recovery factors estimated are based on appropriate metallurgical testwork and are appropriate to the mineralisation types and the selected process route. Based on the 2022 preliminary testwork results on composite samples representative of the Marban and Norlartic deposits, doré can be produced at a high recovery of gold. Additional testwork is recommended at the proposed final grind size of 85 µm to refine recovery curves.

25.5 Mineral Resource Estimates

The mineral resource estimate has been completed using CIM standards and best practice guidelines (CIM 2014, 2019). The data used for the resource estimate and methods employed are considered reasonable for this level of study for the Marban project.

25.6 Mineral Reserve and Mining Methods

Mineral reserve estimations were performed by GMS. Optimization runs were carried out only using measured and indicated mineral resources to define the optimal pit limits. Inferred mineral resources were considered waste. The mine design and mineral reserve estimates have been completed to a prefeasibility level with the pits designs following criteria recommended from the pit slope geotechnical study carried out by Ausenco. Optimization parameters used to create the shells are sourced from previous Marban studies as well as current industry benchmarks and best practices. A gold price of C$2,000/oz was used and a mining cost of 2.40 t/mined and a processing cost of C$16.55/t.

The proven and probable ore reserves estimates are at a cut-off of 0.3 g/t as follows:

- Proven Mineral Reserve: 0 kt at 0 g/t for 0 k in-situ gold ounces;
- Probable Mineral Reserve: 56,437 kt at 0.91 g/t for 1,647 k in-situ gold ounces; and
- Total Proven and Probable Reserve: 56,437 kt at 0.91 g/t for 1,647 k in-situ gold ounces.

The mineral reserve includes a mining dilution of 5.4%, including ore loss and loss due to historical voids.

Mining takes place over 12 years with 2 years of pre-production and construction. A peak mining rate of 52.3 Mt is achieved in Year 4 of commercial production. A total 342.6 Mt of material is mined over the project life with an ore and waste split of 56.4 Mt and 286.1 Mt, respectively, for a resulting stripping ratio of 5.07. The mine consists of six pits; the largest (Marban Pit) is split into three nested phases and the second largest (Norlartic) into two nested phases. All other pits contain no phasing (Norlartic Sub Pit 1, Norlartic Sub Pit 2, Kieren Sub Pit 1 and Kieren Sub Pit 2).

Mining is done using conventional open pit mining techniques. Production drilling of the 10 m benches will be carried out by 6.5-inch (165.1 mm) production drills with the capabilities of rotary drilling and down the hole (DTH) drilling. Blast holes are loaded with high energy bulk emulsion. The majority of the loading of the pit will be carried out by three 16 m^3 electric hydraulic shovels, one 12 m^3 diesel electric shovel, and a 10.7 m^3 diesel front end loader. The loading fleet with be augmented with 60 t shovels that will handle overburden material and scaling. The primary loading fleet with service a fleet of 150 t trucks and the secondary overburden loading units wills service a fleet of 100 t trucks. The mine will be owner operated with blasting activities outsourced to an external contractor.

25.7 Recovery Methods

The process plant is designed to process material at a rate of 6.0 Mt/a (16,438 t/d) to produce doré. The flowsheet was selected after trade off studies that included recovery, production ounces, flowsheet robustness and simplicity, operating and capital cost and resultant financial analysis.

The process plant flowsheet designs were based on testwork results and industry-standard practices. The flowsheet was developed for optimum recovery while minimizing capital expenditure and life of mine operating costs. The process methods are conventional to the industry. The comminution and recovery processes are widely used with no significant elements of technological innovation.

25.8 Infrastructure

25.8.1 Geotechnical

A geotechnical program was performed in 2021 including test pits and boreholes along with the collection of soil and rock samples, laboratory testing and geotechnical surface mapping and geophysical investigations to understand the foundation conditions for the site infrastructure. The information gathered was used to develop the designs for the site infrastructure.

25.8.2 Tailings Storage Facilities

The Marban Engineering Project has two tailings storage facilities that will store approximately 56.4 million tonnes over a 10-year mine life. The initial tailings storage facility (TSF 1) is a traditional ring dike configuration since there is very little topographic relief with a storage capacity of 19.3 Mt. Norlartic open pit (TSF 2) becomes available in Year 4.5 for tailings in-pit disposal with a storage capacity of 37.1 Mt.

TSF 1 consists of a ring dike shell constructed with waste rock, the interior consists of filter zone against the waste rock, then a low permeability layer and finally a geomembrane liner to prevent seepage through the embankment. The facility will be constructed in three phases over the life of the facility to a maximum height of approximately 24 metres. In addition, in the final phase a stability berm will be constructed around the outer perimeter of the embankment. TSF1 has a total storage capacity of 19.3 Mt and has been designed in accordance with CDA (2013, 2019) guidelines. Tailings will be slurried from the process plant to the TSF 1 by way of a pipeline and spigots around the facility discharging tailings and a water reclaim system will pump decant water back to the process plant for reuse.

TSF 2, in-pit tailings disposal, become available in Year 4.5. Tailings will be slurried from the process plant to the TSF 2 by way of a pipeline and spigots around the facility discharging tailings and a water reclaim system will pump decant water back to the process plant for reuse. TSF 2 has a storage capacity of 37.1 Mt.

25.9 Environmental, Permitting, and Social Considerations

The Marban Engineering Project is subject to the provincial and federal environmental impact assessment procedure as forecasted daily production is over the thresholds outlined in the applicable regulation.

Several environmental baseline studies have been completed in 2021 and 2022 to complete those of 2016. As the project continues to be defined, and in regard of the EIA procedure requirements, additional baseline data collection and assessment will be necessary.

Current project definition is sufficient to provide a basis upon which most anticipated environmental and social impacts can be identified. The main impacts anticipated relate to wetlands, fish habitat, air quality, noise/vibrations, visual landscape and water resources. No specific inordinate environmental risk to project development was identified. Although there are some sensitive elements in the proposed layout surroundings, optimization could be made to eliminate or reduce the effect

on these components. Consultation with stakeholders, specifically First Nations communities, may highlight additional impacts.

Based on the static test results from 2020 and the kinetic test results from 2021 on composite samples, waste rock and ore are classified as leachable and would not produce long-term ARD under the conditions of the column tests. Both materials are low in sulfur and have a good neutralization potential. The management facilities will have to comply with Directive 019 groundwater protection measure. Additional geochemistry testing and contaminant dispersion modelling are required to ensure that the storage design meets the requirements of Directive 019 in terms of groundwater protection.

A comprehensive plan to engage with stakeholders has been implemented. Throughout 2021 and 2022, several meetings with all levels of government, residents, First Nations, suppliers, para-governmental bodies, employees, etc. were held. Consultations and engagement activities will continue throughout the project development.

25.10 Markets and Contracts

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 of similar contracts for the sale of doré throughout the world. There are many markets in the world where gold is bought and sold, and it is not difficult to obtain a market price at any particular time. The gold market is very liquid with a large number of buyers and sellers active at any given time.

O3 Mining plans to contract out the transportation, security, insurance, and refining of doré gold bars. O3 Mining may enter into contracts for forward sales of gold or other similar contracts under terms and conditions that would be typical of, and consistent with, normal practices within the industry in Canada and in countries throughout the world. For the PFS, a cost of C$2.5/oz Au was assumed for transportation, and refining.

25.11 Capital Cost Estimates

The total initial capital cost for the Marban Engineering Project is C$435M and LOM sustaining costs are C$283M. Closure costs are estimated at C$48.9M. As a total, 70% of all project costs were derived from first principles, with equipment quotation or contract supply/installation.

25.12 Operating Cost Estimates

The overall LOM operating cost for the Marban Engineering Project is $1,419.0 million over 10 years, or an average of $25.15/t of ore milled in a typical year. Of this total, processing and G&A account for C$520.5 million and mining accounts for C$898.5 million.

25.13 Economic Analysis

An engineering economic model was developed to estimate annual pre-tax and post-tax cash flows and sensitivities of the project. It must be noted, however, that tax estimates involve many complex variables that can only be accurately calculated during operations and, as such, the after-tax results are only approximations. Sensitivity analysis was performed to assess the impact of variations in metal prices, head grades, foreign exchange, operating costs, and capital costs.

The economic analysis was performed assuming a 5% discount rate. The pre-tax net present value discounted at 5% (NPV 5%) is C$775M, the internal rate of return IRR is 30.2%, and the payback period is 2.8 years. On an after-tax basis, the NPV 5% is C$463M, the IRR is 23.2%, and the payback period is 3.5 years.

A sensitivity analysis was conducted on the base case pre-tax and after-tax 5% NPV and IRR of the project, using the following variables: metal price, foreign exchange rate, discount rate, grade, total capital costs, and operating costs.

Analysis revealed that the project is most sensitive to changes in metal prices, head grade, and foreign exchange rate, then a lesser extent, to capital costs and operating costs.

25.14 Risk and Opportunities

25.14.1 Risks

25.14.1.1 Geology and Resource Modelling

The following factors could affect the resource estimate:

- Continuity and orientation of gold mineralization. The deposit has been drilled sufficiently to support indicated category, however there will always be an element of risk pertaining to continuity of high grades at scales smaller than the current drill spacing. The orientation of mineralisation is generally well understood, however in there appears to be greater geological complexity in the fold noses of the Marban deposit that could pose issues during mining
- Historic underground openings location. All known stopes have been accounted for in the mineral resource, however the precise location of these stopes is yet to be determined as the underground openings are currently flooded. This could also impact the mineral reserve.

25.14.1.2 Mining

The following factors could affect the mining costs or methodologies:

- Manpower availability in the region and across Canada could vary drastically depending on the current economic conditions in Canada and in the local area. Not only could salaries be altered, but also the ability to achieve the workforce required to maintain production.
- Many contractor teams available in the region may not have capacity for additional work if not given enough of a lead time.
- Market and logistic conditions, due to COVID, have made acquiring certain equipment and materials difficult. Some major equipment might not be available at all. Expect additional lead times for all critical supplies.
- Construction and production delays due to sickness and outbreak of disease it possible. Ensure proper fail safes and allowances in project timelines.
- Unmapped/unstable historical underground voids require additional safety and mining precautions to be mined efficiently and safely. There is a potential loss of productivity when working in these areas.

25.14.1.3 Processing

The following risks are noted with regards to the Process Plant:

- The metallurgical testwork program associated with the design of the leach-adsorption circuit was conducted primarily at grind sizes between 105 and 108 µm, whereas the Process Plant will be operating at a P80 of 85 µm. An additional 1.5% is added to the recovery to account for the adjustment in grind size, but there is a risk associated with the leach circuit reagent addition rates selected and the leach recovery models assumed based on the metallurgical testwork.
- The use of filtered raw water from the Marban pit for process plant raw water presents a risk of equipment scaling or damage, if the water quality deviates from the test data available. This risk will have to be further evaluated in future studies, or an alternate source of raw water found.

25.14.1.4 Tailings Storage Facilities

The following risks are noted with regards to the general site infrastructure:

- If additional resource is discovered in Norlartic deposit, then TSF 2 may not be available for tailings disposal in Year 4.5. There is the ability to raise TSF 1, which would add sustaining capital to the project.
- The underling soils in TSF 1 have a higher permeability then model in the seepage analysis and requires a geomembrane liner along the floor to meet Quebec guidelines for maximum seepage rates from a tailings facility, which would add capital cost to the facility.

25.14.1.5 Water Management

The following risks are noted with regards to site water management:

- For waste rock and ore contact water, no chemical water treatment has been included in the PFS design based on the geochemical testwork data available, which does not indicate the likelihood of significant acid generation/metal leaching or exceedances of regulatory guidelines. However, the available data are limited and further testwork is required to confirm this assumption as part of the FS and future phases of the project. Further testwork may indicate additional capital expenditure requirements for water treatment.
- For the tailings facility, there is a general water deficit; therefore, at this time, it is not anticipated that there will be any regular discharge from the facility. Consequently, a treatment facility is not required for surplus water from the tailings facility based on the current available information. Water balance predictions will be better refined during feasibility level design to take into consideration the potential for upset conditions caused by seasonal and extreme weather events as well as taking into consideration the dam raise construction schedule and available freeboard over time. The requirement for occasional treatment of tailings decant water due to the above factors will be revisited and re-assessed as part of the FS and future phases of the project. Refinement of the water balance and tailings dam construction schedule may indicate the requirement for occasional treatment, requiring additional capital expenditures.
- At the decision of O3 Mining, the Keriens Creek diversion channel was sized for the 100-year flood event. The channel would be overtopped in the case of a 200-year or bigger flood event, which could result in the flooding of the Kierens and Norlartic pits and consequential risk to personnel and equipment. Appropriate monitoring and safety mitigations will have to be implemented during detailed design and operations.
- The bridge crossing over the Keriens Creek is significantly undersized, which could cause ponding/flooding upstream of the crossing during relatively major storm events. A detailed hydraulic analysis of the bridge crossing is required as part of the FS study to quantify the existing capacity and to propose a bridge design upgrade according to the local guidelines.

25.14.1.6 Environment and Permitting

The following risks are noted with regards to environment and permitting:

- A residual materials landfill is located within the proposed layout. The landfill may present certain environmental (surface runoff or underground contamination) and public health (gases and vapours) risks. Production activities were carried out at three old mines sites located within the proposed layout, and leachable waste rock backfill, and contaminated soil might still be present although the sites were cleaned. Additionally, some waste materials and potentially hazardous materials were found on private property located within the proposed layout. O3 Mining is currently evaluating these sites and performing additional soil characterization on potentially contaminated lands to evaluate the environmental liability.

- A preliminary evaluation estimates that 226 ha of wetlands and 31 ha of shorelines will be directly impacted by the project and will require authorization and monetary compensation.
- To mine the Norlartic and Kierens pitts, the current Keriens Creek will need to be diverted. The diversion will need to be done with minimum alterations, disruption, or destruction of the fish habitat. The Department of Fisheries and Oceans Canada will require compensation of the fish habitat.
- According to available data and preliminary assumptions, the waste rock storage area is located on one intermittent water course that is a confirmed fish habitat. The encroachment of this water course by the waste rock storage area is considered as a risk because an amendment to the Metal and Diamond Mining Effluent Regulations (MDMER; federal legislative action) will be required.
- Due to habitation proximity to the project's location, emissions of particles (total and fine) and gas, which can cause air quality deterioration and/or health concerns, are a risk. Similarly, noise and vibration from blasting could cause nuisances and damage to infrastructure.
- The predicted underground water table drawdown will possibly have an impact on residential wells located south of the project. Additional investigation will be completed in order to determine the actual condition of the residential wells

25.14.2 Opportunities

25.14.2.1 Exploration

Exploration activities are likely to identify additional mineralization, which could provide additional resources within the known shear zones as well as in additional unknown structures. These efforts could result in changes to the style of mineralization to that currently identified, the scale of the project, and the deleterious elemental issues identified.

25.14.2.2 Resource Modelling

Additional re-assaying of historic assays (pre-1980) would increase the data density and therefore the Classification of the mineralization within areas of dense historic drilling, particularly at Kierens-Norlartic.

Near-mine exploration, especially in the Marban Dyke area could add additional ounces to future mine plans

25.14.2.3 Mining

A fleet of two trucks is currently being used to best control overburden and maintain production. More analysis can be completed into better optimizing the fleet to target smaller pits with smaller trucks allowing a redesign of the ramp layouts. This could potentially lead to more aggressive sub pits.

Additional scenarios of pit electrification, trolley systems and autonomous equipment can be looked into in future studies to save costs on fuel, manpower and carbon taxes.

25.14.2.4 Processing

Further cyanide leach testing of Nolartic variability ore samples will refine the recovery forecasts made in the PFS, and may be increased if performance is observed to align to the Marban samples.

25.14.2.5 Tailings Storage Facilities

Conservative foundation conditions were assumed in the area of TSF 1. Additional geotechnical investigation in this area may show better foundation conditions, which could reduce sustaining capital costs by the elimination or reduction of the stability buttress.

25.15 Conclusions

Based on the assumptions and parameters presented in the report, the project has a mine plan that is technically feasible and economically viable. The positive financials of the project ($463 M after-tax NPV5% and 23.2% after-tax IRR) support the mineral reserve.

26 RECOMMENDATIONS

26.1 Summary

The following sections detail recommended future work for the project. The estimated costs are summarized in Table 26-1. Advancing to Phase 2 of the drilling program is not contingent on positive results from Phase 1.

Table 26-1: Recommendation Cost Summary

Area	Estimated Cost (C$M)
Drilling (Phase 1)	6.4
Drilling (Phase 2)	3.2
Sampling / QA/QC	0.0
Mining	0.9
Open Pit Geotechnical	0.8
Metallurgy	0.2
Hydrological	0.2
Geochemistry and Water Management	Included in Environment
Geotechnical	0.5
Tailings	0.3
Environment	1.8
Total	14.3

26.2 Drilling

The recommended drilling can be divided into two programs. A first program aiming to expand and convert into the indicated category all lateral extensions of the near surface mineralization of Marban Engineering. First focus will be on the remaining inferred resources within the pit shells. Then, lateral extensions of the Norlartic zone in the area north of the Marban pit and those of the North and North-North zones are of particular interest. Mineralized zones not considered in this MRE that show near surface economic potential also need follow-up drilling. These zones are Triple North, Gold Hawk, Orion, Malartic Hygrade and Marlartic.

A second program will consist of target testing elsewhere in the Marban project. Two areas are highly prospective. One to the northwest of the Malartic Hygrade zone where the mafic volcanic rocks are folded in a regional structure. The second is the down dip extension of the Marban deposit where only 14 holes reached a vertical depth of 700 m. This program can be considered as exploration drilling, and is not contingent on positive results from the first campaign which is considered resource delineation drilling.

Advancing to Phase 2 is not contingent on positive results from Phase 1. Table 26-2 below summarizes the expected drill expenditures for the next two phases of drilling.

Table 26-2: Drilling Budget

Phase	Description	Metres	Budget (C$M)
1	Expansion of all near surface zones and conversion to indicated category	30,000	6.4
2	Exploration for new zones and UG down-dip extension	15,000	3.2

26.3 Sampling and QA/QC

GMS recommends that additional density measurements are gathered in the mineralized zones to be able to compare with the current density measurements taken mostly in barren lithologies. GMS does not expect any significant differences, as the orebodies are not strongly mineralized with sulphides, however silica flooding can occasionally reduce the bulk density of mineralization.

O3 should include the routine collection of density measurements from drill core to build a more robust database for use in mineral resource estimation. Costs are considered minimal (less than $3.00 per sample) and not material for this section.

26.4 Mining

During the course of the study, items were identified as requiring additional information to further improve precision and information as part of the FS.

- Detailed planning for earthworks and deforestations should detail the tasks performed by owner and contractor fleets to better optimize upfront capital and equipment utilization. Potential overlap can be eliminated and key equipment that would be better owned by the mine can be identified to save costs. The approximate cost of the detailed trade off is $50,000.
- In depth review of the ground water conditions and dewatering plan must take place to ensure that the project is resistant to lifetime rainfall and flooding events. In addition, thorough environmental testing of the ground water should be done to ensure that water sourced from historical stopes are properly treated. The approximate cost of detailed groundwater studies is estimated at $500,000.
- Additional LOM plans can be assessed to explore additional mining options. Smaller pits can be redesigned for a smaller contractor fleet, allowing a larger owner mining fleet for the larger pits. Additional trade-offs between contractor and owner fleets can be assessed. The approximate cost of the additional trade offs is $80,000.
- Additional work on blast rock movements can be done to capture the void loss ore in the models. This material can be properly recovered with better understanding of the post blast movements and underground void structures. Careful work on initial void zones with blast balls and reconciliation programs can potentially recover this ore in the model. The approximate cost of the additional testing is $250,000.

26.5 Open Pit Geotechnical

Ausenco recommends the items in this section to improve information needed for the pit slope designs.

26.5.1 Geotechnical Drilling

Based on the PFS geotechnical drilling, Ausenco recommends drilling an additional nine geomechanical boreholes for a total depth of 2,160 m to support the next phase of work. The program would comprise one borehole in Kierens, four boreholes in Norlartic, and four boreholes in Marban. It is estimated that two to three of the nine proposed geotechnical holes will intersect the faults through Norlartic and Marban. All holes are to be televiewer (oriented core) logged and core should be geotechnically logged at the drill sites. Strength profiling by point load testing should also be continued, as per the current practice at the project. The televiewer (or oriented core) logging of geotechnical holes associated standby costs are estimated to cost approximately $80,000. The estimated cost for drilling and core logging is approximately $550,000.

26.5.2 Surficial Mapping

It is recommended that surface mapping be performed for each of the pits. The information should be used to refine the overburden thickness maps and to update engineering properties of the rock based on laboratory testing. Geophysical

surveys should also be considered to supplement the overburden thickness database. The estimated cost for surficial mapping is approximately $20,000.

26.5.3 Laboratory Testing

Laboratory testing, including moisture content, gradation, triaxial strength, Atterberg limits, and density, should be performed to classify the clays. The site-specific relationship between point load strength and unconfined compressive strength should be refined and shear strength tests should be conducted to determine the basic friction angle for major rock types. The estimated cost for the laboratory testing is approximately $15,000.

26.5.4 Structural Modelling

The structural model should be updated to incorporate all new drilling available at the time the FS is completed. Borehole fault intersections should be traced for continuity to determine if structural control is present in the pit walls of either pit. The cost for updating the model is approximately $40,000. The cost for reviewing the model is estimated to be approximately $10,000.

26.5.5 Rock Engineering Analyses

After the new data is available, the rock engineering analysis should include:

- Update the structural fabric database to incorporate all pit areas. Evaluate the data for spatial variation and/or lithologic correlations.
- Update the rock mass geomechanical parameters and revise design values, as required.
- Update overburden geomechanical parameters
- Evaluate hazard posed by upslope overburden deposits relative to pit crests and design conceptual mitigation measures accordingly.
- Perform slope stability analyses to optimize design for each wall for all pits.
- Look at benefits of depressurizing pit slope.

The estimated cost for the rock engineering analysis, is approximately $85,000.

26.6 Metallurgy

Sample selection for future mining studies should reflect mineralization that would be treated in the first five years of the mine life. Variability samples are required to understand the responses of the various mineralized zones to grind size, leach kinetics and contaminant correlations.

Additional comminution tests (e.g., SMC, Bond ball work index, and abrasion index) are recommended on samples representative of the first years of the planned operation to provide more confidence in equipment selection and to ensure that there is sufficient comminution information that is spatially representative of the variability within the various mineralized zones.

The flowsheet selected for the PFS should be validated by selecting a composite sample representative of Year 1 or Year 1.5. This composite sample should undergo gravity-leach testwork, and the tailings should complete cyanide detoxification optimization testwork and vendor thickener tests.

The estimated cost for the recommended metallurgical testing programs is $190,000.

26.7 Hydrological

It is recommended to plan a water sourcing schedule during different years of the mine life. Pumping and linear water facilities should be designed to source water from the pits and collection ponds on a seasonal basis, as well as during different years.

The following studies are recommended to be considered for the feasibility study:

- Review an option to increase the upstream invert elevation of the diversion channel to possibly reduce the excavation quantity. This option would require environmental review to make sure the upstream ponding and shallow water level in the upstream section of the channel does not impact the aquatic habitat.
- Explore modification to the bridge crossing in the downstream, which currently acts as a constriction within the Keriens Creek. Widening the bridge crossing would help lowering the water level and subsequently reduce the inundation extent upstream.
- A more comprehensive hydrological studies including rainfall-runoff modelling and which may improve water management facility design, and reduce estimations of contact water containment volume, excavation, and excess water rates.
- A stochastic hydrologic modelling to account for uncertainties embedded with weather components and other water inflow/loss components.
- A more detailed groundwater modelling to provide annually varying estimates of groundwater inflow to the pits.
- The cost for the above studies is estimated at $180,000.

26.8 Geochemistry and Water Management

No chemical water treatment has been included in the PFS design based on the geochemical testwork data available, which does not indicate any significant acid generation or exceedances of regulatory guidelines.

However, the available data is very limited in nature. Further testwork is required for the Feasibility Study as follows:

- Additional static test work on a range of samples encompassing all potential geological and mineralogical heterogeneities within the ore and waste rock of the mineral reserve;
- Further kinetic testwork requirements will be defined based on the static test results; and
- Metallurgical test work completed on the ore (planned for H2 2022) can generate process water samples and coarse/fine fractions of tailings that will allow for further test work and characterization/prediction of potential tailings behaviour and water quality.
- The costs for the above work are included in the geochemical characterization in Section 26.10.

26.9 Geotechnical

The following recommendations are made for geotechnical site investigation work.

- Completion of 31 (1,240 m) geotechnical boreholes, 58 test pits, and 5 geophysics in the areas of the TSFs, WRSFs, plant site, haul roads, and ancillary roads to investigate and confirm foundation conditions, specifically the extent of the overburden along with depth to groundwater and to bedrock.

- Additional laboratory index testing, including compaction tests, mechanical strength tests, and permeability tests on foundation soils and potential borrow materials.
- Laboratory testing to confirm the physical characteristics of bedrock.
- Using new data, recommend designs for foundations, borrow sources, construction materials for infrastructure, and tailings and waste rock storage facility exterior slope configurations.
- Perform deterministic and probabilistic local seismic hazard study for the development of design seismic values for infrastructure.

The estimated cost for the fieldwork and lab testing is approximately $450,000 including the drilling and excavator rental. The estimated cost for the seismic analysis is approximately $50,000.

26.10 Tailings Storage Facility

The following work is recommended to support a FS:

- Review and update meteorological and hydrology information, updating surface water and sediment management for the tailings storage facilities.
- Confirm geochemical characterization of tailings and waste rock from additional waste characterization studies.
- Develop seepage predictions and seepage control measures for the TSFs.
- Update the tailings and waste rock deposition strategy to optimize material handling for tailings facilities.
- The stability model should be reviewed and updated, as required, with consideration of the final deposition plan using updated data about the material properties of the waste rock, tailings, and the foundations for both the TSF 1.
- Perform a liquefaction assessment with consideration of updated information on material properties and the updated stacking plan for the TSF 1.

The estimated cost for the recommended testwork is approximately $ 275,000.

26.11 Environment, Permitting, and Community Relations

Recommendations for environment, permitting and community relations to better define the environmental and social impacts are as follows:

- Finalize the surface land acquisition protocol.
- Continue with the negotiation and agreements with private and public stakeholders, including First Nations.
- Continue with the consultations and engagement activities, addressing and documenting concerns.
- Continue with the government involvement to ensure that their expectations are addressed throughout the project development.
- Initiate provincial and federal environmental assessment processes in order to receive the Tailored Impact Statement Guidelines.
- Initiate an exhaustive geochemical characterization to support FS and EIA. The estimated cost is $1,500,000.

- Complete a contaminant dispersion modelling for the waste rock piles, TSF 1, and the tailings in pit storage (groundwater). The estimated cost is $75,000.
- Complete the inventory and the baseline study on private wells in the area with potential impacts. The estimated cost is $160,000.
- Continue soil environmental assessment on potentially contaminated lands. The estimated cost is $75,000.

27 REFERENCES

The following information and references were available and considered during preparation of this report:

27.1 Previous NI 43-101 Technical Reports

Belzile, Elzéar, 2016. Updated Mineral Resource Technical Report on the Marban Block Property, Quebec, Canada. Report prepared by Belzile Solutions Inc. for Osisko Mining Inc.

Gustin, Michael M., and Ronning, Peter, 2013. Updated Mineral Resource Technical Report on the Marban Block Property, Quebec, Canada. Report prepared by Mine Development Associates for NioGold Mining Corporation

27.2 Geology and Resource Estimate

Camus, Y., Dupéré, M., and Rousseau, G., 2012 (October 22). NI 43-101 Technical Report mineral resource update for the Marban block property, Abitibi, Qc, Canada. Report prepared for Aurizon Mines Ltd. and NioGold Mining Corp. by SGS Canada Inc., 157 p.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2019. Estimation of Mineral Resources and Mineral Reserves, Best Practice Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, November 29, 2019.

Canadian Institute of Mining, Metallurgy and Petroleum (CIM), 2014: CIM Standards for Mineral Resources and Mineral Reserves, Definitions and Guidelines: Canadian Institute of Mining, Metallurgy and Petroleum, May 2014

Card, K.D. and Ciesielski, A.,1986. DNAG Subdivisions of the Superior Province of the Canadian Shield: Geoscience Canada, v. 13, p. 5-13.

Carrier, A., 2006. 43-101 Technical Report on the Marban Property. Technical report prepared for NioGold Mining Corporation by InnovExplo Inc., August 16, 2002. 99 p.

Chown E. H., Daigneault, R., Mueller, W., and Mortensen, J., 1992. Tectonic Evolution of the Northern Volcanic Zone, Abitibi belt, Quebec: Can. Jour. Earth Sci., v. 29, p. 2211-2225.

Desrochers, J. P., and Hubert, C., 1996. Structural Evolution and Early Accretion of the Archean Malartic Composite Block, Southern Abitibi Greenstone belt, Quebec, Canada: Can. Jour. Earth Sci., v. 33, p. 1556-1569.

Desrochers, J. P., Hubert, C., Ludden, J. and Pilote, P., 1993. Accretion of Archean Oceanic Plateau Fragments in the Abitibi Greenstone Belt, Canada: Geology, v. 21, p. 451-454.

Dubé, Benoît & Gosselin, Patrice. (2007). Greenstone-hosted quartz-carbonate vein deposits. Mineral Deposits of Canada. 5.

Ducharme, Y., 2008. NioGold Mining Corporation memo on the North-North Zone: Internal memo for NioGold Mining Corp., 2 p.

Ducharme, Y., Gagnon-Lamothe, B., and Pelletier, M., 2009. Activity Report for 2007 and 2008 - Marban Block and Malartic Hydgrade Properties. Internal report for NioGold Mining Corp., March 2009, 54 p.

Feng, R., Kerrich, R., McBride, S., and Farrar, E., 1992. 40Ar/39Ar Age Constraints on the Thermal History of the Archean Abitibi Greenstone Belt and the Pontiac Subprovince: Implications for Terrane Collision, Differential Uplift, and Overprinting of Gold Deposits: Can. Jour. Earth Sci., v. 29, p. 1389-1411.

Imreh, L., 1984. *Sillon de La Motte-Vassan et son avant-pays méridional, in Synthèse volcanologique, lithostratigraphique et gîtologique*. Ministère de l'Énergie et des Ressources du Québec, MM 82-04, 72 p.

Jackson, S. L. and Fyon, J. A., 1991. The Western Abitibi Subprovince in Ontario, in Thurston, P.C., Williams, H.R., Sutcliffe, R.H. & Stott, G.M. (eds), Geology of Ontario: Ontario Geological Survey Special Volume 4, Part I, p. 405—82.

Latulippe, M., 1976. *La région de Val-d'Or – Malartic dans le Nord-Ouest québécois*. Ministère des Richesses Naturelles, excursion géologique Val-d'Or – Malartic, DP-367, p. 3-28.

Lefrançois, R., (October 18) 2006. Aur Resources Inc. Option Norlartic/First Canadian/Marban Properties Quarterly Progress Report, Third Quarter 2006. Internal report for NioGold Mining Corporation, 15 p.

Lefrançois, R., 2007. Aur Resources Inc. Option Norlartic/First Canadian/Marban properties; Thundermin Resources Inc. Option Gold Hawk Property; Quarterly Progress Report, Fourth Quarter 2006. Internal report for NioGold Mining Corp., February 27, 2002, 16 p.

Ludden and Hubert, 1 Ludden, J. and Hubert, C., 1986. Geologic Evolution of the late Archean Abitibi Greenstone Belt of Canada, Geology, v. 14, p. 707-711.

Ludden, J. et al., 1986. The Tectonic Evolution of the Abitibi Greenstone Belt of Canada: Geological Magazine, v. 123, no. 2, p. 153-166.986.

Mueller, W., Daigneault, R., Mortensen, J., and Chown, E. H., 1996. Archean Terrane Docking: Upper Crust Collision Tectonics, Abitibi Greenstone Belt, Quebec, Canada. Tectonophysics, v. 265, p. 127-150.

Poulsen, K. H., Robert, F. and Dubé, B., Geological classification of Canadian gold deposits, Geological Survey of Canada, Bulletin 540, 2000.

Renou, A. S., 2006. *Étude pétrographique, association minéralogique de l'or, projet Norlartic, zone North-North*. Report prepared by Modèlaur for NioGold Mining Corp., 24 p.

Renou, A. S., 2007. *Étude pétrographique, association minéralogique de l'or et alterations, project Marban*. Report prepared by Modèlaur for NioGold Mining Corp.

Rocheleau, M., Gaudreau R., Lacoste P., Hébert R., St-Julien, P., Perrier, B., 1987. *Synthèse stratigraphique, paléographique et gîtologique du secteur de Vauquelin, de Pershing et de Haig*. Ministère de l'Énergie et des Ressources du Québec, MB 87-52, 153 p.

Stuart, E. and Martin, L., 1986. Report on the Winter 1985-86 Diamond Drilling Program, Kierens Zone, First Canadian Property, Project 1902, Malartic Township, Quebec, 32D/1: Internal report of Aur Resources Inc., 44 p. plus appendices and pocket plans.

Stuart, E. and Martin, L., 1988. Report on the Winter 1988 Surface Diamond Drilling Program on the Gold Hawk Horizon & North Zone, Malartic property, Project 2001, Vassan Township, Quebec: Internal report of Aur Resources Inc., 39 p. plus appendices and pocket plans.

Thurston, P. C. and Chivers, K. M., 1990. Secular Variation in Greenstone Sequence Development Emphasizing Superior Province, Canada: Precambrian Research, v. 46, p. 21-58.

27.3 Geological Setting and Mineralisation

Beauchamp, C., 2010. *Origine métasomatique et contrôle structural de la minéralisation aurifère du secteur minier de Marban, canton de Dubuisson, Val d'Or, Abitibi, Québec: Thèse M.Sc., Université du Québec à Montréal, 83 p*.

Bertrand-Blanchette, S., 2016. *Géologie et minéralisations aurifères de la zone no.8 Orion, secteur de Malartic Hygrade, région de Malartic: Thèse M.Sc., Université du Québec à Montréal, 145p*.

Camus, Y., Dupéré, M., and Rousseau, G., 2012. NI 43-101 Technical Report Mineral Resource Update for the Marban Block Property, Abitibi, Qc, Canada: Report prepared for Aurizon Mines Ltd. and NioGold Mining Corp. by SGS Canada Inc., 157 p. October 22, 2012.

Carrier, A., 2006. Technical Report on the Marban Property: 43-101 Technical Report. Prepared for NioGold Mining Corporation by InnovExplo Inc., 99 p. August 16, 2006.

Chown E. H., Daigneault, R., Mueller, W., and Mortensen, J., 1992. Tectonic Evolution of the Northern Volcanic Zone, Abitibi belt, Quebec: Canadian Journal of Earth Sciences, v. 29, p. 2211-2225.

Daigneault, R., Mueller, W.U. et Chown, E.H., 2002. Oblique Archean Subduction: Accretion and Exhumation of an Oceanic Arc During Textral Transpression, Southern Volcanic Zone, Abitibi Subprovince, Canada: Precambrian Research, vol. 115, p. 261-290.

De Souza, S., Perrouty, S., Dubé, B., Mercier-Langevin, P., Linnen, R. L. and Olivo, G. R., 2020. Metallogeny of the Neoarchean Malartic Gold Camp, Québec, Canada. Society of Economic Geologist Special Publications, no. 23, p. 29-52.

Dimroth, E., Imreh, L., Rocheleau, M., Goulet, N., 1982. Evolution of the South-central Part of the Archean Abitibi Belt, Quebec. Part I: Stratigraphy and Paleogeographic Model. Canadian Journal of Earth Sciences, vol. 19, p. 1729–1758.

Ducharme, Y., 2007. NioGold Mining Corporation Memo on the North-North Zone: Internal Memo for NioGold Mining Corp., 3 p. November 15, 2007.

Ducharme, Y., 2008. NioGold Mining Corporation Memo on the North-North Zone: Internal Memo for NioGold Mining Corp., 2 p. June 2008.

Guay, F., Pilote, P., Daigneault, R., and McNicoll, V., 2018. Early Gold-bearing Quartz Veins within the Rivière-Héva Fault Zone, Abitibi Subprovince, Québec, Canada. Canadian Journal of Earth Sciences, vol. 55, p. 1139–1157.

Jemielita, R.A., Davis, D.W., and Krogh, T.E., 1990. U-Pb Evidence for Abitibi Gold Mineralization Postdating Greenstone Magmatism and Metamorphism. Nature, vol. 346, p. 831–834.

Lefrançois, R., 2005. Due diligence – Norlartic and First Canadian Property Acquisitions. Internal NioGold Mining Corp. memorandum, 8 p. November 17, 2005.

Lehouiller, P., Lampron, S., Gagnon, G., Houle, N. and Bouchard, F., 2020. NI 43-101 Technical Report, Canadian Malartic Mine, Québec, Canada: 275 p., December 31, 2020.

Mannard, G. N., and Bubar, D. S., 1986. Report on the 1985-1986 diamond drilling program, Norlartic property, North-North Zone, Project 2001, Vassan Township, Quebec: Internal report for Aur Resources Inc., 28 p. plus appendices. February,1986.

McNicoll, V., Goutier, J., Dubé, B., Mercier-Langevin, P., Ross, P.-S., Dion, C., Monecke, T., Legault, M., Percival, J. and Gibson, H., 2014. U-Pb Geochronology of the Blake River Group, Abitibi Greenstone Belt, Quebec, and Implications for Base Metal Exploration. Metallogeny, Neoarchean Malartic Gold Camp, Quebec, Canada. Economic Geology, vol. 109, p. 27–59.

Mortensen, J.K., 1993a. U-Pb geochronology of the Eastern Abitibi Subprovince: Part I: Chibougamau–Matagami– Joutel region: Canadian Journal of Earth Sciences, vol.30, p.11-28.

Mortensen, J.K., 1993b, U–Pb geochronology of the Eastern Abitibi Subprovince: Part II: Noranda–Kirkland Lake area: Canadian Journal of Earth Sciences, vol. 30, p. 29–41.

Mueller, W., Daigneault, R., Mortensen, J., and Chown, E. H., 1996. Archean Terrane Docking; Upper Crust Collision Tectonics, Abitibi Greenstone Belt, Quebec, Canada. Tectonophysics, v. 265, p. 127-150.

Pilote, P., Mueller, W., Scott, C., Champagne, C., Lavoie, S., and Moorhead, J., 1998. *Volcanologie de la Formation Val-d'Or et du Groupe de Malartic, sous-province de l'Abitibi: contraintes géochimiques et géochronologiques*. Ministère des Ressources naturelles du Québec, DV 98-05, p. 48–49.

Pilote, P., Scott, C.R., Mueller, W., Lavoie, S., and Riopel, P., 1999. *Géologie des formations de Val d'Or, Héva et Jacola: nouvelles interprétations du Groupe de Malartic*. Ministère de l'Énergie et des Ressources, Québec, Report DV 99–03, p. 19.

Samson, B., 2019. Deformation History and Gold Mineralization of the Cadillac Group North of the Larder Lake-Cadillac Deformation Zone, Southern Abitibi Greenstone Belt, Québec. M.Sc. thesis, Laurentian University, 125 p.

Renou, A. S., 2007. *Étude pétrographique, association minéralogique de l'or et alterations, Project Marban*. Report prepared by Modèlaur for NioGold Mining Corp.

Sauvé, P., Imreh, L. and Trudel, P., 1993. *Description des gîtes d'or de la région de Val d'Or*. Ministère de l'Énergie et Ressources naturelles, MM 91-03, 178 p.

Stuart, E. and Martin, L., 1986. Report on the Winter 1985-86 Diamond Drilling Program, Kierens Zone, First Canadian Property, Project 1902, Malartic Township, Quebec, 32D/1. Internal Report of Aur Resources Inc., 44 p. plus appendices and pocket plans.

Stuart, E. and Martin, L., 1988, Report on the Winter 1988 Surface Diamond Drilling Program on the Gold Hawk Horizon & North Zone, Malartic Property, Project 2001, Vassan Township, Quebec. Internal report of Aur Resources Inc., 39 p. plus appendices and pocket plans

Trudeau, Y., Raymond, D., 1992. Geology and Structure, Orion Mine (No. 8 zone), Val-d 'Or District, Quebec. Exploration and Mining Geology, vol. 1, p. 223-230.

Zweng, P.L., Mortensen, J.K., and Dalrymple, G.B., 1993. Thermochronology of the Camflo Gold Deposit, Malartic, Québec: Implications for Magmatic Underplating and the Formation of Gold-bearing Quartz Veins. Economic Geology, vol. 88, p. 1700–1721.

27.4 Metallurgical Testing

Arnold, C., 2020. Gravity Circuit Modelling and scale-up report, prepared by FLSmidth, for NioGold Mining Corp. 16 June 2020.

Chiasson, G., and Imeson, D., 2016. The Grinding Circuit Design Based on Ten Bench-Scale Grindability Test Results from the Marban Project, Report 149647-003. Prepared by SGS Mineral Services, Ontario, Canada, for NioGold Mining Corp. 2 February 2016.

Clark, J.R., 2015. Marban Gold Metallurgy for NioGold Mining Corp. 2 October 2015.

Davis, K. 2015. Gravity Test Work Report KRTS20951, prepared by FLSmidth, UTAH, USA for NioGold Mining Corp. 8 October 2015.

DiLauro, P. and Dymov, I., 2012. Marban Block Samples (revised) Report 13333-001, prepared by SGS Mineral Services, Ontario, Canada, for Aurizon Mines. 14 March 2012.

Jackman, R. and Dymov, I., 2016a. The Recovery of Gold from Marban Project Samples Report 14947-002. Prepared by SGS Mineral Services, Ontario, Canada, for NioGold Mining Corp. 25 February 2016.

Jackman, R. and Dymov, I., 2016b. The Recovery of Gold from Norlartic Project Samples report 14947-004, prepared by SGS Mineral Services, Ontario, Canada, for NioGold Mining Corp. 22 June 2016.

27.5 Tailings Management and Geotechnical Slope Stability

Bonin, M., Limoges Shaigetz, M., Pépin, N., Becker. D., and Karray, M, 2019. Geotechnical Characterization of a Glacio-Lacustrine Clay Deposit of the Abitibi Region – Case Study, GEO St. John's 2019 Conference Proceedings, 72nd Canadian Geotechnical Conference, St. John's, Newfoundland, September 29, 2019 – October 2, 2019.

Canadian Dam Association (CDA, 2013). Dam Safety Guidelines.

Canadian Dam Association (CDA, 2014). Technical Bulletin on "Application of Dam Safety Guidelines to Mining Dams".

Environment and Climate Change Canada (ECCC, 2009). Environmental Code of Practice for Metal Mines.

Geological Survey of Canada Open File 7893, Fifth Generation Seismic Hazard Model for Canada. Grid Values of Mean Hazard to be used with the 2015 National Building Code of Canada.

Golder Associates Ltd., 2016. Preliminary Economic Assessment Marban Pit Rock Slope Assessment, Report Number: 048-1541337-41010-RA-RevB, April 1, 2016.

Ministère du Développement Durable, de l'Environnement et des Parcs (MDDEP, 2012). *Directive 019 sur l'industrie minière*.

National Building Code of Canada, 2015. NRCC No. 56190; Appendix C: Table C-3, Seismic Design Data for Selected Locations in Canada.

Québec Ministère des Ressources naturelles de la Faune (MRNF), *Cartes écoforestières (032C04NO-SO et 032D01NE-SE), 4ième révision.*

Québec Ministère des Ressources naturelles de la Faune (MRNF), 2011. *Habitats fauniques*.

WSP, 2018. *Projet Bloc Marban. Étude géochimique sur les stériles miniers*. Note technique prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 19 p. + tables + maps + appendix.

WSP, 2020. *Préliminaire Carte 1 Desktop baseline secteur N-W, Minière O3 Projet Marban Étude économique préliminaire, Approbation : 151-11330-89_DT_Desktop_baseline.mxd, Août 2020*.

Viellette, J.L., 2003. *Géologie des formations en surface et histoire glaciaire, Lac Blouin, Québec, Paradis, S.J., 2003 (carte 10) et Cadillac, Québec, (carte 9*).

27.6 Environment

Asini. 2021. Étude du potentiel archéologique: Projet minier Marban, Minière O3. Rouyn-Noranda. 35 p.

Beaulieu, M. 2020. *Guide d'intervention – Politique de protection et de réhabilitation des terrains contaminés. Direction des lieux contaminés. Direction générale des politiques en milieu terrestre*. ISBN 978-2-550-83515-8 (PDF). 326 p.

Ducharme, Y., Gagnon-Lamothe, B., and Pelletier, M., 2009. Activity report for 2007 and 2008 - Marban Block and Malartic Hydgrade properties. Internal report for NioGold Mining Corp. 54 p., March 13, 2009.

Golder Associates, Inc., 2013. *Caractérisation environnementale de la propriété Marban, Val-d'Or, Québec*. Report prepared for Mines Aurizon Ltée. Ref. no: 002-12-1221-0031-RF-Rev0, 26 p. + maps + appendix.

Institut de la statistique du Québec (ISQ). 2022. Population et structure par âge et sexe - Municipalités. Estimations de la population des municipalités du Québec, Québec, 1er juillet 2001 à 2021. Excel spread sheet available at: Population et structure par âge et sexe – Municipalités (quebec.ca).Ministère du Développement durable, de l'environnement et des parcs (MDDEP). 2012. *Directive 019 sur l'industrie minière. ISBN: 978-2-550-64507-8 (PDF). 66 p. + appendix*.

Ministère de l'Énergie et des Ressources naturelles (MERN). 2017. *Guide de préparation du plan de réaménagement et de restauration des sites miniers au Québec. Direction de la restauration des sites miniers, ISBN: 978-2-550-77162-3 (PDF). Novembre 2017, 82 p.*

Ministère de l'Environnement et de la Lutte contre les changements climatiques (MELCC), 2020. *Guide de caractérisation des résidus miniers et du minerai. Québec, ISBN: 978-2-550-86820-0 (PDF). 52 p.*

Municipalité Régionale de Comté de la Vallée-De-L'or, 2005. *Schéma d'aménagement et de développement MRC de La Vallée-de-l'Or. Revised in 2019. 339 p*.

Statistics Canada. 2017. Census Profile, 2016 Census. Updated on June 18, 2019. Val-d'Or, Ville [Census subdivision], Quebec and La Vallée-de-l'Or, Municipalité régionale de comté [Census division], Quebec. Available at: Census Profile, 2016 Census - Val-d'Or, Ville [Census subdivision], Quebec and La Vallée-de-l'Or, Municipalité régionale de comté [Census division], Quebec (statcan.gc.ca)

Statistics Canada. 2017. Census Profile, 2016 Census. Updated on June 18, 2019. Malartic, Ville [Census subdivision], Quebec and La Vallée-de-l'Or, Municipalité régionale de comté [Census division], Quebec. Available at: Census Profile, 2016 Census - Malartic, Ville [Census subdivision], Quebec and La Vallée-de-l'Or, Municipalité régionale de comté [Census division], Quebec (statcan.gc.ca)

Robitaille, A., et. Saucier, J.P., 1998. *Paysages régionaux du Québec méridional. Les publications du Québec. 204 p*.

WSP, 2020a. *Projet Bloc Marban. Caractérisation géochimique du minerai et des stériles. Report prepared for O3 Mining. Ref. WSP: 151-11330-89*. Preliminary Version

————, 2018b. *Projet Bloc Marban. Étude sectorielle – Faune aviaire et micromammifères. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 50 p. + appendix*.

————, 2018c. *Projet Bloc Marban. Étude sectorielle – Inventaire des chiroptères. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 18 p. + maps + appendix*.

————, 2018d. *Projet Bloc Marban. Évaluation de la teneur de fond naturelle en métaux dans les sols. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 19 p. + tables + maps + appendix*.

————, 2018e. *Projet Bloc Marban. Étude sectorielle – Inventaire de l'herpétofaune. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 15 p. + maps.*

————, 2018f. *Projet Bloc Marban. Étude sectorielle – Analyse hydrologique. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 23 p. + appendix.*

————, 2018g. *Projet Bloc Marban. Étude sectorielle – Habitats aquatiques et communauté de poisson. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 25 p. + appendix.*

————, 2018h. *Projet Bloc Marban. Étude sectorielle – État initial de la qualité de l'air. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 23 p. + appendix.*

————, 2018i. *Projet Bloc Marban. Étude sectorielle – Mesure du climat sonore et vibratoire existant avant le projet. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 29 p. + appendix.*

————, 2018j. *Projet Bloc Marban. Étude sectorielle – Végétation et milieux humides. Report prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 15 p. + maps + appendix.*

————, 2018k. *Projet Bloc Marban. Étude géochimique sur les stériles miniers. Note technique prepared for Osisko Mining Inc. Ref. WSP: 151-11330-12. 19 p. + tables + maps + appendix.*

27.7 Mining

Canadian Securities Administrators (CSA), 2011. National Instrument 43-101, Standards of Disclosure for Mineral Projects, Canadian Securities Administrators.

Caterpillar Inc., 2016. Caterpillar Performance Handbook: 46th edition, January 2016.

Golder Associates, Ltd., 2016. Pit Rock Slope Assessment, Preliminary Economic Assessment, April 2016.

Golder Associates, Ltd, 2020. Draft – Rock Slope Designs Recommendations – Preliminary Economic Assessment – Marban Gold Project, Malartic, Québec, Technical Memorandum, July 2020.

Gustin M, Ronning P, 2013. Updated Mineral Resource Technical Report Marban Block Property, Québec, Canada, Mine

Guide de caractérisation des résidus miniers et du minerai (Guide); MELCC, July 2020

27.8 Websites

http://mrcvo.qc.ca/administration/administration/portrait/statistiques/ (Observatoire de l'Abitibi-Témiscamingue, 2021. Statistiques)